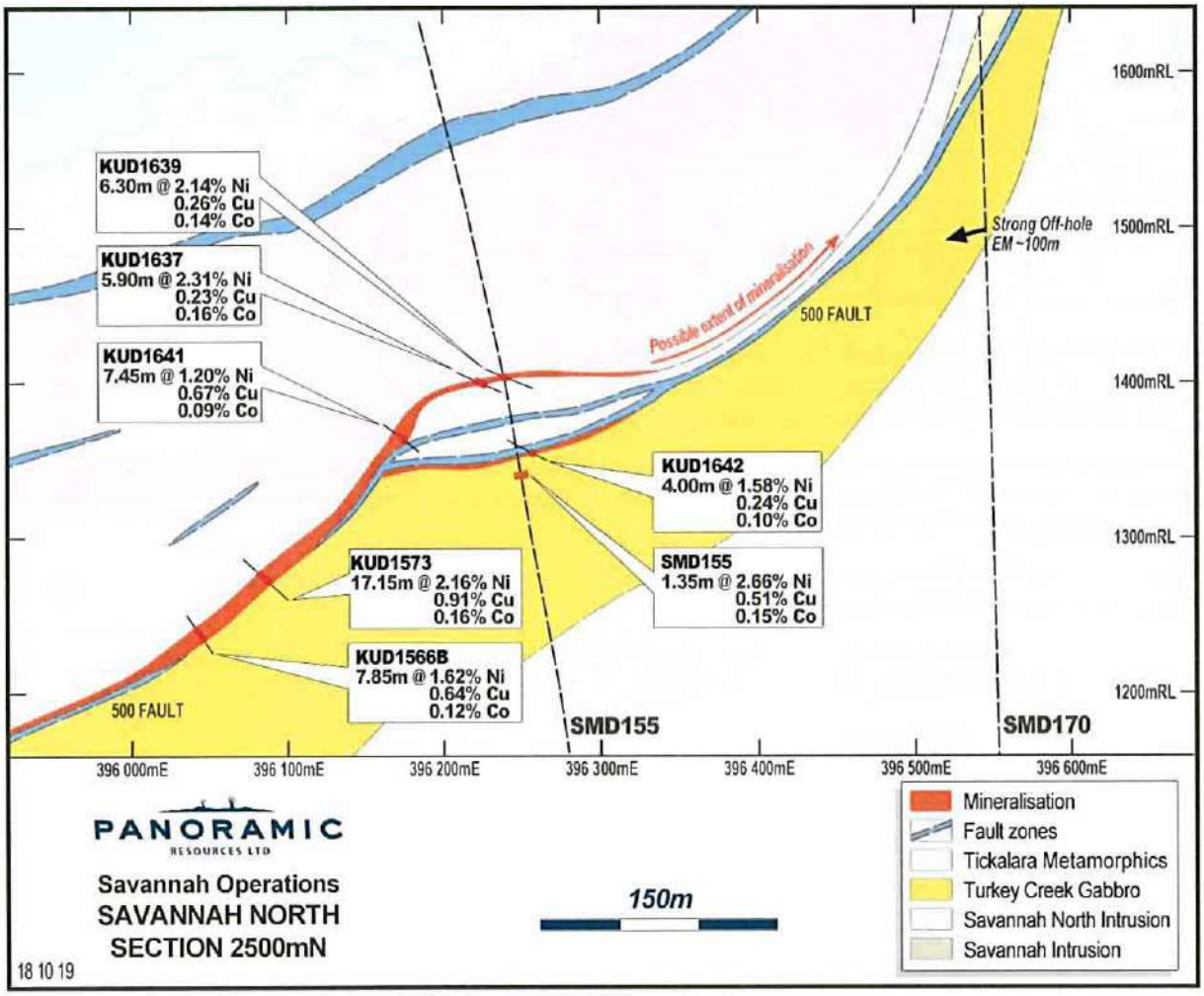
Figure 3 – Savannah North Section 2500mN showing split in mineralisation about 6200mE



Frog Hollow VTM Project

In the December 2018 and March 2019 Quarterly reports, the Company reported on the intersection of broad zones of vanadiferous titanomagnetite (VTM) accumulations in three drill holes at Frog Hollow (*Figure 4*). During the June 2019 quarter, the three Frog Hollow drill holes were sampled in their entirety and three zones of differing grade were selected from drill hole SMD185 to undergo low intensity magnetic separation (LIMS) testing to determine the vanadium grade and recovery that could be achieved to a VTM concentrate using this relatively inexpensive magnetic separation technique. The composited three intervals selected for testing from SMD185 had the following assayed grades:

- **34m @ 0.36% V_2O_5, 5.57% TiO_2, 25.93% Fe_2O_3 from 19m;**
- **45m @ 0.25% V_2O_5, 3.06% TiO_2, 17.73% Fe_2O_3 from 160m; and**
- **53m @ 0.17% V_2O_5, 3.66% TiO_2, 15.40% Fe_2O_3 from 92m.**

The LIMS test-work was completed during the September quarter with the results summarised in Table 5.

The test-work was conducted at 3,000 gauss with the closing screen for the Davis Tube set at 106um.

Table 5: Results of LIMS testing on Frog Hollow Drill Hole Samples

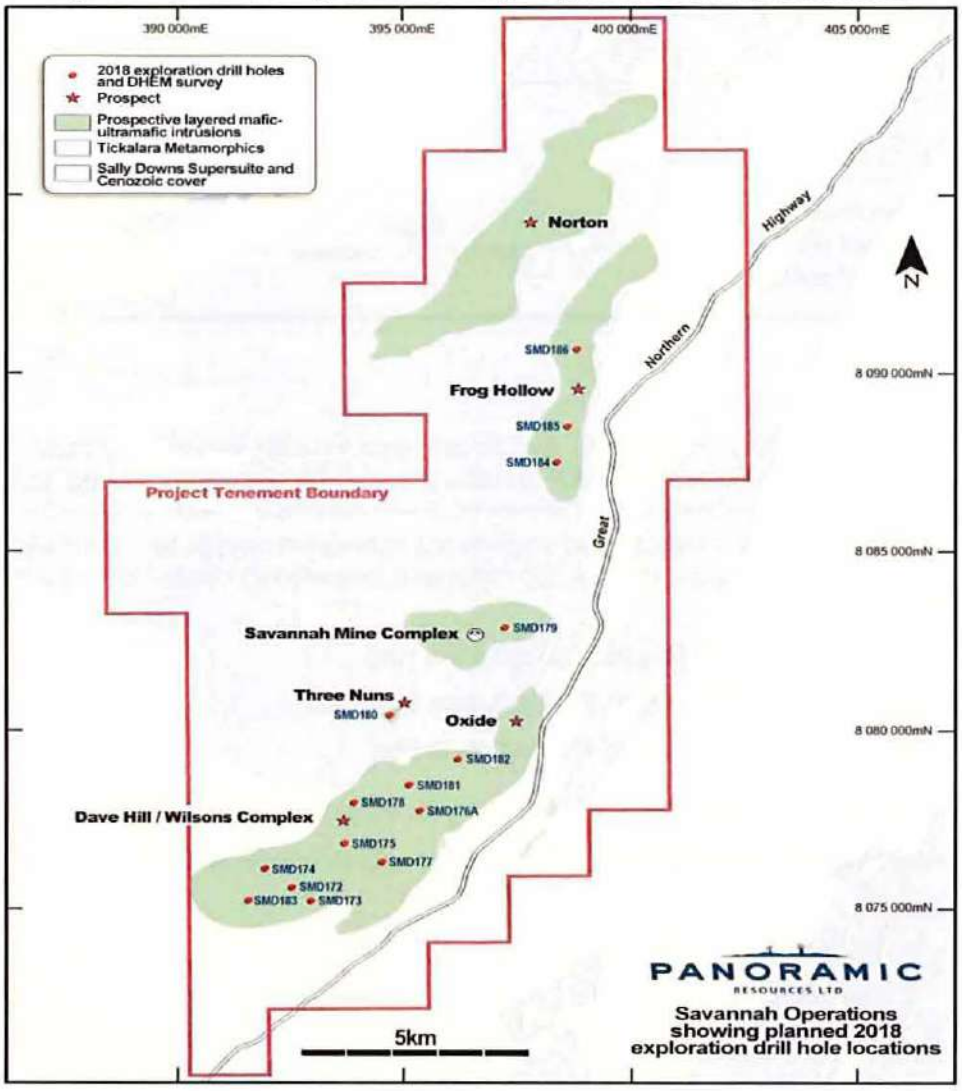
DT Wash @ 3000G										
Mass Dist'n (%)	Fe (%)		SiO_2 (%)		Al_2O_3 (%)		TiO_2 (%)		V_2O_5 (%)	
	Grade	Dist'n	Grade	Dist'n	Grade	Dist'n	Grade	Dist'n	Grade	Dist'n
16.39	60.6	52.09	1.60	0.73	3.00	3.02	6.11	17.72	1.901	85.73
7.45	59.0	34.33	3.33	0.58	2.00	0.84	5.54	13.18	2.199	65.57
5.06	57.3	25.39	3.42	0.39	1.20	0.34	11.25	15.94	1.883	55.63

- The high-grade (0.36% V_2O_5) sample concentrate recovered 16.39% mass with 85.7% of V_2O_5 at a grade of 1.90% V_2O_5.
- The intermediate-grade (0.25% V_2O_5) sample concentrate recovered 7.45% mass with 65.6% of V_2O_5 at a grade of 2.20% V_2O_5.
- The low-grade (0.17% V_2O_5) sample concentrate recovered 5.06% mass with 55.6% of V_2O_5 at a grade of 1.90% V_2O_5.

The reported test results, which are based on single-tests only and are therefore un-optimised, are very positive and demonstrate that high-grade V_2O_5 concentrates at good recoveries can be produced from Frog Hollow samples using inexpensive magnetic separation techniques.

Further work is planned in the December 2019 quarter, with the aim to assess several different process options that could be used to recover the vanadium from the V_2O_5 concentrates.

Figure 4 – Savannah Project Plan showing Frog Hollow VTM Project drill holes



Keller Creek Graphite Project

The Keller Creek Graphite Project is located immediately to the west of Savannah on E80/4834 (*Figure 5*). Regional airborne electromagnetic surveys conducted in the past by the Company in search of nickel sulphide mineralisation, identified several large stratigraphic horizons of graphite bearing meta-sediments (Tickalara Metamorphics) across the Keller Creek tenement.

In June 2019, the Company conducted a preliminary reverse circulation (RC) drill test of the main graphite bearing horizon over a strike length of approximately five kilometres. The program consisted of 14 RC drill holes for a total of 1,368 drill metres, with a total of 1,074 one-metre graphite bearing samples collected and submitted for assay. The aim of the program is to provide an indication of the thickness and Total Graphitic Carbon (TGC) content of the graphite bearing horizon. In addition to the assay samples, representative RC chips were collected from each drill hole and submitted for mineralogical examination to determine the purity and flake size of the graphite.

All results for the Keller Creek program were received during the September 2019 quarter. Using a 3% TGC cut-off grade, the program returned the intercepts shown on Figure 5. The JORC (2012 Edition), Table 1 drill hole details and associated compliance tables are included in Appendix 1. The better intercepts include:

- **10m @ 4.67% from 55m, 6m @ 5.58% from 68m, and 10m @ 4.05% from 82m in SMP180;**
- **4m @ 7.35% from 24m, 8m @ 3.58% from 60m in SMP181;**
- **5m @ 5.76% from 92m in SMP182;**
- **4m @ 6.82% from 75m in SMP183;**
- **11m @ 3.73% from 111m in SMP187; and**
- **8m @ 3.71% from 39m, 8m @ 3.40% from 71m in SMP191.**

In addition to the assay results, petrological descriptions for the samples submitted to investigate graphite flake size and quality, were received during the quarter. The samples described had variable graphite contents (or tenor). While most samples had strong flake graphite concentrations (ie up to 20 vol% flake graphite), a few samples showed no visible flake graphite.

Graphite flake sizes were also variable with large to jumbo sized flake occurring in most samples, correlating with the enhanced upper amphibolite to granulite facies metamorphic grade of the area. In contrast, lithologies that had been subject to strong brittle/ductile deformation tended to exhibit a finer flake graphite size due to comminution.

The grade and flake quality of the Keller Creek graphite appears to be very similar to Hexagon Resources Limited's (ASX: HXG) McIntosh Project, located 40km to the SE of Savannah. The McIntosh Project has a reported Mineral Resource (based on a 3% TGC cut-off grade) of 23.8 million tonnes grading 4.5 % TGC, contained within four separate deposits.

Based on the Company's initial drill test results and the broad extents of the graphitic horizons within the Keller Creek tenement demonstrated by previous electromagnetic surveys, there is a high probability that the Keller Creek project tenement contains large quantities of graphite of a similar grade and quality to the McIntosh Project.

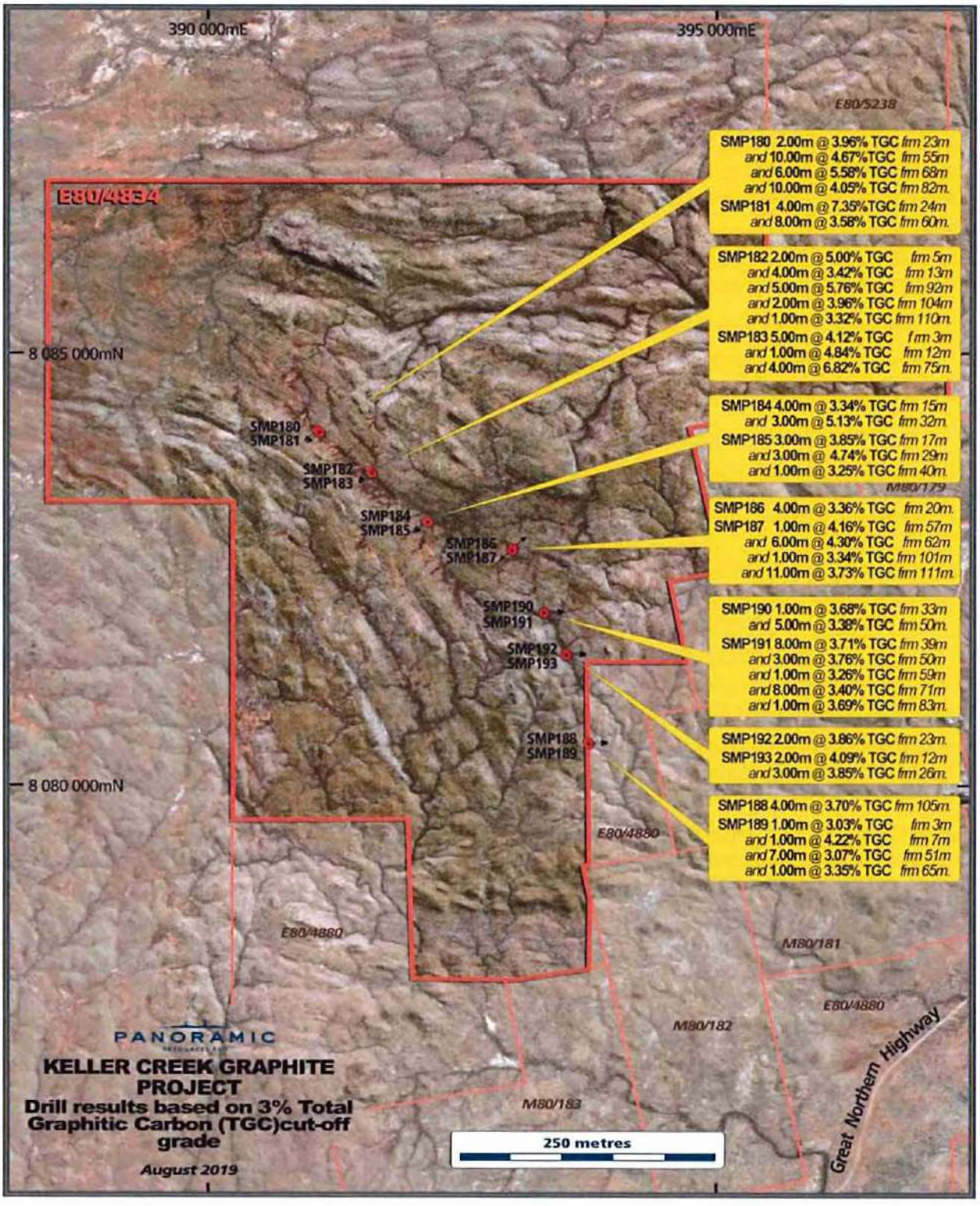
Group Mineral Resource and Reserve Tables (30 June 2019)

On 30 September 2019, the Company released the Group's Mineral Resource and Reserve Tables as at 30 June 2019 (*refer to the Company's ASX announcement of 30 September 2019 for further details*).

Nickel in Mineral Resources decreased by 96,600 tonnes (-31%) between 30 June 2018 and 30 June 2019 due to the removal of the Lanfranchi Nickel Project Resource of 95,500 tonnes following the sale of the project in December 2018 and from depletion for FY2019 Savannah production. The Savannah Nickel Project contains 217,000 tonnes of contained nickel, maintaining the project as one of the largest medium-grade nickel sulphide deposits in Australia.

Nickel in Ore Reserves decreased by 6,400 tonnes (-5%) between 30 June 2018 and 30 June 2019, due to the removal of the Lanfranchi Nickel Project Ore Reserve of 4,200 tonnes following the sale of the project in December 2018 and from depletion for FY2019 Savannah production. The Ore Reserves at the Savannah Project of 110,400 tonnes of contained nickel is able to sustain a mine life of approximately eight years.

Figure 5 – Keller Creek Graphite Project plan showing recent drill hole locations and results



PGM – Thunder Bay North Project

The Thunder Bay North (TBN) Project is located near Thunder Bay in northwest Ontario, Canada. The TBN Project Resource contains 10.4Mt at 1.13g/t Pt and 1.07g/t Pd for ~0.4Moz Pt and ~0.4Moz Pd (*refer to the Company's ASX announcement of 30 September 2019*).

In 2015, Rio Tinto Exploration Canada Inc. ("RTEC") commenced a farm-in whereby RTEC was able to earn a 70% interest in the TBN Project by sole funding C$20 million in expenditure over five years, with a minimum spend of C$5 million. In January 2017, RTEC confirmed that it had achieved the minimum spend of C$5 million on the Project.

With the palladium price at an all time high, in June 2019, the Company was approached by Benton Resources Inc. ("Benton") to purchase the Project. On 2 July 2019, the Company signed a binding Letter Agreement, accepting a purchase price of C$9 million for TBN (*refer to the Company's ASX announcement of 2 July 2019*) subject to the signing of a Definitive Agreement and Benton raising sufficient funds to purchase the Project on a deferred payment basis (*refer to the Company's ASX announcements of 3 September 2019 and 3 October* 2019).

On 9 October 2019, RTEC notified the Company that it had entered into an option agreement with Benton in which RTEC has granted to Benton an option to purchase some, or all, of RTEC's 100% owned Escape Lake Property, which is located in close proximity to the TBN Project. As a result of the RTEC transaction with Benton, the TBN Project farm-in agreement between the Company and RTEC has terminated.

The Company and Benton are currently working towards signing the Definitive Agreement in November 2019 with settlement and the receipt of the first tranche payment of C$4.5 million expected before the end of 2019.

PGM – Panton Project

Panton is located 60km south of the Savannah Nickel Project in the East Kimberley region of Western Australia. Panton is a significant PGM Resource containing **~1.0Moz Pt at 2.2g/t and ~1.1Moz Pd at 2.4g/t** (*refer to the Company's ASX Announcement of 30 September 2015*) with exploration potential at depth and along strike.

Panoramic considers the Panton Project to be a quality development asset which fits within the Company's commodity diversification and growth strategy and is a key part of its Kimberley Hub concept.

In addition to continuing to sponsor research by Curtin University on alternative PGM leaching methods applicable to Panton mineralisation, the Company is studying the viability of producing a high-grade PGM concentrate together with a chromite by-product stream.

In the December 2018 quarter, the Company commenced test-work in conjunction with Curtin University to evaluate the feasibility of producing a value-added direct Pt, Pd and Au refinery feed products from Panton while maintaining the ability to also produce an economic chromite by-product revenue stream.

The aim of the new test-work program was to determine if this innovative extraction and recovery method performs equally well on a high-grade PGE (flotation) concentrate as it did previously on run of mine ore samples. If the result can be replicated on a high-grade PGE concentrate, it will have significant advantages for the Project by reducing the capital and operating cost of the PGE extraction and recovery process (treating concentrate instead of run-of-mine feed), enabling the direct production of more valuable Pt, Pd and Au refinery feed products, while maintaining the ability to produce a chromite by-product from the flotation tails. This testwork phase is now complete and was successful on many fronts.

In May 2019, the Company contracted Mr Len Jubber, a consulting mining engineer, to undertake a detailed review of the Project, bringing together all aspects of the Project (geology, resources, mining and processing) with the aim to produce a financial model based on the latest flow sheet designs and their respective operating and capital costs. The review has also identified where further testwork may be beneficial to the Project's economics. Mr Jubber's final report is due in November 2019.

Gold – 51% investment in Horizon Gold/Gum Creek Project

The Company has an indirect interest in the Gum Creek Project through its 51% majority equity interest in Horizon Gold Limited (ASX Code: HRN). The market value of this investment in Horizon at 30 June 2019 was approximately $6.8 million.

Exploration and evaluation activities are ongoing at Gum Creek (*refer to Horizon's September 2019 quarterly report for further details*). Horizon has reported an increase in Mineral Resources as at 30 June 2019 and has defined the foot print of zinc-copper-silver mineralisation at the Altair Prospect. Horizon has commenced a Scoping Study based on renewed open pit mining on the Swan and Swift free-milling gold Resources (*refer to Horizon's ASX announcement of 15 October 2019*). Under the October 2016 Management Agreement ("Agreement"), Panoramic staff are providing management services to Horizon on a cost recovery basis.

Corporate

Cash

Group Cash (available and restricted) as at 30 September 2019 totalled $20.2 million.

The movement in the cash position during the quarter included the following items:

Income

- $19.5 million net revenue from nickel concentrate sales;
- $4.1 million positive final quotational period (QP) pricing adjustments;
- $27.2 million net proceeds from the September pro-rata Rights Issue (*refer below*);

Expenditure

- $33.2 million on Savannah site costs (capital/operating); and
- $2.9 million on corporate and finance lease costs (net of other income).

Aggregate movements in the Group Cash balance over the quarter are shown in Figure 6.

At 30 September 2019, $180K was cash-backed against the Company's performance bond facility.

Capital Raising

On 27 September 2019, the Company received net proceeds of $27.2 million from the pro-rata renounceable two for eleven Rights Issue at 28c per new Share to eligible existing shareholders that was announced on 5 September 2019.

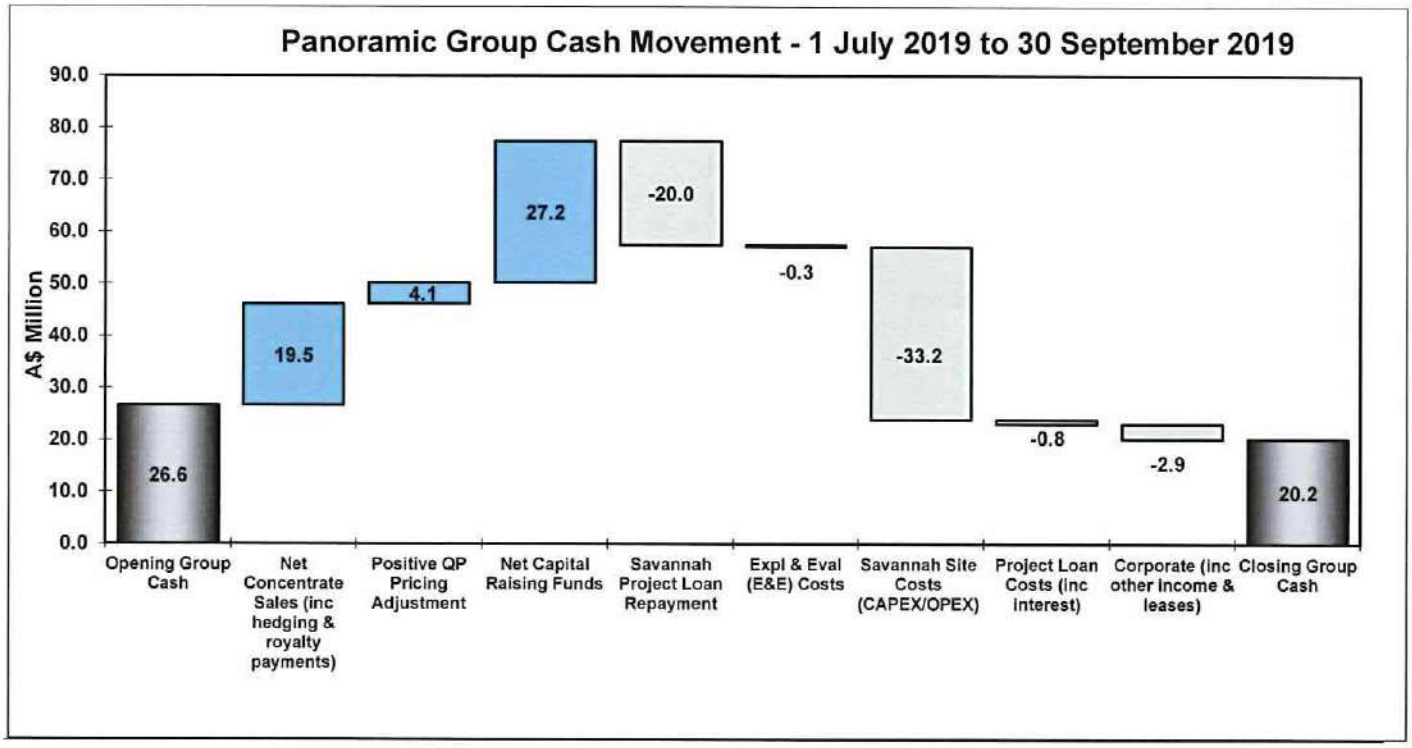
Board and Management Changes

The Company announced several changes to the Board and Management, namely:

- Peter Harold will be leaving after over 18 years as Managing Director of the Company;
- Chairman, Brian Phillips, will step down at the conclusion of the 2019 AGM on 19 November 2019;
- Gillian Swaby was elected as a non-executive director, effective 8 October 2019 and will seek re-election at the 2019 AGM;
- Victor Rajasooriar was appointed as the new Managing Director and CEO, effective from 11 November 2019; and
- CFO and Company Secretary, Trevor Eton, will retire on 28 January 2020 after 16 years in the role.

The Company would like to take the opportunity to welcome Gillian and Victor to their new roles in the Company and to thank Brian, Peter and Trevor for their dedicated contribution and service to the Company and wish them well in the future.

Figure 6 – Panoramic Group Cash Movement (September 2019 Quarter)



Competent Person

The information in this release that relates to exploration results, Mineral Resources and Ore Reserves is based on information compiled by John Hicks. Mr Hicks is a member of the Australasian Institute of Mining and Metallurgy (AusIMM) and is a full-time employee and shareholder of Panoramic Resources Limited.

The aforementioned has sufficient experience that is relevant to the style of mineralisation and type of target/deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Mr Hicks consents to the inclusion in the release of the matters based on the information in the form and context in which it appears.

About the Company

Panoramic Resources Limited (**ASX code: PAN**) is a Western Australian mining company formed in 2001 for the purpose of developing the Savannah Nickel Project in the East Kimberley. Panoramic successfully commissioned the $65 million Savannah Project in late 2004 and then in 2005 purchased and restarted the Lanfranchi Nickel Project, near Kambalda. In FY2014, the Company produced a record 22,256t contained nickel and produced 19,301t contained nickel in FY2015. The Lanfranchi and Savannah Projects were placed on care and maintenance in November 2015 and May 2016 respectively pending a sustained recovery in the nickel price.

After delivering an updated feasibility study on the Savannah Project in October 2017, securing an offtake customer and putting in place project financing in July 2018, the Company made the decision to restart operations at Savannah with first concentrate shipped from Wyndham on 13 February 2019. The Lanfranchi Project was sold in December 2018 for a total cash consideration of $15.1 million, providing additional financial support for the re-commissioning of the Savannah Project.

Apart from the nickel, copper and cobalt inventory at Savannah the Company has a diversified resource base including platinum group metals (PGM) and gold. The PGM Division consists of the Panton Project, located 60km south of the Savannah Project and the Thunder Bay North Project in Northern Ontario, Canada, which is in the process of being sold to Benton Resources for C$9 million. Following the ASX listing of Horizon Gold Limited (ASX Code: HRN) in December 2016, the Company's interest in gold is an indirect investment in the Gum Creek Gold Project located near Wiluna through its 51% shareholding in Horizon.

At 30 September 2019, Panoramic had $20.2 million in available and restricted cash ($180k). The Company has a $20 million project financing facility with Macquarie Bank which was restructured in September 2019 (previously a $40 million facility).

For further information contact:
Peter Harold, Managing Director
+61 8 6266 8600

Appendix 1

Table 1: Keller Creek Graphite Project – Drill Hole Intercepts

Hole	East	North	RL	Dip	Azi	EOH	From	To	TGC Intercept
SMP180	391074.5	8084109.8	462.8	-65.0	221.6	106	23	25	2m @ 3.96 %
							55	65	**10m @ 4.67 %**
							68	74	**6m @ 5.58 %**
							82	92	**10m @ 4.05 %**
SMP181	391079.8	8084109.2	463.0	-50.0	221.6	80	24	28	**4m @ 7.35 %**
							60	68	**8m @ 3.58 %**
SMP182	391597.6	8083636.5	460.1	-80.0	221.6	118	5	7	2m @ 5.00 %
							13	17	4m @ 3.42 %
							92	97	**5m @ 5.76 %**
							104	106	2m @ 3.96 %
							110	111	1m @ 3.32 %
SMP183	391596.7	8083635.5	460.1	-60.0	221.6	100	3	8	**5m @ 4.12 %**
							12	13	1m @ 4.84 %
							75	79	**4m @ 6.82 %**
SMP184	392157.4	8083071.4	453.8	-70.0	221.6	52	15	19	4m @ 3.34 %
							32	35	3m @ 5.13 %
SMP185	392156.6	8083070.6	453.7	-55.0	221.6	45	17	20	3m @ 3.85 %
							29	32	3m @ 4.74 %
							40	41	1m @ 3.25 %
SMP186	393000.0	8082740.0	450.0	-60.0	221.6	82	20	24	4m @ 3.36 %
SMP187	393000.9	8082738.6	458.8	-60.0	41.6	130	57	58	1m @ 4.16 %
							62	68	**6m @ 4.30 %**
							101	102	1m @ 3.34 %
							111	122	**11m @ 3.73 %**
SMP188	393756.9	8080496.6	444.5	-60.0	86.6	136	105	109	4m @ 3.70 %
SMP189	393755.8	8080496.7	444.7	-75.0	86.6	120	3	4	1m @ 3.03 %
							7	8	1m @ 4.22 %
							51	58	**7m @ 3.07 %**
							65	66	1m @ 3.35 %
SMP190	393313.6	8082015.8	459.4	-60.0	86.6	180	33	34	1m @ 3.68 %
							50	55	5m @ 3.38 %
SMP191	393311.9	8082015.8	459.5	-80.0	86.6	100	39	47	**8m @ 3.71 %**
							50	53	3m @ 3.76 %
							59	60	1m @ 3.26 %
							71	79	**8m @ 3.40 %**
							83	84	1m @ 3.69 %
SMP192	393524.2	8081533.7	442.5	-55.0	86.6	52	23	25	2m @ 3.86 %
SMP193	393523.1	8081533.8	442.4	-70.0	86.6	58	12	14	2m @ 4.09 %
							26	29	3m @ 3.85 %

Keller Creek Graphite Project – Table 1, Section 1 - Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• Industry standard reverse circulation (RC) drilling techniques were used to perform this initial drill test of the Keller Creek Graphite Project. • 1m RC sample were collected and split using on-board splitters to produce a ~3kg assay sample.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• All drilling pertaining to this initial Keller Creek drill test was by RC techniques using face sampling hammers.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• RC sample recovery was monitored visually to ensure sample recovery (volume) was consistent throughout the drilling process. • Sample were laid out sequentially on the ground in regularised rows and verified against final drill depth. • There is no apparent relationship between sample recovery and grade
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• All holes and RC chips were geologically logged in full. • Logging protocols dictate lithology, colour, mineralisation and other features are routinely recorded.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in-situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• For each RC drill metre drilled a standard 3kg assay sample was collected using on-board splitters mounted below the cyclone. • Sample sizes are considered appropriate to represent the Savannah style of mineralisation. • No Company QAQC CRM standard or blanks were submitted with the assay samples • Standard Laboratory QAQC checks and procedures were employed

Criteria	JORC Code explanation	Commentary
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. • For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• All samples analyses were performed by ALS Laboratories Pty Ltd at their Perth and Brisbane premises. Sample preparation and assaying of all drill samples involved crushing and pulverising the 3kg assay sample to 80% passing 75µm followed by: • Total Carbon by LECO furnace (LabMethod C_IR07) • Graphitic Carbon (LabMethod) C_IR18) • Total Sulfur by LECO furnace (LabMethod S_IR08) • No other analytical tools or techniques were employed. • Internal QAQC checks and procedures were conducted by ALS.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel. • The use of twinned holes. • Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. • Discuss any adjustment to assay data.	• All holes were logged into EXCEL spreadsheet templates with in-built validation protocols prior to up-loading the data into the Company's SQL Server geology database Further validation checks are performed at the up-loading stage. • No adjustments are made to assay data.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. • Specification of the grid system used. • Quality and adequacy of topographic control.	• All drill hole collars were picked-up using Leica TS15, R1000 instrument by a registered surveyor. • Grid coordinates MGA GDA94 Zone 52. RL is AHD. • No down-hole surveys were performed as part of this drill program.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. • Whether sample compositing has been applied.	• A standard drill spacing was not adopted because the program was designed as an initial test of a particular stratigraphic horizon over several kilometres. • No sample compositing is undertaken.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• Where possible all drill holes were drilled perpendicular to the stratigraphic horizon being targeted. • No orientation sampling bias is apparent.
Sample security	• The measures taken to ensure sample security.	• Drill samples were collected and transported by Company personnel to the Savannah Nickel Mine prior to being road freighted to the ALS Perth laboratory for processing.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits or reviews of the procedures outlined herein have been performed. The procedures are considered to be of the highest industry standard and sufficient for the level of the program being undertaken.

Keller Creek Graphite Project - Table 1, Section 2 - Reporting of Exploration Results

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings. • The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	• The Keller Creek Graphite Project is located on E80/4834 in the East Kimberley region of Western Australia. • Tenure of the EL is held by Pindan Exploration Company Pty Ltd a subsidiary of Panoramic Resources Limited.
Exploration done by other parties	• Acknowledgment and appraisal of exploration by other parties.	• No previous graphite exploration has been conducted on E80/4834.
Geology	• Deposit type, geological setting and style of mineralisation.	• Extensive graphitic horizons have been recognised at Keller Creek since the early 2000s when airborne electromagnetic (EM) surveys were conducted as part of nickel exploration programs conducted in the area. The graphitic horizons form part of the Palaeoproterozoic Tickalara Metamorphics; a complex unit of amphibolite and granulite facies metasedimentary and meta-igneous rock types.
Drill hole Information	• A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes: o easting and northing of the drill hole collar o elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar o dip and azimuth of the hole o down hole length and interception depth o hole length. • If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	• All Keller Creek drilling and results is reported on the MGA GDA94 Zone 52 grid system. RLs are AHD. • All material drill hole information is tabulated in Table 1, accompanying this compliance table.
Data aggregation methods	• In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated. • Where aggregate intercepts incorporate short lengths of high-grade results and longer lengths of low-grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. • The assumptions used for any reporting of metal equivalent values should be clearly stated.	• All analytical drill intercepts in Table 1 are based on a 3% lower Total Graphitic Carbon content cut-off grade and a maximum consecutive internal waste of 2 metres. The minimum sample length was 1 metre
Relationship between mineralisation widths and intercept lengths	• These relationships are particularly important in the reporting of Exploration Results. • If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. • If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').	• All reported intersection lengths are down-hole lengths. • All holes were typically drilled perpendicular to the graphitic horizon being tested. • There is insufficient data at this point to determine True Widths.
Diagrams	• Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	• A simplified plan view showing drill hole positions and reported intercepts is deemed sufficient at this time.
Balanced reporting	• Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.	• Based on the fact that all drill results are reported herein, the report is considered to be sufficiently balanced.
Other substantive exploration data	• Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density,	• No other exploration data is considered material to this release at this stage.

Criteria	JORC Code explanation	Commentary
	groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	
Further work	• The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling). • Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	• The Keller Creek drill results reported herein are the result of an initial assessment of the graphite potential of E80/4834. Further results will be reported if and when they become available.

CLARIFICATION AND RETRACTION ANNOUNCEMENT

Green Critical Minerals Ltd ("GCM" or "the Company") refers to its announcement of 01 July 2024 "Major Mineral Resource Estimate Upgrade for the McIntosh Graphite Project" (the **Announcement**). In discussions with the Company subsequent to the release of the Announcement, the Australian Securities Exchange (**ASX**) have expressed concern that:

- Certain statements made in the Announcement refer to the 20 year mine life and are made not in accordance with the ASX Listing Rule 5.16 and have been retracted; and
- Additional information has been included on page 8 regarding the update mineral resource estimate to comply with ASX Listing Rule 5.8.

To avoid any potential confusion the Company attaches a revised Announcement.

Authorisation

This Announcement is authorised for ASX release by the Board of Green Critical Minerals Limited.

MAJOR MINERAL RESOURCE ESTIMATE UPGRADE FOR THE McINTOSH GRAPHITE PROJECT

- Mineral Resource Estimate at the McIntosh Graphite Project increased to **30.2Mt** grading 4.40% TGC, a **26% upgrade**.
- Project's scale and premium product potential enhanced, supporting GCM's strategy.
- Previously excluded fine flake material now included based on metallurgical test work results and updated marketing information.
- Product suite for fine flake size upstream and downstream qualifications identified, with initial target markets of lubricants, friction components, agriculture, and coatings.
- Expansion of the McIntosh graphite product suite to include spherical purified graphite (SPG) anode qualifications exist, with additional test work to be performed.

Green Critical Minerals Ltd ("GCM" or "the Company") which holds earn-in rights for up to 80% of the McIntosh Graphite Project (see announcement on 15 June 2022) is pleased to announce a major increase to 30.2 million tonnes (Mt) in the Mineral Resource estimate for its flagship McIntosh Graphite Project (**"Project"**) in Western Australia. This represents a **substantial 26% upgrade from the previous estimate** of 23.8Mt, grading 4.45% Total Graphitic Carbon (TGC).

METALLURGICAL BREAKTHROUGHS UNLOCK RESOURCE UPGRADE

The resource upgrade follows an extensive review of existing data, including detailed analysis of historical metallurgical test work programs and the recently concluded metallurgical test work program (see announcements dated 17 June 2024 and 19 June 2024) conducted on graphitic ore from the McIntosh area. Significantly, these studies have demonstrated that the fine flake size component (<150µm) of the graphite concentrate can be upgraded to >95% TGC purity through conventional flotation processes.

Furthermore, analysis of the particle size distribution assay data has revealed that the <150µm component of the concentrate achieves the overall target product concentrate grade. These encouraging outcomes have identified new market opportunities for premium graphite product streams from the McIntosh project.

Commenting on the announcement, **GCM's Managing Director Clinton Booth said:** "*This major resource upgrade, in conjunction with our recently announced metallurgical results, is a transformational milestone for the McIntosh Graphite Project. By unlocking the value of the fine flake component, we have not only increased the total resource inventory but also opened up new potential revenue streams for premium graphite products..*"

INCLUSION OF MAHI MAHI RESOURCE CONTRIBUTES TO UPGRADE

Buoyed by the positive metallurgical results and marketing initiatives, the Company has incorporated the previously modelled but unreported Mahi Mahi resource into the latest estimate. This decision was driven by the high degree of geological and mineralogical consistency observed across the broader McIntosh project area, which extends over 30km of strike length.

Current and historic metallurgical test work programs have been conducted on samples from the Emperor and Wahoo deposits, delivering outstanding concentrate grades. Diamond drill samples from the Longtom, Barracuda, and Mahi Mahi deposits exhibit similar geological and mineralisation characteristics, albeit with varying flake size distributions.

Crucially, analysis of the flake size distribution of the concentrates produced has shown that the fine flake size component generates TGC grades that closely match the total concentrate grade. This observation underpins the decision to include the Mahi Mahi resource in the latest estimate.

Deposit	Resource Classification	Tonnes (Mt)	%TGC	Contained Graphite (Mt)
Emperor	Indicated	12.1	4.28	0.52
	Inferred	3.8	4.35	0.17
	Total	15.9	4.3	0.68
Wahoo	Indicated	1.3	3.97	0.05
	Inferred	0.0	0.00	0.00
	Total	1.3	3.97	0.05
Longtom	Indicated	5.1	4.93	0.25
	Inferred	0.8	5.25	0.04
	Total	5.9	4.97	0.29
Barracuda	Indicated	0.7	4.4	0.03
	Inferred	0.0	0	0.00
	Total	0.7	4.4	0.03
Mahi Mahi	Indicated	0.0	0	0.00
	Inferred	6.3	4.2	0.27
	Total	6.3	4.2	0.27
All	Indicated	19.2	4.44	0.85
	Inferred	10.9	4.33	0.47
	Total	**30.2**	**4.40**	**1.33**

Table 1: Updated Mineral Resource Estimate for the McIntosh Graphite Project

This mineral resource estimate update reflects the company's recent review of historical datasets and findings that the dominate graphite flake size over the area is classified as fine. The Mahi Mahi resource had been modelled previously, but not included in the resource statement due to its fine flake size endowment. GCM is now satisfied that the graphitic ore from the fine flake size Mahi Mahi resource has

the potential to produce a concentrate that can be sold into various end markets. Further details about the Mineral Resource of the McIntosh project deposits are available on the ASX announcements platform (*www2.asx.com.au, Code: HXG, Date:5 April 2018, Title : Revised McIntosh Mineral Resource ~ Amended*).

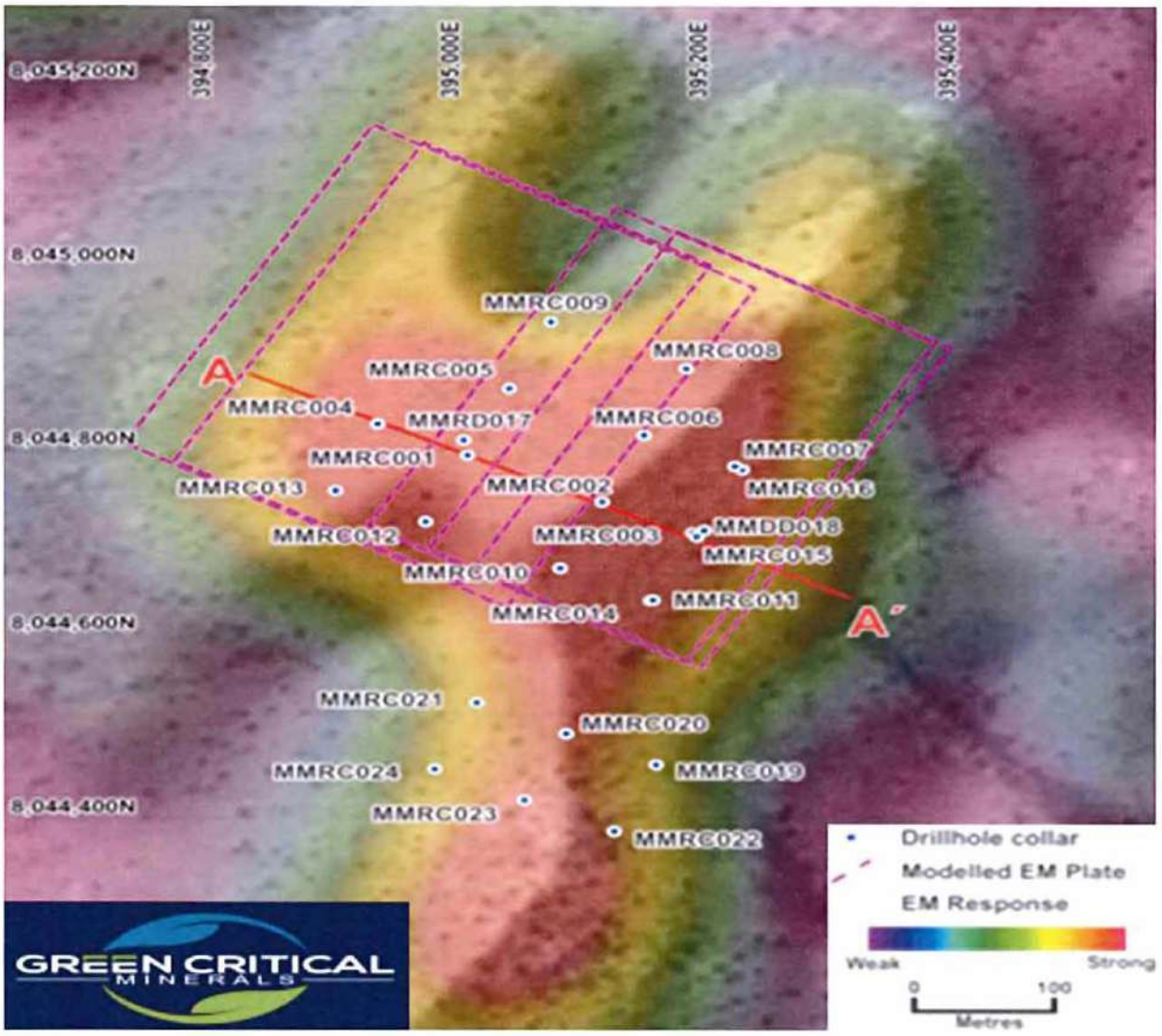


Figure 1. Plan view of drill hole locations and VTEM geophysical anomaly (with modeled plates) for the Mahi Mahi resource.

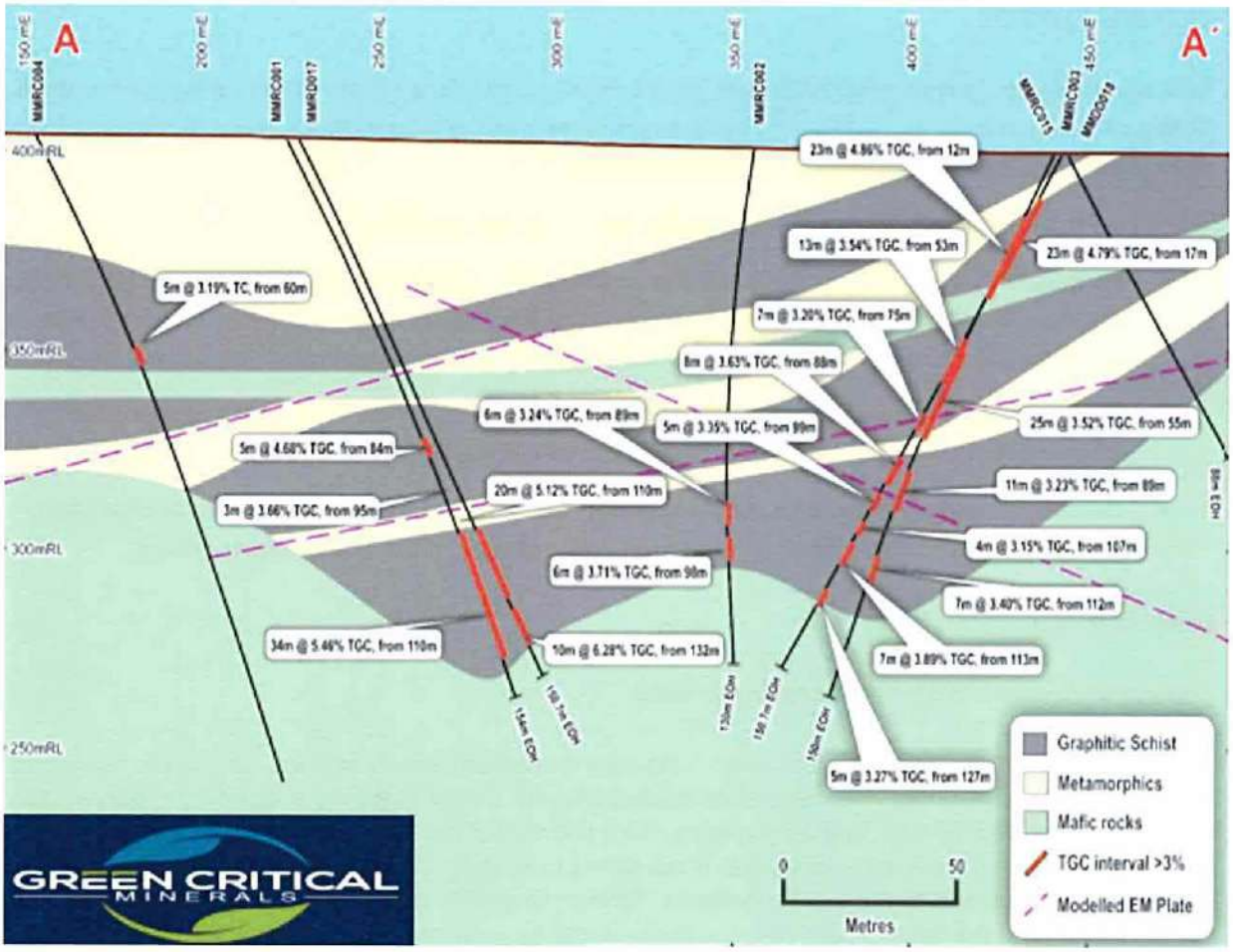


Figure 2. Cross section through the Mahi Mahi resource showing drill hole intercepts and modelled geology looking NNE.

ONGOING METALLURGICAL OPTIMISATION

GCM is committed to an ongoing metallurgical test work program to support the identification and design of an optimised process flow sheet, with the most recent results announced on 17 June 2024.

The Company is now planning further metallurgical test work across the McIntosh resource areas including the Mahi Mahi resource, leveraging the optimised flow sheet. This comprehensive approach aims to validate the potential for premium graphite product streams across the project's entire resource base.

GCM's Managing Director Clinton Booth added: "*Our metallurgical program is advancing, and we are highly encouraged by the results achieved thus far. With ongoing process optimisation, we are confident in our ability to become a leading and reliable global producer of high-quality upstream and downstream graphite powder products.* "

NEXT STEPS UPDATE

GCM reported in its 21 November 2023 announcement the next steps it will be taking to progress McIntosh. GCM is pleased to advise numerous activities are underway, and the information and results gathered from those activities are being used to inform future activities.

Based on the development pathway outlined GCM has therefore prioritised:

- Ongoing marketing analysis and potential customer engagement to identify McIntosh specific sale agreement opportunities, including traditional graphite markets, downstream graphite markets and both primary and secondary battery markets which includes the lithium-ion battery sector.
- Continuation of its comprehensive metallurgical test work program, to optimise each processing step from primary grinding, rougher and various cleaning stages for the McIntosh deposits.
- Development of a process flow sheet to support the design of a graphite concentrate pilot plant, including the possibility of a micronisation circuit and state of the art packaging system.

Competent Person Statement

Exploration Results and Mineral Resource Estimates

The Mineral Resource Estimates set out in this announcement are based on, and fairly represent, information and supporting documentation reviewed by Mr. David Eastman, a competent person. Mr. Eastman is employed full time by the company and is a Member of the Australian Institute of Geoscientists. Mr. Eastman has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to quality as a Competent Person as defined in the 2012 edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves: Mr. Eastman has consented to the inclusion of statements regarding the Mineral Resource Estimates set out in this announcement in the form and context that they appear. Further details about the Mineral Resource of the McIntosh project deposits are available on the ASX announcements platform (www2.asx.com.au, Code: HXG, Date:5 April 2018, Title : Revised McIntosh Mineral Resource ~ Amended)

Metallurgical test work outcomes

The information in this report that relates to the metallurgical activities are based on information compiled by Oliver Peters, who is a Member of the Professional Engineers of Ontario and the Principal Metallurgist and President of Metpro Management Inc. Oliver Peters has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Oliver Peters consents to the inclusion in the report of the matters based on his information in the form and context in which it appears.

Authorisation

The provision of this announcement to the ASX has been authorised by the Board of Green Critical Minerals Limited.

Forward Looking Statements

Statements contained in this release, particularly those regarding possible or assumed future performance, costs, dividends, production levels or rates, prices, resources, reserves or potential growth of Green Critical Minerals Limited, are, or may be, forward looking statements. Such statements relate to future events and expectations and, as such, involve known and unknown risks and uncertainties. Actual results and developments may differ materially from those expressed or implied by these forward-looking statements depending on a variety of factors.

Summary of Mineral Resource Estimate Reporting Criteria for the Mahi Mahi deposit

As per ASX Listing Rules 5.8 and the 2012 JORC reporting guidelines, a summary of the material information used to estimate the Mineral Resource is detailed below.

Mahi Mahi Deposit Mineral Resource Estimate

The estimate was created by Mr. Shane Tomlinson who worked for Mineral Resources Limited in a contractual capacity. Subsequently, it was reviewed by Mr. David Eastman who is a full-time employee of Green Critical Minerals.

Table 1 below shows the mineral resource estimation for the Mahi Mahi deposit

Deposit	Resource Classification	Tonnes (Mt)	%TGC	Contained Graphite (Mt)
Mahi Mahi	Indicated	0.0	0	0.00
	Inferred	6.3	4.2	0.27
	Total	6.3	4.2	0.27

Geology and Geological Interpretation

The McIntosh Project graphite schist horizons occur in the Halls Creek Mobile Zone of Western Australia. The host stratigraphy is the Tickalara Metamorphic which extend for approximately 130 km along the western side of the major Halls Creek Fault. The mineralisation has been described as a series of stacked graphitic schist horizons hosted within amphibolite facies that have been deformed into an anticlinal structure.

The mineralisation at the Mahi Mahi deposit generally occurs in zones between 5 to 90 metres wide, dipping gently to the west and north-west. The mineralisation remains open in all directions.

Sampling and Sub-sampling

The samples were collected via reverse circulation and diamond drilling. Drill holes were down-hole sampled at 1m intervals. The RC sub samples were generated using a rotary-splitter. Diamond drilling was run in the form of a HQ3 diamond tail on the end of an RC drill hole for one hole and continuous HQ3 diamond for a second hole. The core was collected for geotechnical and metallurgical characterisation test work. Core was quarter sampled at 1m downhole intervals for geochemical analysis. All samples were dried, crushed and pulverised to achieve 85% passing 75µm.

Duplicate and standards analysis were completed and no issues identified with sampling reliability.

Sample Analysis Method

Analytical test work was completed by ALS commercial laboratories in Perth, Brisbane, Vancouver and Ireland for Total Graphitic Carbon (TGC) analysis.

Drilling Techniques and hole spacing

Reverse circulation drilling was completed using face sampling hammers.

Diamond drilling was completed using HQ3 sized diamond bits.

All holes were drilled on an approximate 80m by 80m grid pattern.

Estimation Methodology

The estimation methodology used was ordinary kriging (OK). Block model dimensions used are 40m (east) by 40m (north) by 5m (elevation) with sub-blocking down to 5m (east) by 10m (north) by 1.25m (elevation). Block size was based on kriging neighbourhood analysis. The estimation was constrained using a soft 3% TGC cut-off grade to delineate the mineralisation/waste boundary from the resource drillhole dataset. Detailed statistical investigations have been completed on the captured estimation data set. This includes exploration data analysis and grade estimation trials. No top-cutting of composite grades was implemented. The estimation employed a three-pass search strategy. An inverse distance cubed estimate was run to provide an independent check on the OK model. Both models were globally similar.

Resource Classification

The resource has been classified as inferred for mineralisation satisfying the requirement of 'reasonable prospects for eventual economic extraction' in accordance with the JORC (2012) Code. The mineral resource has been classified on the basis of confidence in geological and grade continuity using the drilling density, geological model, modelled grade continuity and conditional bias measures (slope of the regression and kriging efficiency) as criteria.

Cut-off Grade

A cut-off grade of 3% TGC was used for the stated Mineral Resource estimate. The cut-off grade is in line with the standardised parameters used to generate the open pit shell.

Mining Method Selection

It is assumed that extraction will be by open pit mining and that the mineralisation is economic to exploit to currently modelled depths.

Processing Method

Multiple generations of metallurgical test work have been undertaken on the McIntosh Project graphitic ore to arrive at the preliminary process flowsheet for a 6-stage floatation processing circuit. Extensive additional metallurgical test work will be completed during FY24 to confirm mineralisation compatibility of deposits in the McIntosh Project area with the existing flowsheet. This work will include samples from the Mahi Mahi deposit. This test work will also be used to confirm the suitability of a variety of flow sheet optimisations.

Material Modifying Factors

No assumptions about minimum mining widths or dilution have been made as these are not seen as material at this stage.

Eventual Economic Extraction

It is the view of the Competent Persons that at the time of estimation there are no known issues that could materially impact on the reasonable prospects for eventual extraction of the Mineral Resources.

List of attachments

Appendix 1: Table of drill holes used for the Mahi Mahi resource estimation

Appendix 2: JORC Table 1 for Mahi Mahi Resource Estimate

Appendix 1: Table of drill holes used for the Mahi Mahi resource estimation

HoleID	HoleType	Eastng	Northing	mRL	Dip	Depth	Azimuth
MMDD018	DD	395211	8044692	399	-58	151	303
MMRC001	RC	395017	8044790	403	-61	154	115
MMRC002	RC	395134	8044730	401	-80	130	295
MMRC003	RC	395208	8044691	399	-62	150	297
MMRC004	RC	394950	8044815	404	-61	196	120
MMRC005	RC	395047	8044876	404	-61	100	125
MMRC006	RC	395168	8044794	401	-70	130	306
MMRC007	RC	395235	8044757	399	-70	130	303
MMRC008	RC	395197	8044887	401	-70	88	296
MMRC009	RC	395085	8044921	403	-61	94	131
MMRC010	RC	395099	8044659	401	-70	148	302
MMRC011	RC	395171	8044621	400	-56	148	298
MMRC012	RC	394982	8044714	402	-59	172	118
MMRC013	RC	394906	8044742	403	-59	172	117
MMRC014	RC	395179	8044621	399	-60	100	116
MMRC015	RC	395214	8044694	399	-59	88	118
MMRC016	RC	395241	8044762	399	-61	58	123
MMRC019	RC	395177	8044457	399	-57	82	117
MMRC020	RC	395105	8044482	401	-58	124	114
MMRC021	RC	395041	8044506	401	-58	174	112
MMRC022	RC	395145	8044383	400	-59	96	113
MMRC023	RC	395065	8044413	403	-60	142	115
MMRC024	RC	394998	8044443	403	-57	172	117
MMRD017	RCD	395020	8044789	403	-61	151	118
Total	**24**					**3149**	

***Appendix 2*: JORC Table 1 for the Mahi Mahi Resource Estimate – From February 2019 and amended April 2024**

Section 1 Sampling Techniques and Data

Criteria	JORC Code Explanation	Commentary
Sampling techniques	• *Nature and quality of sampling* • *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.*	**1. Reverse Circulation** • RC drilling used high pressure air and a cyclone with a rotary splitter. • Samples were collected at one-metre intervals. • All graphitic intervals were submitted for analyses. • Duplicate and standards analysis were completed and no issues identified with sampling reliability. • Samples were sent to the ALS laboratory in Perth for assay preparation and then sent to ALS in Brisbane, Vancouver and Ireland for Total Graphitic Carbon (TGC) analyses. • All samples were pulverised to better than 85% passing 75µm with a 10g aliquot taken for assay. • Sampling was guided by Hexagon and MRL's protocols and QA/QC procedures. • RC drilling samples of 3 to 5kg weight were shipped to the laboratory in calico bags; samples were pulverised and milled for assay. **2. Diamond Drilling** • HQ3 drill core samples were collected at one-metre intervals. • All graphitic intervals were submitted for analyses. • Core samples were quarter split by ALS using a diamond bladed saw and sent to the ALS laboratory in Perth for assay preparation and then sent to ALS in Brisbane, Vancouver and Ireland for Total Graphitic Carbon (TGC) analyses. • All samples were pulverised to better than 85% passing 75µm with a 10g aliquot taken for assay. • Duplicate samples, CRM standards and blank material (washed quartz sand) were used during the drill programs. Duplicates were collected after each 50 samples. Standards were inserted for samples ending in *00,*20,*40,*60 and *80 and blanks for samples ending in *01,*21,*41,*61 and *81.Sampling was guided by Hexagon and MRL's protocols and QA/QC procedures.

Criteria	JORC Code Explanation	Commentary
Drilling Techniques	• *Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).*	**1. Reverse Circulation** • 22 RC holes have been completed for 2,848 metres. • All RC drilling was completed with face sampling hammers and collected through a cyclone. Sample recovery was estimated as a percentage of the expected sample, sample state recorded (dry, moist or wet), samples tested with 10:1 HCl acid for carbonates and graphite surface float. • Drilling was completed by Mt Magnet Drilling using a Hydco 1300 drill rig. **2. Diamond Drilling** • One RC pre-collar was drilled in preparation for a HQ_3 diamond tail, for a total of 51m. • Two diamond holes for 251 metres were completed. • HQ_3 core was collected using a 3m core barrel. • Drilling was completed Mt Magnet Drilling using a Hydco 650 drill rig. • Core orientation was recorded using a Reflex EZ Shot instrument.
Drill sample recovery	• *Method of recording and assessing core and chip sample recoveries and results assessed.* • *Measures taken to maximise sample recovery and ensure representative nature of the samples.* • *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	**1. RC Drilling** • A face sampling hammer was used to reduce contamination at the face. • 1m drill chip samples, weighing between 3-5kg were collected in sequentially numbered bags. • Split samples were recovered from a cyclone and rig-mounted cone splitter. The sample recovery and physical state were recorded. • Every interval drilled is represented in an industry standard chip tray that provides a check for sample continuity down hole. **2. Diamond drilling** • Core recoveries were measured for each run between core blocks and measurements recorded.
Logging	• *Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.* • *Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.*	• All RC and diamond drilling was logged for geology in the field by qualified geologists. Lithological and mineralogical data was recorded for all drill holes using a coding system developed specifically for the Project. Primary and secondary lithologies are recorded in addition to texture, structure, colour, grain size, alteration type and intensity, estimates of mineral quantities, graphite intensity and sample recovery. The oxidation zone is also recorded. • No adjustments have been made to any assay data. • Geological logging is qualitative in nature.

Criteria	JORC Code Explanation	Commentary
	• *The total length and percentage of the relevant intersections logged.*	• Diamond drill logging also recorded recovery, structure and geotechnical data. • Diamond core was orientated using the Reflex orientation tool. • All core was orientated and marked up in preparation for cutting. • Core was photographed both wet and dry.
Sub-sample techniques and sample preparation	• *If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.* • *For all sample types, the nature, quality and appropriateness of the sample preparation technique.* • *Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.* • *Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.* • *Whether sample sizes are appropriate to the grain size of the material being sampled.*	**1. RC Drilling** • All samples were marked with unique sequential sample number. • RC drilling samples were bagged at the drill site in calico bags with a second outer plastic bag to prevent loss of fines. The sample sizes are considered to be appropriate to the grain size of the material being sampled. • 1m RC drill samples were submitted to ALS laboratories in Perth. The samples were riffle split on a 50:50 basis, with one split pulverised and analysed for Total Graphitic Carbon (TGC), Total Carbon (TC) and Total Sulphur (TS) using a LECO Furnace, and the other split held in storage. • For RC samples, standards and field duplicates were inserted at an approximate rate of 1 in every 20 samples collected. Duplicate assay results exhibit good correlation with the original assays and no consistent bias is evident. • Sample preparation: 1. Coarse crush using a jaw crushed to better than 70% passing 6mm. 2. For samples exceeding 3kg received mass, riffle split using a Jones Riffle Splitter 50:50 3. Pulverise up to 3kg of coarse crushed material to better than 85% passing 75µm particle size. 4. Small aliquot (~10g) taken for assay. **2. Diamond Core** • Diamond drill core was cut into half core (retained for metallurgical testing) and the remaining half sawn into quarter core using diamond blade core-saw. Quarter core was used for samples and duplicates. Core cutting was carried out by ALS in Perth. • Duplicate assay results exhibit good correlation with the original assays and no consistent bias is evident. • Sample preparation: 1. Coarse crush using a jaw crushed to better than 70% passing 6mm.

Criteria	JORC Code Explanation	Commentary
		2. For samples exceeding 3kg received mass, riffle split using a Jones Riffle Splitter 50:50 3. Pulverise up to 3kg of coarse crushed material to better than 85% passing 75µm particle size. 4. Small aliquot (~10g) taken for assay. • Sampling procedures and sample preparation represent industry good practice:
Quality of assay data and laboratory tests	• *The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.* • *Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.*	• The assaying and laboratory procedures used are appropriate for the material tested. • Sampling was guided by Hexagon and MRL's protocols and QA/QC procedures. • For RC samples, standards and field duplicates were inserted at an approximate rate of 1 in every 20 samples collected. • Field duplicates were taken from the coarse reject of processed diamond core samples at a rate of 4 every 100 samples, standards at a rate of 4 every 100 samples and blanks at 2 every 100 samples. • Statistical analysis of standards, blanks and duplicates during the QAQC process showed that the data was satisfactory. • No issues were identified with sampling reliability
Verification of sampling and assaying	• *The verification of significant intersections by either independent or alternative company personnel.* • *The use of twinned holes.* • *Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.* • *Discuss any adjustment to assay data.*	• Hexagon QA/QC checks show that all samples are within acceptable limits. No adjustments to assay data have been made based on the analysis of duplicates, standards and blanks. • Standards from ALS laboratory were found to be acceptable. • Duplicate analysis was completed and no sampling issues were identified. • During a site visit in October 2015, a geological consultant from CSA verified that the diamond drilling, geological logging and sampling practices were of industry standard. The same practices were used for the Mahi Mahi drilling in 2018. • Green Critical Minerals conducted external verification on all data collected during 2018. • The Hexagon database is hosted in a SQL backend database, ensuring that data is validated as it is captured and exports are produced regularly. Assay results are merged into the database from the lab certificates limiting transcription or mapping errors from occurring.

Criteria	JORC Code Explanation	Commentary
		• No adjustments have been made to the results.
Location of Data points	• *Accuracy and quality of surveys used to locate drillholes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.* • *Specification of the grid system used.* • *Quality and adequacy of topographic control.*	• 24 drill collars were surveyed by MNG Survey using a Differential GPS. The degree of accuracy of drill hole collar location and RL is estimated to be within 0.1m for DGPS. • All holes used in the resource have been downhole surveyed using a gyro by ABIM Solutions. • The map projection used is the Australia Geodetic MGA 94 Zone 52.
Data spacing and distribution	• *Data spacing for reporting of Exploration Results.* • *Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.* • *Whether sample compositing has been applied.*	• Drill spacing on an approximate 80m by 80m grid. • Geological interpretation and mineralisation continuity analysis indicates that data spacing is sufficient for definition of a Mineral Resource.
Orientation of data in relation to geological structure	• *Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* • *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	• Holes generally drilled dipping at -60° to the southeast targeting a north westerly dipping body. • Diamond drill core has been orientated using a Reflex ACE tool 9Act II), with α and β angles measured and positioned using a Kenometer. • The relationship between the drilling orientation and the orientation of key mineralised structures is not considered to have introduced a sampling bias.
Sample Security	• *The measures taken to ensure sample security.*	• Unique sample number was retained during the whole process. • RC samples were placed into calico bags and then into plastic bags prior to being put into bulka bags on pallets. The bulka bags were then transported by road to ALS laboratories in Perth. Preparation was completed by ALS in Perth and then transferred through internal systems to ALS Brisbane, Vancouver and Ireland for analysis • Diamond core was sent to ALS in Perth for cutting and preparation. Then transferred through internal systems to ALS Brisbane, Vancouver and Ireland for analysis.

Criteria	JORC Code Explanation	Commentary
		• Drill core was transported to ALS in Perth by road train in stacked core trays, secured to pallets with metal strapping. • The sample security is considered to be adequate.
Audits or reviews	• *The results of any audits or reviews of sampling techniques and data.*	• Sampling techniques and data collected methods have been audited by CSA during a site visit in October 2015. These same practices were adopted in 2018. • Field data is managed by an independent data management consultancy Rocksolid Solutions. • All data collected was subject to internal review. • No audits or reviews were completed on work completed in 2018.

Section 2 Reporting of Exploration Results

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• *Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.* *The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.*	• Drilling at the Mahi Mahi deposit occurred on exploration lease E80/4825. This tenement is held by Hexagon Energy Materials Limited. • Mineral Resources Ltd were the mangers of the 2018 exploration work on the McIntosh Project.
Exploration done by other parties	• *Acknowledgment and appraisal of exploration by other parties.*	• The East Kimberley has been largely explored for base metals and diamonds with no active previous exploration for graphite. Graphite had been noted by Gemutz during regional mapping in the Mabel Downs area for the BMR in 1967, by Rugless mapping and RAB drilling in the vicinity of Melon Patch bore, to the east of the Great Northern Highway in 1993 and has been located during nickel exploration by Australian Anglo American Ltd,

Criteria	JORC Code explanation	Commentary
		Panoramic Resources Ltd and Thundelarra Resources Ltd over the last 20 years.
Geology	• *Deposit type, geological setting and style of mineralisation.*	• The McIntosh Project graphite schist horizons occur in the high grade terrain of the Halls Creek Mobile Zone of Western Australia. The host stratigraphy is the Tickalara Metamorphic which extend for approximately 130 km along the western side of the major Halls Creek Fault. The metamorphic rocks reach granulite metamorphic facies under conditions of high-temperature and high pressure although the metamorphic grade in the McIntosh Project area appears to be largely upper amphibolite facies with the presence of key minerals such as sillimanite and evidence of original cordierite. • Hexagon and GCM have identified potential graphite schist horizons based on GSWA mapping and EM anomalism over a strike length in excess of 15km within the project area, with potential for an additonal 35km strike length of graphite bearing material from lower order EM anomalism.

Drill hole Information	• *A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drillholes:*
	  • *easting and northing of the drillhole collar*
	  • *elevation or RL (elevation above sea level in metres) of the drillhole collar*
	  • *dip and azimuth of the hole*
	  • *down hole length and interception depth*
	  • *hole length.*
		• 22 RC holes have been completed for 2,848 metres.
		• One RC pre collar for 51metres was completed.
		• Two Diamond holes for 251metres were completed.
		• Hole locations tabulated and reported in the body of the report.

Data aggregation methods	• *In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.*	• Data compiled in excel and validated in Datashed by an external data management consultancy. • RC and diamond samples were all 1m in length. • Metal equivalents are not reported as this is an industrial mineral project where the mineral properties define grade (e.g. flake size and purity). • A nominal 3% Total Graphitic Carbon cut-off has been applied in the determination of significant intercepts
Relationship between mineralisation widths and intercept lengths	• *If the geometry of the mineralisation with respect to the drillhole angle is known, its nature should be reported.*	• Mineralised widths at Mahi Mahi are estimated to be typically between 20m and 70m, compared with RC samples of 1m width. There is a very close relationship between the graphitic schist unit and Total Graphitic Carbon

Criteria	JORC Code explanation	Commentary
	• *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect.*	(TGC%) assays. The presence of graphitic schist is clearly evident in both the RC chips and diamond drill core so that the assay widths can be clearly related to the geological logs.
Diagrams	• *Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drillhole collar locations and appropriate sectional views.*	• Relevant diagrams have been included within the Mineral Resource report main body of text.
Balanced reporting	• *Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	• Exploration results have been reported using a nominal 3% TGC cut off, over a minimum interval length of 3m. Internal dilution of no more than 2m sub 3% TGC has been incorporated.
Other substantive exploration data	• *Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	• An Xcite EM survey was completed over the tenement in 2016. Conductors highlighted in this survey were targeted for drill testing and resulted in the discovery of the Mahi Mahi deposit.
Further work	• *The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large-scale step-out drilling).*	• Test EM anomalies along strike for potential extensions to mineralisation.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code explanation	Commentary
Database integrity	• *Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.*	• Primary data was captured into spreadsheet format by the supervising geologist, validated and subsequently loaded into Hexagon's database. • Database extracted as an .mdb access file from Datashed and validated before importing into Surpac.

Criteria	JORC Code explanation	Commentary
	Data validation procedures used.	• Additional data validation by Green Critical Minerals; included checking for out of range assay data and overlapping or missing intervals.
Site Visits	• *Comment on any site visits undertaken by the Competent Person and the outcome of those visits.*	• Numerous site visits were completed by S. Tomlinson during the 2018 drilling period. The diamond and RC drill rigs were inspecting, sampling procedures checked, RC chips and diamond core logged.
Geological interpretation	• *Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.* • *Nature of the data used and of any assumptions made.* • *The effect, if any, of alternative interpretations on Mineral Resource estimation.* • *The use of geology in guiding and controlling Mineral Resource estimation.* *The factors affecting continuity both of grade and geology.*	• Geological interpretation was based on lithology logging, structural logging, geochemical sampling, prospect scale surface mapping and modelled Xcite EM survey data collected in 2016. • Drill coverage to ~80m x 80m. • Mineralisation wireframe produced based on soft 3% TGC cut-off grade delineating ore/waste boundary. Internal dilution in the main mineralised envelope has been modelled as two domains. Further modelling of mafic intrusive bodies have also been modelled. • The base of oxidation and mafic intrusives were also modelled as part of the Mahi Mahi resource. • Confidence in the grade and geological continuity is reflected in the Mineral Resource classification.
Dimensions	• *The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	• The Mahi Mahi deposit resource extends 520m north- northeast to south-southwest. The mineralisation dips ~20^0 degrees to the southwest and is ~70m thick. • Mineralisation is open along strike and at depth.
Estimation and modelling techniques	• *The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.* • *The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes*	• The resource was modelled using Geovia's Surpac v6.7 modelling software. • Drill hole sample data was flagged from interpretations of the top and base of the mineralisation horizon. • Sample length was composited to 1m down hole length. • Top grade cuts were not applied. • Total Graphitic Carbon (TGC) estimated by Ordinary Kriging (OK) for mineralised domains (1 and 4). Sulfur (S) estimated by OK for mineralised domains (1 to 4). • Regression calculations were used to fill attributes SiO_2, CaO, K_2O and Na_2O. • Flake size distribution was assigned to domains 1 and 4 based on petrography analysis.

Criteria	JORC Code explanation	Commentary
	appropriate account of such data. • *The assumptions made regarding recovery of by-products.* • *Estimation of deleterious elements or other non-grade variables of economic significance (e.g. sulphur for acid mine drainage characterisation).* • *In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.* • *Any assumptions behind modelling of selective mining units.* • *Any assumptions about correlation between variables.* • *Description of how the geological interpretation was used to control the resource estimates.* • *Discussion of basis for using or not using grade cutting or capping.* • *The process of validation, the checking process used, the comparison of model data to drillhole data, and use of reconciliation data if available.*	• Density was assigned based on the average of downhole using a Geovista Dual density logging tool. • Statistical analysis was completed to investigate low correlation variances, boundary conditions between domains, fresh/oxide, extrapolation distance, variogram ranges, KNA, parent block size, sub-cell, constraints used for volume model, variable search orientation, sample numbers used, discretisation, validation. • TGC mineralisation continuity was interpreted from variogram analyses to have a horizontal range of 170m (north-west to south-east). • The entire resource has been classified as Inferred material, due to the drill hole spacing being sparse, but still sufficient to assume continuity of mineralisation. Confidence for the resource in these areas is also from the VTEM survey completed over the area. • The maximum extrapolation distance is 40 m along strike and 40 m across strike. • Grade estimation was into parent blocks of 40 mE by 40 mN by 5 mRL. Block size was selected based on kriging neighbourhood analysis. • Estimation was carried out using ordinary kriging at the parent block scale. • The search ellipses were oriented within the plane of the mineralisation. • Three estimation passes were used; the first search was based upon the variogram ranges in the three principal directions; the second search was two times the initial search and the third search was four times the initial search, with reduced sample numbers required for estimation. • Aproximately70% of the block grades were estimated in the first pass for domain 1 (main envelope) and 49% for domain 4. • The estimated TGC block model grades were visually validated against the input drillhole data, comparisons were carried out against the drillhole data and by northing, easting and elevation slices. • There is no production data and so no reconciliation has taken place.
Moisture	• *Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	• The Mahi Mahi deposit sits partially below the water table. • Moisture content has not been tested

Criteria	JORC Code explanation	Commentary
Cut-off parameters	• *The basis of the adopted cut-off grade(s) or quality parameters applied.*	• Based on a statistical analysis of drill data, lower cut-off grade of 3.0% total graphitic carbon was used for determining mineralised material at the Mahi Mahi deposit.
Mining factors or assumptions	• *Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous.*	• It is assumed that extraction will be by open pit mining and that the mineralisation is economic to exploit to currently modelled depths. • Mining factors such as dilution and ore loss have not been applied. • No assumptions about minimum mining widths or dilution have been made.
Metallurgical factors or assumptions	• *The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous.*	• Petrological assessment of graphite flake size was completed. This concluded that the graphite is <50micron. • A review of the geological data suggests that the new targets may have a similar metallurgical response than the Emperor resource. • Flotation testing will have to be conducted to confirm these assumptions.
Environmental factors c assumptions	• *Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation.*	• No assumptions have been made regarding waste and process residue. • Environmental studies will be completed as part of the GCM McIntosh Pre-Feasibility study.
Bulk density	• *Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and*	• Dry density was assigned a value of 2.85 (fresh) and 2.65 (oxide) based on 26 dried core samples and water emersion technique carried out by ALS.

Criteria	JORC Code explanation	Commentary
	representativeness of the samples. • *The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.* *Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	
Classification	• *The basis for the classification of the Mineral Resources into varying confidence categories.* • *Whether appropriate account has been taken of all relevant factors (i.e. relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* *Whether the result appropriately reflects the Competent Person's view of the deposit.*	• The Mahi Mahi Mineral Resource has been classified in the Inferred category, in accordance with the 2012 Australasian Code for Reporting of Mineral Resources and Ore Reserves (JORC Code). • The mineral resource has been classified on the basis of confidence in geological and grade continuity using the drilling density, geological model, modelled grade continuity and conditional bias measures (slope of the regression and kriging efficiency) as criteria. • Measured resource - none defined. • Indicated resource – none defined. • Inferred material includes the entire resource due to drilling data being sparse, but still sufficient to assume continuity of mineralisation. Confidence for the resource in these areas is also from the VTEM survey completed over the area. The classification considers all available data and quality of the estimate and reflects the Competent Person's view of the deposit.
Audits or reviews	• *The results of any audits or reviews of Mineral Resource estimates.*	• A review of all geological data was undertaken by Green Critical Minerals, with no underlining issues being identified.
Discussion of relative accuracy/ confidence	• *Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person.* • *The statement should specify whether it relates to global or local estimates, and, if local, state the*	• The relative accuracy of the Mineral Resource estimate is reflected in the reporting of the Mineral Resource as per the guidelines of the JORC Code (2012 Edition). • The mineral resource is a global estimate of tonnes and grade. • The confidence intervals have been based on a block informing information. • Relative tonnages and grade above the nominated cut-off grades for TGC are provided in the body of this report. Volumes of the collated blocks sub-set by mineralisation domains were multiplied by the dry density value to derive

Criteria	JORC Code explanation	Commentary
	relevant tonnages, which should be relevant to technical and economic evaluation.	the tonnages. The contained graphite values were calculated by multiplying the TGC grades (%) by the estimated tonnage. • No production data is available to reconcile results with.



breakaway

BREAKAWAY RESOURCES LTD

ANNUAL REPORT

ON THE EAST KIMBERLEY PROJECT

FOR THE PERIOD

16 FEBRUARY 2008 TO 15 FEBRUARY 2009

KIMBERLEY MINERAL FIELD, WESTERN AUSTRALIA

Combined Reporting Number C12/2005

Exploration Licences: E80/2607, 2635, 2716, 2746, 2748, 2749, 2817, 2824, 2827, 2835, 2836, 2865, 2866, 2867, 3704, 3705 and 3323
Prospecting Licences: P80/1563
Reversion Licenses, E80/3873, E80/3876, E80/3877 and E80/3878

BY

D. THOMPSON and D. HUTTON

April 2009

DISTRIBUTION:

1. Department of Industry and Resources
2. Breakaway Resources Ltd
3. Thundelarra Exploration Ltd

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1.0 OVERVIEW

Project Name:	East Kimberley Project
Combined Reporting Number:	C12/2005
Prospect Names:	Keller Creek, Sally Downs Bore, Fletcher Creek, Dave Hill, Copernicus, Mabel Downs, Three Nuns and Six Mile Intrusion
Tenement:	Exploration Licences: E80/2607, 2635, 2716, 2746, 2748, 2749, 2817, 2824, 2827, 2835, 2836, 2865, 2866, 2867, 3704, 3705, and 3323. Prospecting Licences: P80/1563
Mineral Field:	Kimberley Mineral Field, Western Australia
Holder:	Thundelarra Exploration Ltd (40%) Altia Resources Pty Ltd (60%)
Operator:	Breakaway Resources Ltd (100%)

Location:

Lissadell	SE5202	1:250 000
Bow	4564	1:100 000
Dixon Range	SE5206	1:250 000
Mt Remarkable	4463	1:100 000
Turkey Creek	4563	1:100 000
McIntosh	4462	1:100 000
Dixon	4562	1:100 000
Gordon Downs	SE5210	1:250 000
Halls Creek	4461	1:100 000
Mt Ramsay	SE5209	1:250 000
Angelo	4361	1:100 000
Dockrell	4360	1:100 000

Reporting Period:	16 February 2008 to 15 February 2009
Commodities:	Nickel-copper sulphide, PGE's

2.0 SUMMARY OF EXPLORATION ACTIVITIES

The following activities were completed during the reporting period:

- Reverse Circulation drilling on the Keller Creek prospect with 7 holes being drilled for a total of 1150m.
- Ground truthing of geophysical anomalies on EL's 80/3323 and at the Gauss prospect.
- A high-resolution aeromagnetic survey was flown over the northern tenement block of the project area.

3.0 INTRODUCTION AND TENURE

The East Kimberley Project is located within the Halls Creek Orogen (HCO), in the East Kimberley region of Western Australia. Several Proterozoic – age intrusive mafic to ultramafic bodies, prospective for nickel-copper-platinum group elements (PGE) mineralisation crop out within the project area.

During 2003, LionOre Australia Pty Ltd, entered into an agreement with Thundelarra to earn a 60% interest in a portion of their East Kimberley tenements, by funding A$5.0 million of exploration over a five year period, commencing 01 January 2004. On 18 August 2006 LionOre sold their earn-in interest, and management of, the Project tenements to Breakaway Resources Limited, a Perth based, ASX listed exploration company. Altia Resources Pty Ltd (a registered company of Breakaway Resources) subsequently earned their 60% in the tenements by completing the $5M earn in during November 2006. On 31 December 2008 all tenements, with the exception of E80/2836 were handed back to Thundelarra.

Tenement details are shown in Table 1.

Exploration tenements E80/3704 and E80/3705 are 100% Altia Resources Pty Ltd owned.

This report describes the exploration activities conducted by LionOre Australia (East Kimberley) Pty Ltd and Breakaway Resources Ltd within the East Kimberley Project (tenements grouped under the combined reporting number C12/2005), during the reporting period 16 February 2008 to 15 February 2009.

Table 1 Tenement Details - East Kimberley Project

Project	Tenement	Approval Date	Expiry Date	Area (blocks)	Expenditure Commitment	Holders/Shares
Corkwood South	E80/2607	23-05-02	22-05-07	8 *	$50,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Corkwood North	E80/2635	23-05-02	22-05-07	12 *	$50,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Eileen Bore	E80/2716	8-02-02	7-02-07	22 *	$64,667	Thundelarra Exploration Ltd 100%
Nortons	E80/2746	18-06-02	17-06-07	12	$50,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Dave Hill	E80/2748	15-01-03	14-01-08	56	$112,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Sally Downs	E80/2749	15-08-03	14-08-08	38	$76,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Mabel Downs	E80/2817	4-03-03	3-03-08	25	$50,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Mabel West	E80/2824	3-09-02	2-09-07	12	$50,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Bow River West	E80/2827	7-10-02	6-10-07	4	$30,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Ord River	E80/2835	7-10-02	6-10-07	2	$30,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Keller Creek	E80/2836	4-07-03	3-07-08	39	$78,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Fletcher Creek	E80/2865	7-10-02	6-10-07	4	$30,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Warmun	E80/2866	16-05-03	15-05-08	7	$50,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
Killarney	E80/2867	4-03-03	3-03-08	11	$50,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
	E80/3704	8-08-07	7-08-12	39	$39,000	Altia Resources Pty Ltd 100%
	E80/3705	8-08-07	7-08-12	36	$36,000	Altia Resources Pty Ltd 100%
Mabel Downs	E80/3323	25-07-05	24-07-10	4	$20,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%
	P80/1563	16-02-07	15-02-11	26Ha	$2,000	Altia Resources Pty Ltd 60% Thundelarra Exploration Ltd 40%

* In accordance with Section 120AA of the Mining Act three tenements were reduced in size during the reporting period (Table 2).

Table 2 Surrendered and partially surrendered tenements

Project	Tenement	Partial expiry date	Area expired	Area retained	Exploration application
Corkwood South	E80/2607	8-04-08	6	8	E80/3873
Corkwood North	E80/2635	28-07-08	10	12	E80/3875
Eileen Bore	E80/2716	7-10-08	14	22	E80/3878

4.0 LOCATION AND ACCESS

The East Kimberley project is located 50km north east of Halls Creek within the East Kimberley region of Western Australia (Figure 1). The project straddles the Great Northern Highway and extends for 110km to the northeast. Access to the project area is from the Great Northern Highway and within the tenements by way of station and exploration tracks (Figure 2).

The project is located within the Lissadell (SE 5202), Dixon Range (SE 5206), Mt Ramsay (SE5209), and Gordon Downs (SE5210) 1:250,000 Geological Map Sheets published by the Geological Survey of Western Australia.

Figure 1 Project location plan

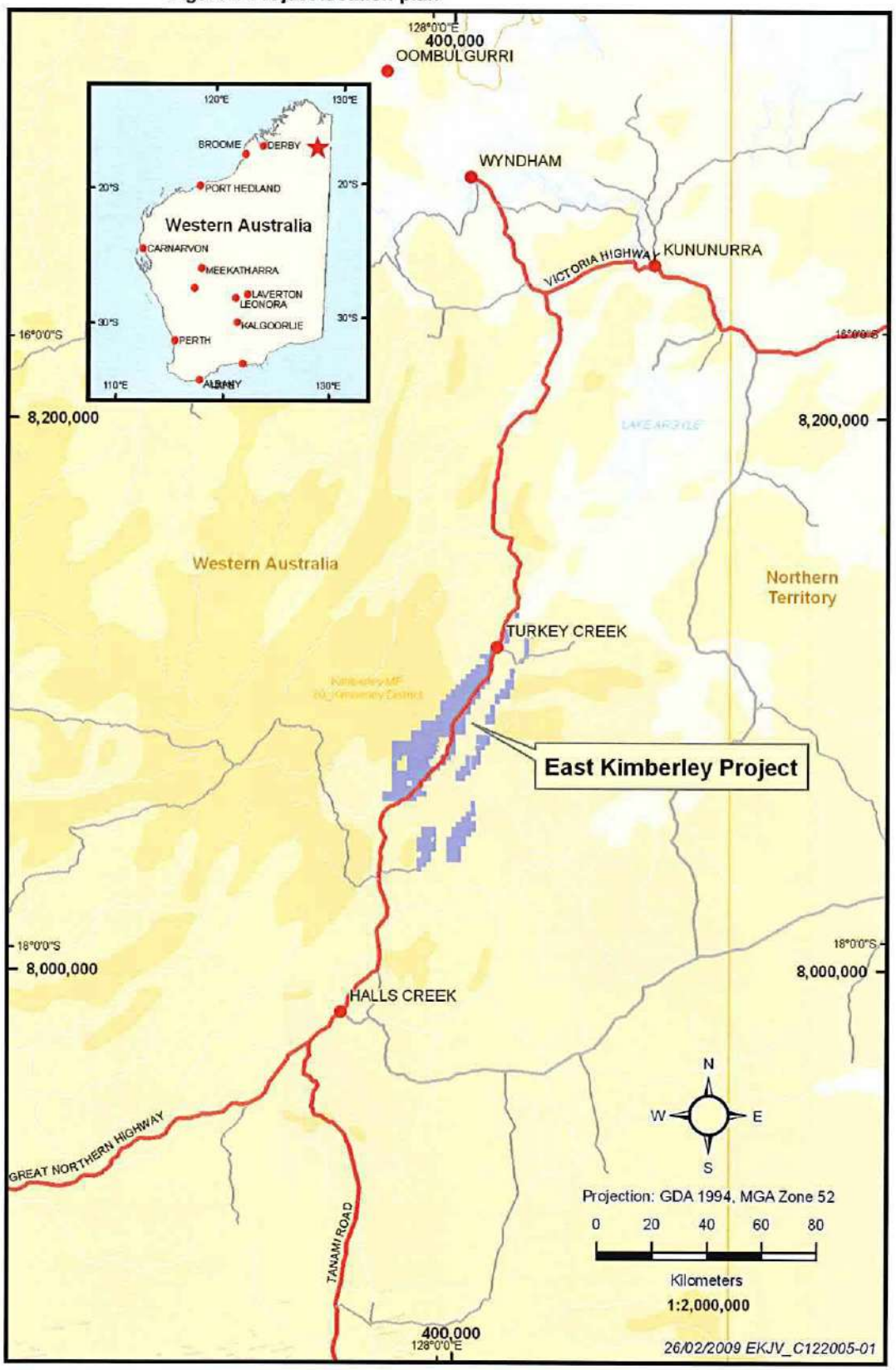
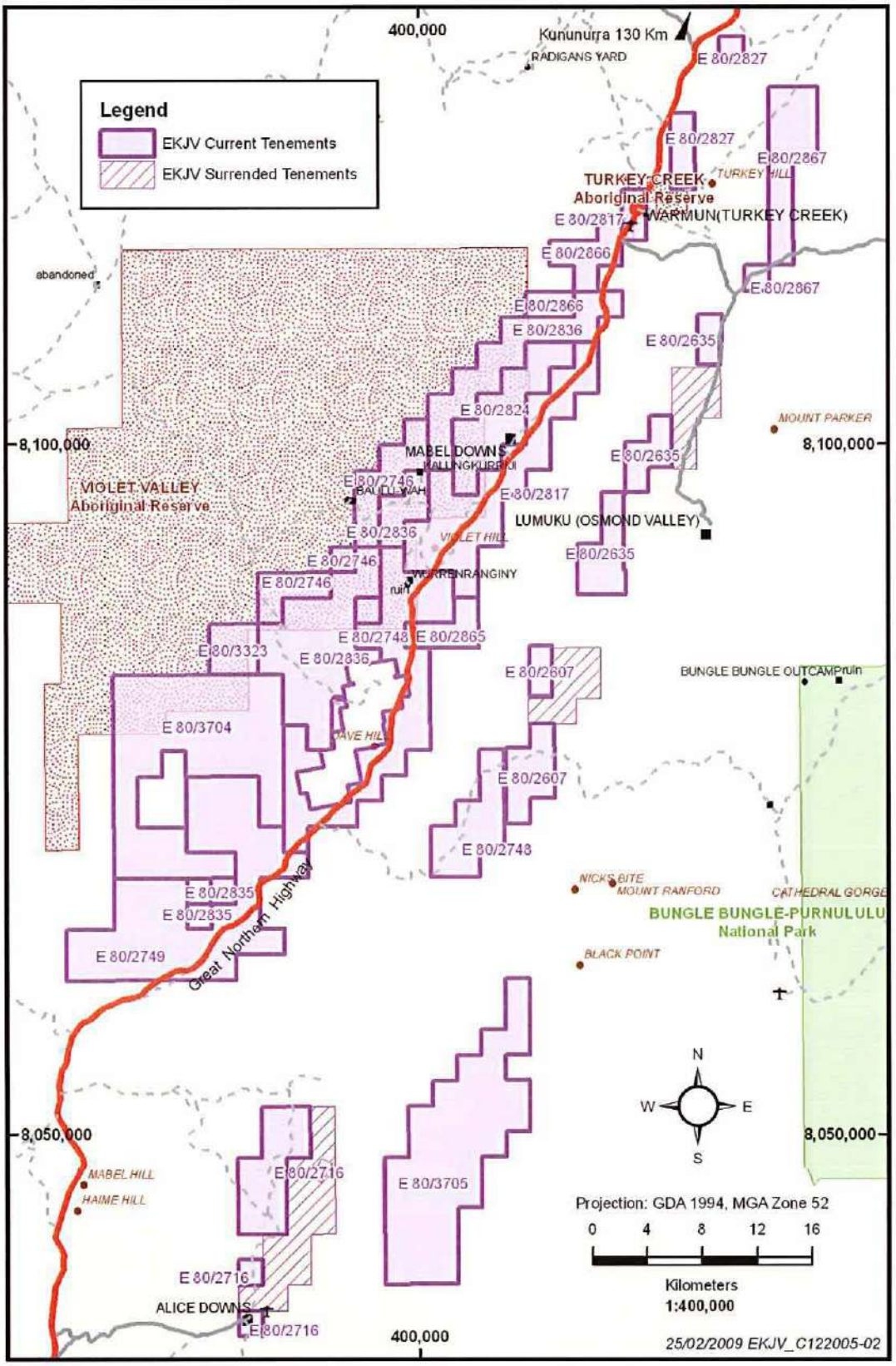


Figure 2 Location and access plan



5.0 REGIONAL GEOLOGY

The National GeoScience Mapping Accord (NGMA), a joint initiative between State (GSWA) and Commonwealth (AGSO, now GA) organizations, has recently completed a major study of the East Kimberley. Resulting publications, especially AGSO Bulletin 246, 'Geology and economic potential of the Palaeoproterozoic layered mafic – ultramafic intrusions in the East Kimberley, Western Australia' (Hoatson and Blake, 2000), comprehensively describes the geological setting of the Halls Creek Orogen.

The Halls Creek Orogen (HCO) is a complex Palaeoproterozoic terrain comprising low to high grade metasedimentary and metavolcanic rocks, and voluminous granitic, mafic and mafic-ultramafic intrusions that collectively range in age from about 1910 Ma to 1790 Ma (Page et al. 1995). The HCO is a well-exposed north-northeast trending orogenic belt approximately 120 km long by 45 km wide. The package as a whole is termed the Lamboo Complex and can be divided into three zones (Figure 3).

The Central Zone comprises felsic to mafic and ultramafic intrusions within high-grade metamorphic sediments and mafic units of the Tickalara Metamorphics, together with sediments of the Koongie Park Formation to the south. The Central Zone contains numerous nickel-copper and platinum group elements occurrences associated with the intrusive suites.

The Eastern Zone comprises Halls Creek Group sediments of the Olympio and Biscay formation, broadly correlated in age with the c.1865ma Tickalara Metamorphics. These are intruded by the mafic to ultramafic Woodward Dolerite. The Eastern Zone has a number of gold occurrences.

The Western Zone comprises granitic and gabbroic rock of the Paperbark Supersuite (1865-1850Ma) with the Whitewater Volcanics cropping out on its western flank. The Western Zone has a number of nickel-copper and platinum group elements occurrences associated with intrusive suites.

The HCO is underlain by a regional gravity high of unusually large magnitude is present when compared to other anomalies of the Australian continent, with only the Giles Complex of the Musgrave Block of comparable intensity (Hoatson and Blake 2000). This is interpreted to be related to mantle-derived magma pooled at or near the crust-mantle boundary. The various intrusions within the HCO appear to be related to this accumulation. Mafic-ultramafic intrusions are confined to the central part of the Halls Creek Orogen, and crystallized at depths of between 8-23km in a variety of forms including sheets, basinal forms, funnels, plugs and multi-chambered bodies. A total of 60 intrusions have been recognized by AGSO. It is anticipated that as the geological knowledge of the area evolves, more intrusions will be identified.

SHRIMP U-Pb age dating undertaken by AGSO (Hoatson and Blake, 2000) has determined some of the major intrusions were emplaced between 1830-1856Ma. Hoatson and Blake, (2000) has proposed division of the mafic-ultramafic intrusives into seven groups, at least three of which are coeval, plus three other associations and undivided units. These are listed in Table 3.

Hoatson and Blake (2000) document a high S content within HCO intrusions, with all groups with the possible exception of VII being derived from S-saturated parent magmas. They have potential to host nickel-copper sulphide mineralisation.

Sourced from Phyland (2004).

Figure 3 Regional geological setting

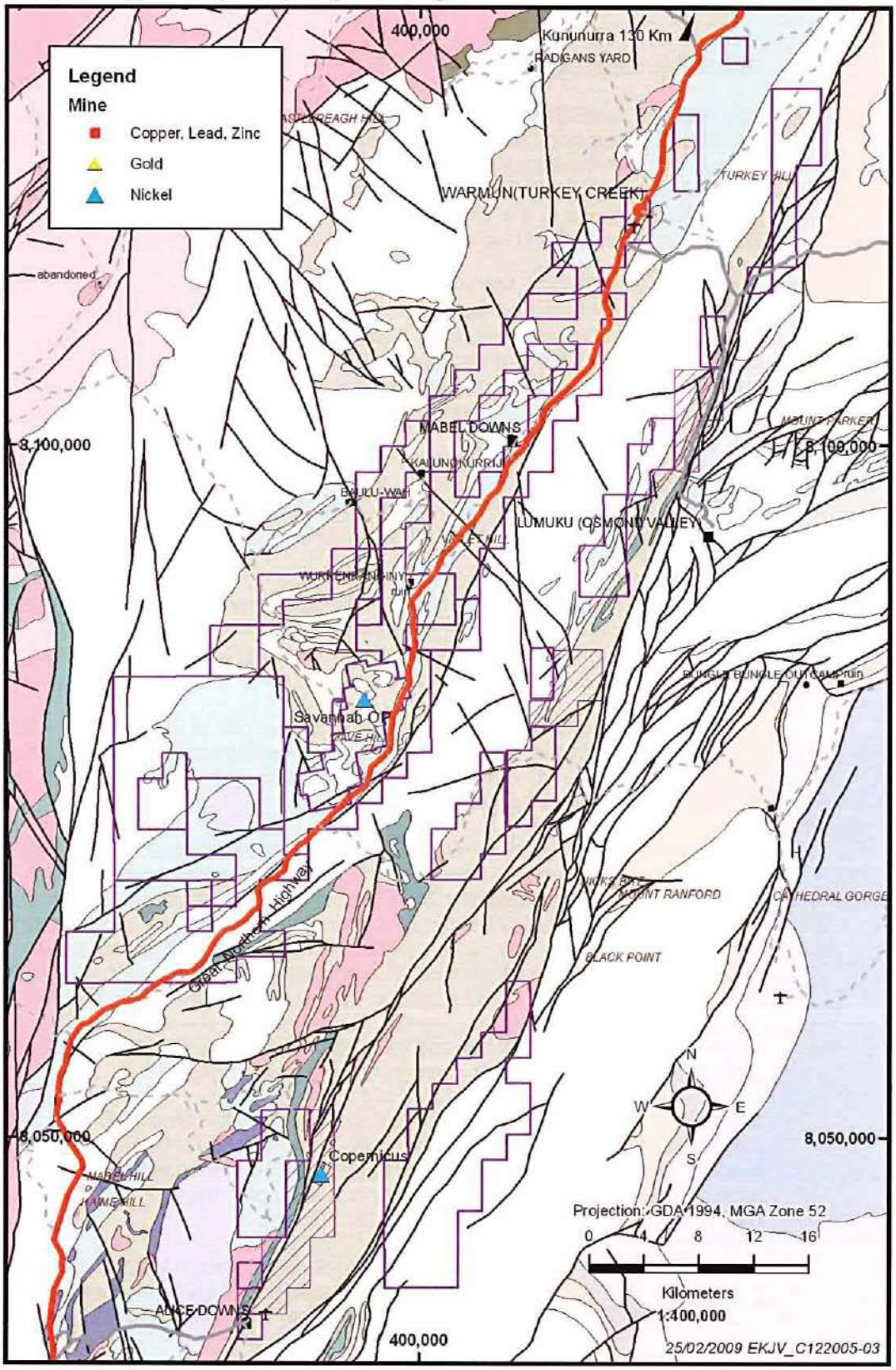


Table 3 Mafic ultramafic intrusions: Types, ages and depths of emplacement

GROUP	NAME	AGE Ma	DEPTH	THICKNESS
Group VII	Black Yard Hills (intrudes Corridor Gabbro)			
Group VI	McIntosh	1830 ± 3	18km	7.8km
	Turkey Creek Microgabbro-norite (intrudes Group V)			
Group V	Sally Malay	1844 ± 3	22km	1.7km
	Koongie Park Felsic Volcanics	1843		
Group IV	Wild Dog Creek	Between 1843-1830		
Group III	Toby	1855 ± 2	8km	2km
	Corridor Gabbro Association (intrudes Group II)			
Group II	Springvale, Willagee	1857 ± 2	18km	2km
Group I	Panton	1856 ± 2	11km	2km
	Mafic granulites of Tickalara Metamorphics, pre-dates peak high grade metamorphism	Pre 1850		

6.0 LOCAL GEOLOGY

Bow River West (E80 / 2827)

The Bow River West tenement secures part of the Central Zone of the Halls Creek Orogen terrain. The tenement area has little Palaeoproterozoic bedrock outcrop, with a thin veneer of colluvium present over much of the tenement. A small portion of mafic granulite crops out and appears to be a portion of the Bow River Intrusive which has good exposure to the east of the tenement area. Outcrop of pelitic gneiss in the north and southern tenement areas belong to the Tickalara Metamorphics.

Warmun (E80 / 2866)

The Warmun tenement lies within the Central Zone of the Halls Creek Orogen, west of the Highway Shear Zone, a major crustal feature linked in the north to the Halls Creek and Alice Downs Faults.

West of the Highway Shear, norite, olivine norite and gabbro norite lithologies are present within a large semi circular zone, which potentially represents a mafic – ultramafic intrusion complex ("Faulted Intrusion"). The intrusion is present within a wall rock sequence of pelitic gneiss, granulite and amphibolite of the Tickalara Metamorphics. East of the structure, the tenement covers pelitic gneiss and undifferentiated mafic rocks of the Tickalara Formation.

The western portion of the tenement lies north along strike from the Sally Malay nickel mine, and is considered prospective for potentially economic, nickel copper mineralised gabbro – pyroxenite intrusives. This Sally Malay Corridor represents a major exploration target for mineral exploration companies working within the district.

Killarney (E80 / 2867)

The Killarney tenement covers the eastern boundary of the Central Zone of the Halls Creek Orogen. The boundary is defined by the Alice Downs Fault, a major crustal feature linked in the north to the Halls Creek Fault Zone. Stratigraphy and fabric of the

rock package within the tenement area parallel the NNE strike of this fault. Numerous NNE and NE trending subsidiary structures cut the stratigraphy on the eastern side of the fault.

West of the Alice Downs Fault, the tenement comprises monzogranite, granodiorite and tonalite of the Sally Downs Supersuite. Texas Downs Formation sediments lie east of the Alice Downs Fault. Along the fault, mafic – ultramafic lithologies of the Tickalara Metamorphics are present within structurally controlled domains.

The tenement lies north along strike from known nickel copper bearing gabbro – pyroxenite intrusives. This trend, known locally as the "Corkwood Corridor" represents a major exploration target for mineral exploration companies working within the district.

Nortons (E80 / 2746)

The Nortons tenement lies within the Central Zone of the Halls Creek Orogen, west of the Highway Shear Zone, a major crustal feature linked in the north to the Halls Creek and Alice Downs Faults. Stratigraphy and fabric of the rock package within the tenement area parallel the NNE strike of this fault.

Within the tenement, mafic lithologies of the Tickalara Metamorphics are present within structurally controlled domains. No mafic intrusive units have been identified to date however pyroxenite – bearing gabbros of the Nortons Intrusion lie along the eastern tenement boundary. East of the structure the tenement comprises monzogranite, granodiorite and tonalite of the Sally Downs Supersuite.

Keller Creek (E80 / 2836)

The Keller Creek tenement lies within the Central Zone of the Halls Creek Orogen west of the Highway Shear Zone. Stratigraphy and fabric of the rock package within the tenement area parallel the NNE strike of this fault.

West of the Highway Shear, mafic – ultramafic lithologies of the Tickalara Metamorphics are present within structurally controlled domains. East of the structure, the tenement comprises monzogranite, granodiorite and tonalite of the Sally Downs Supersuite.

The tenement lies northwest of the Sally Malay nickel mine, and is considered prospective for intrusive - hosted nickel copper mineralisation.

The tenement covers known mineralisation at the Keller Creek, Nortons and Six Mile prospects. At the Keller Creek prospect, disseminated and massive nickel sulphide mineralization occurs towards the base of a poorly differentiated, inwards dipping, arcuate shaped, mafic intrusion, located immediately west of a prominent, outcropping granite (NB – the granite hill represents a significant Aboriginal Heritage Site and has been excluded from all exploration activities). Lithologies within the intrusion include gabbronorites, olivine norite, troctolite, pyroxenite with rare serpentinite locally present at the base of the intrusive sequence. Numerous nickeliferous gossans mark the surface position of the sulphide mineralisation. A prominent gossan (the "Footwall Gossan") lies immediately west of the basal contact gossans and is interpreted to represent possible structurally controlled remobilisation of sulphides into the stratigraphic footwall of the Intrusion.

Previous exploration by LionOre / Breakaway has primarily focused on testing down dip of the nickeliferous gossans, using a combination of electrical geophysics, surface

sampling and RC / diamond drilling. The exploration history of Keller Creek prior to LionOre / Breakaway's involvement is best described in Hoatson, 2000.

Mabel Downs and Mabel West (EL's 80 / 2817, & 80 / 2824)

The lenticular Mabel Downs and Mabel West tenement package lie within the Central Zone of the Halls Creek Orogen and along the Highway Shear Zone, a major crustal feature linked to the north to the Halls Creek Alice Downs Faults. Stratigraphy and fabric of the rock package within the tenement area parallel the NNE strike of this fault. West of the Highway Shear, mafic – ultramafic lithologies of the Tickalara Metamorphics are present within structurally controlled domains. East of the structure the tenement comprises monzogranite, granodiorite and tonalite of the Sally Downs Supersuite.

The tenements lie north along strike from the Sally Malay nickel mine and are considered prospective for intrusive - hosted nickel copper mineralisation. Mapping by Hoatson and Blake (2000) has identified three intrusions within the tenements – Mabel Downs, Mabel West, and Frog Hollow. The 6 Mile noritic intrusion also occurs within this area.

Fletcher Creek (E80 / 2865)

The Fletcher Creek tenement secures part of the Central Zone of the Halls Creek Orogen. The tenement area has good outcrop of the Palaeoproterozoic bedrock.

The central tenement corridor is dominated by pelitic gneiss of the Tickalara Metamorphics. Within this sequence a 2.5km lenticular leucogabbro unit crops out. Its north-northeast strike parallels the adjacent Highway Shear Zone. This intrusion has affinities to the Group 5 Sally Malay suite recognized by Hoatson.

East of the Highway Shear the Mabel Downs Tonalite, part of a more extensive suite of felsic rocks that comprise the Sally Downs Supersuite crops out.

Corkwood, North and South (E80 / 2635 & E80 / 2607)

The lenticular Corkwood area lies adjacent to the eastern margin of the Central Zone of the Halls Creek Orogen and covers the Halls Creek Fault Zone, a NNE trending major crustal feature that controls the fabric of the rock package throughout the area.

The central and eastern side of the tenements is dominated by metamorphosed and strongly deformed metasediments and mafic (amphibolite) rocks of the Tickalara Formation, with a broad package of tonalite and granite to the west (Mabel Downs Tonalite). The Halls Creek Fault Zone marks the eastern boundary of the Tickalara package. Locally the NNE trending package of Tickalara Formation rocks is known as the "Corkwood Corridor" and can be traced from the Alice Downs homestead in the south to Texas Downs in the north.

Numerous ultramafic units (pyroxenite, olivine gabbro and gabbro norite) lie within the Tickalara host rocks and are interpreted to represent intrusive phases, although the contact relationships are often unclear. At the northern end of the tenement, surface nickel and copper mineralisation occurs at the Sally Bore prospect within an ultramafic unit. The similarity of Sally Bore mineralisation to other intrusive hosted mineralisation elsewhere within the Corkwood Corridor, and the presence of other relatively under explored intrusions within the tenement make the Corkwood area prospective for nickel copper mineralisation.

Dave Hill (E80 / 2748)

The Dave Hill tenement secures part of the Halls Creek Orogen Central Zone. The tenement encompasses rock units that form part of both the Sally Malay and Corkwood Corridor structural blocks. It straddles the Alice Downs Fault, a major crustal feature linked to the north to the Halls Creek Fault. Stratigraphy and fabric of the rock package within the tenement area parallel the NNE strike of this fault.

The central tenement sector comprises Mabel Downs Tonalite, locally invaded or with remnant enclaves of biotite bearing norite and olivine gabbronorite.

The eastern and western tenement area comprises Tickalara Metamorphics, high-grade granulite metamorphosed sediments and mafic rocks which comprise the basement to the central zone. Within this package several weakly mineralised troctolite, olivine gabbro, gabbro norite and leucogabbro mafic intrusions are recognized (e.g. Fletchers Creek, Frog Hollow & Dave Hill Intrusions). These are up to 0.4 x 6km in extent. Due to the minor fabric displayed these are regarded as post peak metamorphism. Hoatson assigns them to the Group V Sally Malay suite. The western sector also comprises biotite bearing norite and olivine gabbronorite that comprise a more mafic component of the Sally Downs Supersuite.

Potentially significant malachite-stained pyroxenite units have been identified within the eastern portion of the tenement, at the Black Bull Prospect. Further along strike to the north east, secondary copper has also been observed at the Bull Dust Prospect. Mineralisation occurs as fracture-fill within the weathered ferruginous pyroxenite. Drilling conducted at both prospects suggests that the mineralisation can be correlated with weakly disseminated pyrrhotite-chalcopyrite that occurs in the same pyroxenite below the base of weathering.

Eileen Bore (E80 / 2716)

The north-northeast trending Central Zone of the Halls Creek Orogen underlies the Eileen Bore tenement. The southeast boundary of the tenement area approximately coincides with the margin between the Central and Eastern Zones of the Halls Creek Orogen.

Broadly speaking, the western flank of the Eileen Bore tenement area covers the eastern half of the McIntosh Intrusion, classified as a Group VI intrusion (Hoatson and Blake 2000). In its entirety it covers a considerable area of approximately 5 x 16km and is an ovoid shaped body featuring a distinctive ovoid ridge and valley pattern formed by macro-rhythmic cyclic units. Petrology of the intrusion comprises troctolite, and rare peridotite (comprising the valleys) and olivine gabbros, olivine gabbronorite, and gabbro (the ridges).

The central portion of the tenement is underlain by the Dead Finish Tonalite of the Tickalara Suite in the southern half while Wild Dog Creek Gabbro predominates in the north. The eastern flank of the tenement is host to a variety of lithologies belonging to the Tickalara Metamorphics.

Mapping in the northern tenement area identified four main groups of rocks; 1) meta-sediments and mafic granulites of the Tickalara Metamorphics, 2) Ni-Cu mineralized intrusives, 3) McIntosh Intrusion equivalents and 4) Pegmatites (Cotton 2002).

Five known Ni-Cu mineralised intrusions occur within the tenement; Copernicus, Salk, Gauss, Maxwell, and Eileen Bore; all appear to be quite similar. The main component for the Copernicus, Salk and Maxwell intrusives is a cumulate textured pyroxenite with

a differentiated gabbroic cumulate zone to the west. The mineralized intrusives are unfoliated. Gossans have been located at surface within these intrusions.

7.0 REGIONAL MINERALISATION AND EXPLORATION MODELS

The East Kimberley Halls creek Orogen is widely regarded as having excellent potential for magmatic Ni-Cu sulphide and PGE mineralisation, and Hoatson and Blake (2000) considers it one of the most extensively mineralized igneous associations in Australia. Hoatson also recognized broad similarities between the HCO intrusions and the major mineralised layered intrusions at Sudbury, the Bushveld Complex and the Stillwater Complex.

Two mineralised bodies of potential economic interest have been discovered to date within the intrusive complexes of the HCO. These include the Panton resource, with a current resource of 4.5 Moz's of PGE+Au and the Sally Malay Cu-Ni sulphide deposit with a current resource of 3.74 Mt @ 1.72% Ni, 0.72% Cu and 0.09% Co.

The HCO has a number of similarities to the Tornio-Narankavaara (T-N) intrusive belt in northern Finland. The T-N belt contains at least 20 intrusive complexes of varying forms, ranging in area from 1 km² to 200 km². This belt contains the Portimo and Penikat intrusive complexes that are known to host PGE mineralisation of potential economic grade and size.

The mineralisation in the Penikat intrusive is analogous to the PGE mineralisation at the Panton. The Penikat intrusive complex is a 23 km by 1.5-3.5 km sill-like layered intrusive. PGE-bearing chromite (+ sulphide) reefs have been traced along its entire 23 km of strike. The AP1 reef averages 2.0 g/t Pt, 7.1g/t Pd, 0.35 g/t Au with 0.53 % Ni and 0.14 % Cu. The AP2 reef averages 3.4 g/t Pt, 11.4 g/t Pd, 0.60 g/t Au with 0.06 %Ni and 0.20 % Cu. The Portimo intrusive complex occurs as 7 separate fault blocks that originally formed two structurally complex interconnected magma chambers. Significant PGE reserves of 50 Mt containing 3.0 Moz of PGE+Au have been announced.

The proximity of the projects to the Argyle diamond mine indicates a potential for further mineralisation of this type to be present.

A significant gold mining tradition in the area also indicated potential for epigenetic gold mineralisation.

7.1. Nickel - Copper Sulphide Deposit Types

The vast majority of Ni-Cu sulphide deposits are magmatic in origin and are hosted or linked to igneous rocks that formed from magma ranging from ultramafic to mafic in composition. The vast majority of the world's mafic and ultramafic rocks are not associated with any sulphide occurrences however and special processes are required to form magmatic sulphides with nickel and copper and to concentrate them into economic deposits. Three processes are viewed as critical. First is the generation of suitable sulphur undersaturated magma from a mantle derived source. Secondly the host magma has to have been contaminated by crustal material with resulting sulphur saturation and the formation of an immiscible sulphide melt scavenging copper and nickel from the magma. Thirdly the presence of a suitable physical trap site is required to allow the dense sulphide melt to separate from the rest of the magma body and form an ore body.

Within the Halls Creek Orogen all three critical processes have occurred. Intrusions of Group 1-6 of Hoatson and Blake (2000), being almost all of the very numerous intrusions in the HCO, are sulphur saturated and thus highly prospective for magmatic sulphide deposits. Some 180 orthomagmatic Ni, Cu, and PGE occurrences are recorded from within the HCO (Hassan 2001). The presence of these sulphide occurrences is also strong evidence that many host intrusions within the East Kimberley have fertile sulphide contents. The Sally Malay resource demonstrates these sulphide occurrences can be focused into economic deposits.

7.2. Nickel Deposit Models of Relevance to the HCO

The Sally Malay Intrusion (held by Sally Malay Mining Ltd) hosts the largest Ni-Cu-Co sulphide resource with in the East Kimberley with an indicated and inferred resource of the Sally Malay deposit of 3.85Mt @ 1.79% Ni, 0.73% Cu and 0.1% Co (Shedden and Barnes 1996). This deposit commenced production in 2004. The Sally Malay complex is a multichambered body 1.5 x 3km in extent. Four sub chambers are linked by a network of feeder conduits within migmatites of the Tickalara Metamorphics. Primary mineralisation consists of two steeply dipping massive sulphide lenses 220m by 20m in thickness. Most sulphides are hosted within the basal contact zone of sub chamber A within gabbroic rocks, predominantly microgabbro and norite with troctolite present within other portions of the intrusion. Hoatson and Blake (2000) have commented on the similar tectonic, stratigraphic and mineralisation features between Sally Malay and the world class Voisey's Bay deposit in Labrador, Canada. Within the East Kimberley the Voisey's Bay deposit provides a robust and realistic model to guide exploration strategies and targeting. Importantly the Voisey's Bay and Sally Malay models indicate even small intrusive bodies can host giant nickel deposits and that mineralisation may not outcrop at surface.

Sourced from Phyland (2004).

8.0 EXPLORATION ACTIVITIES

Exploration activities in the East Kimberley Joint Venture project included the following:

- Reverse Circulation drilling on the Keller Creek prospect with 7 holes being drilled for a total of 1150m.
- Ground truthing of geophysical anomalies on EL's 80/3323 and at the Gauss prospect.
- A high-resolution aeromagnetic survey was flown over the northern tenement block of the project area.

Exploration activities are shown on the Index Plan (Figure 9) and drillhole collars are shown in Figure 10.

8.1. RC sampling and analytical methodology

The RC drillholes were completed by Arrinooka Drilling All Reverse Circulation drill hole results were obtained from analysis of 1 metre samples. Sample quality in all cases was good and dry. All samples were prepared and analysed at UltraTrace's Perth facility using dual stage crush and pulverisation. Analysis was by four acid digest and ICP/OES finish (ICP102, Ni, Cu, Cr) and fire assay/ICPMS finish (FA003, Au, Pt, Pd).

The location of drill holes were determined using a handheld GPS achieving +/- 4-metre accuracy and reading AMG coordinates (AGD datum). Downhole surveys were obtained every 30 metres downhole using an Eastman single shot survey camera.

8.2. Keller Creek RC drilling

Seven Reverse Circulation holes (1,150 metres – 08BEKC0014 to 08BEKC0020) were drilled to test the significance of several historical DHTEM/FLTEM conductors present within the basal contact zone of the Keller Creek Intrusions where sulphides were first discovered in the early 1970s. Additionally the holes also tested the continuity of the basal contact massive sulphide horizon away from existing pierce points. All digital data is contained in Appendix 1 and the geological legend is in Appendix 2.

All holes intersected sulphides, mostly pyrrhotite with some pyrite on the basal contact of the Keller Creek Gabbro. Sulphides were also intersected higher in the gabbro associated with silicification, possibly indicating that they have been remobilized. Pyrrhotite, including semi-massive to massive mineralisation was also intersected in the footwall Tickalara Metamorphics, both mafics and metasediments in 08BEKC0017, 18, 19 and 20. A collar location plan showing intrusion geology and EM conductor plates is shown in Figure 4.

The intersections are summarised in Table 4 and each hole is summarised below.

08BEKC0014 & 15

These two holes were drilled on the same section and in the same direction to test an interpreted down-hole EM feature on the basal contact of the Keller Creek mafic body (Gabbro). The holes intersected a sequence very similar to that interpreted from earlier drilling. There are two mineralised intersections in each hole; the Upper Sulphides and the Basal (contact) Sulphides. However, in both holes these intersections were approximately 30m higher than was expected suggesting that the sequence had been raised by faulting.

The mineralisation in the Upper Sulphides showed a metre intersection massive sulphide within disseminated sulphide. The Basal Sulphides were composed of disseminated and matrix sulphides. The footwall Tickalara Metamorphics on this section are fine grained mafic composed of hornblende, quartz and biotite.

The sulphides intersected were sufficient to have caused the EM response.

The following significant results from 08BEKC0015 were returned:

- 2.0m @ 1.04%Ni, 0.17%Cu from 71 metres,
- 2.0m @ 1.00%Ni, 0.10%Cu from 94 metres,
- 2.0m @ 1.03%Ni, 0.18%Cu from 99 metres.

08BEKC0016 & 17

These holes are located approximately 80m west of holes 14 and 15 but drilled vertically and steeply to the south. As with the two previous holes, Upper and Basal Sulphides were intersected but the quantity of sulphides was not as great, particularly in hole 16 where it only reached an estimated 10 to 15% although massive pyrrhotite and pyrite were intersected in hole 17 and 15 to 20% disseminated sulphides were intersected in the footwall mafics. The quantity and style of sulphides in hole 17 at least were sufficient to have caused the down-hole EM feature.

08BEKC0018

Hole 18 intersected three bands of matrix sulphides, see Table 4, although the intersections in the Upper Sulphides were probably one intersection that had been separated by quartz veining. The sequence and the mineralisation are in keeping with historical hole THX_05, shown on the section as being collared 22m to the north that intersected the mineralisation approximately 60m up-dip. The sulphides intersected in hole 18 were sufficient to have caused the EM feature.

08BEKC0019

This hole was drilled at 80° to the east and inters ected two bands of massive sulphide at the basal contact of the Gabbro and within the footwall metasediments. These were at slightly shallower depths than was anticipated from the earlier drilling. Because of the topography, this hole had to be drilled down-dip in order to intersect the mineralisation at this depth. The style of sulphides was different in this hole. They tended to be semi-friable – granular. There was also a grey, non-magnetic sulphide, possibly arsenopyrite. The footwall Tickalara Metamorphics here are composed of very fine grained, foliated, quartz-amphibole-biotite rock interpreted to be a metasediment. The sulphides were sufficient to have caused the EM response.

08BEKC0020

Hole 20 was the surprise hole. It was originally planned to be 120m deep based on the intersection in historical hole THX_KC4 and a surface gossan. However when sulphides were first intersected from 51m to 54m it was apparent that they were high in the Gabbro so the hole was extended until four metres of massive sulphide were intersected between 119m and 123m the latter depth is the contact of the Gabbro and the footwall metasediments. However strongly disseminated sulphides continued to 149m where, once more, matrix sulphides were intersected over four metres. Further strongly disseminated sulphides were intersected between 160m and 163m. Between the above sulphide intersections there are extensive, but lower grade disseminated pyrrhotite. As shown in Table 4 the intersections in hole 20 are impressive and when combined with the lack of drilling in this part of the western side of the prospect are encouraging. The sulphides intersected in hole 20 are sufficient to have caused the down-hole EM feature.

From the drilling, the following conclusions can be drawn:

- The Keller Creek prospect is clearly a large sulphide system albeit of low nickel tenor. The sulphides seen in drill chips were primarily pyrrhotite with pyrite and traces of chalcopyrite. The grey, non-magnetic sulphide seen in 08BEKC0019 could be arsenopyrite or simply non-magnetic pyrrhotite.

- The mineralisation seen on the western side of the prospect in holes 19 and 20 looked interesting and subject to looking at the analyses could well justify further drilling. Specifically 30m in front of, behind 08BEKC0020, and about the same distance on either side.

- Quartz biotite pegmatite was intersected in most holes, commonly in the area of the gabbro's basal contact and in the footwall Tickalara Metamorphics.

Table 4. East Kimberley Project - Keller Creek RC Drilling
Summary of Mineralised Intersections (logged with ≥20% Sulphide)

Hole 08BEKC00-	Depth From	Depth To	Intercept	Description	Position
14	48	49	1	30% Disseminated Po	Upper Sulphides
	49	50	1	50% Massive Po	Upper Sulphides
	50	51	1	25% Disseminated Po	Upper Sulphides
	64	65	1	20% Disseminated Po	Upper Sulphides
	73	**74**[1]	1	20% Disseminated Po	Basal Sulphides
15	71	73	2	Massive to Semi-massive Po	Upper Sulphides
	93	95	2	25% Stringer Sulphides	Basal Sulphides
	98	**101**	3	30% Matrix/Semi-massive	Basal Sulphides

[1] Depths shown in **blue** are at the basal contact of the gabbro.

Hole 08BEKC00-	Depth From	Depth To	Intercept	Description	Position
16	-	136	-	-	-
17	160	161	1	Massive Po	Upper Sulphides
	161	163	2	~30% Matrix Sulphide	Upper Sulphides
	165	166	1	~30% Matrix Sulphide	Basal Sulphides
	166	167	1	~50% Semi-massive Po	Basal Sulphides
	203	204	1	~20% Disseminated Po	Footwall Sulphides
18	81	85	1	~30% Matrix Po & Py	Upper Sulphides
	86	88	2	40% Matrix Po	Upper Sulphides
	91	94	3	40% Matrix & Stringer Po	Basal Sulphides
19	109	110	1	Massive Po, Py & ?Apy	Basal Sulphides
	115	120	5	Massive Po, Py & ?Apy	Footwall Sulphides
20	51	52	1	Massive Po	Upper Sulphides
	52	54	2	30-40% Matrix Po	Upper Sulphides
	119	123	4	Massive Po with Py	Basal Sulphide
	123	130	7	20% Disseminated Po & Py	Footwall Sulphides
	130	132	2	Matrix – Diss. Po & Py	Footwall Sulphides
	132	133	1	20% Disseminated Po & Py	Footwall Sulphides
	139	149	10	25% Disseminated Po & Py	Footwall Sulphides
	149	153	4	40% Diss. – Matrix Po & Py	Footwall Sulphides
	160	163	3	25% Disseminated Po	Footwall Sulphides

Using a 0.5%Ni cut off, the following significant intercepts were obtained.

Table 5 Significant intersections

Hole Id	Northing	Easting	From	Width	Ni%	Cu%	Co%	weighting	Target
08BEKC0014	8086483	390520	49.0	1.0	0.53	0.16	0.05	length	HW Zone
			58.0	3.0	0.59	0.22	0.06	length	HW Zone
			73.0	1.0	0.78	0.16	0.07	length	basal
08BEKC0015	8086489	390520	69.0	4.0	0.80	0.23	0.10	length	HW Zone
			including						
			71.0	2.0	1.04	0.17	0.12	length	
			82.0	1.0	0.55	0.21	0.07	length	HW Zone
			93.0	8.0	0.72	0.24	0.08	length	basal
			including						
			94.0	2.0	1.00	0.10	0.11	length	
			99.0	2.0	1.03	0.18	0.11	length	
08BEKC0016	8086500	390600	NSA						
08BEKC0017	8086500	390600	129.0	1.0	0.59	0.43	0.05	length	HW Zone
08BEKC0018	8086451	390480	82.0	3.0	0.70	0.30	0.09	length	HW Zone
			87.0	1.0	0.92	0.10	0.12	length	HW Zone
			91.0	3.0	0.75	0.15	0.10	length	basal
			including						
			91.0	1.0	1.06	0.18	0.13	length	
08BEKC0019	8086400	390450	NSA						
08BEKC0020	8086328	390418	119.0	4.0	0.78	0.38	0.08	length	basal

The drill results confirm the occurrence of broad nickel-copper-cobalt system in which the mineralisation is mainly concentrated towards the base of the Keller Creek intrusion. The drill results confirm that the known mineralisation covers an area measuring 600 metres and over 300 metres down-dip on the northwestern margin of the intrusion. The margins of the system have not yet been defined and there is every probability that it could be more extensive. In addition, there is every potential for zones of greater sulphide concentration, which remain to be discovered.

Follow-up downhole TEM geophysics carried out on selected holes has identified several additional conductors, which are under evaluation (geophysical data is contained in Appendix 3).

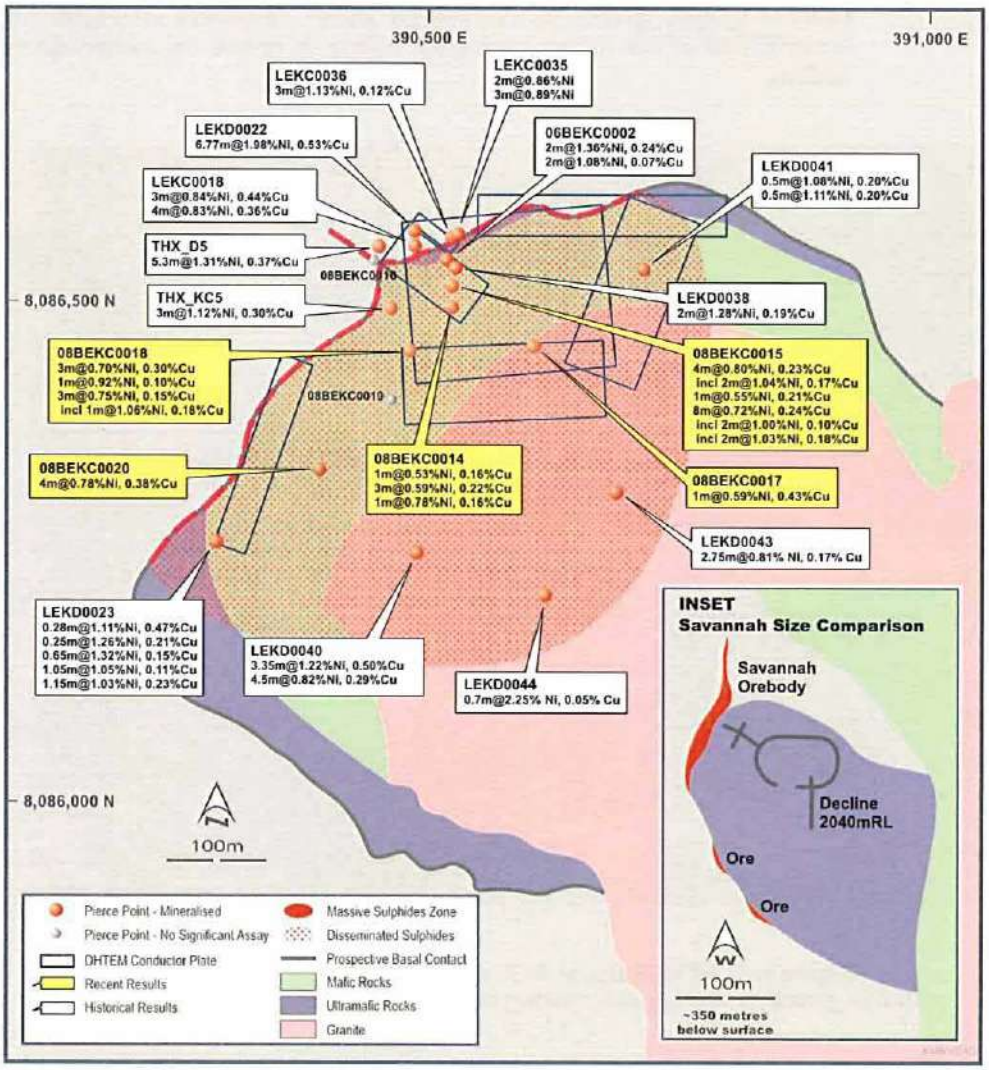


Figure 4. Keller Creek Collar Location Plan
Collar Location Plan showing intrusion geology and EM conductor plates

8.3. Ground Reconnaissance

8.3.1. E80/3323 Aeromagnetic Feature

The aeromagnetic feature was visited on 13th July 2008. It is located 4.5km west southwest of the Keller Creek prospect and is coincident with what is mapped on the Mount Remarkable sheet as Cainozoic/Recent sediments in a valley, see Figure 5. The centre of the feature is narrow, steep-sided valley, (Figures 6 and 7).

The feature has been caused by the accumulation of colluvial magnetite in the sediments that cover the valley floor. Magnetite could be readily picked-up on a pen magnet on the thin (? centimetre-scale) terraces on the sides of the valley-fill. The source of the magnetite is from the quartz-biotite-magnetite gneiss of the valley's sides. Sufficient magnetite has accumulated in the recent geological history to create the aeromagnetic feature.

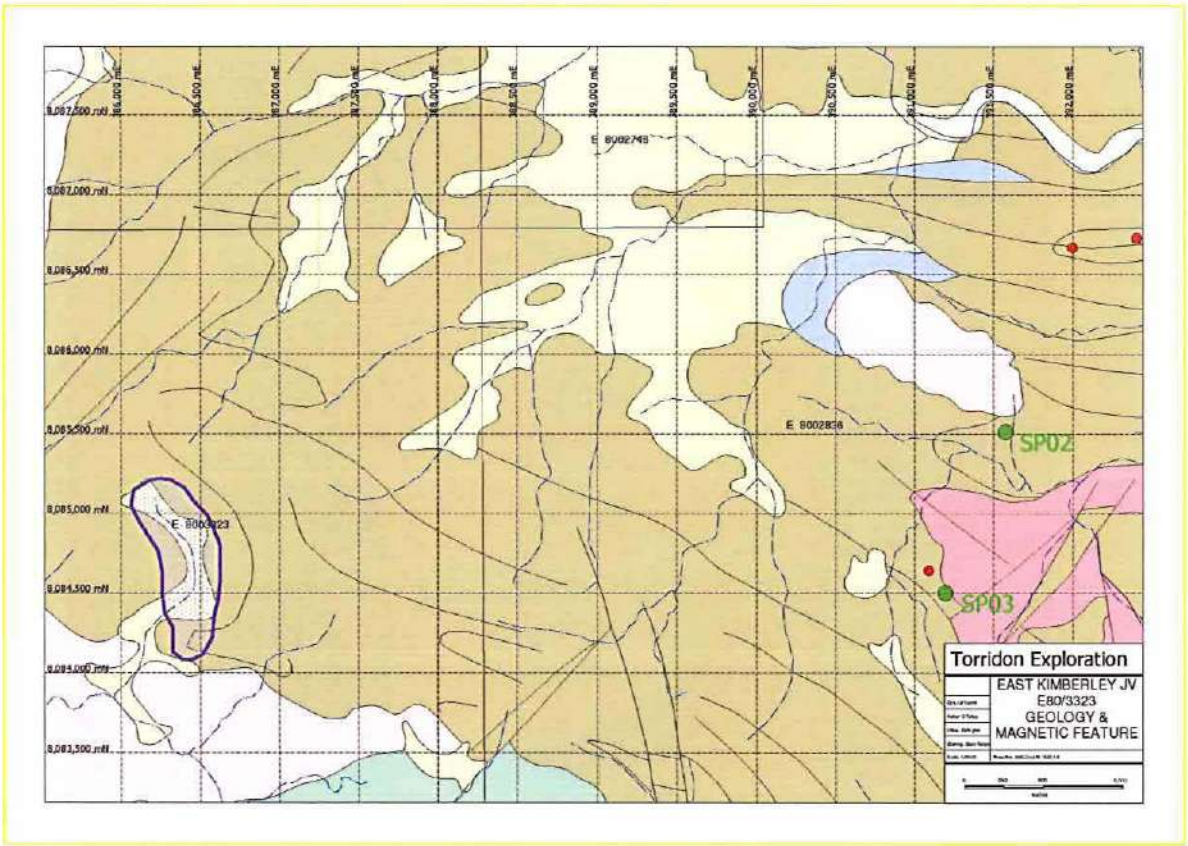


Figure 5. EL80/3323 Magnetic Feature Geology
GSWA geological plan showing location of magnetic feature (purple outline).



Figure 6. EL80/3323 Magnetic Feature Photograph
Photograph of topography at magnetic feature.

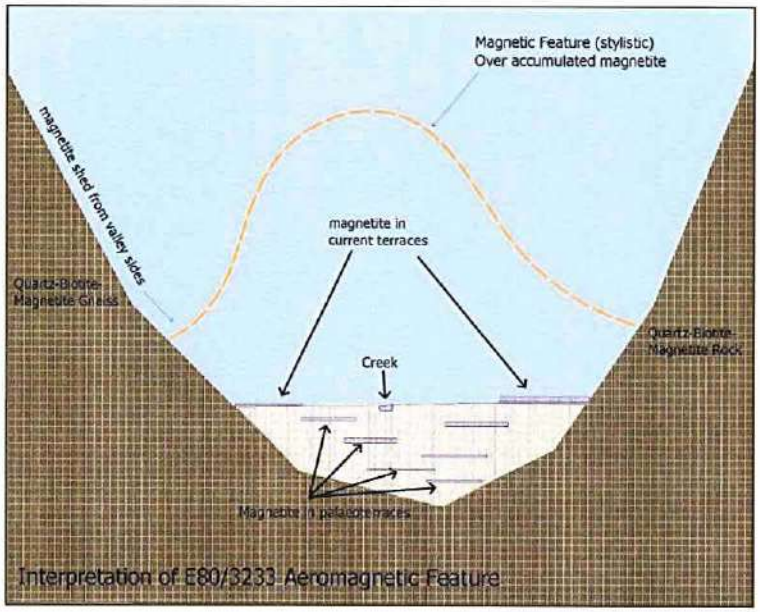


Figure 7. EL80/3323 Magnetic Feature Section
Schematic section through magnetic feature.

8.3.2. E80/3878 (M80/580), Gauss EM Feature

The interpreted EM feature from the 2007 fixed loop survey at the Gauss prospect appears to be caused by graphite within calcsilicate horizons, of which there are many in the area. The source of the feature was interpreted as being 20m below the surface at (AGD 84, AMG Zone 52) 391 850E, 8 047 650N. This point is between two small (<1m) calcsilicate horizons on a west-facing slope overlooking the access track that heads south from Panoramic Resources Ltd's Copernicus haul road.

No further work is recommended on this feature.

8.4. Aeromagnetics

The high-resolution aeromagnetic survey covering the northern tenement block was completed by GPX Airborne. As shown in Figure 9, the survey data offers a significant improvement in spatial resolution compared to the previously available data sets. Also evident is a leveling error that has resulted in a "stripey" nature to the imagery. GPX have reprocessed the aeromagnetic data in an attempt to remove / reduce the effects of leveling error.

The survey data has been lodged into the GSWA MAGIX database and its registration number is 70033.

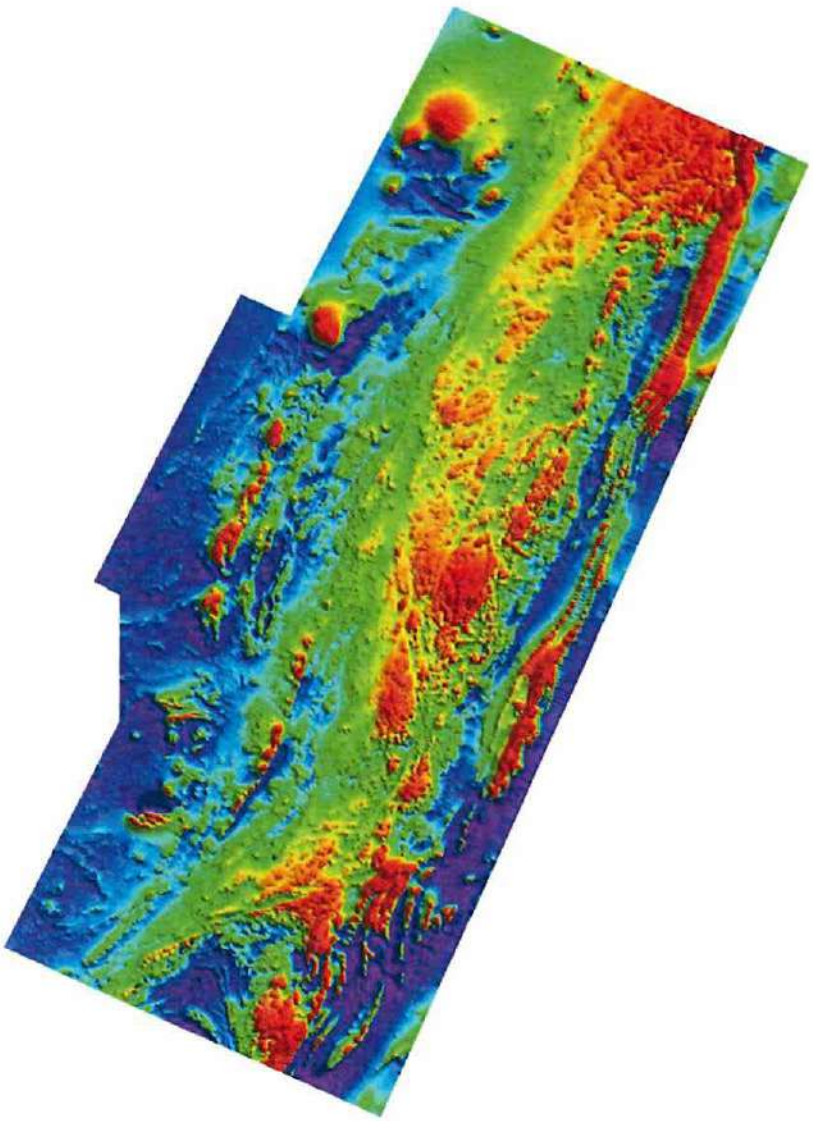


Figure 8. 2008 Aeromagnetic Survey.
TMI image

Figure 9 Exploration Index Plan

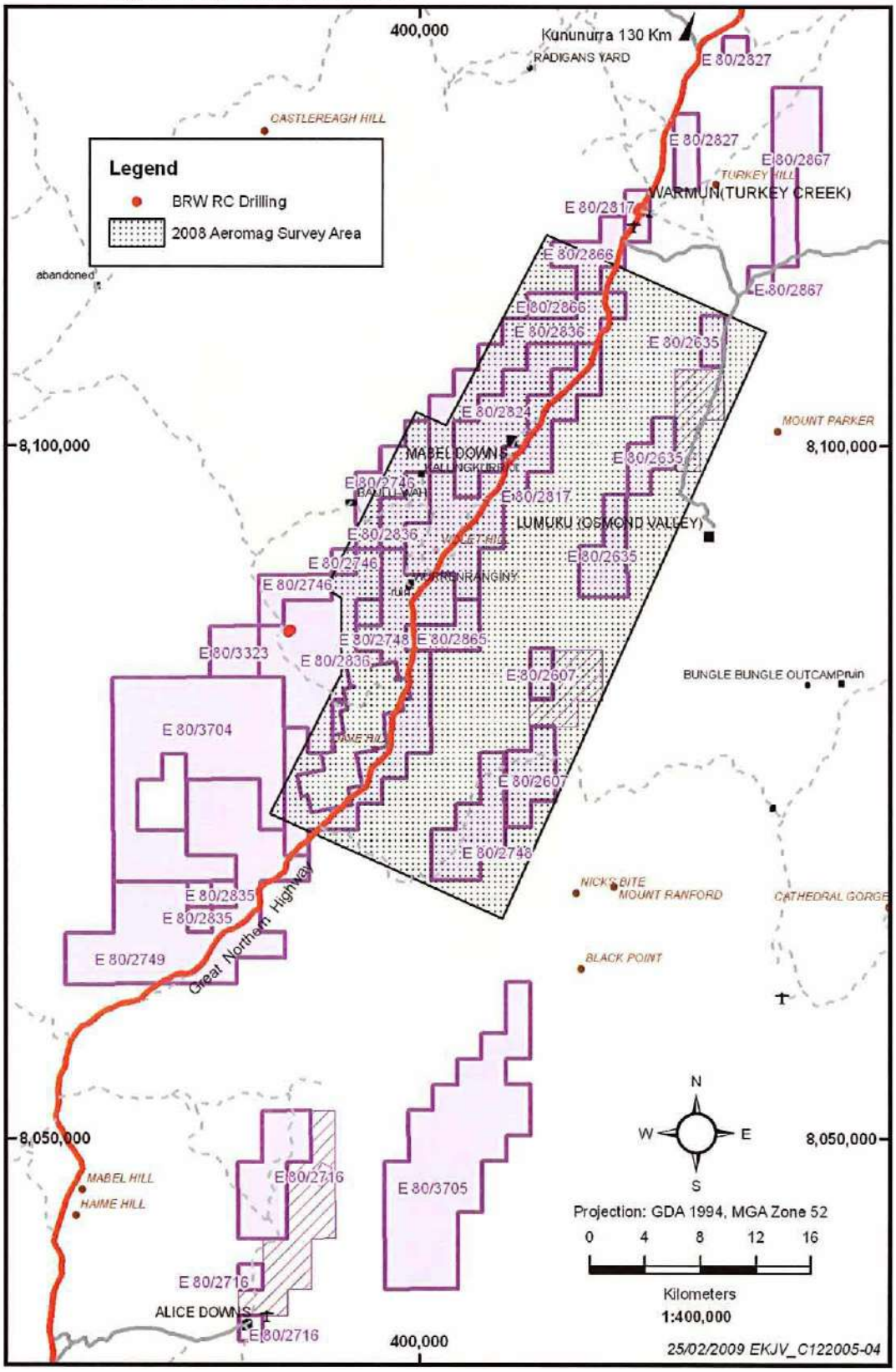
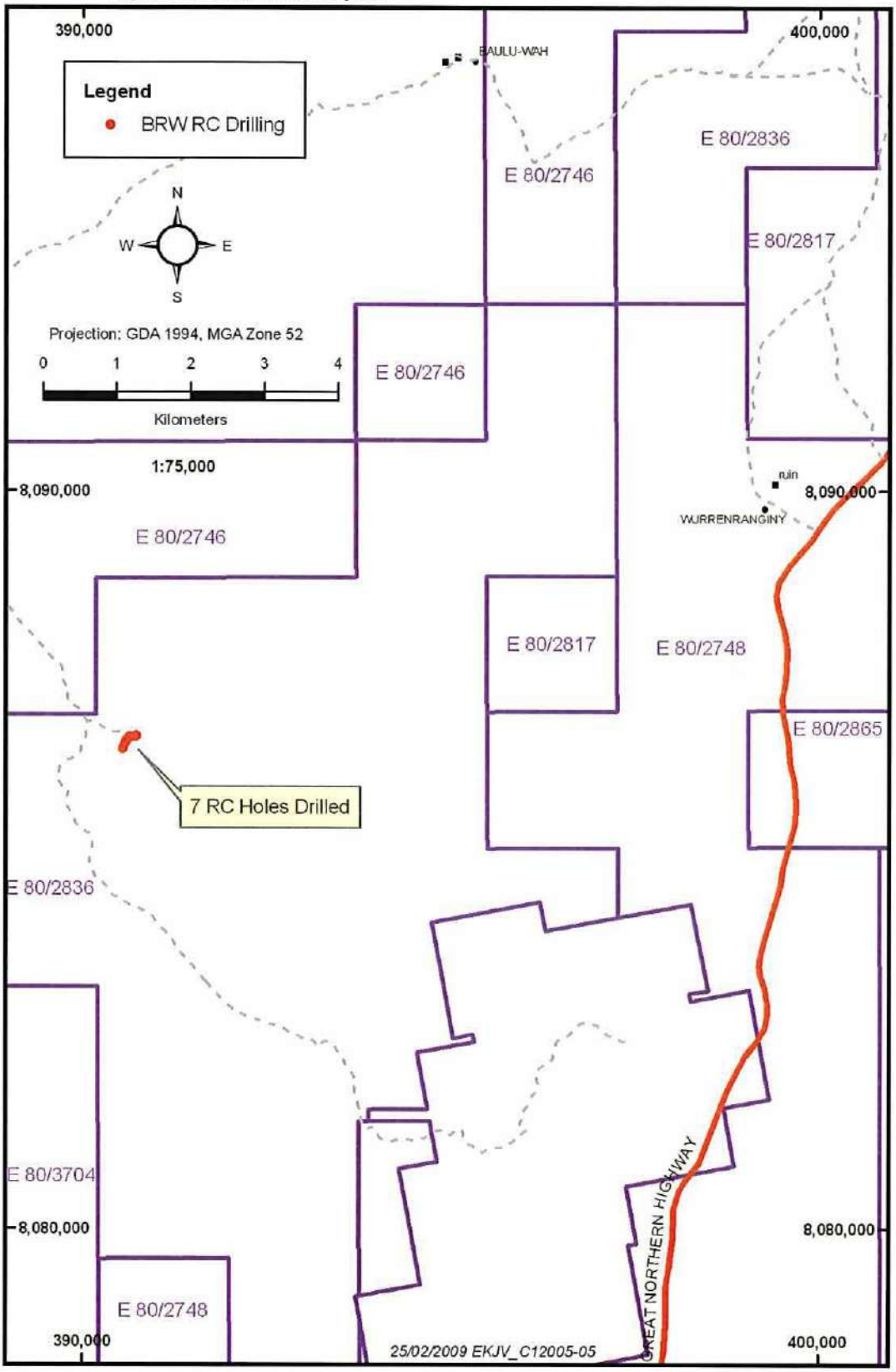


Figure 10 Drillhole collar plan



9.0 CONCLUSIONS AND RECOMENDATIONS

Exploration activities in the East Kimberley Joint Venture project included the following:

- Reverse Circulation drilling on the Keller Creek prospect with 7 holes being drilled for a total of 1150m.
- Ground truthing of geophysical anomalies on EL's 80/3323 and at the Gauss prospect.
- A high-resolution aeromagnetic survey was flown over the northern tenement block of the project area.

The drill results confirm the occurrence of broad nickel-copper-cobalt system in which the mineralisation is mainly concentrated towards the base of the Keller Creek intrusion. The drill results confirm that the known mineralisation covers an area measuring 600 metres and over 300 metres down-dip on the northwestern margin of the intrusion. The margins of the system have not yet been defined and there is every probability that it could be more extensive. In addition, there is every potential for zones of greater sulphide concentration, which remain to be discovered.

The aeromagnetic survey provides a level of data resolution not previously available for the East Kimberley and will underpin ongoing exploration activities for the foreseeable future.

Future work on the project should be directed towards interpreting the aeromagnetic survey, and establishing the significance of the recently obtained nickel sulphide intersections at Keller Creek.

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Valuation of mineral properties which do not contain exploitable reserves

Lionel C. Kilburn

ABSTRACT

This geological engineering method is based on four main characteristics of mineral properties, viz; location; inclusion of valuable mineralization; inclusion of geophysical and/or geochemical targets; and inclusion of geological targets. These are subdivided into a total of 19 subcategories, which are used to determine the value of the property.

The nineteen subcategories are prioritized and assigned relative value factors from 1.3 to 10. Value of each mineral claim-unit is determined by applying the appropriate subcategory factors to a base vaue of $400 per 16.2 hectare unit. Property value is calculated by totalling the values of such individual claim-units.

The boundary between ore grade and sub-ore grade must be estimated for each property under valuation, thereby making the method sensitive to mineral commodity prices.

A clear and disciplined sequence of decisions based on personal experience forms the basis of this method. The method utilizes all factors normally used to value a mineral property.

Introduction

It is generally accepted in the exploration industry that valuation of exploration properties is a subjective rather than objective process (Roscoe, 1986; Lawrence, 1988; Lawrence, 1989; Woodcock, 1985), and that it is greatly influenced by the experience and qualifications of the person who is doing the valuation. However, the significantly low degree of objectivity in mineral property valuation at this point in time invites effort to achieve a more quantitative engineering basis.

Value of a mineral property depends basically upon metal prices, recoverable grades, mining and metallurgical practices, commodity supply and demand, and political-economic conditions, all of which vary with time. Therefore, any valuation is valid for only that point in time when it is made. Mineral deposits can decrease as well as increase in technical value with time.

Five fields of expertise influence the value of a mineral property:

1. Geological engineering
2. Commodity markets
3. Financial markets
4. Stock markets
5. Mineral property markets.

The skills of geologists and engineers properly apply to the first field of expertise. Knowledge and consideration of the other four fields is important, but such advice is more properly sought from marketing experts, economists and stock brokers. It is important that any or all of these five fields should be applied separately, and that any value assigned because of any of them should be identified separately from geological engineering. The method of mineral property valuation described here restricts itself to geological engineering considerations, for properties which do not contain economically exploitable mineral deposits.

The purpose here is not to restrict the individual's method of valuation, but to discipline his/her thought process, to require each geologist or engineer to analyze his/her method, and to determine the specific reasons why value is assigned.

Keywords: Property valuation, Exploration, Economics.

Paper reviewed and approved for publication by the Geological Society of CIM.



Lionel C. Kilburn received a B.Sc. (Honours) in geology, an M.Sc. and a Ph.D. from the University of Manitoba. He is the author of several scientific papers, received a Gold Medal for outstanding research in geology in 1953 and was the recipient of an Inco Fellowship from 1954 to 1956.

Dr. Kilburn began his career at Falconbridge Mine in 1956 as a geologist, became assistant chief geologist at Falconbridge Nickel Mines Ltd. in 1966 and chief geologist in 1968. In 1969, he was promoted to assistant vice-president, Exploration and Development, and became president and chief executive officer of Falconbridge Copper Ltd. (later Corporation Falconbridge Copper) in September 1976. In September 1982, he took on the responsibilities of vice-president, Exploration, Development and Precious Metals Operations, with corporate responsibility for United Keno Hill Mines Ltd., Giant Yellowknife Mines Ltd., Western Platinum Ltd., Kiena Gold Mines Ltd., and Falconbridge Zimbabwe. He currently performs the duties of prospector, consulting geologist, advisor to the Ontario Securities Commission, technical consultant to the Toronto Stock Exchange, managing editor of *PDAC Digest* and director of Giant Yellowknife Mines Limited.

Dr. Kilburn is a member of CIM; the Associations of Professional Engineers of Ontario and Manitoba; the Geological Association of Canada; he was president of the Prospectors and Developers Association of Canada from 1985-1987; and president of the Engineers Club of Toronto from 1987-1988.

Valuation of Mineral Property Characteristics

The method proposed here is based on the four main characteristics which most engineers and geologists use consciously or subconsciously when valuing a mineral property. This method allows the user to apply his/her experience and knowledge obtained from the examination of other mineral deposits to assess the importance of each of the characteristics used. These four mineral property characeristics are:

1. **location** with respect to any off-property, mineral occurrence of value; or any favourable geological, geophysical, and/or geochemical pattern;
2. **grade** and amount of mineralization known to exist on the property under valuation;
3. **geophysical and/or geochemical targets** present on the property under valuation and their position with respect to one another;
4. **geological patterns** present on the property under valuation, which may be considered favourable for the occurrence of exploitable mineralization, based on previous experience or comparison with known mineable deposits.

These four major characteristics may be subdivided further into 19 subcategories as follows:

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Location of Property Under Valuation with Respect to Known Off-property Mineral Occurrence(s)

The off-property mineral occurrence(s) may contain:

1. interesting* but sub-ore grade material, that has been measured in two horizontal dimensions;
2. ore grade** material, that has been measured in two horizontal dimensions;
3. an interesting but sub-ore grade mineralized zone, that has been measured in three dimensions;
4. an ore grade mineralized zone, that has been measured in three dimensions, (not yet shown to be economically exploitable);
5. a mine — past or present producer;
6. a major mine — past or present producer. The distiction between a mine and a major mine is made here, only to recognize the "giants" of the industry such as Sullivan, Horne, Kidd Creek, Flin Flon, Hollinger, and Hemlo. Experience has shown that greater value is attributed to their proximity, than to that of lesser deposits.

Location of Property Under Valuation with Respect to Known Off-property Geological, Geophysical and/or Geochemical Targets

These targets are similar to those indicative of known exploited mineral deposits of which:

7. one such target exists or two such targets based on different methods correlate with one another;
8. three or more such targets correlate with one another.

Grade of Mineralization on the Property Under Valuation

Such mineralization is:

9. interesting but sub-ore grade material, which has been measured in two horizontal dimensions;
10. ore grade mineralization, which has been measured in two horizontal dimensions, at a size which is economically interesting;
11. an interesting but sub-ore grade zone, which has been measured in three dimensions;
12. an ore grade mineralized zone, which has been measured in three dimensions, at a size which is economically interesting (not yet shown to be economically exploitable);
13. a mine — past producer, with ore grade mineralization which has been measured in three dimensions, at a size which is economically interesting (not yet shown to be economically exploitable);
14. a major mine — past producer, with ore grade mineralization which has been measured in three dimensions, at a size which is economically interesting (not yet shown to be economically exploitable).

Geophysical and/or Geochemical Targets on the Property Under Valuation

These targets are similar to those indicative of known exploited mineral deposits, and of which there is/are:

15. one such geophysical/geochemical target;
16. 2 to 3 such geophysical/geochemical targets which correlate with each other;
17. 4 or more such geophysical/geochemical targets which correlate with each other.

Geological Patterns† on the Property Under Valuation

These patterns are recognizeable compared to those associated with known exploited mineral deposits:

18. one or two such patterns;
19. three or more such patterns.

Prioritization of Characteristics

The 19 sub-categories must be assigned a relative importance with respect to each of the others. For example, the following prioritization is recommended here:

14 > 13 > 12 > 6 = 11 > 5 > 17 > 4 = 10 = 16 = 19 > 3 > 2 = 9 = 15 = 18 > 1 = 8 > 7

The relative importance which is attributed to each subcategory will be a function of personal experience, knowledge and preference. Some people may prefer to change these priorities, for example to reverse subcategories No. 11 and No. 5 or No. 17 and No. 4. However, each geologist and engineer must set down his/her own set of priorities, which will be assigned value factors later.

For example, the prioritization used in this text maintains:

- that an "ore grade mineralized zone on the property" (No. 12) is better than being located well with respect to a "Major Mine — past or present producer" (No. 6).
- that being located well with respect to a "Major Mine .. etc." (No. 6) is equal to having a three dimensional "interesting but sub-ore grade" mineralized zone on one's property (No. 11);
- that having three or more geophysical and/or geochemical anomalies that correlate well with one another (No. 17) is better than being located well with respect to a three dimensional ore grade but unexploitable zone" (No. 4).
- that "three or more positive geological patterns" on one's property (No. 19) is better than being located well with respect to a "three dimensional sub-ore grade mineralized zone" (No. 3);
- that a "two dimensional, interesting, but sub-ore grade" showing on one's property (No. 9) is equal to having geophysical or geochemical anomalies on one's property which do not correlate with one another (No. 15), which in turn is equal to having one or two positive geological patterns on one's property (No. 18);
- and so on.

Assignment of Value Factors

Once the relative importance (prioritization) of each subcategory has been established, then one must determine value

*"Interesting" is used in this text to describe sub-ore grades which are at least one half of those exploitable at the time of valuation, based on quantities and dimensions reasonably expected to yield an economically exploitable deposit.

**Ore grade material is not necessarily "ore". It is a descriptive term which indicates the amount of a commodity required in a body of rock, which has size and dimensions reasonably expected from available measurements.

†*Geological Pattern*—the distribution of certain rock types; their size, shape and contacts; their alteration; their structural features such as folds and faults; without limiting the number of geological features which frequently have been recognized to be associated closely with certain mineral deposits. "Patterns" has been chosen as a general term, so as not to limit the possibilities. Certain patterns are accepted as better indicators of mineralization than others, at different times in history.

factors to be used in relating the subcategories to one another. The following factors, which reflect the priorities chosen above, are recommended as follows:

Subcategory	Factor	Subcategory	Factor
1	1.5	11	5.0
2	2.0	12	6-8
3	2.5	13	7-8
4	3.0	14	9-10
5	4.0	15	2.0
6	5.0	16	3.0
7	1.3	17	3.5
8	1.5	18	2.0
9	2.0	19	3.0
10	3.0		

It should be noted that the size of a factor must be consistent with its priority which was decided previously. For example, in the priority 12 > 6 the respective factors are 6X-8X and 5X, whereas in the priority 6 = 11 the respective factors are equal at 5X.

Each individual geologist or engineer may modify these factors based on personal experience, but always keeping in mind the priorities he/she has assigned to them. *Each subcategory MUST be compared and rated independently with EVERY other subcategory, in isolation from the others.*

Base Cost

The value of a property is a function of its size, *viz:* the larger the property, the more room there is to find a mineral deposit, all other things being equal. Most properties consist of mineral claims and the most common claim size in Canada is approximately 16.2 hectares (40 acres). A unit equivalent to 16 hectares is used here with a base value related to acquisition cost. Even though it changes with location and terrane, the basic acquisition cost should approximate average Canadian prospecting and staking costs. It is recommended here that a fair *base cost* which is reflective of country-wide operations would be $400 per 16 hectare unit. The 16 hectare size is only a "unit" to be factored into the actual size of a claim in any particular jurisdiction. For example, in areas where claims are 48 hectares, the base cost would be $1,200 per claim. However, valuation is recommended on units of 40 hectares or less, to prevent unreasonable assignment of localized value over broad areas.

Periodically, this base cost may be adjusted for inflation. However, the same base cost must be used for all current valuations, even though actual costs vary from location to location. There must be a constant basis for comparison, which will give rise to comparable valuations across the country.

Valuation

The base cost is factored by using the subcategories that apply to the property under consideration. For example, a 16 hectare unit which is located well with respect to a three-dimensional sub-ore grade mineralized zone (No. 3), which contains a two-dimensional sub-ore grade showing (No. 9), which contains EM anomalies unrelated to other anomalies (No. 15), and which has one or two positive geological patterns (No. 18), would be valued as follows:

Base cost × (SC-3)* × (SC-9) × (SC-15) × (SC-18) = $
$400 × 2.5 × 2.0 × 2.0 × 2.0 = $8,000

The procedure is followed by calculating and assigning a separate value to each claim. For most claim groups, this is a lot less complex than one might expect. For example, many claims which have been staked for "protection" purposes, may have the same rating, little value except for location, and may be grouped for calculation purposes.

Certain claims may have an added value, because they are patented or leased. This added value would arise from security of title, surveyed boundaries, and relief from the need to carry out assessment work in the future. Such added value can be calculated easily for each province, is not a part of the technical valuation, and should be added to the value of a claim *after it has been valued technically.*

Valuation based on location (subcategories No. 1 to No. 3) is a good example of how the experience and qualifications of a geologist or engineer enter the process. Each person must decide how closely a claim must be located to an off-property valuable mineral deposit, in order to apply the "full" factor.

Practice shows that one selects "factors" usually on the basis of clear, distinctive examples from one's past experience. Certain properties may not warrant the full value of such factors, and, therefore, they must be applied on a qualified basis "up to" a maximum, depending upon how closely they meet the standards of experience.

Similarly, comparisons would have to be made for two claims which are equidistant from an off-property valuable mineral deposit, but in one case along strike and in the other case across strike of the mineralized structure. Based on distinctions such as these, each of the 19 subcategories may be subdivided again based on more detailed information. This refinement of the valuation process will be pursued in the future by the author, but for the present, a simple and more easily applied method is recommended to encourage experimentation and adoption by others.

Geologists and engineers must decide on their priorities and their factors for each subcategory, before applying the method.

It is believed that with useage, a consensus of priorities and factors will grow within the industry; if one does not already exist.

Because it is a numerical system, this method can be computer adapted and applied to each claim individually.

Discussion

This method sets a maximum value for a claim. If the highest rating for all categories is given to a claim, its value is as follows:

	Factor	Subcategory
Location	5 ×	6
Grade	10 ×	14
Geophy/geoch	3.5 ×	17
Geology	3.0 ×	19
Total	525.0 × $400 = $210,000	

There are claims which have been sold for a value in excess of this amount. It is argued here that these larger values arise from other than technical factors, such as those listed at the beginning of this paper.

There are 756 combinations possible within the 19 categories presented above. The allotment of this many combinations within the range $400 to $210,000 provides considerable latitude for flexibility, variety, variation and presentation of the individual geologist's or engineer's preferences.

There may be a reluctance on the part of some geologists and engineers to place a monetary value on a claim. In such an instance, this method still can be of value by rating on a comparative point count basis. It then becomes a simple matter of using a point count base value rather than a dollar base value. Thus, the method can be used to place relative non-monetary value on claims (or properties).

The relative importance attributed to each of the main categories by a geologist or engineer in the valuation of a claim (or property) can be obtained by totalling the four value factors and showing

*SC = subcategory

each category on a percentage basis. For example, the maximum rating shown above would be presented as follows:

Location	5	23%
Grade	10	47%
Geophy/geoch	3.5	16%
Geology	3.0	14%
Total	21.5	100%

This rating shows not only the priorities and preferences of the maximum value case, but also the relative importance attributed to each of these categories by the user of the method.

Infrastructure

Value of a mineral property is very much a function of location with respect to infrastructural facilities. From the very beginning of mineral assessment, the explorationist creates a "model" of the minimum size and grade needed for exploitation based on location with respect to infrastructure. Later in the exploration sequence, grade, width, length and depth of mineralization are determined; and parameters of ore and sub-ore are adjusted accordingly. In this manner, the minimum requirements for exploitation are built into the model for that particular location. The boundary of tonnage and grade between ore and sub-ore, may change as exploration progresses, but it exists from the very beginning of prospecting and exploration, in the minds of those who examine and value the property.

Therefore, every explorationist automatically adjusts a property's value to its location with respect to infrastructure, by determining its ore and sub-ore values as required in subcategories No. 9 to No. 12 of the method described here.

Conclusion

In conclusion, it cannot be emphasized enough, that a clear and disciplined sequence of decisions must be made, which separate and identify the reasons for adding or subtracting any value to/from a claim or a property. **"Fair market value"** arises from several fields of expertise, many of which are not geologically or engineering related. There may be good reasons for adding value because of stock market activity, mineral property availability, or commodity price changes (or forecasts); but every valuation should keep them separate and identify them clearly from geological engineering considerations. The geological engineering valuation must be able to stand alone for its own technical reasons.

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International toxic air measurement symposium

"Measurement of Toxic and Related Air Pollutants," an international symposium sponsored by the Air & Waste Management Association and the U.S. Environmental Protection Agency's Atmospheric Research Exposure Assessment Laboratory will be held **May 7 to 10, 1991**, at the Omni Hotel and Convention Center in Durham, North Carolina, U.S.A.

The technical program includes invited and contributed papers that emphasize emerging technology for measuring toxic and related air pollutants (gases and particulates).

Papers are invited on the following topics: hazardous waste emissions, source measurements, quality assurance, ambient measurements, radon measurements, air pollution dispersion modelling, indoor air pollution, exposure assessments, atmospheric chemistry and fate of toxic pollutants, complex chemical data analysis, Superfund site air monitoring, municipal waste combustion emissions, asbestos measurement, remote sensing, automotive emissions, state and local agency toxic monitoring needs, air transport of toxics, and health effects.

Send abstracts of 200 words (typed and double spaced) by **November 30, 1990**, to: Bruce Gay, Atmospheric Research Exposure, Assessment Laboratory, U.S. EPA, Research Triangle Park, NC, 27709, U.S.A., or Dr. R.K.M. Jayanty, Research Triangle Institute, P.O. Box, 12194, Research Triangle Park, NC, 27709, U.S.A.

CIM

Seminar on advances in surface mining equipment performance monitoring

A one-day seminar on Advances in Surface Mining Equipment Performance Monitoring will be presented in conjunction with the Second International Symposium on Mine Planing and Equipment Selection in Surface Mining. The symposium will be held at the Hotel Palliser, Calgary, Alberta, **November 7 to 9, 1990**. This seminar will be held on November 7. The organizers of the symposium are Faculty of Continuing Education, The University of Calgary; *International Journal of Surface Mining and Reclamation* (IJSM) and the Calgary and Edmonton Branches of The Canadian Institute of Mining, Metallurgy and Petroleum (CIM).

This seminar aims to draw together in a workshop environment those engineers and maintenance personnel interested in the application of equipment monitoring technology. Sessions will be led by engineers and researchers currently developing and implementing instrumentation systems on surface mining equipment. Emphasis will be placed on practical application relating to instrumentation, software and interpretational techniques. The working mode of the course aims to promote participation and discussion to facilitate the exchange of experience in technology development and application.

Topics covered include: overview of current technology; future trends; instrumentation for electric and hydraulic blast hole drills; equipment monitoring techniques for electric cable, and hydraulic shovels, front-end loaders, draglines, haul trucks and dispatch systems.

Seminar leaders for this workshop are: Dr. J. Peck, laboratory manager of the Canadian Centre for Automation and Robotics in Mining (CCARM) at McGill University and Dr. M.J. Scoble, professor of mining engineering at McGill University.

For further information and registration, please contact:
Dr. Raj K. Singhal, Symposium Chairman
Faculty of Continuing Education
The University of Calgary
2500 University Drive, N.W.
Calgary, Alberta T2N 1N4
Tel.: (403) 220-6069 Fax: (403) 289-7287

CIM

PJD-39

Multiples of Exploration Expenditure as a Basis for Mineral Valuation

P. G. Onley

ABSTRACT

The "multiple of exploration expenditure" method of mineral valuation is applicable to exploration properties from the earliest stage of exploration to a moderately advanced stage but for which no resources have been delineated. While not recommended as a primary valuation method by some workers, others believe it to be the most satisfactory method of valuing exploration properties until it is possible to employ a DCF technique. In many situations facing the professional valuer, the method may represent the only semi-quantitative option available and is frequently used. Significantly it often forms the starting point in discussions to farm-in on an exploration tenement.

The paper briefly reviews the definition of value as applied to mineral valuation and in particular examines the relationship between fair value and market clearing price in "fair", bull and bear markets.

The multiple of exploration expenditure method uses previous exploration expenditure and/or future committed exploration expenditure to derive a base value for the tenements. This base value is multiplied by a prospectivity enhancement multiplier (PEM) with adjustments for market premium or discount and consideration of the quality of the exploration team to derive a fair value for the tenements. The paper will demonstrate that there is no necessary nexus between *fair value derived from the method and present book* values.

The method is most subject to criticism for the subjective and apparently arbitrary selection of a PEM. The derivation of an appropriate PEM is examined with emphasis on the issues of the relevance and effectiveness of past exploration expenditure and holding costs. The paper will examine current practice and present a semi-quantitative approach to defining such factors, including a proposed model for employing all previous relevant expenditure. Appropriate methods for escalating that expenditure to current dollars are addressed together with consideration of committed and/or proposed future exploration expenditure.

The paper discusses the application of market premium or discount to the base value. It also discusses the importance of an assessment of the quality of the exploration team and its management to the validity of the method, particularly in relation to differences in efficiency and effectiveness of exploration carried out by different teams.

Finally, the limitations of the method are outlined.

Peter Onley graduated from the University of Exeter with the degree of BSc (Hons) in Geology, from the University of Leeds with the Degree of MSc in Engineering Geology and from the University of Western Australia with the degree of MBA.

He is a Fellow of the AusIMM and Vice-Chairman of the Perth Branch, Member of the AIG and a Member and Board Member of MICA.

Peter has twenty five years postgraduate experience including mineral exploration, mining geology and major project evaluation mainly in Australia, but has worked throughout the world. After working for almost fifteen years with CRA Limited he established a Consultancy in Perth in 1985 to provide geological advice to exploration and mining companies, banks and other financial institutions and to other service providers associated with those industries. Peter has prepared numerous independent mineral property assessments, valuations, loan application reviews and independent reports for prospectuses and other corporate compliance documents. Peter joined Mining & Resource Technology Pty Ltd in September 1993 and was recently appointed a director.

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INTRODUCTION

A great many people, both within the mining industry and on the outside, regard the valuation of exploration tenements as being akin to reading tea leaves or a chicken's entrails!

While not going quite that far, the author would like to put clearly on the record his opinion that the requirement to place a hard-number dollar value on exploration tenements should continue to be vigorously resisted by The Australasian Institute of Mining and Metallurgy and by all individual practitioners.

Several prospectuses have adopted this approach this year including one in which the valuer stated:

None of the properties have Identified Resources or Reserves (under the "Joint Committee Code") and the

successful conclusion of exploration of these properties is subject to the inherent high risks of diamond exploration. Thus it is not possible to place a meaningful value on [the] exploration properties.

So far, so good. Unfortunately, the writer then went on:

Nevertheless, the perceived prospectivity of [the] current exploration property portfolio, and the experience the company has gained over the past five years provide good reason to conclude that the proposed share and option issue is fair and reasonable.

It seems to this author that it is illogical to conclude that the consideration paid for a portfolio is fair if it is impossible to place a value on that portfolio! On the other hand, the holder of a securities licence might be able to argue that it was reasonable although it may or may not be fair.

The author entirely supports the view that it is often not possible to place a meaningful value on an exploration property. For any reasonable confidence level the realistic range of values for a particular exploration property is frequently too wide to be useful and conversely if a range of values is sufficiently narrow to be useful, then the confidence level of that range is too low for it to be relied upon.

Having said that, it is intended to assess the multiple of exploration expenditure (MEE) method of mineral valuation. The method has been used to value exploration properties ranging from those in the earliest stage of exploration to those where although considerable moneys may have been spent on exploration, no resources have been delineated.

The method is widely used and enjoys considerable support. For example, Butler (1991) stated that he believed that the most satisfactory method of valuing exploration properties is to relate the value to the accrued exploration expenditure until it is possible to employ a discounted cash flow (DCF) technique.

Others have criticised the method as being too subjective with respect to both selection of past expenditure that should be included and the prospectivity enhancement multiplier (PEM) applied to that expenditure. One well-known media commentator on business matters (Pierpont, 1994) expressed concern that the principle behind the MEE valuation method seemed to be that the more money companies had spent exploring the ground in the past before abandoning it, the more valuable it became.

In this author's opinion, that comment arises from a common misunderstanding with regard to both the principle and practice associated with the method, rather than any inherent weakness in the underlying principles.

It is accepted that the method is subjective and the MEE is not recommended as a primary valuation method if better options are available. However, in many situations facing the professional valuer, the technique may represent the only semi-quantitative method available and is frequently used.

Perhaps the most cogent reason for reviewing MEE as a basis for mineral valuation is that it frequently forms the starting point in discussions between two parties, one of whom is to farm-in on an exploration tenement: *We have spent a million dollars on these tenements and you can earn a 50% interest by matching that expenditure.*

But before describing the method in more detail a brief discussion of the definition of value is in order since this is obviously of fundamental importance.

VALUE - WHAT IS IT?

DEFINITIONS OF VALUE

The former National Companies and Securities Commission (NCSC), defined fair value as:

Usually the price a willing, knowledgeable buyer is willing to pay in cash to a willing, knowledgeable seller in an arm's length transaction. (NCSC, 1990)

This definition has been modified in the second exposure draft of the Valmin Code (AusIMM, 1994) to read:

Value is the estimated amount of money or the cash equivalent of some other consideration for which, in the opinion of the Expert reached in accordance with the provisions of this Code, the mineral asset, i.e. tenement, project or security should change hands on the date of the Valuation between a willing buyer and a willing seller in an 'arm's length' transaction, wherein the parties had each acted equally knowledgeably, prudently and without compulsion.

Value is usually comprised of two components, the fundamental or intrinsic value of the asset, in effect an assessment of the asset's future net economic benefit and a market premium, either positive or negative, depending on the circumstances.

The clarity of this definition was somewhat blurred by the "Explanatory Comments" that accompanied the exposure draft. These offered a differing view of value suggesting that:

The Code is primarily concerned with assessing the fundamental underlying technical value of an asset, to which any speculative value may be added or subtracted.

This apparent difference of opinion reflects the differing views (and practices) among valuers, although it is unfortunate to have two views expressed in a single document package!

Some practitioners clearly take the view that "value" equals intrinsic value and that "price" is value plus or minus market premium. Other valuers include the market premium and equate the "fair value" to the "market clearing price". This appears to be the position taken in NCSC Release 149 and subsequently in the Valmin Code.

It has been shown, then, that two basic ideas of value prevail:

- value-in-use or value to the owner (intrinsic value).
- value-in-exchange (fair market value).

There is usually little disagreement that the value-in-exchange may be higher or lower than the value-in-use depending on market sentiment at the time of the valuation.

Most relevant jurisdictions appear to have accepted that a "value" means the value-in-exchange, with US courts going a stage further in accepting the view that market price equals value. US courts generally accept that recent sales form the best basis of valuations and place evidence of such transactions above all "methodological" valuations. (Loucke, 1990) This is of relevance because our current definition of value in Australia in fact stems from a California Supreme Court definition of "fair market value".

Most writers accept this definition unequivocally. For example, Ellis (1992) stated:

The value of property is its market value.

Ellis went on to state:

The market value of property is the price which a willing but not anxious purchaser, fully acquainted with the property, would have to pay to a willing but not anxious vendor.

But there is more argument about whether fair market value is the same as the market clearing price - that is the price at which a property changes hands. While most would expect that the market clearing price should equal the fair market value, this is not necessarily the case, the essential additional consideration usually being the state of "anxiety" of the participants in any particular exchange. For example, in a forced sale situation, it may not be possible to find a buyer willing to pay the fair market value within the time-frame of the sale although, given adequate time and proper marketing, it may be that a price equal to the fair market value could have been achieved. Similarly, it may be asked whether in a boom market a purchaser may be unduly anxious to purchase so that he is prepared to bid the market clearing price above the fair value.

This clearly highlights the dangers of accepting recent market transactions as a measure of value without having full cognisance of the circumstances surrounding those transactions. Nonetheless, in most circumstances it is reasonable to accept that fair market value is generally equivalent to the market clearing price, unless it is possible to identify specific factors that cause a disparity between those numbers.

DETERMINATION OF VALUE

Separate from the definition of value (once the concept is agreed) is how to arrive at a determination of that value.

If it is accepted that the market clearing price represents the best practical measure of value, how does one derive a measure of that value in the absence of a market transaction? An obvious method is to look at comparable sales. The difficulty here is that mineral properties are not like project housing where the sale of one property in the recent past will closely define the market clearing price for the remaining properties. There can be no such close comparison between any two mineral exploration properties and determination of value from "comparable" sales can be a risky business.

The necessity of deriving dollar values for properties in the absence of an adequate market has resulted in the development of the various indirect measures of value used today. These "engineering valuations" can only be regarded as "analogues" of the real value and it is these analogues that are being discussed at this conference.

Well-recognised methods such as DCF analysis have been developed to determine the "value in use" of a property with established reserves; unfortunately, these cannot be used to value exploration properties.

While less precise than DCF analysis applied to reserves that have been estimated appropriately, the MEE can be used as a measure of value of an exploration property if the method is applied judiciously.

MULTIPLE OF EXPLORATION EXPENDITURE

OUTLINE OF TECHNIQUE

It must be emphasised from the start that the method is highly subjective despite the initial reliance on apparently objective exploration expenditures.

In essence the method uses previous exploration expenditure and/or future committed exploration expenditure to derive a base value for the tenements. Obviously there are situations where previous expenditure will have added to value and other times when expenditure will have closed-off the potential and reduced the value. It is therefore necessary to introduce another factor into the equation. This factor is generally termed the "prospectivity enhancement multiplier" or more simply the "prospectivity multiplier". The product of the base expenditure and prospectivity enhancement multiplier provides a first-order estimate of the intrinsic value of the property. This may then be adjusted for market premium/discount and the quality of the exploration team to derive an analogue of the fair value of the tenements; alternatively, these factors may be incorporated in the PEM.

RELEVANCE AND EFFECTIVENESS OF EXPLORATION

In deriving the base value for exploration expenditure it is important that only relevant and effective past exploration be included.

There is some discussion about whether prior expenditure by other parties should be included. The first draft of the Valmin Code stated that the base value should *generally not include ANY prior expenditure by other parties.....* It is argued that prior expenditure by previous owners should never be included as this should be reflected in the purchase price if the properties are traded, and that if the ground is open and the data held in Mines Department files, that no value should be attributed to the past expenditure. In practice, a number of valuers have been including expenditure by parties other than the current tenement holder and there has been some criticism levelled at the practice.

The author believes that this is unnecessarily restrictive, and, while great care should be taken in using prior expenditure by other parties, there are valid reasons for including some such past expenditure even when the data is in the public domain and the ground is open. A model will be proposed later in this presentation for incorporating such data.

Old data may have value not recognised by former explorers resulting from:

- Failure to recognise the significance of exploration results;
- new exploration concepts;
- significant moves in commodity prices;
- improvements in metallurgical processes which have improved recovery, or mining methods which have reduced costs;
- development of new markets.

One good example known personally to the author is worth relating. Part of the supergene gold blanket surrounding the Hannan South gold mine was intersected, but not recognised, by a previous explorer in the early 1960s. During a review of that data looking for a valid low-cost exploration target in 1981 to meet a minimum expenditure commitment on a tenement, the author recognised gold-arsenic anomalies. These were rapidly drilled out and subsequently developed into a profitable mine. Clearly, the past expenditure, even though it was very old and the results in the public domain, enhanced the value of those tenements.

A more recent example is the current redevelopment by a junior miner of iron ore deposits on leases resumed by the Western Australian Government. The development is based largely on work previously carried out by a major iron ore producer. A total ban on use of previous expenditure by other parties would presumably place no value on this data which now forms the basis for a multi-million dollar development.

To discount completely all former exploration by parties other than the current tenement holders may significantly undervalue some exploration properties; on the other hand indiscriminate inclusion of all past expenditure may well result in over-valuation.

In using past expenditure, even when all the expenditure has been by the current tenement holder there arises the problem selecting relevant expenditure.

Obviously, if the target mineral for the present explorer is gold, then exploration expenditure for diamonds may be totally irrelevant and should be excluded from the accrued expenditure. Some data, such as detailed geological mapping, good quality topographic base plans and geophysical data may be applicable to any commodity and can be considered relevant. Care should be taken with all old data, however, since it may have been collected using outmoded equipment or using inappropriate geological models. Examples include: aeromagnetic data collected using low resolution fluxgate magnetometers; or, in the Eastern Goldfields of Western Australia, almost ubiquitous geochemical sampling surveys dependent upon rotary airblast (RAB) drill samples collected from the point of drill refusal, prior to wide recognition of the importance of the lateritic profile on metal distribution. This data may be almost valueless despite the high cost of acquisition in the not-too-distant past. Consideration of such data is important, however, as the argument can be extended to say that poor results from such surveys may not impact negatively on the present value of the property.

TREATMENT OF PAST EXPLORATION EXPENDITURE

Once relevant past exploration expenditure has been identified, it must be quantified in current dollars. Too often practitioners simply escalate actual costs using Consumer Price Index (CPI) or some other inflator without considering whether this is appropriate. The cost of many exploration techniques have not moved in line with inflation indices. While some may have increased in real terms, the majority have become considerably cheaper. It is considered more appropriate to re-cost the program at today's prices rather than inflate actual costs.

TREATMENT OF FUTURE COMMITTED EXPLORATION EXPENDITURE

Planned expenditure as well as past expenditure can be used to develop a base for valuation. The justification for using future expenditure is that the rational person would not commit to further expenditure unless he believed that this expenditure will at least add directly to the present value of the tenement. Some practitioners believe that only

committed expenditure can be used. Again, the author believes that this may be too restrictive. There are numerous examples of farm-in arrangements which allow an incoming party to earn an interest in a property by expenditure without the incoming party being committed to a minimum expenditure prior to withdrawal. Nonetheless, it seems at least arguable that no-one would go to the trouble and expense of executing a joint venture arrangement without expecting to expend at least some funds on the project.

There is some scope here for the expert to make an appropriate allocation from the uncommitted future expenditure. In consideration of both past and future expenditure, some allowance must be made for the costs of holding the tenements for long enough to allow the work to be carried out. Such costs may include tenement rents, rates and overheads directly applicable to these holding costs. It should be noted here that if a company took ten years to carry out a program that should have been completed in two years it is more appropriate to allow holding costs of two years rather than the actual ten years costs in deriving the base costs to be used in the valuation. Present costs are again considered to be the appropriate measure to be used rather than employing escalation factors, unless one is attempting to derive some future value.

RELATIONSHIP TO BOOK VALUE

There is no necessary nexus between the base expenditure derived from past expenditure and current book value of the project. This was recognised in NCSC Policy Release 149 which stated "Such carrying values may not, and frequently will not, appropriately reflect the fair market value of the property or lease".

The book value is usually generated from the acquisition cost and subsequent capitalised expenditure and amortisation or depreciation. This book value may be higher than the base expenditure if all previous expenditure has been capitalised against the project and the valuer has discarded significant elements of that past expenditure as being irrelevant. On the other hand, if the holding company has written off previous expenditure which is subsequently considered relevant by the valuer, the book value may be lower than the base expenditure.

PRACTICAL DETERMINATION AND LIMITATIONS OF PEM

Deriving the base cost of exploration completed and committed on the tenement is still only the first stage in determining a fair value.

The most important (and subjective) element of deriving a valuation using the multiple of exploration expenditure method is the application of a prospectivity enhancement multiplier (PEM) to the base expenditure.

The PEM is based on the results of previous exploration and the present perception of the probability for discovery of an economic deposit by further exploration.

Lawrence (1993) suggests that the selected PEM would usually lie in the range of 0.5 to 3.0 but could be as low as zero or as high as 5.0. What does this mean? Values below 1.0 indicate that the relevant work undertaken to date has reduced rather than enhanced the prospectivity of the project; as potential is exhausted, the PEM approaches zero. It cannot be a negative number. There appears to be little agreement about how PEMs greater than 1.0 should be applied. In the past, many workers (including the present author) have generally failed to define the method whereby the multiplier is derived. The author is only aware of one example where the parameters have been explicitly defined. Duncan (1994) employed the following scale for determining the upper limit of the PEM:

1. Work to date or historic data justifies the next stage of exploration.
2. Strong indications of potential for economic mineralisation identified.
3. "Ore grade" intersections or exposures indicative of economic resources.

Duncan believes that values greater than 3.0 should not be used because it should then be possible to use a more reliable method such as DCF analysis.

The author considers that, while these definitions may require some refinement, they represent a significant advance on the purely subjective approach and all valuers should consider providing a definition of the categories which is as rigorous as possible. It may be possible in the longer term to adopt a uniform approach to such definition.

A point of concern is the apparent reluctance of valuers to use a PEM of less than one, particularly where the party for whom the valuation is being performed has acquired the property at little cost and has not incurred significant expenditure since acquisition. An analysis of 37 valuations by six authors in nine specialist valuations indicated only four examples of a PEM of less than one being employed.

Values greater than one are clearly justified where expenditure by the present explorer has significantly upgraded the prospectivity of the tenement. Where a significant proportion of the exploration expenditure has been spent by an earlier party who subsequently "abandoned" the property, the valuer should take great care to justify employing a PEM of greater than one.

MARKET PREMIUM / DISCOUNT

In principle, the application of an appropriate market premium or discount to the intrinsic value derived from the Multiple of Exploration Expenditure is no different from that applying to any other technique, but in practice most practitioners appear to incorporate the market premium or

discount in the PEM. A cautious approach is favoured by the author resulting in values that may fall short of the peaks and troughs of the market, but this policy is adopted because such market conditions tend to prevail for much shorter periods of time than other parts of the cycle. Avoiding the extremes provides a slightly longer period of validity for the valuation.

QUALITY OF THE EXPLORATION TEAM

An assessment of the quality of both the historic exploration and that of the incoming party is perhaps the single most important factor in determining the value of an exploration tenement apart from its inherent prospectivity. It may be the explanation for previous explorers failing to make a discovery and this failure may provide value for the subsequent tenement holder, but only if his team is smarter! The quality of the incoming team provides significant input to the probability of future exploration success and flows through to any premium that might be applied to the intrinsic value.

LIMITATIONS OF A METHOD BASED ON EXPENDITURE

Valuation based on expenditure is subject to a number of limitations, the most important of which is probably the **subjectivity** involved in applying the prospectivity enhancement multiplier, but which also include consideration of how **efficiently** the exploration has been carried out and how **effectively** it has been carried out. The difference between efficiency and effectiveness may be illustrated by turning to the example of a shallow RAB drilling program to test a deeply weathered terrain in the Eastern Goldfields. The area may have been explored very efficiently by employing an excellent drilling contractor at a cheap rate (exploration activity outcome per unit of expenditure) but the program may have been totally ineffective because the drilling failed to test below the leached portion of the profile (exploration outcomes achieved per unit of exploration activity). While it may be easy to recognise such an obvious example, it may not be so easy to determine whether the last few diamond drill holes on a project that proved not, to be viable were warranted and therefore effective.

PROPOSED MODEL FOR USING ALL RELEVANT EXPLORATION EXPENDITURE

Since effective expenditure will increase the value of a property that contains economic mineralisation, it should also reduce the value of a property that in fact contains no economic minerals. The author therefore proposes that consideration should be given to including all previous relevant expenditure by all parties and using the prospectivity enhancement multiplier to "value" that expenditure. It is, after all, very important to know that $10M has already been spent on a particular property and the results of that work. While this should be covered in the description of the prospectivity accompanying the valuation, the requirement to analyse previous expenditure in detail may provide additional rigour.

Work Carried Out	Exp ($)	Results of Work	PEM
Previous Tenement Holder (1)			
Diamond exploration	5.0M	Not relevant	NA
Previous Tenement Holder (2)			
Diamond Drilling	5.0M	Generated good targets following commodity price rise	1.5
RC Drilling	3.0M	Discounted some prospective strike	0.5
Mapping, geophysical surveys etc	1.0M	Too old to be used without additional work	0.2
Current Tenement Holder			
Assay previously un-split core	0.5M	Recognition of wide zones of low grade mineralisation	2.0
Geophysical surveys	0.5M	Excellent coincident geochemical/ geophysical anomalies in areas of known mineralisation	2.0
TOTAL RELEVANT EXP	**10.0M**	**Expenditure weighted**	**1.12**

In the proposed model, each work category would be itemised and its impact on the current prospectivity would be valued using the PEM. Discussion and criticism of the valuation would then focus solely on the PEM employed, rather than on whether or not previous expenditure by other parties should be included.

An analysis of past expenditure on a gold prospect could look something like this:

The author believes that development of this approach could satisfy the concerns previously expressed about the apparently ever-increasing value of a property the more money that has been spent and add to the rigour and credibility of the method.

CONCLUSIONS

- Valuation of exploration properties is highly subjective and justifiably regarded with considerable scepticism.
- In determining a value for a property, the market clearing price is recommended as the best measure of value.
- It is considered unfortunate that present regulations still require a valuation of exploration tenements in a number of circumstances.
- Since it is often not possible to place a meaningful value on an exploration property, it may be preferable to permit the statement that consideration for an exploration tenement is reasonable although it may not be possible to determine whether the transaction is fair.
- The MEE method for valuing exploration tenements is widely used and frequently forms the starting point in discussions between two parties in joint venture and farm-in negotiations but has been criticised as being too subjective and is not recommended as a primary valuation method if better options are available.
- In deriving the base value for exploration expenditure it is important that only relevant and effective past exploration be included. It is recommended that the program should be re-costed at today's prices rather than inflate actual costs.
- A model is proposed which would include all past relevant and effective exploration whether carried out by previous tenement holders or the current tenement holder
- Planned expenditure as well as past expenditure can be used to develop a base for valuation. Both committed and uncommitted expenditure can be used provided the uncommitted expenditure is discounted for probability that it will not be spent.
- The method is highly sensitive to the prospectivity enhancement multiplier ("PEM") applied to the base expenditure. The PEM as currently applied is far too subjective and practitioners should be rigorous in their definition of the PEM factor used.
- Where a significant proportion of the exploration expenditure has been spent by an earlier party the valuer should take great care to justify employing a PEM of greater than one.
- Provided that the limitations of the method are recognised and the technique is used judiciously, it can provide a useful measure of value in the absence of better techniques.

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BHP

Exchange release

15 February 2024

Half Year 2024 Exceptional Items Update

BHP announces two items which will be recognised as exceptional items in BHP's financial results for the half year ended 31 December 2023 (**HY24 Results**).

These items relate to an impairment of the carrying value of the Nickel West operations and West Musgrave project (**Western Australia Nickel**) and an increase to the provision for the Samarco dam failure.

The HY24 Results remain subject to review by BHP's external auditor and final review and approval by the BHP Board.

The HY24 Results will be released on 20 February 2024.

BHP Chief Executive Officer, Mike Henry, said "This is an uncertain time for the Western Australia nickel industry and we are taking action to address the current market conditions. We are reducing operating costs at Western Australia Nickel and reviewing our capital plans for Nickel West and West Musgrave.

BHP Brasil along with Samarco and Vale continue to progress negotiations towards a settlement of the Federal Public Prosecutor Office Claim and Framework Agreement obligations in Brazil. The Renova Foundation has made good progress on reparation and compensation programs and over 84% of the community resettlement cases have been completed."

Impairment of Western Australia Nickel

During the period, BHP integrated the West Musgrave project (acquired from OZ Minerals) with the Nickel West operations to create the Western Australia Nickel business unit. Western Australia Nickel will be reported in the Group and Unallocated segment.

BHP will recognise a non-cash impairment charge of approximately US$2.5 billion (post tax) (approximately US$3.5 billion pre-tax) against the carrying value of Western Australia Nickel.

Developments for HY24

As disclosed in the January 2024 Operational Review, the nickel industry is facing challenges and there has been a sharp fall in nickel prices.

During CY2023, London Metals Exchange benchmark nickel prices fell considerably as both the supply of nickel from Indonesia significantly increased and the London Metals Exchange began accepting Indonesian-origin nickel products as part of its efforts to respond to evolving industry dynamics. These unfavourable operating conditions are expected to endure for a considerable time.

Due to the deterioration in the short-term and medium-term outlook for nickel, BHP has lowered its nickel price assumptions. In addition, capital costs for Western Australia Nickel have increased due to inflation. BHP has undertaken a carrying value assessment of Western Australia Nickel having regard to these factors and will recognise an impairment as at 31 December 2023.

The impairment reduces the carrying value of Western Australia Nickel's net operating assets to negative US$0.3 billion, including closure and rehabilitation provisions of approximately US$0.9 billion. BHP will also record Underlying EBITDA of approximately negative US$0.2 billion at Western Australia Nickel in the HY24 Results.

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As noted in the January 2024 Operational Review, operations at Nickel West are being optimised, and options are being evaluated to mitigate the impacts of the current low realised prices. Nickel West has responded by reducing discretionary expenditure and reviewing capital plans.

Nickel West's Kambalda concentrator will be placed into care and maintenance in June 2024 following Wyloo's decision to suspend its Cassini and Northern Operations mines from 31 May 2024. The Cassini and Northern Operations mines provide the majority of ore feed into the Kambalda concentrator and it will be no longer viable for Nickel West to continue operating the milling circuit after those mines cease operating.

BHP is also reviewing the development plans for Western Australia Nickel with a focus on preserving cash, which includes the potential to place Nickel West into a period of care and maintenance. The West Musgrave project, which BHP acquired as part of the OZ Minerals acquisition, is in execution and is 21% complete. BHP is currently assessing phasing and capital spend for the development of the West Musgrave project as part of this review.

Samarco dam failure provision

BHP will also recognise an income statement charge of US$3.2 billion (post tax) (approximately US$3.1 billion pre-tax) in relation to the Samarco dam failure. BHP Brasil's provision for the Samarco dam failure will be US$6.5 billion as at 31 December 2023.

As disclosed in BHP's 2023 Annual Report, BHP Brasil's provision for the Samarco dam failure as at 30 June 2023 was US$3.7 billion, of which US$0.3 billion was utilised in the period to 31 December 2023.

The exceptional item to be recognised in BHP's HY24 Results predominantly reflects the assessment of the estimated costs to resolve all aspects of the Federal Public Prosecution Office Claim and the Framework Agreement obligations.

For further information on the Framework Agreement, the Federal Public Prosecution Office Claim and the Samarco dam failure, refer to BHP's FY23 Financial Statements, note 4 'Significant events – Samarco dam failure' on page 144 of the 2023 Annual Report.

Developments for HY24

The Federal Public Prosecution Office Claim is a claim brought against Samarco Mineração S.A. (**Samarco**), BHP Billiton Brasil Ltda (**BHP Brasil**) (a subsidiary of BHP Group Limited) and Vale S.A. (**Vale**) by the Brazilian Federal Public Prosecution Office on 3 May 2016, seeking R$155 billion (approximately US$32 billion) for reparation, compensation and moral damages in relation to the Samarco dam failure (**Federal Public Prosecution Office Claim**).

As at 30 June 2023, the provision for the Samarco dam failure only reflected the future cost estimates associated with the delivery of the programs under the Framework Agreement to provide reparation and compensation. In the 2023 Annual Report, BHP also disclosed a contingent liability in relation to the Federal Public Prosecution Office Claim given that it was not possible to reliably estimate the potential outcomes of the claim beyond the estimated costs of completing the programs under the Framework Agreement.

Since 30 June 2023:

- Samarco, Vale and BHP Brasil have been engaging in negotiations with the Brazilian State and Federal Government and other public entities to seek a settlement of obligations under the Framework Agreement, the Federal Public Prosecution Office Claim, and other claims by government entities relating to the Samarco dam failure; and
- as previously disclosed, on 25 January 2024, the Federal Court of Brazil issued a decision in relation to the Federal Public Prosecution Office Claim quantifying collective moral damages arising from the Samarco dam failure. The decision found that Samarco, Vale and BHP Brasil are jointly and severally liable to pay collective moral damages in the amount of R$47.6 billion (US$9.75 billion) (to be adjusted for interest and inflation) when any and all appeals are finally determined. On 1 February 2024, Samarco, Vale and BHP Brasil filed a clarification motion with the Federal Court of Brazil in respect of certain factual inaccuracies in the decision, including the calculation of damages. A decision remains pending. BHP Brasil also intends to

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appeal the decision, challenging the merits and amount of damages. The appeal process is estimated to take approximately two to five years.

In preparing the HY24 Results, BHP has considered the additional information available from the status of settlement negotiations, the judicial decision regarding collective moral damages, updates to the estimated costs of executing the Framework Agreement programs, and the extent to which Samarco may be in a position to fund any future outflows.

As a result, the Group will recognise an income statement charge of US$3.2 billion (post tax), reflecting the change in BHP Brasil's assessment of potential outflows to resolve all aspects of the Federal Public Prosecution Office Claim and Framework Agreement obligations, covering reparation, compensation and moral damages. The dam failure provision at 31 December 2023 will increase to US$6.5 billion. The Federal Public Prosecution Office Claim will be reflected in the measurement of the provision in the HY24 Results and will no longer be disclosed as a contingent liability.

Uncertainties

Significant uncertainty remains around the resolution of the Federal Public Prosecution Office Claim and the Framework Agreement obligations, and there is a risk that outcomes may be materially higher or lower than amounts reflected in BHP Brasil's provision for the Samarco dam failure.

Key areas of uncertainty include:

- the scope and cost of completing the programs under the Framework Agreement, including as a result of ongoing legal actions in relation to the number of people eligible for compensation and the amount of damages to which they are entitled;
- the outcomes of appeals relating to the judicial decision regarding collective moral damages, including any appeals that may be lodged by the Brazilian Federal Public Prosecution Office;
- the terms of any potential future settlement agreement in respect of the Federal Public Prosecution Office Claim, including amounts payable, obligations of the parties to perform ongoing programs of work in relation to reparation and compensation, and the period of time over which any settlement amounts may be payable; and
- the extent to which Samarco is able to directly fund any future obligations relating to reparation, compensation or moral damages. Samarco's long-term cash flow generation remains highly sensitive to factors including its ability to return to full production capacity, commodity prices and foreign exchange rates.

Other claims and proceedings related to the Samarco dam failure are reflected as contingent liabilities in BHP's 2023 Annual Report. The outcomes of these claims and proceedings remain uncertain and are expected to continue to be reflected as contingent liabilities in the HY24 Results. An update on their status will be provided in the notes to the financial statements in the HY24 Results.

Samarco and Renova Foundation

BHP Brasil remains fully committed to supporting the extensive ongoing remediation and compensation efforts in Brazil through the Renova Foundation. The Renova Foundation has spent R$34.8 billion (approximately US$7.2 billion) on comprehensive reparation and compensation programs to 31 December 2023. This includes approximately R$16.6 billion (over US$3.4 billion) paid in indemnities and emergency financial assistance to approximately 430,000 people. In addition, approximately 84% of the resettlement cases for the communities impacted by the Samarco dam failure have been completed.

Authorised for release by Stefanie Wilkinson, Group Company Secretary.

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Forward-looking statements

This release contains forward-looking statements, which involve risks and uncertainties. Forward-looking statements include all statements other than statements of historical or present facts, including: statements regarding: trends in commodity prices and currency exchange rates; demand for commodities; global market conditions, guidance; reserves and resources and production forecasts; expectations, plans, strategies and objectives of management; our expectations, commitments and objectives with respect to social value or sustainability; approval of certain projects and consummation of certain transactions; closure or divestment of certain assets, operations or facilities (including associated costs or benefits); anticipated production or construction commencement dates; capital expenditure or costs and scheduling; operating costs; anticipated productive lives of projects, mines and facilities; provisions and contingent liabilities; and tax, legal and other regulatory developments.

Forward-looking statements may be identified by the use of terminology, including, but not limited to, 'intend', 'aim', 'ambition', 'aspiration', 'goal', 'target', 'prospect', 'project', 'see', 'anticipate', 'estimate', 'plan', 'objective', 'believe', 'expect', 'commit', 'may', 'should', 'need', 'must', 'will', 'would', 'continue', 'forecast', 'guidance', 'outlook', 'trend' or similar words. These statements discuss future expectations or performance, or provide other forward-looking information.

Forward-looking statements are based on management's expectations and reflect judgements, assumptions, estimates and other information available, as at the date made. BHP cautions against reliance on any forward-looking statements. These statements do not represent guarantees or predictions of future financial or operational performance and involve known and unknown risks, uncertainties and other factors, many of which are beyond our control and which may cause actual results to differ materially from those expressed in the statements contained in this release.

For example, our future revenues from our assets, projects or mines described in this release will be based, in part, on the market price of the commodities produced, which may vary significantly from current levels. These variations, if materially adverse, may affect the timing or the feasibility of the development of a particular project, the expansion of certain facilities or mines, or the continuation of existing assets.

In addition, there are limitations with respect to scenario analysis, and it is difficult to predict which, if any, of the scenarios might eventuate. Scenario analysis is not an indication of probable outcomes and relies on assumptions that may or may not prove to be correct or eventuate.

Other factors that may affect the actual construction or production commencement dates, revenues, costs or production output and anticipated lives of assets, mines or facilities include our ability to profitably produce and deliver the products extracted to applicable markets; the impact of economic and geopolitical factors, including foreign currency exchange rates on the market prices of the commodities we produce and competition in the markets in which we operate; activities of government authorities in the countries where we sell our products and in the countries where we are exploring or developing projects, facilities or mines, including increases in taxes and royalties or implementation of trade or export restrictions; changes in environmental and other regulations, political or geopolitical uncertainty; labour unrest; weather, climate variability or other manifestations of climate change; and other factors identified in the risk factors discussed in OFR 8.1 in the Annual Report and BHP's filings with the U.S. Securities and Exchange Commission (the 'SEC') (including in Annual Reports on Form 20-F) which are available on the SEC's website at www.sec.gov.

Except as required by applicable regulations or by law, BHP does not undertake to publicly update or review any forward-looking statements, whether as a result of new information or future events.

Past performance cannot be relied on as a guide to future performance.

BHP

Exchange release

11 July 2024

Western Australia Nickel to temporarily suspend operations

BHP announces that the Nickel West operations and West Musgrave project (**Western Australia Nickel**) will be temporarily suspended from October 2024. BHP intends to review the decision to temporarily suspend Western Australia Nickel by February 2027.

The decision to temporarily suspend Western Australia Nickel follows oversupply in the global nickel market. Forward consensus nickel prices over the next half of the decade have fallen sharply reflecting strong growth of alternative low-cost nickel supply.

During the temporary suspension, BHP will continue to support its workforce and local communities. BHP will invest approximately US$300 million (AU$450 million) per annum following completion of a transition period to support a potential re-start of Western Australia Nickel.

The transition period will commence from July 2024. Operations will be suspended in October 2024 and handover activities for temporary suspension will be completed by December 2024.

During the temporary suspension, BHP will:

- suspend mining and processing operations at the Kwinana nickel refinery, Kalgoorlie nickel smelter and Mt Keith and Leinster operations and the development of the West Musgrave project;
- implement a care and maintenance program of work to ensure the ongoing safety and integrity of its mines and related infrastructure;
- continue to invest in exploration to extend the resource life of Western Australia Nickel and preserve optionality; and
- offer Western Australia Nickel frontline employees another role within BHP or the choice of a redundancy and establish a AU$20 million Community Fund to support local communities during the temporary suspension.

Since FY2020 BHP has invested approximately US$3 billion (AU$4.4 billion) to sustain Western Australia Nickel as an ongoing business and to reorient its production to the battery and electric vehicle market. This includes establishing Australia's first nickel sulphate plant to enhance downstream infrastructure, building two new mines and investing in the development of two solar farms and battery storage. Western Australia Nickel has recorded negative cash flow every year during this period.

Despite the significant capital investments, lower global nickel prices have contributed to Western Australia Nickel expecting to report an underlying EBITDA loss of approximately US$300 million in the financial year to 30 June 2024.[1]

In February 2024, BHP announced plans to review Western Australia Nickel and a non-cash impairment charge of approximately US$3.5 billion (AU$5.1 billion) pre-tax against the carrying value of Western Australia Nickel. As a result of the decision to temporarily suspend operations, BHP

[1] The underlying EBITDA loss figure is preliminary and subject to review and finalisation in BHP's full year results for the period ending 30 June 2024.

expects to recognise a further non-cash impairment charge of US$0.3 billion (AU$0.45 billion) pre-tax as an exceptional item in the Group's FY2024 Financial Statements.

Any redundancy payments and other contractual costs triggered by the decision to place Western Australia Nickel into temporary suspension would be recognised in the Group's HY2025 Financial Statements.

BHP's President Australia, Geraldine Slattery said:

"We understand this is a challenging period for the Western Australia Nickel team and surrounding communities. Every frontline employee will be offered another role within BHP, and best endeavours will also be made to identify redeployment opportunities for other employees engaged in the day-to-day operations of Western Australia Nickel.

We will also work closely with local communities, Traditional Owners and suppliers to support a responsible transition process, which will include the establishment of a AU$20 million Community Fund.

Since BHP announced a review of Western Australia Nickel in February, we have explored options to stem losses in the short-term and identify a viable path forward for the business.

Like others in the Australian nickel sector, we have not been able to overcome the substantial economic challenges driven by a global oversupply of nickel.

We have made the difficult but necessary decision to temporarily suspend the Nickel West operation and West Musgrave project.

We will continue to invest approximately AU$450 million (US$300 million) per annum in the Western Australia Nickel facilities to enable a potential re-start. Western Australia remains an important investment destination for BHP globally, with investment in the State expected to be greater than AU$12 billion over the next five years and we will continue to work with all of our Western Australian partners to advance the economic prosperity of the State."

Authorised for release by Stefanie Wilkinson, Group General Counsel and Group Company Secretary

BHP

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BHP Group is headquartered in Australia

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20 February 2024

Results for announcement to the market

Name of Company: BHP Group Limited (ABN 49 004 028 077)

Report for the half year ended 31 December 2023

This statement includes the consolidated results of BHP for the half year ended 31 December 2023 compared with the half year ended 31 December 2022.

This page and the following 68 pages comprise the half year end information given to the ASX under Listing Rule 4.2A and should be read in conjunction with the 2023 BHP Group annual financial report.

The results are prepared in accordance with IFRS and are presented in US dollars.

				US$ Million
Revenue	**Up**	**6%**	**to**	**27,232**
Profit after taxation attributable to the members of the BHP Group	**Down**	**86%**	**to**	**927**

Net Tangible Asset Backing:

Net tangible assets[1] per fully paid share were US$8.68 as at 31 December 2023, compared with US$8.91 as at 31 December 2022.

[1] Net Tangible Assets includes right of use assets with a carrying value of US$3,194 million as at 31 December 2023 (December 2022: US$2,466 million).

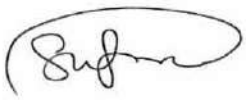
Dividends per share:

Interim dividend for current period (record date 8 March 2024; payment date 28 March 2024)	US 72 cents fully franked
Interim dividend for previous corresponding period	US 90 cents fully franked

Dividend Reinvestment Plan

Any eligible shareholder who wishes to participate in the dividend reinvestment plan, or to vary a participation election, should do so by 5 March 2024 or, in the case of shareholdings on the South African branch register of BHP Group Limited, in accordance with the instructions of your CSDP or broker. Full terms and conditions of the dividend reinvestment plan and details about how to participate can be found at: bhp.com

This statement was approved by the Board of Directors.



Stefanie Wilkinson
Group Company Secretary
BHP Group Limited

BHP

20 February 2024

Financial results for the half year ended 31 December 2023

Strong underlying financial performance underpinned by reliable operations and disciplined cost control.

We are saddened by the loss of one of our sub-contractors who was fatally injured at BMA's Saraji mine in January. An investigation is underway into the circumstances of this tragic incident and we are resolute in our commitment to prevent fatalities and serious injuries in our workplace.

Today, we announced underlying attributable profit of US$6.6 billion for the half year. We also announced an interim dividend of 72 US cents per share – a total of US$3.6 billion, equating to a payout ratio of 56%.

The period also had its challenges, with adjustments relating to Nickel West, West Musgrave and Samarco offsetting an otherwise solid operational performance and overall healthy commodity prices.

At our Western Australia Iron Ore operations, we remain the lowest cost major producer globally and in copper we set new production records at our operations in South Australia and Chile. In South Australia, our consolidated copper province has performed strongly and we are pursuing future growth options. In Canada, we've sanctioned Jansen Stage 2, which will almost double our planned potash production capacity.

We've seen volatility in global commodity prices and demand in the developed world has been softer than expected. That said, China demand is healthy despite weakness in housing and India remains a bright spot. In Australia, the mining industry is facing near-term headwinds in developing resources and it's essential that the right industrial relations and fiscal settings are in place to support the sector's ability to compete and win in global markets.

Long term, the mega-trends playing out in the world around us continue to underline our confidence in future demand for steel, non-ferrous metals and fertilisers.

Mike Henry
BHP Chief Executive Officer

Safety

Fatality at BMA

A team member from one of BHP Mitsubishi Alliance (BMA)'s sub-contracting partners was fatally injured in an incident at BMA's Saraji mine in January. Investigations into the incident are underway.

Social value

Female employee representation[i]

36.2% ↑2.6% pts

HY23 33.6%

We have more than doubled our female employee representation since announcing in 2016 our aspiration to achieve a gender-balanced workforce, and more than 30% of our people leaders are female.

Financial performance

Underlying attributable profit[ii]

US$6.6 bn

HY23 US$6.6 bn

Our underlying attributable profit was in line with HY23, as a result of strong revenue generation and disciplined cost control. All assets are on track to meet their FY24 production and unit cost guidance. FY24 production and unit cost guidance for BMA was revised at the Q2 Operational Review.

Attributable profit

US$0.9 bn ↓86%

HY23 US$6.5 bn

Our attributable profit decreased as a result of an exceptional loss of US$5.6 bn following an impairment of Western Australia Nickel, and an increase to the provision related to the Samarco dam failure.

Capital management

Capital and exploration expenditure[iii]

US$4.7 bn ↑57%

HY23 US$3.0 bn

We are investing in growth and increased our capital and exploration spend by 57% including at Jansen and Copper South Australia.

Fully franked interim dividend

US$0.72 per share

56% payout ratio

We have determined an interim dividend of US$3.6 bn. This follows our strong returns to shareholders in CY23, when we were the highest dividend payer on the ASX.

Group financial performance

Earnings and margins

Solid operational performance and disciplined cost control, aided by higher iron ore and copper prices in the period, maintains strong underlying financials.

Revenue

US$27.2 bn ↑6%

HY23 US$25.7 bn

Attributable profit

US$0.9 bn ↓86%

HY23 US$6.5 bn

Underlying attributable profit

US$6.6 bn

HY23 US$6.6 bn

Profit from operations

US$4.8 bn ↓56%

HY23 US$10.8 bn

Underlying EBITDA[ii]

US$13.9 bn ↑5%

HY23 US$13.2 bn

Underlying EBITDA margin[ii]

53.3%

HY23 53.5%

Adjusted effective tax rate[ii]

31.0%

HY23 29.5%

FY24e 30 – 35%

BHP's revenue increased by US$1.5 bn primarily as a result of higher iron ore and copper prices, as well as the contribution of new mines Prominent Hill and Carrapateena. These were partially offset by New South Wales Energy Coal (NSWEC), where despite a 43% increase in sales volumes, realised prices decreased by 65%.

We continue to experience the impacts of inflation on our underlying cost base, particularly on labour and parts, as reflected in a global inflation rate of 6.3% across our operating jurisdictions during CY23.

Unit costs[ii] were however approximately 5.4%[iii] higher across our major assets during HY24 reflecting our disciplined cost and reliable operational performance and the normalisation of commodity linked consumable prices such as diesel and acid.

This strong operational performance saw Western Australia Iron Ore (WAIO) maintain its lead as the lowest cost major iron ore producer globally and overall Group **Underlying EBITDA** increase by 5%, with an Underlying EBITDA margin of 53.3%.

For further details see Underlying EBITDA waterfall.

We expect the lagged impact of global inflation to continue into H2, particularly in relation to labour, and as we negotiate long-term supply arrangements.

We continue to assess the impact of the Australian Federal Government's 'Same Job Same Pay' industrial relations reforms which will add to our labour costs.

Underlying attributable profit was in line with the prior period, however we reported an exceptional loss, which decreased **attributable profit** by US$5.6 bn, due to:

- a US$2.5 bn impairment of Western Australia Nickel; and
- a US$3.2 bn charge related to the Samarco dam failure.

For further details see note 2 – Exceptional items and note 9 – Significant events – Samarco dam failure.

Our adjusted effective tax rate of 31% was above the 30% Australian corporate tax rate. The adjusted effective tax rate for FY24 is still expected to be in the range of 30 to 35%.

Our operating costs include US$1.8 bn of revenue or production based royalties. Once these are included, our Group effective tax rate was 40.9%. The average comparable rate of ASX50 companies is approximately 30%[iv].

For further details see Adjusted effective tax rate.

Detailed financial information is included in Appendix 1

Cash flow and balance sheet

Strong cash flow generation underpinned US$5.1 bn of investments in the period.

Net operating cash flow

US$8.9bn ↑31%

HY23 US$6.8 bn

Capital and exploration expenditure

US$4.7 bn ↑57%

HY23 US$3.0 bn

Free cash flow[iii]

US$3.8bn ↑9%

HY23 US$3.5 bn

Net debt[iii]

US$12.6 bn

FY23 US$11.2 bn

HY23 US$6.9 bn

Gearing ratio[iii]

21.7%

FY23 18.7%

HY23 12.9%

Our net operating cash flow increased by 31% as a result of the higher Underlying EBITDA and lower income tax and royalty-related taxation payments, partially offset by an increase in working capital.

In line with our **Capital Allocation Framework (CAF)**, we generated **free cash flow** of US$3.8 bn after investing US$5.1 bn. Our investments in the period included:

- US$3.4 bn in organic development including US$1.7 bn on improvement projects; US$1.3 bn major capital in future facing-commodities; and US$199 m of exploration spend; and
- US$1.4 bn of maintenance and decarbonisation expenditure[v].

Capital and exploration expenditure is expected to be[vi]:

- For FY24 and FY25, ~US$10 bn per annum, including US$0.4 bn of exploration in FY24; and
- In the medium term, ~US$11 bn per annum on average[vii].

We have flexibility to adjust capital spend and phasing of projects to accommodate market dynamics and cash flow generation.

BHP's **balance sheet** remains strong. During the half, BHP issued US$4.8 bn of new bonds and retired US$5.7 bn of debt of which US$5 bn related to the OZL acquisition facility.

Our net debt increased by US$1.5 bn to US$12.6 bn from 30 June 2023, largely reflecting net operating cash flow more than offset by:

- Payment of dividends to BHP shareholders of US$4.0 bn, and to non-controlling interests of US$0.6 bn; and
- Capital and exploration expenditure of US$4.7 bn.

Our net debt target range of between US$5 and US$15 bn enables us to maintain a resilient balance sheet during periods of change and external uncertainty while retaining the flexibility to allocate capital within our CAF towards shareholder returns and growth opportunities.

For further details see Net debt waterfall.

Detailed financial information is included in Appendix 1

Value and returns

Continuing to balance investing in the business and cash returns to shareholders.

Interim dividend

72 US cps

Fully franked

56% payout ratio

Earnings per share - basic

18.3 US cps

HY23 127.5 US cps

Underlying return on capital employed (ROCE)[ii]

26.4%

HY23 29.4%

Earnings per share - Underlying[ii]

129.6 US cps

HY23 130.3 US cps

Our operations continued to generate strong **Underlying ROCE** of 26.4%, including 62% at WAIO.

An **interim dividend** of US$0.72 per share (US$3.6 bn), equivalent to a 56% payout ratio will be paid to shareholders on 28 March 2024.

This extends our track record of strong returns. Including the determined dividend, we will have returned ~US$44 bn cash to shareholders since 1 January 2021.

Important dates for shareholders

BHP's Dividend Reinvestment Plan (DRP) will operate in respect of the interim dividend. Full terms and conditions of the DRP and details about how to participate can be found at: bhp.com

Events in respect of the interim dividend	Date
Announcement of currency conversion into RAND	27 February 2024
Last day to trade cum dividend on Johannesburg Stock Exchange Limited (JSE)	5 March 2024
Ex-dividend Date JSE	6 March 2024
Ex-dividend Date Australian Securities Exchange (ASX), London Stock Exchange (LSE) and New York Stock Exchange (NYSE)	7 March 2024
Record Date	8 March 2024
Announcement of currency conversion into AUD, GBP and NZD	11 March 2024
DRP and Currency Election date	11 March 2024[1]
Payment Date	28 March 2024
DRP Allocation Date[2]	15 April 2024

1 5:00pm AEDT.

2 Allocation dates may vary between registers but all allocations will be completed on or before 15 April 2024.

Shareholders registered on the South African branch register will not be able to dematerialise or rematerialise their shareholdings between the dates of 5 March 2024 and 8 March 2024 (inclusive), and transfers between the Australian register and the South African branch register will not be permitted between the dates of 27 February 2024 and 8 March 2024 (inclusive). American Depositary Shares (ADSs) each represent two fully paid ordinary shares and receive dividends accordingly.

Any eligible shareholder who wishes to participate in the DRP, or to vary a participation election should do so before 5:00pm (AEDT) on 11 March 2024, or, in the case of shareholdings on the South African branch register of BHP Group Limited, in accordance with the instructions of your CSDP or broker. The DRP Allocation Price will be calculated in each jurisdiction as an average of the price paid for all shares actually purchased to satisfy DRP elections. The DRP Allocation Price applicable to each exchange will be made available at: bhp.com/DRP

Corporate governance

Terry Bowen retired from the Board following the 2023 Annual General Meeting on 1 November 2023.

The current members of the Board's committees are:

Risk and Audit Committee	Nomination and Governance Committee	People and Remuneration Committee	Sustainability Committee
Michelle Hinchliffe (Chair)	Ken MacKenzie (Chair)	Christine O'Reilly (Chair)	Gary Goldberg (SID) (Chair)
Xiaoqun Clever-Steg	Gary Goldberg (SID)[1]	Catherine Tanna	Ian Cockerill
Ian Cockerill	Michelle Hinchliffe	Dion Weisler	Catherine Tanna
Christine O'Reilly	Christine O'Reilly		Dion Weisler

1 Senior Independent Director (SID).

Economic outlook[viii]

As was the case in recent periods, BHP's external operating environment in CY23 was relatively volatile. Our key commodity prices were slightly higher overall but with significant variation in performance between individual commodities. We also continued to manage external cost inflation across the business.

In the long run, we expect that population growth, rising living standards, and the infrastructure required for global decarbonisation will drive demand for steel, non-ferrous metals and fertilisers.

In the near term, the economic outlook for the developed world is expected to improve modestly after a difficult year for both steel and non-ferrous metals demand in CY23. China and India are expected to remain relative sources of stability for commodity demand, as they have been over the last 12 months. The prices of steel-making raw materials, which are reliant on the physical fundamentals of trade into Asia, have performed better than the prices of non-ferrous metals over the last half year. Non-ferrous metals have greater exposure to weaker demand in the developed world and also to general investor sentiment, despite low exchange inventories. We anticipate that a more balanced global economy and evidence that the worst of the general inflationary wave is behind us, will have a positive impact on our industry in CY24.

On the cost front, while it is positive to see economy-wide inflation in our operating regions coming under control, we expect the lagged impacts from the inflation peak observed in FY23, as well as continued labour market tightness, to impact our cost base throughout the remainder of FY24. Wage inflation is especially problematic in the context of historically poor productivity performance across the resources industry.

Commodity demand

The demand for commodities in the developed world has been soft over the last 12 months due to anti-inflationary policies and the lagged impacts of the energy crisis. We believe that the lag effect of higher interest rates will continue to restrain household consumption in the developed world in the first half of CY24, but we expect that steel, copper and nickel demand will all be modestly firmer across the Organisation for Economic Cooperation and Development (OECD) in the coming 12 months. We also forecast another solid year for commodity demand in China, while India has considerable positive momentum behind it.

The Chinese economy has been volatile since the zero-COVID policy was eased in December 2022. CY23 saw a solid recovery in a range of sectors important to commodity demand including conventional infrastructure, lower GHG emission technology, manufacturing capital, automotive, shipbuilding and consumer durables. However, weakness continued in the steel-intensive real estate sector and in non-steel exports, and overall corporate profitability has been challenged. Throughout the year authorities have acknowledged that additional policies will be needed to support China's economic recovery. For the balance of FY24 and into FY25, the key question remains how effective the policy push will be. Until we see greater coherence between the policies and their effective implementation, our outlook will remain cautious and conditional.

The demand picture has been more balanced in India supported by increasing capital investment, and commodity demand has been accordingly robust. The Indian economy has shown continued healthy momentum as the country moves towards a general election, which is expected to be held in the first half of CY24.

For the review and outlook relating to our individual commodities please refer to the relevant segment sections from page 7.

Costs and inflation

At our full year results in August 2023, we noted the lower rate of inflation on our cost base compared to recent periods. Whilst the spot price of certain input costs has normalised, the lagged effect of inflation continued to be felt through the business. Pressures in non-energy raw materials, logistics and manufacturing supply chains and energy risks have continued to ease, but labour costs remain a key forward-looking inflationary risk. The direct and indirect impact of current events in the Red Sea are not expected to alter the downward trend in global inflation, and at this stage have not had any material impact on our business.

With economy-wide inflation having noticeably eased in our main operational regions, additional pressure has also come out of industry-specific supply chains. We continue to expect some lagged effect of non-labour inflation (including pricing in contracts that reset periodically based on historical outcomes) to impact the business in the balance of FY24 and into FY25. The labour market remains a core inflationary concern, with aggregate wage outcomes in Australia increasingly disconnected from underlying productivity performance, which has been historically weak. This concern is amplified by regulatory reform underway in Australia, which will add to our labour costs and reduce the international competitiveness of the Australian economy.

Overall, the cost of mining production continues to be higher than it was prior to the pandemic. This implies that price support is also expected to be higher than in previous cycles and low-cost operators stand to capture potentially higher relative margins in certain commodities.

For more detail, please refer to our website.

Segment and asset performance

Detailed financial information on all business segments in the Financial performance summary

Copper

Production

894 kt ↑7%

HY23 834 kt

FY24e 1,720 – 1,910 kt

Average realised price

US$3.66/lb ↑5%

HY23 US$3.49/lb

Underlying EBITDA

US$3.5 bn ↑23%

HY23 US$2.8 bn

25% contribution to the Group's Underlying EBITDA

46% Underlying EBITDA margin

Underlying ROCE

10%

HY23 11%

Capital and exploration expenditure

US$2.0 bn

HY23 US$1.3 bn

FY24e US$4.2 bn

Commodity review and outlook[viii]

Despite a 5% increase in prices from HY23 to HY24, copper prices declined slightly over the CY23 period, with intra-year movement driven by shifting expectations of China's recovery, and demand risk in the OECD from manufacturing weakness, global inflation and tighter financial conditions. Extremely low global copper inventories, rising mine costs and the sector's vulnerability to operational disruptions helped to establish a higher floor for prices than seen in prior downturns in OECD manufacturing.

In the near term, we expect broad-based end-user demand growth in China to continue, albeit at a somewhat slower pace than the 6% YoY rate seen in CY23. That view includes an assumption of rapidly increasing investment in lower GHG technology for energy and transport. We anticipate a modest recovery in OECD copper demand. A broadly balanced market is the most likely outcome for CY24. That compares to our view of six months ago that a modest surplus was likely. The major difference in the two forecasts are reductions in primary supply announced by copper producers late in CY23. Uncertainty in primary supply remains a key swing factor. In the medium and longer term, traditional demand (such as home building, electrical equipment and household appliances) is expected to remain solid while the decarbonisation mega-trend is expected to bolster demand. In terms of meeting that demand, we anticipate that the cost curve is likely to steepen as challenges to the development of new resources (such as societal expectations, decarbonisation and water challenges) progressively increase. We anticipate that the industry is likely to enter the final third of this decade with a low inventory buffer, which implies that should deficits occur in this phase, as we expect they will, elevated pricing that is well above the cost curve may well occur during this period. Recent negative surprises to the primary supply outlook could bring deficits forward.

Segment outlook

Integration of the Copper South Australia province has successfully achieved more than US$50 m of annualised EBITDA synergies six months ahead of schedule, with each of Olympic Dam, Carrapateena and Prominent Hill delivering strong safety, production and development outcomes. We have also had exploration success in South Australia, announcing an Exploration Target at Oak Dam in July and multiple intersects of copper grade above 1% (and areas above 2%) beneath Olympic Dam (OD Deeps) in January. This operational performance, coupled with the significant resource base, provide a solid foundation on which to study further value realisation, including the potential to grow annual copper production at Copper South Australia to above 500 ktpa.

In Chile, we continue to progress a range of studies to unlock the next phase of value from our significant resource endowment and utilise latent capacity across our Escondida, Spence and Cerro Colorado operations. These include studying concentrator options at Escondida and the evaluation of multiple leaching technologies which could be applied across all three operations. We expect to provide further information on potential development pathways during CY24.

In Peru, Antamina has received environmental approval to extend operations from 2028 until 2036.

Escondida

Copper production	Unit cost[1,2]	Underlying EBITDA
528 kt ↑3%	**US$1.51/lb** ↑5%	**US$2.3 bn** ↑9%
HY23 511 kt	**HY23** US$1.44/lb	**HY23** US$2.2 bn
FY24e 1,080 – 1,180 kt	**FY24e** US$1.40 – US$1.70/lb	
FY25 and FY26e 1,200 – 1,300 ktpa	**FY25 and FY26e** US$1.30 – US$1.60/lb	

1 Based on average exchange rates of: HY24 USD/CLP 874 (realised); HY23 USD/CLP 920 (realised); FY24e – FY26e USD/CLP 810 (guidance).
2 Refer to Non-IFRS financial information for detailed unit cost reconciliation.

Financial performance

Underlying EBITDA increased by 9% primarily as a result of:

- Higher copper prices, which had a favourable US$0.2 bn impact; and
- Increased sales volumes in line with improved concentrator feed grade and higher concentrator throughput.

These were partially offset by higher operating costs, primarily reflecting inventory drawdowns to maintain concentrator ore feed and the impacts of inflation.

Asset outlook

Escondida production guidance for FY24 remains unchanged at between 1,080 and 1,180 kt, increasing to between 1,200 and 1,300 ktpa in FY25 and FY26, after which production is expected to decline to between 900 and 1,000 ktpa for a period in line with lower concentrator feed grades.

Escondida is assessing multiple options to offset the impact of lower concentrator feed grade, which is expected from FY27. These include the potential for a new concentrator to replace the current Los Colorados facility and the application of one or more leaching technologies to improve recoveries and unlock primary sulphide resources. We expect costs associated with the studies, which are captured as operating costs, to increase to ~US$140 m per year in both FY24 and FY25, from ~US$60 m in FY23.

Full SaL, a BHP designed leaching technology, is on track for first production in FY25 and is expected to unlock ~410 kt in copper cathodes over a 10-year period through improved recoveries and shorter leach cycle times. The capital expenditure to implement Full SaL is expected to be approximately US$300 m.

Deployment of autonomous haulage is expected to begin in the Escondida Norte pit in H2 FY24 and ramp up to approximately 50 autonomous trucks over the next three years. Escondida is evaluating transitioning its fleet of approximately 160 conventional haul trucks to autonomous operations over the next decade.

New royalties came into effect for Escondida from 1 January 2024. These will not be included in Escondida unit costs.

Pampa Norte

Copper production	Spence unit cost[1,2,3]	Underlying EBITDA
138 kt ⬇6%	**US$1.98/lb** ⬇10%	**US$0.4 bn** ⬆37%
HY23 147 kt	**HY23** US$2.19/lb	**HY23** US$0.3 bn
FY24e 210 – 250 kt[1]	**FY24e** US$2.00 – US$2.30/lb	
Medium-term ~250 ktpa		

1 Production and unit cost guidance for FY24 is provided for Spence only. Cerro Colorado produced 11 kt before ceasing production on 9 November 2023.
2 Based on average exchange rates of: HY24 USD/CLP 874 (realised); HY23 USD/CLP 920 (realised); FY24e USD/CLP 810 (guidance).
3 Refer to Non-IFRS financial information for detailed unit cost reconciliation.

Financial performance

Underlying EBITDA increased by 37% predominately as a result of:

- Increased sales volumes at Spence driven in part by record concentrate production following improvement in concentrator throughput and recoveries;
- Higher copper prices, which had a favourable US$51 m impact; and
- Lower commodity-linked consumable prices.

These were partially offset by costs associated with Cerro Colorado's transition to temporary closure in December 2023.

Asset outlook

Our application for environmental approval to extend the life of the Spence leaching facilities to 2039, was lodged in FY23. If approved, this would involve the implementation of a novel approach to re-processing previously leached ores followed by a planned medium-term transition to chalcopyrite ore leaching.

Spence remains on-track to achieve fully autonomous mine haulage operations in Q4 FY24 following the successful extension of autonomous truck operations to the cathode ore crushing circuit in January 2024. In total, Spence has deployed 23 of 33 planned autonomous trucks.

The concentrator plant modifications, which commenced in August 2022, are expected to be completed in FY24. In the Q2 FY24 Operational Review, we announced approval of an incremental US$570 m in sustaining capital to progress remediation of previously identified anomalies in the Spence Tailings Storage Facility (TSF). These remediation plans have been developed in consultation with the Engineer of Record, Independent Tailings Review Board and expert consultants.

Production guidance for Spence for FY24 remains unchanged at between 210 and 250 kt, and is expected to average 250 ktpa over the next five years. This guidance remains subject to successful remediation of the TSF anomalies.

Cerro Colorado entered temporary care and maintenance in December 2023. Cerro Colorado's estimated expenditure of US$45 m for H2 FY24 remains unchanged. We are exploring options to extend the life of Cerro Colorado, including through the use of novel leaching technologies and desalinated water, which could see the operation restart after 2030, subject to environmental approvals.

Copper South Australia

Copper production	Underlying EBITDA
154 kt ↑48%	**US$0.6 bn** ↑128%
HY23 104 kt	**HY23** US$0.3 bn
FY24e 310 – 340 kt	

Financial performance

Underlying EBITDA increased 128% predominantly as a result of:

- The contribution of US$0.3 bn from Carrapateena and Prominent Hill which were acquired in May 2023 as part of the acquisition of OZL; and
- Higher average realised prices for copper, uranium, gold and silver, which had an impact of US$0.1 bn.

This was partially offset by inflationary impacts on labour and contractors, additional planned maintenance and increased exploration activity at Oak Dam and OD Deeps.

Asset outlook

Integration of OZL has successfully delivered more than US$50 m of annualised EBITDA synergies ahead of schedule through supply chain optimisation and reducing corporate overheads. The operations are performing well with record copper concentrate production at Carrapateena, record gold production at Olympic Dam and record development metres at Prominent Hill. We are looking to apply best practice from across the operations to further optimise operational performance and lift productivity.

Production guidance for FY24 remains unchanged at between 310 and 340 kt, net of the planned transfer of copper concentrate from Prominent Hill to Olympic Dam for processing to realise improved margins by processing high-uranium concentrate into cathode.

We are assessing options for a new two-stage smelter which could produce more than 500 ktpa of copper, with a Final Investment Decision expected between FY26 and FY27. In addition to productivity improvements mentioned above, we are undertaking the following growth projects:

- At Prominent Hill, the Wira shaft mine expansion project is under construction. After a review of this project, it is expected to come online in FY26, for a total investment of US$673 m, both in-line with our estimates as part of the acquisition. The hoisting shaft is expected to extend the mine life to at least 2036 and may provide access to potential mineralisation outside the current mine plan.
- At Carrapateena, the Block Cave Expansion project is progressing and is expected to (i) extend the mine life beyond the existing sub-level cave and (ii) increase production at Carrapateena from FY29. Crusher 2 is still expected to come online in Q3 FY24 and to ramp up in Q4 FY24, which will provide an uplift in mine productivity.

We have had continued exploration success at Oak Dam and OD Deeps, and are progressing the external approval process for an access decline at Oak Dam to enable faster and lower cost resource definition drilling of the mineral deposit.

Iron ore

Production

129 Mt ▼2%

HY23 132 Mt

FY24e 254 – 264.5 Mt

Average realised price

US$103.70/wmt ▲21%

HY23 US$85.46/wmt

Underlying EBITDA

US$9.7 bn ▲27%

HY23 US$7.6 bn

68% contribution to the Group's Underlying EBITDA

69% Underlying EBITDA margin

Underlying ROCE

85%

HY23 61%

Capital and exploration expenditure

US$1.0 bn

HY23 US$0.9 bn

FY24e US$2.0 bn

Commodity review and outlook[viii]

China posted record iron ore consumption and near-record blast furnace utilisation rates in CY23, while India continued to exhibit rapid growth in steel production. In contrast developed regions saw steel output contract once again, albeit at a lesser rate than in CY22. The global result was a considerable improvement from the 4% decline in CY22, although the precise scale of the turnaround in China for CY23 is not clear from publicly reported data.

Over the next two years we expect a small further improvement in global steel production with growth led by India, Southeast Asia and to a lesser extent China. We expect that reduced drag from developed regions will also help.

In China, steel production was resilient at high levels in CY23, making it the 5th consecutive year the country produced more than 1 Bt of steel. Weakness in the real estate sector was more than offset by healthy growth in infrastructure, machinery and autos, as well as net steel exports reaching a seven-year high. In CY24 we expect modest growth in steel production in line with our long-held view that China's steel production would sit at a plateau in the 1.0 to 1.1 Btpa range in the first half of the 2020s.

India was a continued bright spot with steel production rising around 12% to ~140 Mt in CY23. That is a 40% increase since the beginning of the decade. In CY24 we expect another year of strong growth as construction demand remains robust. Medium term, we note that the Indian government is targeting 300 Mtpa of steel-making capacity by 2030.

In iron ore, conditions improved in CY23 with a broadly balanced market overall, notwithstanding volatility in pricing within the year. For CY24 we expect similar dynamics for the mass balance, but with both supply and demand growth to slow somewhat from the rates of the prior year. Low-cost supply is not growing at a rate sufficient to displace the high-cost production which is required to keep the market balanced. Our estimate of real-time cost support sits in the US$80 – US$100/t range on a 62% Fe CFR basis, unchanged from our previous reporting period. How effectively China's stimulus policy is implemented, especially with regards to real estate, and how the government chooses to regulate steel production, are both swing factors in CY24.

In the medium term, China's demand for iron ore is expected to be lower than it is today as it moves beyond its crude steel production plateau and the scrap-to-steel ratio rises, though we expect demand for our products from elsewhere in developing Asia will offset this to a degree.

Segment outlook

At WAIO, we are focused on increasing production to greater than 305 Mtpa over the medium term. We are also studying options for growth of the WAIO business up to 330 Mtpa and we expect to complete these studies in CY25. Options under consideration include developing new mines, expanding and leveraging existing infrastructure, including at Yandi and Port Hedland, increasing ore beneficiation and building a new hub.

In Brazil, Samarco is ramping up production and supporting the local community through jobs, investment and taxes. The Renova Foundation continues to make progress on remediation activity and compensation.

Western Australia Iron Ore

Iron ore production	Unit cost[1,2]	Underlying EBITDA
126 Mt ↓3%	**US$18.46/t** ↑1%	**US$9.6 bn** ↑27%
	C1[ix] **US$15.98/t**[3]	
HY23 130 Mt	**HY23** US$18.30/t	**HY23** US$7.6 bn
FY24e 282 – 294 Mt (100% basis)	**FY24e** US$17.40 – US$18.90/t	
Medium-term >305 Mtpa (100% basis)	**Medium-term** <US$17.00/t	

1 Based on average exchange rates of: HY24 AUD/USD 0.65 (realised); HY23 AUD/USD 0.67 (realised); FY24e and medium-term AUD/USD 0.67 (guidance).
2 Refer to Non-IFRS financial information for detailed unit cost reconciliation.
3 C1 unit costs for HY23 were US$15.50/t. WAIO C1 unit cost excludes third party royalties of US$1.99 per tonne (HY23: US$1.52 per tonne), net inventory movements US$(0.43) per tonne (HY23: US$0.57 per tonne), depletion of production stripping US$0.77 per tonne (HY23: US$0.84 per tonne), exploration expenses, marketing purchases, demurrage, exchange rate gains/losses, and other income US$0.15 per tonne (HY23: US$(0.14) per tonne).

Financial performance

Underlying EBITDA increased by 27%, primarily driven by:

- Higher average realised prices for iron ore, which increased by 21% and had a favourable impact of US$2.3 bn; and
- Lower diesel prices.

This was partially offset by higher price-linked royalties and the impacts of inflation.

During HY24, WAIO maintained its lead as the lowest cost major iron ore producer globally with a C1 unit cost of US$15.98/t.

Asset outlook

South Flank remains on track to ramp up to full production capacity of 80 Mtpa (100% basis) by the end of FY24. South Flank's high quality ore is expected to maintain WAIO's average iron ore grade of at least ~61% (excluding Yandi) and increase the overall proportion of lump to between 30% and 33%.

WAIO production guidance for FY24 remains unchanged at between 282 and 294 Mt (100% basis). In support of growing the business to produce >305 Mtpa in the medium term, we are undertaking the following projects:

- The Port Debottlenecking Project (PDP1) which remains on track to be completed in CY24 following commissioning in December 2023, and is expected to deliver an uplift in port throughput;
- The Rail Technology Programme (RTP) will be rolled out over the next few years, and is expected to improve communications and signalling, enhance operational safety and reduce variability on our WAIO rail network;
- The Western Ridge Crusher Project (WRC) was approved by the Board in February 2024 for an expected investment of US$943 m (100% basis). It is expected to deliver ~25 Mtpa (100% basis) from FY28 at a capital intensity of US$38/t, to replace production from the depleting orebodies around Newman.

Average annual sustaining capital expenditure guidance over the medium term, excluding costs associated with operational decarbonisation and our automation programs, remains unchanged at US$5.50/t[vi].

Samarco

Iron ore production

2.5 Mt ↑13%

HY23 2.2 Mt

FY24e 4 – 4.5 Mt

Total Renova Foundation spend

US$7.2 bn[1] ↑22%

HY23 US$5.9 bn[1]

1 Refers to total Renova spend since 2016 (100% basis).

Performance

Samarco continues to operate safely and efficiently since re-starting operations in December 2020. Production increased 13% in HY24 to 2.5 Mt (5.1 Mt on a 100% basis), as a result of higher concentrator throughput. Production guidance for FY24 remains unchanged at between 4 and 4.5 Mt.

The restart of the second concentrator, which will increase pellet production capacity to approximately 16 Mtpa (100% basis) through a filtration and dry stack tailings solution, is expected to deliver first production in Q3 FY25.

In December 2023, Samarco completed the restructure of its finances by issuing new debt under the Judicial Reorganisation process. All eligible employees and small suppliers who had claims under the Judicial Reorganisation process have been paid in full. The restructure provides Samarco with a stable financial position to support the funding of its remediation and compensation obligations, as well as to fund the expansion of its operations and continue to benefit its communities through job creation, investment and taxes.

Financials

BHP Brasil remains committed to supporting the Renova Foundation and its work to progress the remediation and compensatory programs to restore the environment and re-establish communities affected by the Samarco dam failure. Renova made strong progress during HY24, and since March 2016, it has paid compensation and financial assistance to approximately 430,000 people and completed approximately 84% of resettlement cases[x].

For the half year ended 31 December 2023, BHP Brasil has recognised an income statement charge of US$3.2 bn in relation to the Samarco dam failure, which predominantly reflects the change in the assessment of the estimated costs to resolve all aspects of the Federal Public Prosecution Office Claim and the Framework Agreement obligations. As at 31 December 2023 BHP Brasil's provision for the Samarco dam failure is US$6.5 bn.

For further information, please see note 9 – Significant events – Samarco dam failure for the Samarco dam failure provision.

Coal

Production

Metallurgical coal

11.3 Mt ↓17%

HY23 13.6 Mt

Energy coal

7.5 Mt ↑36%

HY23 5.5 Mt

Average realised price

Metallurgical coal

US$266.43/t ↓1%

HY23 US$268.73/t

Thermal coal – export

US$123.29/t ↓65%

HY23 US$354.30/t

Underlying EBITDA

US$1.0 bn ↓63%

HY23 US$2.6 bn

7% contribution to the Group's Underlying EBITDA

26% Underlying EBITDA margin

Underlying ROCE

15%

HY23 55%

Capital and exploration expenditure

US$0.4 bn

HY23 US$0.2 bn

FY24e US$0.7 bn

Commodity review and outlook – Metallurgical coal[viii]

Metallurgical coal prices moved higher in the second half of CY23 on tight fundamentals, particularly on the supply side of the industry. Pig iron production grew strongly in India and China, while it contracted heavily in Europe and was flat in developed Asia. Australian seaborne supply fell for a 4th consecutive year, while Mongolian exports doubled year-on-year.

In the near term, based on public company guidance we expect a modest supply recovery from Australia into the seaborne market. The availability of land borne imports and the operational performance of Chinese domestic mines are key uncertainties for assessing China's role in the seaborne trade in CY24. On seaborne demand, India is expected to maintain its current strong momentum while we believe that OECD importing regions are likely to see a gradual pickup in their steel industries.

Over the longer term, we believe that higher quality metallurgical coals (such as those produced by our BMA assets) will continue to be required in blast furnace steel-making for decades, driven by the growth of the steel industry in hard coking coal importing countries such as India. In particular, such higher quality coking coals are expected to be valued for their potential to help reduce the GHG emissions intensity of blast furnaces. And with the major seaborne supply region of Queensland having become less conducive to long-life capital investment as a result of changes to the royalty regime, the scarcity value of higher quality coking coals may well increase over time.

Segment outlook

Aligned with this longer-term forecast, our strategic objective is to focus on producing higher quality metallurgical coal. In October 2023, we announced that together with Mitsubishi Development Pty Ltd (our 50:50 joint venture partner in BMA) the sale of our Blackwater and Daunia mines to Whitehaven Coal for up to US$4.1 bn (100%). The sale is expected to complete on 2 April 2024.

In thermal coal, we are seeking the relevant approvals to continue mining at NSWEC beyond the current mining consent that expires at the end of FY26, and proceed with a managed process to cease mining at the asset by the end of FY30.

BMA

Metallurgical coal production	Unit cost[1,2]	Underlying EBITDA
11.3 Mt ↓17%	**US$129.00/t** ↑29%	**US$0.8 bn** ↓43%
HY23 13.6 Mt	**HY23** US$100.23/t	**HY23** US$1.4 bn
FY24e 46 – 50 Mt (100% basis)	**FY24e** US$110 – US$116/t	

1 Based on average exchange rates of: HY24 AUD/USD 0.65 (realised); HY23 AUD/USD 0.67 (realised); FY24e AUD/USD 0.67 (guidance).
2 Refer to Non-IFRS financial information for detailed unit cost reconciliation.

Financial performance

Underlying EBITDA decreased by 43% predominately driven by:

- Lower sales volumes in line with lower production volumes, which had an unfavourable impact of US$0.4 bn; and
- Higher costs as a result of the increase in prime stripping to recover depleted inventory positions following extended weather impacts and labour constraints in prior periods, the impacts of inflation, and planned higher maintenance activity, which were partially offset by lower diesel prices.

Queensland remains one of the highest royalty jurisdictions in the world. The change to the royalty regime in CY22 increased coal royalties to the highest maximum rate in the world, and resulted in an additional US$0.3 bn in royalties paid to the Queensland Government by BHP in relation to HY24 than would have otherwise been paid. Combined with income taxes, this equates to an adjusted effective tax rate including royalties of 62%.

Asset outlook

We are focused on improving value chain stability following depleted inventory positions arising from extended weather impacts and labour constraints over recent years, and we expect this will continue into CY25.

Following the planned sale of the Blackwater and Daunia mines, we expect 86% of BMA's products will be sold by reference to the Platts PLV HCC FOB Qld index, the highest quality metallurgical coal index, increasing from 64% currently. BMA is also expected to benefit from simplified operations and transport logistics, including the sale and shipment of all products through the 100% owned Hay Point Coal Terminal. After the completion of the sale, we will no longer recognise the closure and rehabilitation provisions for the two sold mines of US$0.6 bn.

Given the negative impact on investment economics resulting from the change in coal royalty rates, and the increase in sovereign risk due to the decision to raise royalties without consultation, we will not be investing in any further growth in Queensland, however we will sustain and optimise our existing operations.

New South Wales Energy Coal

Energy coal production

7.5 Mt ↑36%

HY23 5.5 Mt

FY24e 13 – 15 Mt

Underlying EBITDA

US$0.2 bn ↓84%

HY23 US$1.2 bn

Financial performance

Despite higher sales volumes in line with strong production, Underlying EBITDA decreased by 84%, primarily due to lower average realised prices for thermal coal of 65% which had an unfavourable impact of US$1.7 bn.

The additional volumes resulted in higher operating costs, however these were more than offset by lower royalties (linked to lower prices).

Asset outlook

As announced in June 2022, we made the decision to retain NSWEC in our portfolio and proceed with a managed process to cease mining by the end of FY30.

We submitted a modification request to the NSW Government to extend mining approval to 30 June 2030 in support of the 2030 closure plan. The modification submission went on public exhibition for four weeks in November 2023. The approval process will continue through FY24 alongside continued stakeholder and community engagement on closure plans.

Subject to receiving the necessary approvals, as we look ahead to 2030 we will not be allocating any growth capital to NSWEC. We will continue to optimise the operation for value, with absolute costs expected to be stable in the medium term after a period of higher inflation and input prices. The royalty rates in NSW will rise from 8.2% to 10.8% for open cut mines, effective from 1 July 2024.

Group & Unallocated

Western Australia Nickel

Production

40 kt ▲4%

HY23 38 kt

FY24e 77 – 87 kt

Average realised price[xi]

US$18,602/t ▼24%

HY23 US$24,362/t

Underlying EBITDA

US$(0.2) bn ▼276%

HY23 US$0.1 bn

Capital and exploration expenditure

US$0.8 bn

HY23 US$0.3 bn

FY24e US$1.4 bn

Commodity review and outlook[viii]

The nickel industry moved into significant surplus over the course of CY23 as Indonesian supply continued to grow at pace at a time of weak traditional end use demand in the OECD, and mixed outcomes across the electric vehicle value chain versus expectations. Indonesian production of Class-II nickel products is up around 3.6 times since CY19, and its production of intermediates has increased dramatically in the last two years. Whilst electric vehicle and battery demand grew across CY23, a significant destocking cycle in the battery value-chain impacted nickel and all battery metals. The conversion of Indonesian-origin nickel products into cathode, which the London Metal Exchange (LME) chose to allow onto its platforms, saw the significant non-Class-I surplus spill over into visible Class-I inventory, and then LME pricing. LME prices had fallen deep into the cost curve by the end of CY23. Whilst voluntary curtailments have been publicly announced by various operators, we estimate that we are in a multi-year run of surpluses that are likely to average out well over 5% of annual demand.

Longer term, we continue to believe nickel will be a core beneficiary of the electrification mega-trend and that nickel sulphides will be particularly attractive. Notwithstanding that, the industry is expected to experience a difficult multi-year run as excess current and committed supply is gradually absorbed by rising demand. Among the many cases we consider for this timing, our base case is that the market may rebalance by the late 2020s.

Financial performance

During HY24 we integrated the West Musgrave project with the Nickel West operations to create the Western Australia Nickel business unit. As the West Musgrave project is still in execution and is not yet operational, its financial impact for the period is seen in capital and exploration expenditure.

Despite higher production volumes at Nickel West, Underlying EBITDA decreased by US$0.3 bn, to a loss of US$(0.2) bn, predominantly as a result of:

- Significantly lower realised prices for nickel metal and intermediate products, which had an unfavourable impact of US$0.3 bn; and
- The unfavourable impact of inflation on the cost base including increased labour and contractor costs.

Nickel West also had increased deliveries of third party concentrate and ore, however the lower nickel prices in the period largely offset the impact of these volumes.

Due to the deterioration in the short-term and medium-term outlook for nickel, BHP has lowered its nickel price assumptions. In addition, capital costs for Western Australia Nickel have increased due to inflation. With regard to these factors, we have recognised an impairment of US$2.5 bn (post tax) against the carrying value of Western Australia Nickel. For further details please refer to note 2 – Exceptional items.

Business outlook

In the context of the above, we are assessing our plans for Western Australia Nickel. These include optimising operations, reducing discretionary expenditure and reviewing capital plans. We are also looking at the longer-term future of Western Australia Nickel, including potentially entering a period of care and maintenance at Nickel West, and assessing phasing and capital spend for the development of the West Musgrave project.

Potash

Capital expenditure

US$0.53 bn ↑64%

HY23 US$0.33 bn

FY24e US$1.2 bn

Underlying EBITDA

US$(0.13) bn

HY23 US$(0.09) bn

Commodity review and outlook[viii]

Potash prices declined across CY23 as prices reverted to more normal ranges with both demand and supply adjusting following the major supply related shocks of the last few years. The existing operations in Russia and Belarus are now back to around four-fifths of pre-shock capacity and global shipments are tracking at 93% of CY20 levels.

In the medium-term we expect existing capacity in Russia and Belarus to return to normal operating rates. However, new projects in the region are expected to face significant delays versus pre-sanctions timelines. Longer term, we believe that potash stands to benefit from the intersection of global mega-trends: rising populations, changing diets and the need for the more sustainable intensification of agriculture on finite arable land. We consider this compelling demand picture, rising geopolitical uncertainty and the maturity of the existing asset base to be an attractive, accelerated entry opportunity in a lower-risk supply jurisdiction such as Saskatchewan, Canada.

Business outlook

In October 2023 we announced an investment of US$4.9 bn for stage 2 of the Jansen potash project, which will increase our total planned potash production capacity to ~8.5 Mtpa (representing approximately 10% of forecast FY30 supply). Transitioning directly from Jansen Stage 1 (JS1) to Jansen Stage 2 (JS2) during the construction period brings a number of operational benefits including leveraging the experience of our integrated project team and continued use of our existing suppliers and contractors. This will allow us to realise potential synergies of ~US$300 m, which are embedded in JS2's economics.

Jansen

	Progress	Production target date	Investment estimate
Jansen Stage 1	**38%**	**End-CY26**	**US$5.7 bn**

Project update

JS1 is now 38% complete and on track to deliver first production at the end of CY26, with a two year ramp up period. In HY24, earthworks at JS1 continued, with the concrete foundations for the mill nearing completion. In H2 FY24, we expect to award all remaining major equipment and construction packages for JS1 and will continue to progress underground mine construction activities.

Construction of JS2 is expected to commence in Q4 FY24 and to take approximately six years, with first production expected in FY29, followed by a three year ramp up period. JS2 will have a capital intensity of US$1,050/t[xii], which is lower than JS1, due to leveraging existing and planned infrastructure. At third party consensus prices, the project is expected to generate an IRR of 15 – 18%, and EBITDA margins of 65 – 70%.

Capital expenditure in HY24 for JS1 and JS2 was US$0.53 bn and we remain on track to spend US$1.2 bn for the full year.

The US$0.13 bn of operating costs incurred at Jansen relate to site services, overheads, studies and social investments, which are expected to continue throughout the project.

Minerals exploration and early-stage entry

Exploration expenditure

US$199 m ▲28%

HY23 US$156 m

As well as the successes seen in Copper South Australia, we continued to build our portfolio of options in future facing commodities via high potential exploration projects, equity investments, joint ventures and farm-in agreements. Greenfield minerals exploration was also undertaken to advance copper targets in Sweden, Chile, Serbia, Peru, Canada, Australia and the United States.

We have announced the second cohort of our accelerator program, Xplor. After reviewing several hundred applications, six early-stage minerals exploration companies were selected to join the program. They will each receive funding of up to US$500,000 and will have access to internal and external industry experts to support their growth. Additionally, several companies were selected from the first Xplor cohort for follow on investments, after successfully completing last year's program.

We closed the acquisition of Ragnar Metals' Swedish projects, which included the Tullsta nickel sulphide project. Early drilling results at the project have demonstrated several significant intercepts. Elsewhere, we continue to progress nickel exploration activities in Australia and Canada. We also continued to progress our strategy of partnering with mining companies focused on early-stage copper and nickel exploration projects.

In H2, we intend to pilot exploration trials using generative artificial intelligence. This work extends from a successful proof of concept of the technology in HY24 which achieved early success in retrieving information from large, unstructured government datasets.

Appendix 1

Financial Report for the half year ended 31 December 2023

Financial performance summary[1]

A summary of performance for the HY24 and HY23 financial years is presented below.

Key group metrics

	HY24 US$M	HY23 US$M	Change %
Revenue	27,232	25,713	6%
Profit from operations	4,803	10,833	(56%)
Attributable profit	927	6,457	(86%)
Basic earnings per share (cents)	18.3	127.5	(86%)
Interim dividend per share (cents)	72	90	(20%)
Net operating cash flow	8,884	6,770	31%
Capital and exploration expenditure	4,744	3,027	57%
Net debt	12,648	6,910	83%
Underlying EBITDA	13,875	13,230	5%
Underlying attributable profit	6,569	6,597	(0%)
Underlying basic earnings per ordinary share (cents)	129.6	130.3	(1%)

Key asset metrics

Half year ended 31 December 2023 US$M	Revenue[2]	Underlying EBITDA[3]	Underlying EBIT[3]	Exceptional items[4]	Net operating assets[3]	Capital expenditure	Exploration gross	Exploration to profit
Copper								
Escondida	4,427	2,347	1,897		12,737	853		
Pampa Norte[5]	1,133	408	199		4,740	392		
Antamina[6]	730	469	364		1,454	258		
Copper South Australia[7]	1,853	591	232		16,061	581		
Other[6]	22	(107)	(141)		322	68		
Total Copper from Group production	**8,165**	**3,708**	**2,551**	**–**	**35,314**	**2,152**		
Third party products	1,223	28	28	–	–	–		
Total Copper	**9,388**	**3,736**	**2,579**	**–**	**35,314**	**2,152**	**89**	**89**
Adjustment for equity accounted investments[6]	(730)	(263)	(141)	–	–	(258)	(1)	(1)
Total Copper statutory result	**8,658**	**3,473**	**2,438**	**–**	**35,314**	**1,894**	**88**	**88**
Iron Ore								
Western Australia Iron Ore	13,991	9,646	8,679		20,937	927		
Samarco[8]	–	–	–		(6,272)	–		
Other	59	21	9		(127)	–		
Total Iron Ore from Group production	**14,050**	**9,667**	**8,688**	**(2,899)**	**14,538**	**927**		
Third party products	12	(1)	(1)	–	–	–		
Total Iron Ore	**14,062**	**9,666**	**8,687**	**(2,899)**	**14,538**	**927**	**44**	**22**
Adjustment for equity accounted investments	–	–	–	–	–	–	–	–
Total Iron Ore statutory result	**14,062**	**9,666**	**8,687**	**(2,899)**	**14,538**	**927**	**44**	**22**
Coal								
BHP Mitsubishi Alliance[9]	2,882	810	529		6,863	303		
New South Wales Energy Coal[10]	980	257	216		(144)	43		
Other	–	(31)	(44)		(27)	6		
Total Coal from Group production	**3,862**	**1,036**	**701**	**–**	**6,692**	**352**		
Third party products	–	–	–	–	–	–		
Total Coal	**3,862**	**1,036**	**701**	**–**	**6,692**	**352**	**7**	**2**
Adjustment for equity accounted investments[10]	(76)	(61)	(49)	–	–	–	–	–
Total Coal statutory result	**3,786**	**975**	**652**	**–**	**6,692**	**352**	**7**	**2**
Group and unallocated items								
Potash	–	(129)	(130)		5,247	533	1	1
Western Australia Nickel[11]	725	(174)	(240)		(311)	780	27	25
Other[12]	1	64	(174)		(666)	59	32	32
Total Group and unallocated items	**726**	**(239)**	**(544)**	**(3,531)**	**4,270**	**1,372**	**60**	**58**
Inter-segment adjustment	–	–	–	–	–	–	–	–
Total Group	**27,232**	**13,875**	**11,233**	**(6,430)**	**60,814**	**4,545**	**199**	**170**

Half year ended 31 December 2022 US$M	Revenue[2]	Underlying EBITDA[1]	Underlying EBIT[1]	Exceptional items[4]	Net operating assets[3]	Capital expenditure	Exploration gross	Exploration to profit
Copper								
Escondida	4,089	2,160	1,737		12,103	605		
Pampa Norte[5]	1,094	298	50		4,693	307		
Antamina[6]	752	432	343		1,417	204		
Copper South Australia[7]	1,133	259	62		9,911	270		
Other[6]	1	(99)	(108)		(51)	9		
Total Copper from Group production	**7,069**	**3,050**	**2,084**	**–**	**28,073**	**1,395**		
Third party products	988	8	8	–	–	–		
Total Copper	**8,057**	**3,058**	**2,092**	**–**	**28,073**	**1,395**	**64**	**61**
Adjustment for equity accounted investments[6]	(752)	(244)	(154)	–	–	(204)	(3)	–
Total Copper statutory result	**7,305**	**2,814**	**1,938**	**–**	**28,073**	**1,191**	**61**	**61**
Iron Ore								
Western Australia Iron Ore	11,756	7,623	6,651		20,669	805		
Samarco[8]	–	–	–		(3,235)	–		
Other	57	18	6		(41)	7		
Total Iron Ore from Group production	**11,813**	**7,641**	**6,657**	**111**	**17,393**	**812**		
Third party products	9	–	–	–	–	–		
Total Iron Ore	**11,822**	**7,641**	**6,657**	**111**	**17,393**	**812**	**50**	**26**
Adjustment for equity accounted investments	–	–	–	–	–	–	–	–
Total Iron Ore statutory result	**11,822**	**7,641**	**6,657**	**111**	**17,393**	**812**	**50**	**26**
Coal								
BHP Mitsubishi Alliance[9]	3,598	1,426	1,125		7,426	183		
New South Wales Energy Coal[10]	2,016	1,288	1,247		(209)	50		
Other	–	(29)	(37)		(11)	4		
Total Coal from Group production	**5,614**	**2,685**	**2,335**	**–**	**7,206**	**237**		
Third party products	–	–	–	–	–	–		
Total Coal	**5,614**	**2,685**	**2,335**	**–**	**7,206**	**237**	**4**	**1**
Adjustment for equity accounted investments[10]	(48)	(54)	(41)	–	–	–	–	–
Total Coal statutory result	**5,566**	**2,631**	**2,294**	**–**	**7,206**	**237**	**4**	**1**
Group and unallocated items								
Potash	–	(87)	(88)		4,008	325	–	–
Western Australia Nickel[11]	1,010	99	50		1,100	267	26	24
Other[12]	10	132	(98)		(1,545)	39	15	15
Total Group and unallocated items	**1,020**	**144**	**(136)**	**(31)**	**3,563**	**631**	**41**	**39**
Inter-segment adjustment	–	–	–	–	–	–	–	–
Total Group	**25,713**	**13,230**	**10,753**	**80**	**56,235**	**2,871**	**156**	**127**

1 Group profit before taxation comprised Underlying EBITDA; exceptional items, depreciation, amortisation and impairments of US$9,072 m (HY23: US$2,397 m) and net finance costs of US$821 m (HY23: US$652 m).

2 Total revenue from thermal coal sales, including BMA and NSWEC, was US$980 m (HY23: US$2,123 m).

3 For more information on the reconciliation of non-IFRS financial information to our statutory measures, reasons for usefulness and calculation methodology, please refer to non-IFRS financial information.

4 Excludes exceptional items relating to Net finance costs US$190 m and Income tax benefit US$978 m (HY23: Net finance costs US$222 m and Income tax benefit US$2 m).

5 Includes Spence and Cerro Colorado.

6 Antamina, SolGold and Resolution (the latter two included in Other) are equity accounted investments and their financial information presented above with the exception of net operating assets reflects BHP Group's share. Group and Copper level information is reported on a statutory basis which reflects the application of the equity accounting method in preparing the Group financial statements – in accordance with IFRS. Underlying EBITDA of the Group and the Copper segment, includes depreciation, amortisation and impairments (D&A), net finance costs and taxation expense of US$263 m (HY23: US$244 m) related to equity accounted investments.

7 Includes Olympic Dam as well as Prominent Hill and Carrapateena which were acquired on 2 May 2023 as part of the acquisition of OZL.

8 Samarco is an equity accounted investment and its financial information presented above, with the exception of net operating assets, reflects BHP Billiton Brasil Ltda's share. All financial impacts following the Samarco dam failure have been reported as exceptional items in both reporting periods.

9 On 18 October 2023, BHP announced the execution of Asset Sale Agreements for the divestment of BHP's and Mitsubishi Development Pty Ltd's respective interests in Blackwater and Daunia mines (part of the BHP Mitsubishi Alliance) to Whitehaven Coal. While BHP continues to report its share of profit and loss within the Coal Segment and asset tables, Blackwater and Daunia assets and liabilities have been classified as 'Held for Sale' and therefore excluded from Net Operating Assets. Refer to financial statements note 11 – 'Assets and liabilities held for sale' for further information.

10 Includes Newcastle Coal Infrastructure Group (NCIG) which is an equity accounted investment and its financial information presented above, with the exception of net operating assets, reflects BHP Group's share. Total Coal statutory result excludes contribution related to NCIG until future profits exceed accumulated losses.

11 Includes Nickel West and West Musgrave which was acquired on 2 May 2023 as part of the acquisition of OZL.

12 Other includes functions, other unallocated operations including legacy assets and consolidation adjustments. Revenue not attributable to reportable segments comprises the sale of freight and fuel to third parties, as well as revenues from unallocated operations. Exploration and technology activities are recognised within relevant segments.

Underlying EBITDA waterfall

The following table and commentary describe the impact of the principal factors[1] that affected Underlying EBITDA for the half year ended 31 December 2023 compared with the half year ended 31 December 2022:

US$M	Total Group	Copper	Iron ore	Coal	Group and unallocated
Half year ended 31 December 2022	13,230	2,814	7,641	2,631	144
Net price impact	**776**	**406**	**2,065**	**(1,548)**	**(147)**
Change in sales prices	629	411	2,300	(1,778)	(304)
			Refer to Segment and asset performance for average realised prices		
Price-linked costs	147	(5)	(235)	230	157
			WAIO: Higher royalties in line with higher prices.	NSWEC: Lower royalties in line with lower prices.	Western Australia Nickel: Favourable impact of lower nickel price on third party volumes.
Change in volumes	**260**	**164**	**10**	**68**	**18**
		Escondida: Higher volumes due to higher concentrator feed grade and higher concentrator throughput.	WAIO: Stronger demand in China for portside sales, combined with a favourable product mix at Jimblebar partially offset by continued tie-in activity for the Rail Technology Programme.	NSWEC: Higher volumes reflecting eased labour constraints and improved weather conditions enabling uplift in truck productivity. Partially offset by: BMA: Lower volumes due to a significant increase in planned maintenance across the asset, an extended longwall move and geotechnical faulting. This was partially offset by improved weather conditions.	Western Australia Nickel: Higher volumes due to improved performance, and a shorter shutdown period at the Kalgoorlie Smelter offsetting downtime at the Kwinana Refinery.
Change in controllable cash costs	**(436)**	**(105)**	**17**	**(130)**	**(218)**
Operating cash costs	(372)	(52)	19	(129)	(210)
		Escondida & Spence: Minor inventory drawdowns.	WAIO: Net favourable inventory movements, largely due to a build in pre-crush stocks, partially offset by increased maintenance activity.	BMA: Higher costs as a result of the increase in stripping to recover depleted inventory positions following extended weather impacts and labour constraints in prior periods. NSWEC: Higher stripping costs in line with higher volumes.	Western Australia Nickel: Increased deliveries of third party nickel concentrate and ore.
Exploration and business development	(64)	(53)	(2)	(1)	(8)
		Copper South Australia: Higher exploration spend for drilling activities at Oak Dam and Olympic Dam.			
Change in other costs	**(206)**	**(105)**	**(42)**	**(42)**	**(17)**
Exchange rates	80	80	(15)	(10)	25
Inflation	(507)	(234)	(106)	(107)	(60)
			Global inflation rate of 6.3%		
Fuel, energy, and consumable price movements	340	176	71	75	18
		Escondida, Spence and Copper South Australia: Primarily due to lower diesel, acid, and electricity prices.	WAIO: Primarily due to lower diesel prices.	BMA & NSWEC: Primarily due to lower diesel prices.	Western Australia Nickel: Primarily due to lower diesel and ammonia prices.
Non-Cash	(119)	(127)	8	-	-
		Escondida: Lower stripping capitalisation reflecting phase of mine plan.			
One-off items	-	-	-	-	-
Asset sales	38	-	1	(1)	38
Ceased and sold operations	(23)	(23)	-	-	-
		Cerro Colorado: Entered temporary care and maintenance in December 2023.			
New and acquired operations	240	257	-	-	(17)
		Copper South Australia: Contribution from OZL assets acquired in May 2023.			
Other	(4)	65	(26)	(3)	(40)
		Antamina: Higher profit from higher capitalised stripping reflecting phase of mine plan.			G&U: Prior period fair value gain on OZL holdings, partially offset by HY24 VAT refund received in relation to previously divested Petroleum operations.
Half year ended 31 December 2023	13,875	3,473	9,668	975	(239)

Exchange rates

The following exchange rates relative to the US dollar have been applied in the financial information:

	Average Half year ended 31 December 2023	Average Half year ended 31 December 2022	As at 31 December 2023	As at 31 December 2022	As at 30 June 2023
Australian dollar[1]	0.65	0.67	0.68	0.68	0.66
Chilean peso	874	920	877	860	803

1 Displayed as US$ to A$1 based on common convention.

Capital and exploration expenditure

Historical capital and exploration expenditure and guidance are summarised below:

	FY24e US$ bn	HY24 US$M	HY23 US$M	FY23 US$M
Maintenance and decarbonisation[1]	3.1	1,350	1,128	2,981
Development - Minerals	6.5	3,195	1,743	3,752
Capital expenditure (purchases of property, plant and equipment)	**9.6**	**4,545**	**2,871**	**6,733**
Add: exploration expenditure	0.4	199	156	350
Capital and exploration expenditure	**~10**	**4,744**	**3,027**	**7,083**

1 Includes capitalised deferred stripping of US$441 m for HY24 (HY23: US$432 m) and US$0.9 bn estimated for FY24.

Major Projects

Commodity	Project and ownership	Project scope / capacity	Capital expenditure US$M	First production target date	Progress
Potash	Jansen Stage 1 (Canada) 100%	Design, engineering and construction of an underground potash mine and surface infrastructure, with capacity to produce 4.15 Mtpa.	5,723	End-CY26	Project is 38% complete
Potash	Jansen Stage 2 (Canada) 100%	Development of additional mining districts, completion of the second shaft hoist infrastructure, expansion of processing facilities and addition of rail cars to facilitate production of an incremental 4.36 Mtpa.	4,859	FY29	Approval announced October 2023.

Production and unit cost guidance

Historical production and production guidance are summarised below:

Production	Medium-term guidance	FY24 guidance		HY24	HY23	HY24 vs HY23
Copper (kt)		**1,720 – 1,910**		**894.4**	**834.4**	**7%**
Escondida (kt)	1,200 – 1,300[1]	1,080 – 1,180	Unchanged	527.9	510.7	3%
Pampa Norte (kt)	~250[2]	210 – 250[2]	Unchanged	138.1	147.3	(6%)
Copper South Australia (kt)		310 – 340	Unchanged	153.7	104.1	48%
Antamina (kt)		120 – 140	Unchanged	71.7	72.3	(1%)
Carajás (kt)		–	Unchanged	3.0		
Iron ore (Mt)		**254 – 264.5**		**129.0**	**132.0**	**(2%)**
WAIO (Mt)		250 – 260	Unchanged	126.5	129.7	(3%)
WAIO (100% basis) (Mt)	>305	282 – 294	Unchanged	142.1	146.4	(3%)
Samarco (Mt)		4 – 4.5	Unchanged	2.5	2.2	13%
Metallurgical coal - BMA (Mt)		**23 – 25[3]**	**Lowered**	**11.3**	**13.6**	**(17%)**
BMA (100% basis) (Mt)		46 – 50[3]	Lowered	22.6	27.2	(17%)
Energy coal – NSWEC (Mt)		**13 – 15**	Top end	**7.5**	**5.5**	**36%**
Nickel (kt)		**77 – 87**	Unchanged	**39.8**	**38.4**	**4%**

1 Medium term refers to FY25 and FY26.

2 Production guidance is provided for Spence only. Average of 250 ktpa over five years on the basis that remediation of the previously identified TSF anomalies does not impact operations. Cerro Colorado produced 11 kt before ceasing production on 9 November 2023.

3 Excludes production from Blackwater and Daunia mines from the expected date of sale completion, 2 April 2024.

Historical costs[1] and cost guidance for our major assets are summarised below:

	Medium-term guidance[2]	FY24 guidance[2]	HY24 at guidance exchange rates[2]	HY24 at realised exchange rates[3]	HY23[3]	HY24[3] Vs HY23
Escondida unit cost (US$/lb)[4,5]	1.30 – 1.60	1.40 – 1.70	1.59	1.51	1.44	5%
Spence unit cost (US$/lb)		2.00 – 2.30	2.10	1.98	2.19	-10%
WAIO unit cost (US$/t)[6]	<17	17.40 – 18.90	18.43	18.46	18.30	1%
BMA unit cost (US$/t)		110 – 116	128.76	129.00	100.23	29%

1 Refer to Non-IFRS financial information for detailed unit cost reconciliations and definitions.

2 FY24 and medium-term unit cost guidance are based on exchange rates of AUD/USD 0.67 and USD/CLP 810.

3 Based on average exchange rates of: HY24 AUD/USD 0.65 USD/CLP 874 (realised); HY23 AUD/USD 0.67 USD/CLP 920 (realised).

4 Escondida unit costs for FY24 onwards exclude revenue based government royalties.

5 Medium term refers to FY25 and FY26.

6 The breakdown of C1 unit costs, excluding third party royalties, are detailed on page 12.

Health, safety and social value

Key safety indicators[1]

	Target/Goal	HY24	FY23	HY23
Fatalities[2]	Zero work-related fatalities	0	2	0
High-potential injury (HPI) frequency[2]	Year-on-year improvement in HPI frequency	0.10	0.18	0.13
Total recordable injury frequency (TRIF)[2]	Year-on-year improvement in TRIF	4.3	4.4	4.1

A fatal incident occurred subsequent to the reporting period at our BMA operations in January.

Social value: key indicators scorecard[1,3]

	Target/Goal	HY24	FY23	HY23
Operational greenhouse gas (GHG) emissions (Mt CO_2-e)[4]	Reduce operational GHG emissions by at least 30 per cent from FY20 levels[5] by FY30	5.2	9.8	4.9
Value chain emissions: Financial value committed in steelmaking partnerships and ventures to date (US$ m)	Steelmaking: 2030 goal to support industry to develop technologies and pathways capable of 30 per cent GHG emissions intensity reduction in integrated steelmaking, with widespread adoption expected post-2030	✓	114	✓
Value chain emissions: Reduction[6] in emissions intensity of BHP-chartered shipping of our products (%)	Maritime transportation: 2030 goal to support 40 per cent GHG emissions intensity reduction of BHP-chartered shipping of BHP products	43	41	42
Social investment (US$M BHP equity share)	Voluntary social investment aligned to the six pillars of our Social Value Framework	36.1	149.6	41.1
Indigenous procurement spend (US$M)	Purchases from Indigenous vendors target of US$297 million in FY24	289	332.6	141.1
Female employee representation (%)[7]	Aspirational goal for gender balance[8] by the end of FY25	36.2[9]	35.2	33.6
Indigenous employee representation (%)[7]	Australia[10]: aim to achieve 9.7 per cent by the end of FY27	8.4[11]	8.6	8.3
	Chile[12]: aim to achieve 10.0 per cent by the end of FY25	10.2	9.7	8.9
	Canada[13]: aim to achieve 20.0 per cent by the end of FY26	9.4	7.7	6.7
Area under nature-positive management practices[14] (hectares)	2030 goal of having at least 30 per cent of the land and water we steward[15] under conservation, restoration or regenerative practices	–	82,132	–

1 All data points are presented on a total operations basis, unless otherwise noted, and are indicative and subject to non-financial assurance reviews. Excludes former OZL, unless otherwise noted.

2 Former OZL is included in HY24 fatalities and HPI frequency. HY24 TRIF includes former OZL Exploration from 1 December 2023, reflecting progressive migration of employee data onto the BHP systems (updated data will be provided in the full year results for FY24). HPI frequency and TRIF is combined employee and contractor frequency per 1 million hours worked.

3 Includes selection of key social value framework metrics. Additional metrics are included in OFR 6 in the Annual Report 2023.

4 HY24 operational GHG emissions includes emissions from former OZL. HY23 operational GHG emissions have been restated after data finalisation for Annual Report 2023 and do not include former OZL emissions.

5 Our operational GHG emissions are the Scope 1 and Scope 2 GHG emissions from our operated assets. For our baseline year of FY20, our operational GHG emissions were 14.5 Mt CO2-e. FY20 baseline has been adjusted for divestment of our Petroleum business (merger with Woodside completed on 1 June 2022) and our interest in BMC (completed on 3 May 2022), and for methodological changes (use of Intergovernmental Panel on Climate Change (IPCC) Assessment Report 5 (AR5) Global Warming Potentials and the transition to a facility-specific GHG emission calculation methodology for fugitives at Caval Ridge). Our FY20 baseline year emissions will be updated to include the acquisition of OZL.

6 Against CY08. CY08 was selected as the baseline year for this goal to align with the base year for the International Maritime Organisation's 2030 emissions intensity goal and its corresponding reasoning and strategy.

7 Based on a 'point in time' snapshot of employees as at the end of the relevant reporting period.

8 We define gender balance as a minimum 40% women and 40% men in line with the definitions used by entities such as the International Labour Organisation.

9 Includes some but not all former OZL reflecting progressive migration of employee data onto BHP systems. Updated data will be provided in the full year results for FY24.

10 Indigenous employee representation at Minerals Australia operations.

11 Indigenous employee representation in Australia, including Minerals Australia operations and some but not all former OZL (operational and non-operational roles) reflecting progressive migration of employee data onto BHP systems. Updated data will be provided in the full year results for FY24.

12 Indigenous employee representation at Minerals Americas operations in Chile.

13 Indigenous employee representation at the Jansen Potash project and operations in Canada.

14 Area under our stewardship that has a formal management plan including conservation, restoration or regenerative practices. 82,132 hectares is the area as at 30 June 2023. This metric (which was previously reported as a percentage of the areas of land and water that we stewarded at 30 June 2023) is measured on an annual basis and an update, including restatement of the FY23 percentage to reflect a correction to underlying data, will be provided in the full year results for FY24.

15 Excluding greenfield exploration licences (or equivalent tenements), which are outside the area of influence of our existing mine operations. 30% will be calculated based on the areas of land and water that we steward at the end of FY30.

The Financial Report for the half year ended 31 December 2023 has been prepared on the basis of accounting policies and methods of computation consistent with those applied in the 30 June 2023 financial statements contained within the Annual Report of the Group. This news release including the Financial Report is unaudited. Variance analysis relates to the relative financial and/or production performance of BHP and/or its operations during the December 2023 half year compared with the December 2022 half year, unless otherwise noted. Medium term refers to a five-year horizon, unless otherwise noted. Numbers presented may not add up precisely to the totals provided due to rounding.

The following abbreviations may have been used throughout this report: billion tonnes (Bt); cost and freight (CFR); cost, insurance and freight (CIF); carbon dioxide equivalent (CO2-e); dry metric tonne unit (dmtu); free on board (FOB); giga litres (GL); greenhouse gas (GHG); grams per tonne (g/t); high-potential injury (HPI); kilograms per tonne (kg/t); kilometre (km); million ounces per annum (Mozpa); million pounds (Mlb); million tonnes (Mt); million tonnes per annum (Mtpa); ounces (oz); OZ Minerals Ltd (OZL); pounds (lb); thousand ounces (koz); thousand ounces per annum (kozpa); thousand tonnes (kt); thousand tonnes per annum (ktpa); thousand tonnes per day (ktpd); tonnes (t); total recordable injury frequency (TRIF); and wet metric tonnes (wmt).

The following footnotes apply to this Results Announcement:

i Based on a 'point in time' snapshot of employees as at 31 December 2023, including employees on extended absence, as used in internal management reporting for the purposes of monitoring progress against our goals. We define gender balance as a minimum 40% women and 40% men in line with the definitions used by entities such as the International Labour Organization. This includes some but not all former OZL reflecting progressive migration of employee data onto BHP systems. Updated data will be provided in the full year results for FY24. 'People leaders' are defined as employees with one or more direct reports.

ii We use various non-IFRS financial information to reflect our underlying performance. For further information on the reconciliations of certain non-IFRS financial information measures to our statutory measures, reasons for usefulness and calculation methodology, please refer to non-IFRS financial information.

iii Calculated on a copper equivalent production weighted average basis.

iv Data sourced from the most recent published financial statements of the ASX50 companies and excludes companies that recorded an accounting loss.

v Maintenance capital includes non-discretionary spend for the following purposes: deferred development and production stripping; risk reduction; compliance and asset integrity.

vi Subject to movements in exchange rates; +/- 50% in any given year.

vii Average for FY26-FY28.

viii The information in this section is based on BHP data, analysis and desk top research on public data sources.

ix There may be differences in the manner that third parties calculate or report unit costs data compared to BHP, which means that third-party data may not be comparable to our data. WAIO C1 unit costs exclude third party royalties, net inventory movements, depletion of production stripping, exploration expenses, marketing purchases, demurrage, exchange rate gains/losses, and other income.

x Resettlement cases completed includes completed construction (families either moved in or handover to families in progress) or cash payment made. Overall figures calculated considering total of 727 cases, which is the total of known cases as at 31 December 2023.

xi Relates to refined nickel metal only. Excludes intermediate products and nickel sulphate.

xii Expected capital intensity for Jansen Stage 2, US$/product tonne, Real 1 July 2023

Forward-looking statements

This release contains forward-looking statements, which involve risks and uncertainties. Forward-looking statements include all statements other than statements of historical or present facts, including: statements regarding: trends in commodity prices and currency exchange rates; demand for commodities; global market conditions, guidance; reserves and resources and production forecasts; expectations, plans, strategies and objectives of management; our expectations, commitments, targets, goals and objectives with respect to social value or sustainability; climate scenarios; approval of certain projects and consummation of certain transactions; closure, divestment, acquisition or integration of certain assets, operations or facilities (including associated costs or benefits); anticipated production or construction commencement dates; capital expenditure or costs and scheduling; operating costs, and supply of materials and skilled employees; anticipated productive lives of projects, mines and facilities; the availability, implementation and adoption of new technologies; provisions and contingent liabilities; and tax, legal and other regulatory developments.

Forward-looking statements may be identified by the use of terminology, including, but not limited to, 'intend', 'aim', 'ambition', 'aspiration', 'goal', 'target', 'prospect', 'project', 'see', 'anticipate', 'estimate', 'plan', 'objective', 'believe', 'expect', 'commit', 'may', 'should', 'need', 'must', 'will', 'would', 'continue', 'forecast', 'guidance', 'outlook', 'trend' or similar words. These statements discuss future expectations or performance, or provide other forward-looking information.

Forward-looking statements are based on management's expectations and reflect judgements, assumptions, estimates and other information available, as at the date made. BHP cautions against reliance on any forward-looking statements. These statements do not represent guarantees or predictions of future financial or operational performance and involve known and unknown risks, uncertainties and other factors, many of which are beyond our control and which may cause actual results to differ materially from those expressed in the statements contained in this release.

For example, our future revenues from our assets, projects or mines described in this release will be based, in part, on the market price of the commodities produced, which may vary significantly from current levels. These variations, if materially adverse, may affect the timing or the feasibility of the development of a particular project, the expansion of certain facilities or mines, or the continuation of existing assets.

In addition, there are limitations with respect to scenario analysis, including any climate-related scenario analysis, and it is difficult to predict which, if any, of the scenarios might eventuate. Scenario analysis is not an indication of probable outcomes and relies on assumptions that may or may not prove to be correct or eventuate.

Other factors that may affect the actual construction or production commencement dates, revenues, costs or production output and anticipated lives of assets, mines or facilities include our ability to profitably produce and deliver the products extracted to applicable markets; the impact of economic and geopolitical factors, including foreign currency exchange rates on the market prices of the commodities we produce and competition in the markets in which we operate; activities of government authorities in the countries where we sell our products and in the countries where we are exploring or developing projects, facilities or mines, including increases in taxes and royalties or implementation of trade or export restrictions; changes in environmental and other regulations, political or geopolitical uncertainty; labour unrest; weather, climate variability or other manifestations of climate change; and other factors

identified in the risk factors discussed in OFR 8.1 in the Annual Report and BHP's filings with the U.S. Securities and Exchange Commission (the 'SEC') (including in Annual Reports on Form 20-F) which are available on the SEC's website at www.sec.gov.

Except as required by applicable regulations or by law, BHP does not undertake to publicly update or review any forward-looking statements, whether as a result of new information or future events.

Past performance cannot be relied on as a guide to future performance.

No offer of securities

Nothing in this release should be construed as either an offer, or a solicitation of an offer, to buy or sell BHP securities in any jurisdiction, or be treated or relied upon as a recommendation or advice by BHP.

Reliance on third party information

The views expressed in this release contain information that has been derived from publicly available sources that have not been independently verified. No representation or warranty is made as to the accuracy, completeness or reliability of the information. This release should not be relied upon as a recommendation or forecast by BHP.

No financial or investment advice – South Africa

BHP does not provide any financial or investment 'advice' as that term is defined in the South African Financial Advisory and Intermediary Services Act, 37 of 2002, and we strongly recommend that you seek professional advice.

Emissions and energy consumption data

Due to the inherent uncertainty and limitations in measuring greenhouse gas (GHG) emissions and operational energy consumption under the calculation methodologies used in the preparation of such data, all GHG emissions and operational energy consumption data or references to GHG emissions and operational energy consumption volumes (including ratios or percentages) in this Report are estimates. There may also be differences in the manner that third parties calculate or report GHG emissions or operational energy consumption data compared to BHP, which means third-party data may not be comparable to our data. For information on how we calculate our GHG emissions and operational energy consumption data refer to the BHP Scopes 1, 2 and 3 GHG Emissions Calculation Methodology 2023 available at bhp.com/climate.

BHP and its subsidiaries

In this release, the terms 'BHP', the 'Company, the 'Group', 'BHP Group', 'our business', 'organisation', 'we', 'us', 'our' and ourselves' refer to BHP Group Limited and, except where the context otherwise requires, our subsidiaries. Refer to note 30 'Subsidiaries' of the Financial Statements in the Annual Report for a list of our significant subsidiaries. Those terms do not include non-operated assets.

This release covers BHP's functions and assets (including those under exploration, projects in development or execution phases, sites and closed operations) that have been wholly owned and/or operated by BHP or that have been owned as a joint venture[1] operated by BHP (referred to in this release as 'operated assets' or 'operations') during the period from 1 July 2023 to 31 December 2023.

BHP also holds interests in assets that are owned as a joint venture but not operated by BHP (referred to in this release as 'non-operated joint ventures' or 'non-operated assets'). Notwithstanding that this release may include production, financial and other information from non-operated assets, non-operated assets are not included in the BHP Group and, as a result, statements regarding our operations, assets and values apply only to our operated assets unless stated otherwise.

1 References in this release to a 'joint venture' are used for convenience to collectively describe assets that are not wholly owned by BHP. Such references are not intended to characterise the legal relationship between the owners of the asset.

Further information on BHP can be found at bhp.com

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Financial Report

Half year ended 31 December 2023

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Consolidated Income Statement for the half year ended 31 December 2023

	Notes	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Revenue		**27,232**	25,713	53,817
Other income		**261**	269	394
Expenses excluding net finance costs		**(19,982)**	(15,429)	(31,873)
(Loss)/profit from equity accounted investments, related impairments and expenses	3	**(2,708)**	280	594
Profit from operations		**4,803**	10,833	22,932
Financial expenses		**(1,164)**	(863)	(2,060)
Financial income		**343**	211	529
Net finance costs	4	**(821)**	(652)	(1,531)
Profit before taxation		**3,982**	10,181	21,401
Income tax expense		**(2,215)**	(3,038)	(6,691)
Royalty-related taxation (net of income tax benefit)		**(61)**	(17)	(386)
Total taxation expense	5	**(2,276)**	(3,055)	(7,077)
Profit after taxation		**1,706**	7,126	14,324
Attributable to non-controlling interests		**779**	669	1,403
Attributable to BHP shareholders		**927**	6,457	12,921
Basic earnings per ordinary share (cents)	6	**18.3**	127.5	255.2
Diluted earnings per ordinary share (cents)	6	**18.3**	127.3	254.7

The accompanying notes form part of this half year Financial Report.

Consolidated Statement of Comprehensive Income for the half year ended 31 December 2023

	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Profit after taxation	**1,706**	7,126	14,324
Other comprehensive income			
Items that may be reclassified subsequently to the income statement:			
Hedges:			
Gains/(losses) taken to equity	**114**	100	95
(Gains)/losses transferred to the income statement	**(92)**	(125)	(148)
Loss transferred to initial carrying amount of hedged item	**–**	–	35
Tax recognised within other comprehensive income	**(7)**	8	5
Total items that may be reclassified subsequently to the income statement	**15**	(17)	(13)
Items that will not be reclassified to the income statement:			
Re-measurement gains/(losses) on pension and medical schemes	**2**	6	(18)
Equity investments held at fair value	**(47)**	(7)	17
Tax recognised within other comprehensive income	**–**	–	7
Total items that will not be reclassified to the income statement	**(45)**	(1)	6
Total other comprehensive (loss)/income	**(30)**	(18)	(7)
Total comprehensive income	**1,676**	7,108	14,317
Attributable to non-controlling interests	**779**	669	1,400
Attributable to BHP shareholders	**897**	6,439	12,917

The accompanying notes form part of this half year Financial Report.

Consolidated Balance Sheet as at 31 December 2023

		31 Dec 2023 US$M	30 June 2023 US$M
ASSETS			
Current assets			
Cash and cash equivalents		**10,319**	12,428
Trade and other receivables		**5,352**	4,594
Other financial assets		**607**	470
Inventories		**5,313**	5,220
Assets held for sale	11	**1,570**	–
Current tax assets		**446**	508
Other		**193**	131
Total current assets		**23,800**	23,351
Non-current assets			
Trade and other receivables		**169**	148
Other financial assets		**1,050**	1,115
Inventories		**1,426**	1,403
Property, plant and equipment		**69,994**	71,818
Intangible assets		**1,615**	1,610
Investments accounted for using the equity method		**1,618**	1,620
Deferred tax assets		**76**	56
Other		**240**	175
Total non-current assets		**76,188**	77,945
Total assets		**99,988**	101,296
LIABILITIES			
Current liabilities			
Trade and other payables		**6,092**	6,296
Interest bearing liabilities		**2,839**	7,173
Liabilities directly associated with the assets held for sale	11	**752**	–
Other financial liabilities		**712**	402
Current tax payable		**290**	611
Provisions		**4,405**	4,514
Deferred income		**85**	47
Total current liabilities		**15,175**	19,043
Non-current liabilities			
Trade and other payables		**42**	4
Interest bearing liabilities		**19,565**	15,172
Other financial liabilities		**1,607**	2,157
Non-current tax payable		**39**	68
Deferred tax liabilities		**3,325**	4,299
Provisions		**14,594**	11,973
Deferred income		**50**	50
Total non-current liabilities		**39,222**	33,723
Total liabilities		**54,397**	52,766
Net assets		**45,591**	48,530
EQUITY			
Share capital		**4,819**	4,737
Treasury shares		**(33)**	(41)
Reserves		**(26)**	13
Retained earnings		**36,632**	39,787
Total equity attributable to BHP shareholders		**41,392**	44,496
Non-controlling interests		**4,199**	4,034
Total equity		**45,591**	48,530

The accompanying notes form part of this half year Financial Report.

Consolidated Cash Flow Statement for the half year ended 31 December 2023

	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Operating activities			
Profit before taxation	**3,982**	10,181	21,401
Adjustments for:			
Depreciation and amortisation expense	**2,629**	2,456	5,061
Impairments of property, plant and equipment, financial assets and intangibles	**3,513**	21	75
Net finance costs	**821**	652	1,531
Loss/(profit) from equity accounted investments, related impairments and expenses	**2,708**	(280)	(594)
Other	**290**	258	546
Changes in assets and liabilities:			
Trade and other receivables	**(763)**	888	867
Inventories	**(255)**	(53)	(44)
Trade and other payables	**(33)**	(1,598)	(1,086)
Provisions and other assets and liabilities	**(519)**	(399)	131
Cash generated from operations	**12,373**	12,126	27,888
Dividends received	**199**	75	347
Interest received	**352**	218	545
Interest paid	**(800)**	(434)	(1,090)
Proceeds from cash management related instruments	**311**	274	331
Net income tax and royalty-related taxation refunded	**175**	55	232
Net income tax and royalty-related taxation paid	**(3,726)**	(5,544)	(9,552)
Net operating cash flows	**8,884**	6,770	18,701
Investing activities			
Purchases of property, plant and equipment	**(4,545)**	(2,871)	(6,733)
Exploration and evaluation expenditure	**(199)**	(156)	(350)
Exploration and evaluation expenditure expensed and included in operating cash flows	**170**	127	294
Investment in subsidiaries, operations and joint operations, net of cash	**–**	–	(5,868)
Net investment and funding of equity accounted investments	**(474)**	(369)	(557)
Proceeds from sale of assets	**59**	81	444
Proceeds from sale of subsidiaries, operations and joint operations net of their cash	**55**	74	82
Other investing	**(145)**	(175)	(377)
Net investing cash flows	**(5,079)**	(3,289)	(13,065)
Financing activities			
Proceeds from interest bearing liabilities	**4,991**	350	8,182
Settlements of debt related instruments	**–**	(383)	(677)
Repayment of interest bearing liabilities	**(6,315)**	(1,690)	(3,289)
Purchase of shares by Employee Share Ownership Plan (ESOP) Trusts	**–**	(1)	(88)
Dividends paid	**(4,045)**	(8,660)	(13,268)
Dividends paid to non-controlling interests	**(614)**	(527)	(1,175)
Net financing cash flows	**(5,983)**	(10,911)	(10,315)
Net (decrease)/increase in cash and cash equivalents	**(2,178)**	(7,430)	(4,679)
Cash and cash equivalents, net of overdrafts, at the beginning of the period	**12,423**	17,236	17,236
Foreign currency exchange rate changes on cash and cash equivalents	**74**	(201)	(134)
Cash and cash equivalents, net of overdrafts, at the end of the period	**10,319**	9,605	12,423

The accompanying notes form part of this half year Financial Report.

Consolidated Statement of Changes in Equity for the half year ended 31 December 2023

	Attributable to BHP shareholders						
US$M	Share capital	Treasury shares	Reserves	Retained earnings	Total equity attributable to BHP shareholders	Non-controlling interests	Total equity
Balance as at 1 July 2023	**4,737**	**(41)**	**13**	**39,787**	**44,496**	**4,034**	**48,530**
Total comprehensive income	**–**	**–**	**(32)**	**929**	**897**	**779**	**1,676**
Transactions with owners:							
BHP Group Limited shares issued[1]	**82**	**(82)**	**–**	**–**	**–**	**–**	**–**
Purchase of shares by ESOP Trusts	**–**	**–**	**–**	**–**	**–**	**–**	**–**
Employee share awards exercised net of employee contributions net of tax	**–**	**90**	**(71)**	**(19)**	**–**	**–**	**–**
Vested employee share awards that have lapsed, been cancelled or forfeited	**–**	**–**	**–**	**–**	**–**	**–**	**–**
Accrued employee entitlement for unexercised awards net of tax	**–**	**–**	**64**	**–**	**64**	**–**	**64**
Dividends	**–**	**–**	**–**	**(4,065)**	**(4,065)**	**(614)**	**(4,679)**
Balance as at 31 December 2023	**4,819**	**(33)**	**(26)**	**36,632**	**41,392**	**4,199**	**45,591**
Balance as at 1 July 2022	4,638	(31)	12	40,338	44,957	3,809	48,766
Total comprehensive income	–	–	(24)	6,463	6,439	669	7,108
Transactions with owners:							
BHP Group Limited shares issued[1]	99	(99)	–	–	–	–	–
Purchase of shares by ESOP Trusts	–	(1)	–	–	(1)	–	(1)
Employee share awards exercised net of employee contributions net of tax	–	111	(80)	(31)	–	–	–
Vested employee share awards that have lapsed, been cancelled or forfeited	–	–	–	–	–	–	–
Accrued employee entitlement for unexercised awards net of tax	–	–	64	–	64	–	64
Dividends	–	–	–	(8,858)	(8,858)	(527)	(9,385)
Balance as at 31 December 2022	4,737	(20)	(28)	37,912	42,601	3,951	46,552

1 During the period, BHP Group Limited issued 2,919,231 fully paid ordinary shares to the BHP Group Limited Employee Equity Trust and to Solium Capital (Australia) Pty Ltd at A$43.52 per share (31 December 2022: 3,497,366 fully paid ordinary shares at A$40.51 per share), to satisfy the vesting of employee share awards and related dividend equivalent entitlements under those employee share plans.

The accompanying notes form part of this half year Financial Report.

Notes to the Financial Statements

1. Basis of preparation

This general purpose Financial Report for the half year ended 31 December 2023 is unaudited and has been prepared in accordance with IAS 34 'Interim Financial Reporting' as issued by the International Accounting Standards Board (IASB) and AASB 134 'Interim Financial Reporting' as issued by the Australian Accounting Standards Board (AASB) and the Australian Corporations Act 2001 as applicable to interim financial reporting. The general purpose Financial Report for the half year ended 31 December 2023 does not include all of the notes of the type normally included in an annual report. Accordingly, this report should be read in conjunction with the annual consolidated Financial Statements for the year ended 30 June 2023 and any public announcements made by the Group in accordance with the continuous disclosure obligations of the ASX Listing Rules.

Segment Reporting disclosures from IAS 34/AASB 134 'Interim Financial Reporting' have been disclosed within the Financial performance summary on pages 20 and 21 outside of this Financial Report.

The half year Financial Statements have been prepared on a basis of accounting policies and methods of computation consistent with those applied in the 30 June 2023 annual consolidated Financial Statements contained within the Annual Report of the Group, with the exception of new accounting standards that became effective for the Group from 1 July 2023. The adoption of these new accounting standards has not had a significant impact on the Group. A number of accounting standards and interpretations have been issued, and will be applicable in future periods. While these remain subject to ongoing assessment, no significant impacts have been identified to date. These standards have not been applied in the preparation of these half year Financial Statements.

All amounts are expressed in US dollars unless otherwise stated. The Group's presentation currency and the functional currency of the majority of its operations is US dollars as this is the principal currency of the economic environment in which it operates. Amounts in this Financial Report have, unless otherwise indicated, been rounded to the nearest million dollars.

The Directors have made an assessment of the Group's ability to continue as a going concern for the 12 months from the date of this report and consider it appropriate to adopt the going concern basis of accounting in preparing the half year Financial Statements.

2. Exceptional items

Exceptional items are those gains or losses where their nature, including the expected frequency of the events giving rise to them, and impact is considered material to the Financial Statements. Such items included within the Group's profit for the half year are detailed below.

Half year ended 31 December 2023	Gross US$M	Tax US$M	Net US$M
Exceptional items by category			
Samarco dam failure	**(3,120)**	**(53)**	**(3,173)**
Impairment of Western Australia Nickel assets	**(3,500)**	**1,031**	**(2,469)**
Total	**(6,620)**	**978**	**(5,642)**
Attributable to non-controlling interests	–	–	–
Attributable to BHP shareholders	**(6,620)**	**978**	**(5,642)**

Samarco Mineração SA (Samarco) dam failure

The loss of US$3,173 million (after tax) relates to the Samarco dam failure, which occurred in November 2015, and comprises the following:

Half year ended 31 December 2023	US$M
Expenses excluding net finance costs:	
Costs incurred directly by BHP Brasil and other BHP entities in relation to the Samarco dam failure	**(54)**
(Loss)/profit from equity accounted investments, related impairments and expenses:	
Samarco dam failure provision	**(2,982)**
Fair value change on forward exchange derivatives	**106**
Net finance costs	**(190)**
Income tax expense	**(53)**
Total[1]	**(3,173)**

1 Refer to note 9 'Significant events – Samarco dam failure' for further information.

Western Australia Nickel impairment

The Group determined that the overall recoverable amount of the Western Australia Nickel cash generating unit (CGU) to be approximately negative US$700 million including closure provisions, resulting in an impairment to property, plant and equipment of US$3,437 million and intangible assets of US$63 million. The impairment is predominantly driven by lower nickel price assumptions, uncertainty in the short-term and medium-term outlook for nickel, escalation in capital costs for Western Australia Nickel, and changes to the development plans for Western Australia Nickel. The Western Australia Nickel CGU is part of the 'Group and unallocated items' reportable segment.

The post-impairment carrying value of Western Australia Nickel property, plant and equipment is not material.

Recoverable amount used for the impairment assessment was determined using a fair value less costs of disposal (FVLCD) methodology, applying discounted cash flow techniques utilising a post-tax real discount rate of 7.5% per cent. FVLCD may take into consideration market-based indicators of fair value. FVLCD is based primarily on Level 3 inputs as defined in note 8 'Financial risk management – Fair values'.

The exceptional items relating to the half year ended 31 December 2022 and the year ended 30 June 2023 are detailed below.

Half year ended 31 December 2022	Gross US$M	Tax US$M	Net US$M
Exceptional items by category			
Samarco dam failure	(142)	2	(140)
Total	(142)	2	(140)
Attributable to non-controlling interests	–	–	–
Attributable to BHP shareholders	(142)	2	(140)

Year ended 30 June 2023	Gross US$M	Tax US$M	Net US$M
Exceptional items by category			
Samarco dam failure	(340)	17	(323)
Chilean tax reform	–	(283)	(283)
Total	(340)	(266)	(606)
Attributable to non-controlling interests	–	(107)	(107)
Attributable to BHP shareholders	(340)	(159)	(499)

3. Interests in associates and joint venture entities

The Group's major shareholdings in associates and joint venture entities, including their profit/(loss), are listed below:

	Ownership interest at the Group's reporting date			(Loss)/profit from equity accounted investments, related impairments and expenses		
	31 Dec 2023 %	31 Dec 2022 %	30 June 2023 %	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Share of profit/(loss) of equity accounted investments:						
Compañia Minera Antamina SA	**33.75**	33.75	33.75	**224**	185	451
Samarco Mineração SA[1]	**50.00**	50.00	50.00	**–**	–	–
Other				**(56)**	(32)	(72)
Share of profit of equity accounted investments				**168**	153	379
Samarco dam failure provision[1]				**(2,982)**	18	(256)
Fair value change on forward exchange derivatives[1]				**106**	109	471
(Loss)/profit from equity accounted investments, related impairments and expenses				**(2,708)**	280	594

1 Refer to note 9 'Significant events – Samarco dam failure' for further information.

4. Net finance costs

	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Financial expenses			
Interest expense using the effective interest rate method:			
Interest on bank loans, overdrafts and all other borrowings	**764**	487	997
Interest capitalised at 6.79% (31 December 2022: 5.20%; 30 June 2023: 5.71%)[1]	**(217)**	(114)	(271)
Interest on lease liabilities	**81**	62	130
Discounting on provisions and other liabilities	**479**	613	1,293
Other gains and losses:			
Fair value change on hedged loans	**345**	(754)	(803)
Fair value change on hedging derivatives	**(323)**	659	691
Exchange variations on net debt	**35**	(90)	9
Other	**–**	–	14
Total financial expenses	**1,164**	863	2,060
Financial income			
Interest income	**(343)**	(211)	(529)
Net finance costs	**821**	652	1,531

1 Interest has been capitalised at the rate of interest applicable to the specific borrowings financing the assets under construction or, where financed through general borrowings, at a capitalisation rate representing the average interest rate on such borrowings.

5. Income tax expense

	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Total taxation expense comprises:			
Current tax expense	**3,271**	2,738	6,690
Deferred tax (benefit)/expense	**(995)**	317	387
Total taxation expense	**2,276**	3,055	7,077

	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Factors affecting income tax expense for the year			
Income tax expense differs to the standard rate of corporation tax as follows:			
Profit before taxation	**3,982**	10,181	21,401
Tax on profit at Australian prima facie tax rate of 30 per cent	**1,195**	3,054	6,420
Tax effect of (loss)/profit from equity accounted investments, related impairments and expenses[1]	**844**	(52)	(37)
Derecognition of deferred tax assets and current year tax losses	**237**	162	526
Tax on remitted and unremitted foreign earnings	**96**	37	137
Amounts over provided in prior years	**(8)**	(5)	(18)
Foreign exchange adjustments	**(29)**	11	94
Recognition of previously unrecognised tax assets	**(73)**	(28)	(109)
Impact of tax rates applicable outside of Australia	**(244)**	(189)	(558)
Other	**197**	48	236
Income tax expense	**2,215**	3,038	6,691
Royalty-related taxation (net of income tax benefit)[2]	**61**	17	386
Total taxation expense	**2,276**	3,055	7,077

1 This item removes the prima facie tax effect on (loss)/profit from equity accounted investments, related impairments and expenses that are net of tax, with the exception of the Samarco forward exchange derivatives described in note 3 'Interests in associates and joint venture entities' which are taxable.

2 Includes the revaluation of deferred tax balances in the year ended 30 June 2023, following the substantive enactment of the Chilean Royalty Bill, as presented in note 2 'Exceptional items'.

6. Earnings per share

	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Earnings attributable to BHP shareholders (US$M)[1]	**927**	6,457	12,921
Weighted average number of shares (Million)			
- Basic[2]	**5,067**	5,064	5,064
- Diluted[3]	**5,078**	5,073	5,073
Earnings per ordinary share (US cents)[4]			
- Basic	**18.3**	127.5	255.2
- Diluted	**18.3**	127.3	254.7
Headline earnings per ordinary share (US cents)[5]			
- Basic	**67.0**	127.9	256.1
- Diluted	**66.9**	127.7	255.7

1 Diluted earnings attributable to BHP shareholders are equal to earnings attributable to BHP shareholders.

2 The calculation of the number of ordinary shares used in the computation of basic earnings per share is the weighted average number of ordinary shares of BHP Group Limited outstanding during the period after deduction of the number of shares held by the BHP Group Limited Employee Equity Trust.

3 For the purposes of calculating diluted earnings per share, the effect of 11 million dilutive shares has been taken into account for the half year ended 31 December 2023 (31 December 2022: 9 million shares; 30 June 2023: 9 million shares). The Group's only potential dilutive ordinary shares are share awards granted under employee share ownership plans. Diluted earnings per share calculation excludes instruments which are considered antidilutive.

At 31 December 2023, there are no instruments which are considered antidilutive (31 December 2022: nil; 30 June 2023: nil).

4 Each American Depositary Share (ADS) represents twice the earnings for BHP Group Limited ordinary share.

5 Headline earnings is a Johannesburg Stock Exchange defined performance measure and is reconciled from earnings attributable to ordinary shareholders as follows:

	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Earnings attributable to BHP shareholders	927	6,457	12,921
Adjusted for:			
(Gain)/loss on sales of PP&E, Investments and Operations[i]	**(37)**	2	(9)
Impairments of property, plant and equipment, financial assets and intangibles	**3,530**	21	75
Tax effect of above adjustments	**(1,023)**	(1)	(17)
Subtotal of adjustments	**2,470**	22	49
Headline earnings	**3,397**	6,479	12,970
Diluted headline earnings	**3,397**	6,479	12,970

i Included in other income.

7. Dividends

	Half year ended 31 Dec 2023		Half year ended 31 Dec 2022		Year ended 30 June 2023	
	Per share US cents	Total US$M	Per share US cents	Total US$M	Per share US cents	Total US$M
Dividends paid during the period						
Prior year final dividend	**80.0**	**4,065**	175.0	8,858	175.0	8,858
Interim dividend	**N/A**	**–**	N/A	–	90.0	4,562
	80.0	**4,065**	175.0	8,858	265.0	13,420

Dividends paid during the period differs from the amount of dividends paid in the Consolidated Cash Flow Statement as a result of foreign exchange gains and losses relating to the timing of equity distributions between the record date and the payment date. An additional derivative settlement of US$23 million was made as part of the funding of the final dividend paid during the period and is disclosed in 'Proceeds from cash management related instruments' in the Consolidated Cash Flow Statement.

Each American Depositary Share (ADS) represents two ordinary shares of BHP Group Limited. Dividends determined on each ADS represent twice the dividend determined on each BHP Group Limited ordinary share.

Dividends are determined after period-end and announced with the results for the period. Interim dividends are determined in February and paid in March. Final dividends are determined in August and paid in September. Dividends determined are not recorded as a liability at the end of the period to which they relate. Subsequent to the half year, on 20 February 2024, BHP Group Limited determined an interim ordinary dividend of 72 US cents per share (US$3,649 million), which will be paid on 28 March 2024 (31 December 2022: interim dividend of 90 US cents per share – US$4,559 million; 30 June 2023: final dividend of 80 US cents per share – US$4,052 million).

BHP Group Limited dividends for all periods presented are, or will be, fully franked based on a tax rate of 30 per cent.

8. Financial risk management – Fair values

Recognition and measurement

All financial assets and liabilities, other than derivatives and trade receivables, are initially recognised at the fair value of consideration paid or received, net of transaction costs as appropriate. Financial assets are initially recognised on their trade date.

Financial assets are subsequently carried at fair value or amortised cost based on:

- the Group's purpose, or business model, for holding the financial asset;
- whether the financial asset's contractual terms give rise to cash flows that are solely payments of principal and interest.

The resulting Financial Statements classifications of financial assets can be summarised as follows:

Contractual cash flows	Business model	Category
Solely principal and interest	Hold in order to collect contractual cash flows	Amortised cost
Solely principal and interest	Hold in order to collect contractual cash flows and sell	Fair value through other comprehensive income
Solely principal and interest	Hold in order to sell	Fair value through profit or loss
Other	Any of those mentioned above	Fair value through profit or loss

Solely principal and interest refers to the Group receiving returns only for the time value of money and the credit risk of the counterparty for financial assets held. The main exceptions for the Group are provisionally priced receivables and derivatives which are measured at fair value through profit or loss under IFRS 9/AASB 9 'Financial Instruments'.

The Group has the intention of collecting payment directly from its customers in most cases, however the Group also participates in receivables financing programs in respect of selected customers. Receivables in these portfolios which are classified as 'hold in order to sell', are provisionally priced receivables and are therefore held at fair value through profit or loss prior to sale to the financial institution.

With the exception of derivative contracts and provisionally priced trade payables which are carried at fair value through profit or loss, the Group's financial liabilities are classified as subsequently measured at amortised cost.

The Group may in addition elect to designate certain financial assets or liabilities at fair value through profit or loss or to apply hedge accounting where they are not mandatorily held at fair value through profit or loss.

Fair value measurement

The carrying amount of financial assets and liabilities measured at fair value is principally calculated based on inputs other than quoted prices that are observable for these financial assets or liabilities, either directly (i.e. as unquoted prices) or indirectly (i.e. derived from prices). Where no price information is available from a quoted market source, alternative market mechanisms or recent comparable transactions, fair value is estimated based on the Group's views on relevant future prices, net of valuation allowances to accommodate liquidity, modelling and other risks implicit in such estimates.

The inputs used in fair value calculations are determined by the relevant segment or function. The functions support the assets and operate under a defined set of accountabilities authorised by the Executive Leadership Team. Movements in the fair value of financial assets and liabilities may be recognised through the income statement or in other comprehensive income according to the designation of the underlying instrument.

For financial assets and liabilities carried at fair value, the Group uses the following to categorise the inputs to the valuation method used based on the lowest level input that is significant to the fair value measurement as a whole:

IFRS 13 Fair value hierarchy	Level 1	Level 2	Level 3
Valuation inputs	Based on quoted prices (unadjusted) in active markets for identical financial assets and liabilities.	Based on inputs other than quoted prices included within Level 1 that are observable for the financial asset or liability, either directly (i.e. as unquoted prices) or indirectly (i.e. derived from prices).	Based on inputs not observable in the market using appropriate valuation models, including discounted cash flow modelling.

Financial assets and liabilities

The financial assets and liabilities are presented by class in the table below at their carrying amounts.

	IFRS 13 Fair value hierarchy level[1]	IFRS 9 Classification	31 Dec 2023 US$M	30 June 2023 US$M
Current cross currency and interest rate swaps[2]	2	Fair value through profit or loss	**40**	34
Current other derivative contracts[3]	2,3	Fair value through profit or loss	**281**	407
Current other financial assets[4]	3	Fair value through profit or loss	**251**	–
Current other investments[5]	1,2	Fair value through profit or loss	**35**	29
Non-current cross currency and interest rate swaps[2]	2	Fair value through profit or loss	**318**	149
Non-current other derivative contracts[3]	2,3	Fair value through profit or loss	**179**	228
Non-current other financial assets[4]	3	Fair value through profit or loss	**–**	246
Non-current other financial assets[6]		Amortised cost	**130**	–
Non-current investment in shares	1,3	Fair value through other comprehensive income	**181**	224
Non-current other investments[5]	1,2	Fair value through profit or loss	**242**	268
Total other financial assets			**1,657**	1,585
Cash and cash equivalents		Amortised cost	**10,319**	12,428
Trade and other receivables[7]		Amortised cost	**1,919**	1,506
Provisionally priced trade receivables	2	Fair value through profit or loss	**3,007**	2,705
Total financial assets			**16,902**	18,224
Non-financial assets			**83,086**	83,072
Total assets			**99,988**	101,296
Current cross currency and interest rate swaps[2]	2	Fair value through profit or loss	**316**	147
Current other derivative contracts	2	Fair value through profit or loss	**325**	176
Current other financial liabilities[8]		Amortised cost	**71**	79
Non-current cross currency and interest rate swaps[2]	2	Fair value through profit or loss	**1,225**	1,608
Non-current other derivative contracts[3]	2,3	Fair value through profit or loss	**14**	82
Non-current other financial liabilities[8]		Amortised cost	**368**	467
Total other financial liabilities			**2,319**	2,559
Trade and other payables[9]		Amortised cost	**5,429**	5,338
Provisionally priced trade payables	2	Fair value through profit or loss	**591**	841
Bank overdrafts and short-term borrowings[10]		Amortised cost	**–**	5
Bank loans[10]		Amortised cost	**2,543**	7,502
Notes and debentures[10]		Amortised cost	**16,209**	11,819
Lease liabilities[11]			**3,578**	3,019
Other[10]		Amortised cost	**74**	–
Total financial liabilities			**30,743**	31,083
Non-financial liabilities			**23,654**	21,683
Total liabilities			**54,397**	52,766

1 All of the Group's financial assets and financial liabilities recognised at fair value were valued using market observable inputs categorised as Level 2 unless specified otherwise in the following footnotes.

2 Cross currency and interest rate swaps are valued using market data including interest rate curves and foreign exchange rates. A discounted cash flow approach is used to derive the fair value of cross currency and interest rate swaps at the reporting date.

3 Includes net other derivative assets of US$90 million related to power purchase contract agreements that are categorised as Level 3 (30 June 2023: US$46 million).

4 Includes receivables contingent on future coal price and on outcome of future events relating to mining and regulatory approvals of US$251 million (30 June 2023: US$246 million).

5 Includes investments held by BHP Foundation which are restricted and not available for general use by the Group of US$271 million (30 June 2023: US$290 million) of which other investments (mainly US Treasury Notes) of US$133 million is categorised as Level 1 (30 June 2023: US$138 million).

6 Includes Senior notes relating to Samarco with a maturity date of 30 June 2031. Refer to note 9 'Significant events – Samarco dam failure' for further information.

7 Excludes input taxes of US$595 million (30 June 2023: US$531 million) included in other receivables.

8 Includes the discounted settlement liability in relation to the cancellation of power contracts at the Group's Escondida operations.

9 Excludes input taxes of US$114 million (30 June 2023: US$121 million) included in other payables.

10 All interest bearing liabilities, excluding lease liabilities, are unsecured.

11 Lease liabilities are measured in accordance with IFRS 16/AASB 16 'Leases'.

The carrying amounts in the table above generally approximate to fair value. In the case of US$533 million (30 June 2023: US$534 million) of fixed rate debt not swapped to floating rate, the fair value at 31 December 2023 was US$550 million (30 June 2023: US$538 million). The fair value is determined using a method that can be categorised as Level 2 and uses inputs based on benchmark interest rates, alternative market mechanisms or recent comparable transactions.

For financial instruments that are carried at fair value on a recurring basis, the Group determines whether transfers have occurred between levels in the fair value hierarchy by reassessing categorisation at the end of each reporting period. There were no transfers between categories during the period.

9. Significant events – Samarco dam failure

As a result of the Samarco dam failure on 5 November 2015, BHP Billiton Brasil Ltda (BHP Brasil) and other Group entities continue to incur costs and maintain liabilities for future costs. The information presented in this note should be read in conjunction with section 7 'Samarco', Financial Statements note 4 'Significant events – Samarco dam failure' and Additional information section 8 'Legal proceedings' in the 30 June 2023 Annual Report.

The financial impacts of the Samarco dam failure on the Group's income statement, balance sheet and cash flow statement for the half year ended 31 December 2023 are shown below and have been treated as an exceptional item.

Financial impacts of Samarco dam failure	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Income statement			
Expenses excluding net finance costs:			
Costs incurred directly by BHP Brasil and other BHP entities in relation to the Samarco dam failure[1]	(54)	(47)	(103)
(Loss)/profit from equity accounted investments, related impairments and expenses:			
Samarco dam failure provision[2]	(2,982)	18	(256)
Fair value change on forward exchange derivatives[3]	106	109	471
(Loss)/profit from operations	(2,930)	80	112
Net finance costs[4]	(190)	(222)	(452)
Loss before taxation	(3,120)	(142)	(340)
Income tax (expense)/benefit[5]	(53)	2	17
Loss after taxation	(3,173)	(140)	(323)
Balance sheet movement			
Other financial assets[6]	130	–	–
Trade and other payables	(12)	–	(6)
Derivatives	(36)	83	337
Tax liabilities	(53)	2	17
Provisions	(2,851)	135	(260)
Net (increase)/decrease in liabilities	(2,822)	220	88

	Half year ended 31 Dec 2023 US$M	Half year ended 31 Dec 2022 US$M	Year ended 30 June 2023 US$M
Cash flow statement			
Loss before taxation	**(3,120)**	(142)	(340)
Adjustments for:			
Samarco dam failure provision[2]	**2,982**	(18)	256
Fair value change on forward exchange derivatives[3]	**(106)**	(109)	(471)
Proceeds from cash management related instruments	**142**	26	134
Net finance costs[4]	**190**	222	452
Changes in assets and liabilities:			
Trade and other payables	**12**	–	6
Net operating cash flows	**100**	(21)	37
Net investment and funding of equity accounted investments[7]	**(446)**	(339)	(448)
Net investing cash flows	**(446)**	(339)	(448)
Net decrease in cash and cash equivalents	**(346)**	(360)	(411)

1 Includes legal and advisor costs incurred.

2 US$3,000 million (31 December 2022: US$(33) million; 30 June 2023: US$(33) million) change in estimate and US$(18) million (31 December 2022: US$15 million; 30 June 2023: US$289 million) exchange translation.

3 The Group enters into forward exchange contracts to limit the Brazilian reais exposure on the dam failure provisions. While not applying hedge accounting, the fair value changes in the forward exchange instruments are recorded within (Loss)/profit from equity accounted investments, related impairments and expenses in the Income Statement.

4 Amortisation of discounting of provision.

5 Includes tax on forward exchange derivatives and other taxes incurred during the period.

6 Senior notes relating to the Judicial Reorganisation (JR), described below, with a maturity date of 30 June 2031.

7 Includes US$(321) million (31 December 2022: US$(339) million; 30 June 2023: US$(448) million) utilisation of the Samarco dam failure provision and US$(125) million provided to Samarco following approval of the JR (31 Dec 2022: US$ nil; 30 June 2023: US$ nil) which subsequently converted into the Senior notes referred to in footnote 6, inclusive of accrued interest.

Equity accounted investment in Samarco

BHP Brasil's investment in Samarco remains at US$ nil. No dividends have been received by BHP Brasil from Samarco during the period and Samarco currently does not have profits available for distribution.

Provision related to the Samarco dam failure

		31 Dec 2023 US$M		30 June 2023 US$M
At the beginning of the reporting period		**3,681**		3,421
Movement in provisions		**2,851**		260
Comprising:				
Utilised	**(321)**		(448)	
Adjustments charged to the income statement:				
Change in cost estimate	**3,000**		(33)	
Amortisation of discounting impacting net finance costs	**190**		452	
Exchange translation	**(18)**		289	
At the end of the reporting period		**6,532**		3,681
Comprising:				
Current		**1,871**		1,876
Non-current		**4,661**		1,805
At the end of the reporting period		**6,532**		3,681

Samarco dam failure provision and contingencies

As at 31 December 2023, BHP Brasil has identified a provision and certain contingent liabilities arising as a consequence of the Samarco dam failure. The provision related to the Samarco dam failure recognised as at 31 December 2023 is US$6,532 million and reflects the Group's best estimate of the potential outflows necessary to resolve all aspects of the Federal Public Prosecution Office BRL$155 billion claim and Framework Agreement obligations (see below).

Contingent liabilities will only be resolved when one or more uncertain future events occur or related impacts become capable of reliable measurement and, as such, determination of contingent liabilities disclosed in the financial statements requires significant judgement regarding the outcome of future events. A number of the claims below do not specify the amount of damages sought and, where this is specified, amounts could change as the matter progresses.

Ultimately, future changes in all those matters for which a provision has been recognised or contingent liability disclosed could have a material adverse impact on BHP's business, competitive position, cash flows, prospects, liquidity and shareholder returns.

The following table summarises the current status of significant ongoing matters relating to the Samarco dam failure, along with developments during the period, and the associated treatment in the Financial Statements:

Item	Provision	Contingent liability
Samarco dam failure – Framework Agreement		
On 2 March 2016, BHP Brasil, Samarco and Vale S.A. (Vale) entered into a Framework Agreement with the Federal Government of Brazil, the states of Espirito Santo and Minas Gerais, and certain other public authorities to establish a foundation (Fundação Renova) that is developing and executing environmental and socio-economic programs (Programs) to remediate and provide compensation for damage caused by the Samarco dam failure (the Framework Agreement). Key programs include those for financial assistance and compensation of impacted persons and those for remediation of impacted areas and resettlement of impacted communities. Samarco has primary responsibility for funding Fundação Renova with each of BHP Brasil and Vale having secondary funding obligations in proportion to their 50 per cent shareholding in Samarco. While Samarco has recommenced operations, Samarco's long-term cash flow generation remains highly sensitive to factors including returning to full production capacity, commodity prices and foreign exchange rates. Further, under the Samarco Judicial Reorganisation (refer to Samarco Judicial Reorganisation (JR) below), Samarco's funding of obligations to remediate and compensate the damages resulting from the dam failure, including funding Fundação Renova, is capped at US$1 billion for the period CY2024 to CY2030. Notwithstanding this cap, and subject to certain conditions, to the extent that Samarco each year has a positive cash balance after meeting its various obligations, during this period Samarco's shareholders are able to direct 50 per cent of Samarco's year end excess cash balance to fund remediation and compensation obligations. Execution of the Programs is a key component in the resolution of the Federal Public Prosecution Office claim (outlined below). Therefore, the expected cost of completion of the Programs and Samarco's potential ability to contribute to remediation and compensation obligations have been considered when determining BHP Brasil's provision in relation to the Samarco dam failure at 31 December 2023. Uncertainty exists around the scope and cost of the Programs, including as a result of ongoing legal actions in relation to the number of individuals eligible for compensation and the amount of damages to which they are entitled. Further information on the key areas of estimation uncertainty is provided in the 'Key judgements and estimates' section below.	✓	✗
Federal Public Prosecution Office claim		
BHP Brasil is among the defendants named in a claim brought by the Brazilian Federal Public Prosecution Office on 3 May 2016, seeking R$155 billion (approximately US$32 billion) for reparation, compensation and moral damages in relation to the Samarco dam failure. Since early CY2021, BHP Brasil, Samarco and, Vale have been engaging in negotiations with the Brazilian State and Federal Government and other public entities to seek a settlement of obligations under the Framework Agreement, the Federal Public Prosecution Office Claim, and other claims by government entities relating to the Samarco dam failure. On 25 January 2024, the Federal Court of Brazil issued a decision in relation to the Federal Public Prosecution Office Claim quantifying collective moral damages arising from the Samarco dam failure. The decision found that Samarco, Vale and BHP Brasil are jointly and severally liable to pay collective moral damages in the amount of R$47.6 billion (US$9.75 billion) (to be adjusted for interest and inflation) when any and all appeals are finally determined. On 1 February 2024, Samarco, Vale and BHP Brasil filed a clarification motion with the Federal Court of Brazil in respect of certain factual inaccuracies in the decision, including the calculation of damages. A decision remains pending. BHP Brasil also intends to appeal the decision, challenging the merits and amount of damages. The appeal process is estimated to take approximately two to five years.	✓	✗

Item	Provision	Contingent liability
As at 30 June 2023, the Group disclosed a contingent liability in relation to the Federal Public Prosecution Office claim as, given the status of the claim and ongoing settlement negotiations, it was not possible to reliably estimate the potential outcomes of the claim beyond the estimated costs of completing the programs under the Framework Agreement, which are being executed in relation to financial assistance and compensation of impacted persons, remediation of impacted areas and resettlement of impacted communities. The Group has considered the additional information available from the status of settlement negotiations, the judicial decision regarding collective moral damages, updates to the estimated costs of executing the Framework Agreement programs, and the extent to which Samarco may be in a position to fund any future outflows to increase the provision related to the Samarco dam failure to US$6,532 million at 31 December 2023. The provision at 31 December 2023 reflects the Group's best estimate of outflows required to resolve all aspects of the Federal Public Prosecution Office claim, being reparation, compensation and moral damages. Significant uncertainty remains around the resolution of the Federal Public Prosecution Office Claim and the Framework Agreement obligations, and there is a risk that outcomes may be materially higher or lower than amounts reflected in BHP Brasil's provision for the Samarco dam failure. Key areas of uncertainty include the outcomes of appeals relating to the judicial decision regarding collective moral damages, the terms of any potential future settlement agreement in respect of the Federal Public Prosecution Office claim and the extent to which Samarco is able to directly fund any future obligations relating to reparation, compensation and moral damages. Further information on the key areas of estimation uncertainty is provided in the 'Key judgements and estimates' section below. BHP Brasil, Samarco and Vale continue to maintain security, as required by a Governance Agreement, ratified on 8 August 2018, with the security currently comprising insurance bonds and a charge over certain Samarco assets.		
Australian class action claim BHP Group Limited is named as a defendant in a shareholder class action filed in the Federal Court of Australia on behalf of persons who acquired shares in BHP Group Limited on the Australian Securities Exchange (ASX) or shares in BHP Group Plc (now BHP Group (UK) Ltd) on the London Stock Exchange (LSE) and Johannesburg Stock Exchange (JSE) in periods prior to the Samarco dam failure. The amount of damages sought is unspecified.	✗	✓
United Kingdom group action claims BHP Group (UK) Ltd (formerly BHP Group Plc) and BHP Group Limited (BHP Defendants) are named as defendants in group action claims for damages filed in the courts of England. These claims were filed on behalf of certain individuals, municipalities, businesses, faith-based institutions and communities in Brazil allegedly impacted by the Samarco dam failure. The amount of damages sought in these claims is unspecified. A trial in relation to the BHP Defendant's liability for the dam failure is listed for October 2024. In December 2022, the BHP Defendants filed their defence and a contribution claim against Vale. The contribution claim contends that if the BHP Defendant's defence is not successful and the BHP Defendants are ordered to pay damages to the claimants, Vale should contribute to any amount payable. Vale contested the jurisdiction of the English courts to determine this contribution claim on the basis that the Brazilian courts are a more appropriate forum, with the court dismissing Vale's application on 7 August 2023. In November 2023, the English courts refused Vale's request for permission to appeal that decision. Vale also challenged the jurisdiction of the English courts on the basis that the claims are covered by an arbitration clause and should be submitted to arbitration in Brazil. In December 2023, the English courts dismissed Vale's arbitration challenge, and Vale has not sought permission to appeal from that decision. Accordingly, the contribution claim will proceed in the UK. In January 2024, the BHP Defendants were served with a new group action filed in the courts of England on behalf of additional individuals and businesses in Brazil allegedly impacted by the Samarco dam failure. The new action makes broadly the same claims as the original action and the amount of damages sought in these claims is unspecified.	✗	✓

Item	Provision	Contingent liability
Criminal charges		
The Federal Prosecutors' Office has filed criminal charges against BHP Brasil, Samarco and Vale and certain employees and former employees of BHP Brasil (Affected Individuals) in the Federal Court of Ponte Nova, Minas Gerais. BHP Brasil rejects outright the charges against the company and the Affected Individuals and is defending itself from all charges while fully supporting each of the Affected Individuals in their defence of the charges.	✗	✓
Civil public action commenced by Associations concerning the use of TANFLOC for water treatment		
The Vila Lenira Residents Association, State of Espirito Santo Rural Producers and Artisans Association, Colatina Velha Neighbourhood Residents Association, and United for the Progress of Palmeiras Neighbourhood Association have filed a lawsuit against Samarco, BHP Brasil and Vale and others, including the State of Minas Gerais, the State of Espirito Santo and the Federal Government. The plaintiffs allege that the defendants carried out a clandestine study on the citizens of the locations affected by the Fundão Dam Failure, using TANFLOC – a tannin-based flocculant/coagulant – that is currently used for wastewater treatment applications. The plaintiffs claim that this product allegedly put the population at risk due to its alleged experimental qualities. The plaintiffs are seeking multiple kinds of relief – material damages, moral damages, loss of profits – and that the defendants should pay for water supply in all locations where there is no water source other than the Doce River. On 17 November 2023, the Court dismissed the case without prejudice on the basis of a lack of standing by the Associations to sue and procedural deficiencies in the complaint filed. The plaintiffs filed a motion for clarification and a decision is pending.	✗	✓
Other claims		
BHP Brasil is among the companies named as defendants in a number of legal proceedings initiated by individuals, non-governmental organisations, corporations and governmental entities in Brazilian Federal and State courts following the Samarco dam failure. The other defendants include Vale, Samarco and Fundação Renova. The lawsuits include claims for compensation, environmental reparation and violations of Brazilian environmental and other laws, among other matters. The lawsuits seek various remedies including reparation costs, compensation to injured individuals and families of the deceased, recovery of personal and property losses, moral damages and injunctive relief. In addition, government inquiries and investigations relating to the Samarco dam failure have been commenced by numerous agencies and individuals of the Brazilian government and are ongoing. Additional lawsuits and government investigations relating to the Samarco dam failure could be brought against BHP Brasil and other Group entities in Brazil or other jurisdictions. The outcomes of these claims and proceedings remain uncertain and continue to be disclosed as contingent liabilities.	✗	✓

Commitments

Under the terms of the Samarco joint venture agreement, BHP Brasil does not have an existing obligation to fund Samarco.

However, BHP Brasil has agreed to fund a total of up to US$925 million for the Fundação Renova programs during calendar year 2024. Any additional requests for funding or future investment provided would be subject to a future decision by BHP Brasil, accounted for at that time.

Samarco judicial reorganisation

Samarco filed for Judicial Reorganisation (JR) in April 2021, with the Second Business State Court for the Belo Horizonte District of Minas Gerais, State of Minas Gerais, Brazil (JR Court), following enforcement actions taken by certain financial creditors of Samarco which threatened Samarco's operations.

The JR was an insolvency proceeding that provided a means for Samarco to restructure its financial debts and establish a stable financial position to allow Samarco to continue to rebuild its operations and strengthen its ability to meet obligations in relation to reparation, compensation and moral damages in relation to the Samarco dam failure. Samarco's operations continued during the JR proceeding.

On 28 July 2023, Samarco and one of the financial creditors jointly filed a consensual Judicial Reorganisation Plan (Consensual Plan) with the JR Court. The parties also filed terms of adhesion that demonstrate approval of the Consensual Plan by the majority of Samarco's creditors as required under Brazilian Bankruptcy Law.

On 1 September 2023, the JR Court ratified the Consensual Plan. Following the ratification, Samarco entered into definitive debt restructure agreements with its financial creditors to implement the debt restructure, including the exchange of Samarco's existing financial debt for US$3.6 billion of long-term unsecured debt that matures in June 2031 and remains non-recourse to Samarco's shareholders. Further, as part of the agreement Samarco issued Senior notes to its Shareholders which also mature in June 2031.

The debt restructure does not impact Fundação Renova's ability to undertake the Programs under the Framework Agreement. Samarco continues to have primary responsibility for funding Fundação Renova and each of BHP Brasil and Vale will continue to have secondary responsibility to fund 50% of Fundação Renova if Samarco does not meet its funding obligations under the Framework Agreement. Under the Consensual Plan, Samarco's funding obligation to remediate and compensate the damages resulting from the dam failure, including funding Fundação Renova, is capped at US$1 billion for the period CY2024 to CY2030 (Renova Cap). Notwithstanding the Renova Cap, and subject to certain conditions, to the extent that Samarco each year has a positive cash balance after meeting its various obligations including operating capital requirements, debt service and Renova Cap requirements, Samarco's shareholders are able to direct 50% of Samarco's year end excess cash balance to fund remediation and compensation obligations.

BHP Brasil has considered the extent to which Samarco may be in a position to fund any future outflows, when determining the dam failure related provision at 31 December 2023.

Key judgements and estimates

Judgements

The outcomes of litigation are inherently difficult to predict and significant judgement has been applied in assessing the likely outcome of legal claims and determining which legal claims require recognition of a provision or disclosure of a contingent liability. The facts and circumstances relating to these cases are regularly evaluated in determining whether a provision for any specific claim is required.

Management has determined that a provision can be recognised at 31 December 2023 to reflect the estimated costs to resolve all aspects of the Federal Public Prosecution Office claim and the Framework Agreement. It is not yet possible to provide a range of possible outcomes or a reliable estimate of potential future exposures to BHP in connection to the contingent liabilities noted above, given their status.

Estimates

The provision for the Samarco dam failure reflects the Group's estimate of the costs to resolve all aspects of the Federal Public Prosecution Office claim and Framework Agreement and requires the use of significant judgements, estimates and assumptions.

While the provision has been measured based on the latest information available, changes in facts and circumstances are likely in future reporting periods and may lead to material revisions to these estimates and there is a risk that outcomes may be materially higher or lower than amounts currently reflected in the provision. However, it is currently not possible to determine what facts and circumstances may change, therefore revisions in future reporting periods due to the key estimates and factors outlined below cannot be reliably measured.

The key estimates that may have a material impact upon the provision in the next and future reporting periods include the:

- the scope and cost of completing the programs under the Framework Agreement, including as a result of ongoing legal actions in relation to the number of people eligible for compensation and the amount of damages to which they are entitled;
- the outcomes of appeals relating to the judicial decision regarding collective moral damages, including any appeals that may be lodged by the Brazilian Federal Public Prosecution Office;
- the terms of any potential future settlement agreement in respect of the Federal Public Prosecution Office Claim, including amounts payable, obligations of the parties to perform ongoing programs of work in relation to reparation and compensation, and the period of time over which any settlement amounts may be payable; and
- the extent to which Samarco is able to directly fund any future obligations relating to reparation, compensation or moral damages. Samarco's long-term cash flow generation remains highly sensitive to factors including its ability to return to full production capacity, commodity prices and foreign exchange rates.

The provision may also be affected by factors including but not limited to updates to discount and foreign exchange rates.

In addition, the provision may be impacted by decisions in, or resolution of, existing and potential legal claims in Brazil and other jurisdictions, including the outcome of the United Kingdom group action claims.

Given these factors, future actual cash outflows may differ from the amounts currently provided and changes to any of the key assumptions and estimates outlined above could result in a material impact to the provision in the next and future reporting periods.

10. Business combinations

On 2 May 2023 (Acquisition Date), the Group acquired 100 per cent of the issued share capital of OZ Minerals Limited (OZL) for a net cash consideration of US$5.9 billion. The terms of the acquisition did not include any contingent consideration.

There have been no significant adjustments to the provisional fair values as at 31 December 2023. Due to the size, complexity and timing of the acquisition, the valuation process is ongoing and will be completed within 12 months of the acquisition.

Refer to note 29 of the 30 June 2023 annual consolidated Financial Statements contained within the Annual Report of the Group for details of the acquisition.

11. Assets and liabilities directly associated with assets held for sale

On 18 October 2023, the Group announced that BHP and Mitsubishi Development Pty Ltd (MDP) had entered into an Asset Sale Agreement to divest the Blackwater and Daunia mines (which are part of the BHP Mitsubishi Alliance (BMA) metallurgical coal joint venture in Queensland) to Whitehaven Coal (Buyer). Each of BHP and MDP hold a 50% interest in BMA.

The purchase price comprises US$2.1 billion cash on completion, US$1.1 billion in cash over 3 years after completion and the potential for up to US$0.9 billion in a price-linked earnout payable over 3 years on a 100% interest basis. Subject to the satisfaction of certain conditions, including customary competition and regulatory requirements, transaction completion is expected to occur in the June 2024 quarter. The Buyer has paid a US$100 million deposit (US$50 million BHP share) on signing which BHP and MDP are entitled to retain in certain limited circumstances if the proposed divestment is terminated.

BMA will continue to operate the assets until completion. The Buyer will assume economic and operating control of the Blackwater and Daunia mines on completion of the sale, including all current and future environmental liabilities and rehabilitation obligations.

At 31 December 2023, the Group's share of assets and liabilities have been classified as 'Assets held for sale' and 'Liabilities directly associated with the assets held for sale'. Blackwater and Daunia mines are not considered to meet the criteria for classification as a discontinued operation given their relative size to the Group and the Coal segment.

The assets and liabilities classified as current assets and liabilities held for sale are presented in the table below:

	31 Dec 2023 US$M
Assets	
Inventories	140
Property, plant and equipment	1,382
Intangible assets	45
Other	3
Total assets	1,570
Liabilities	
Interest bearing liabilities	69
Other financial liabilities	45
Provisions	638
Total liabilities	752
Net assets	818

12. Subsequent events

Other than the matters outlined elsewhere in this Financial Report, no matters or circumstances have arisen since the end of the half year that have significantly affected, or may significantly affect, the operations, results of operations or state of affairs of the Group in subsequent accounting periods.

Directors' Report

The Directors present their report together with the half year Financial Statements for the half year ended 31 December 2023 and the auditor's review report thereon.

Review of Operations

A detailed review of the Group's operated and non-operated assets, the results of those operations during the half year ended 31 December 2023 and likely future developments are given on pages 1 to 26. The Review of Operations has been incorporated into, and forms part of, this Directors' Report.

Principal Risks and Uncertainties

The principal risks affecting the Group are described on pages 73 to 81 of the Group's Annual Report for the year ended 30 June 2023 (a copy of which is available on the Group's website at www.bhp.com) and are grouped into the categories of risks listed below. Our principal risks may occur as a result of our activities globally, including in connection with our operated and non-operated assets, third parties engaged by BHP or through our value chain. Our principal risks, individually or collectively, could threaten our viability, strategy, business model, future performance, solvency or liquidity and reputation. They could also materially and adversely affect the health and safety of our people or members of the public, the environment, the communities where we or our third-party partners operate, or the interests of our stakeholders, which could in each case, lead to litigation, regulatory investigation or enforcement action (including class actions or actions arising from contractual, legacy or other liabilities associated with divested assets), or a loss of stakeholder and/or investor confidence. There are no material changes in those risk factors for the first six months of this financial year except to the extent described in the 'Outlook' section.

- Operational events: Risks associated with operational events in connection with our activities globally, resulting in significant adverse impacts on our people, communities, the environment or our business.
- Significant social or environmental impacts: Risks associated with significant impacts of our operations on and contributions to communities and environments throughout the life cycle of our assets and across our value chain.
- Low-carbon transition: Risks associated with the transition to a low-carbon economy.
- Adopting technologies and maintaining digital security: Risks associated with adopting and implementing new technologies, and maintaining the effectiveness of our existing digital landscape (including cyber defences) across our value chain.
- Ethical misconduct: Risks associated with actual or alleged deviation from societal or business expectations of ethical behaviour (including breaches of laws or regulations) and wider or cumulative organisational cultural failings, resulting in significant reputational impacts.
- Optimising growth and portfolio returns: Risks associated with our ability to position our asset portfolio to generate returns and value for shareholders, including through acquisitions, mergers and divestments. Long-term price volatility, sustained low prices or increases in costs may adversely affect BHP's financial performance as our products are usually sold at prevailing market prices and generally BHP does not have the ability to offset costs through price increases.
- Accessing key markets: Risks associated with market concentration and our ability to sell and deliver products into existing and future key markets, impacting our economic efficiency.
- Inadequate business resilience: Risks associated with unanticipated or unforeseeable adverse events and a failure of planning and preparedness to respond to, manage and recover from adverse events (including potential physical climate-related impacts).

Dividend

Full details of dividends are given on page 4.

Board of Directors

The Directors of BHP at any time during or since the end of the half year ended 31 December 2023 are:

Ken MacKenzie – Chairman since 1 September 2017 (a Director since 22 September 2016)

Mike Henry – an Executive Director since 1 January 2020

Terry Bowen – a former Director from 1 October 2017 to 1 November 2023

Xiaoqun Clever-Steg – a Director since 1 October 2020

Ian Cockerill – a Director since 1 April 2019

Gary Goldberg – a Director since 1 February 2020

Michelle Hinchliffe – a Director since 1 March 2022

Christine O'Reilly – a Director since 12 October 2020

Catherine Tanna – a Director since 4 April 2022

Dion Weisler – a Director since 1 June 2020

Auditor's independence declaration

Ernst & Young in Australia are the auditors of BHP Group Limited. Their auditor's independence declaration under Section 307C of the Australian Corporations Act 2001 is set out on page 52 and forms part of this Directors' Report.

Rounding of amounts

BHP Group Limited is an entity to which Australian Securities and Investments Commission (ASIC) Corporations (Rounding in Financial/Directors' Reports) Instrument 2016/191 applies. Amounts in the Directors' Report and half year Financial Statements have been rounded to the nearest million dollars in accordance with ASIC Instrument 2016/191.

Signed in accordance with a resolution of the Board of Directors.



Ken MacKenzie – Chair



Mike Henry – Chief Executive Officer

Dated this 20th day of February 2024

Directors' Declaration of Responsibility

The half year Financial Report is the responsibility of, and has been approved by, the Directors. In accordance with a resolution of the Directors of BHP Group Limited, the Directors declare that:

(a) in the Directors' opinion and to the best of their knowledge, the half year Financial Statements and notes, set out on pages 27 to 48, have been prepared in accordance with the Australian Corporations Act 2001, including:

  (i) complying with applicable accounting standards and the Australian Corporations Regulations 2001; and

  (ii) giving a true and fair view of the assets, liabilities, financial position and profit or loss of the Group as at 31 December 2023 and of its performance for the half year ended on that date;

(b) for the purposes of the Disclosure Guidance and Transparency Rules in the United Kingdom, to the best of the Directors' knowledge, the Directors' Report, which incorporates the Review of Operations on pages 1 to 26, includes: a fair review of (i) the important events during the first six months of the current financial year and their impact on the half year Financial Statements; (ii) a description of the principal risks and uncertainties for the remaining six months of the year; and (iii) related party transactions that have taken place in the first six months of the current financial year and that have materially affected the financial position or performance of the Group during that period, and changes in the related party transactions described in the last annual report that could have such a material effect; and

(c) in the Directors' opinion, there are reasonable grounds to believe that BHP Group Limited will be able to pay its debts as and when they become due and payable.

Signed on behalf of the Directors in accordance with a resolution of the Board of Directors.



Ken MacKenzie – Chair



Mike Henry – Chief Executive Officer

Dated this 20th day of February 2024

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Auditor's Independence Declaration to the Directors of BHP Group Limited

As lead auditor for the review of the financial report of BHP Group Limited for the half year ended 31 December 2023, I declare to the best of my knowledge and belief, there have been:

a. No contraventions of the auditor independence requirements of the *Corporations Act 2001* in relation to the review;

b. No contraventions of any applicable code of professional conduct in relation to the review; and

c. No non-audit services provided that contravene any applicable code of professional conduct in relation to the review.

This declaration is in respect of BHP Group Limited and the entities it controlled during the financial period.

Ernst & Young

Rodney Piltz
Partner
20 February 2024

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Independent auditor's review report to the members of BHP Group Limited

Conclusion

We have reviewed the accompanying half year financial report of BHP Group Limited and its subsidiaries (collectively the Group), which comprises the consolidated balance sheet as at 31 December 2023, the consolidated income statement, consolidated statement of comprehensive income, consolidated statement of changes in equity and consolidated cash flow statement for the half year ended on that date, notes comprising a summary of significant accounting policies and other explanatory information, and the directors' declaration.

Based on our review, which is not an audit, we have not become aware of any matter that makes us believe that the half year financial report of the Group does not comply with the *Corporations Act 2001*, including:

a. Giving a true and fair view of the consolidated financial position of the Group as at 31 December 2023 and of its consolidated financial performance for the half year ended on that date; and

b. Complying with International Accounting Standard IAS 34 *Interim Financial Reporting* as issued by the International Accounting Standards Board (IASB), Australian Accounting Standard AASB 134 *Interim Financial Reporting* and the *Corporations Regulations 2001*.

Basis for conclusion

We conducted our review in accordance with Australian Auditing Standard on Review Engagements ASRE 2410 *Review of a Financial Report Performed by the Independent Auditor of the Entity* (ASRE 2410). Our responsibilities are further described in the *Auditor's responsibilities for the review of the half-year financial report* section of our report. We are independent of the Group in accordance with the auditor independence requirements of the *Corporations Act 2001* and the ethical requirements of the Accounting Professional and Ethical Standards Board's APES 110 *Code of Ethics for Professional Accountants (including Independence Standards)* (the Code) that are relevant to our audit of the annual financial report in Australia. We have also fulfilled our other ethical responsibilities in accordance with the Code.

Directors' responsibilities for the half year financial report

The directors of the Company are responsible for the preparation of the half year financial report that gives a true and fair view in accordance with International Accounting Standards as issued by the IASB, Australian Accounting Standards and the *Corporations Act 2001* and for such internal control as the directors determine is necessary to enable the preparation of the half year financial report that gives a true and fair view and is free from material misstatement, whether due to fraud or error.

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Auditor's responsibilities for the review of the half year financial report

Our responsibility is to express a conclusion on the half year financial report based on our review. ASRE 2410 requires us to conclude whether we have become aware of any matter that makes us believe that the half year financial report is not in accordance with the *Corporations Act 2001* including giving a true and fair view of the Group's financial position as at 31 December 2023 and its performance for the half year ended on that date, and complying with International Accounting Standard IAS 34 *Interim Financial Reporting* as issued by the IASB, Australian Accounting Standard AASB 134 *Interim Financial Reporting* and the *Corporations Regulations 2001*.

A review of a half year financial report consists of making enquiries, primarily of persons responsible for financial and accounting matters, and applying analytical and other review procedures. A review is substantially less in scope than an audit conducted in accordance with Australian Auditing Standards and consequently does not enable us to obtain assurance that we would become aware of all significant matters that might be identified in an audit. Accordingly, we do not express an audit opinion.

Ernst & Young

Ernst & Young

Rodney Piltz
Partner
Melbourne
20 February 2024

BHP

Non-IFRS Financial Information

Half year ended 31 December 2023

Non-IFRS financial information

We use various non-IFRS financial information to reflect our underlying financial performance.

Non-IFRS financial information is not defined or specified under the requirements of IFRS, but is derived from the Group's Consolidated Financial Statements prepared in accordance with IFRS. The non-IFRS financial information and the below reconciliations included in this document are unaudited. The non-IFRS financial information presented is consistent with how management review financial performance of the Group with the Board and the investment community.

The "Definition and calculation of non-IFRS financial information" section outlines why we believe non-IFRS financial information is useful and the calculation methodology. We believe non-IFRS financial information provides useful information, however should not be considered as an indication of, or as a substitute for, statutory measures as an indicator of actual operating performance (such as profit or net operating cash flow) or any other measure of financial performance or position presented in accordance with IFRS, or as a measure of a company's profitability, liquidity or financial position.

The following tables provide reconciliations between non-IFRS financial information and their nearest respective IFRS measure.

Exceptional items

To improve the comparability of underlying financial performance between reporting periods, some of our non-IFRS financial information adjusts the relevant IFRS measures for exceptional items. Refer to the Group's Financial Report for further information on exceptional items.

Exceptional items are those gains or losses where their nature, including the expected frequency of the events giving rise to them, and impact is considered material to the Group's Consolidated Financial Statements. The exceptional items included within the Group's profit for the financial periods are detailed below.

Half year ended 31 December	2023 US$M	2022 US$M
Revenue	–	–
Other income	–	–
Expenses excluding net finance costs, depreciation, amortisation and impairments	(54)	(47)
Depreciation and amortisation	–	–
Net impairments	(3,500)	–
(Loss)/profit from equity accounted investments, related impairments and expenses	(2,876)	127
Profit/(loss) from operations	**(6,430)**	80
Financial expenses	(190)	(222)
Financial income	–	–
Net finance costs	**(190)**	(222)
Profit/(loss) before taxation	**(6,620)**	(142)
Income tax (expense)/benefit	978	2
Royalty-related taxation (net of income tax benefit)	–	–
Total taxation (expense)/benefit	**978**	2
Profit/(loss) after taxation	**(5,642)**	(140)
Total exceptional items attributable to non-controlling interests	–	–
Total exceptional items attributable to BHP shareholders	(5,642)	(140)
Exceptional items attributable to BHP shareholders per share (US cents)	(111.3)	(2.8)
Weighted basic average number of shares (Million)	5,067	5,064

Non-IFRS financial information derived from Consolidated Income Statement

Underlying attributable profit

Half year ended 31 December	2023 US$M	2022 US$M
Profit after taxation attributable to BHP shareholders	**927**	6,457
Total exceptional items attributable to BHP shareholders[1]	**5,642**	140
Underlying attributable profit	**6,569**	6,597

1 Refer to Exceptional items for further information.

Underlying basic earnings per ordinary share

Half year ended 31 December	2023 US cents	2022 US cents
Basic earnings per ordinary share	**18.3**	127.5
Exceptional items attributable to BHP shareholders per share[1]	**111.3**	2.8
Underlying basic earnings per ordinary share	**129.6**	130.3

1 Refer to Exceptional items for further information.

Underlying EBITDA

Half year ended 31 December	2023 US$M	2022 US$M
Profit from operations	**4,803**	10,833
Exceptional items included in profit from operations[1]	**6,430**	(80)
Underlying EBIT	**11,233**	10,753
Depreciation and amortisation expense	**2,629**	2,456
Net impairments	**3,513**	21
Exceptional items included in Depreciation, amortisation and impairments[1]	**(3,500)**	–
Underlying EBITDA	**13,875**	13,230

1 Refer to Exceptional items for further information.

Underlying EBITDA – Segment

Half year ended 31 December 2023 US$M	Copper	Iron Ore	Coal	Group and unallocated items/ eliminations[2]	Total Group
Profit from operations	2,438	5,788	652	(4,075)	4,803
Exceptional items included in profit from operations[1]	–	2,899	–	3,531	6,430
Depreciation and amortisation expense	1,031	971	322	305	2,629
Net impairments	4	8	1	3,500	3,513
Exceptional items included in Depreciation, amortisation and impairments[1]	–	–	–	(3,500)	(3,500)
Underlying EBITDA	3,473	9,666	975	(239)	13,875

Half year ended 31 December 2022 US$M	Copper	Iron Ore	Coal	Group and unallocated items/ eliminations[2]	Total Group
Profit from operations	1,938	6,768	2,294	(167)	10,833
Exceptional items included in profit from operations[1]	–	(111)	–	31	(80)
Depreciation and amortisation expense	875	980	336	265	2,456
Net impairments	1	4	1	15	21
Exceptional items included in Depreciation, amortisation and impairments[1]	–	–	–	–	–
Underlying EBITDA	2,814	7,641	2,631	144	13,230

1 Refer to Exceptional items for further information.

2 Group and unallocated items includes functions, other unallocated operations, including Potash, Western Australia Nickel (which is made up of Nickel West and West Musgrave which was acquired on 2 May 2023 as part of the acquisition of OZ Minerals Ltd), legacy assets and consolidation adjustments.

Underlying EBITDA – Group and unallocated items

Half year ended 31 December 2023 US$M	Profit from operations	Exceptional items included in profit from operations[1]	Depreciation and amortisation	Net impairments	Exceptional items included in Depreciation, amortisation and impairments[1]	Underlying EBITDA
Potash	(130)	–	1	–	–	(129)
Western Australia Nickel[2]	(3,740)	3,500	66	3,500	(3,500)	(174)
Other[3]	(205)	31	238	–	–	64
Total	(4,075)	3,531	305	3,500	(3,500)	(239)

Half year ended 31 December 2022 US$M	Profit from operations	Exceptional items included in profit from operations[1]	Depreciation and amortisation	Net impairments	Exceptional items included in Depreciation, amortisation and impairments[1]	Underlying EBITDA
Potash	(88)	–	1	–	–	(87)
Western Australia Nickel[2]	50	–	49	–	–	99
Other[3]	(129)	31	215	15	–	132
Total	(167)	31	265	15	–	144

1 Refer to Exceptional items for further information.

2 Western Australia Nickel includes Nickel West and West Musgrave (acquired on 2 May 2023 as part of the acquisition of OZ Minerals Ltd).

3 Other includes functions, other unallocated operations, legacy assets and consolidation adjustments.

Underlying EBITDA margin

Half year ended 31 December 2023 US$M	Copper	Iron Ore	Coal	Group and unallocated items/ eliminations[1]	Total Group
Revenue – Group production	7,435	14,050	3,786	726	25,997
Revenue – Third-party products	1,223	12	–	–	1,235
Revenue	8,658	14,062	3,786	726	27,232
Underlying EBITDA – Group production	3,445	9,667	975	(239)	13,848
Underlying EBITDA – Third-party products	28	(1)	–	–	27
Underlying EBITDA[2]	3,473	9,666	975	(239)	13,875
Segment contribution to the Group's Underlying EBITDA[3]	25%	68%	7%		100%
Underlying EBITDA margin[4]	46%	69%	26%		53.3%

Half year ended 31 December 2022 US$M	Copper	Iron Ore	Coal	Group and unallocated items/ eliminations[1]	Total Group
Revenue – Group production	6,317	11,813	5,566	1,010	24,706
Revenue – Third-party products	988	9	–	10	1,007
Revenue	7,305	11,822	5,566	1,020	25,713
Underlying EBITDA – Group production	2,806	7,641	2,631	144	13,222
Underlying EBITDA – Third-party products	8	–	–	–	8
Underlying EBITDA[2]	2,814	7,641	2,631	144	13,230
Segment contribution to the Group's Underlying EBITDA[3]	22%	58%	20%		100%
Underlying EBITDA margin[4]	44%	65%	47%		53.5%

1 Group and unallocated items includes functions, other unallocated operations, including Potash, Western Australia Nickel (which is made up of Nickel West and West Musgrave which was acquired on 2 May 2023 as part of the acquisition of OZ Minerals Ltd), legacy assets and consolidation adjustments.

2 We differentiate sales of our production (which may include third-party product feed) from direct sales of third-party products to better measure our operational profitability as a percentage of revenue. We may buy and sell third-party products to ensure a steady supply of product to our customers where there is occasional production variability or shortfalls from our assets.

3 Percentage contribution to Group Underlying EBITDA, excluding Group and unallocated items.

4 Underlying EBITDA margin excludes third-party products.

Effective tax rate

	2023			2022		
Half year ended 31 December	Profit before taxation US$M	Income tax expense US$M	%	Profit before taxation US$M	Income tax expense US$M	%
Statutory effective tax rate	3,982	(2,276)	57.2	10,181	(3,055)	30.0
Adjusted for:						
Exchange rate movements	–	(29)		–	11	
Exceptional items[1]	6,620	(978)		142	(2)	
Adjusted effective tax rate	10,602	(3,283)	31.0	10,323	(3,046)	29.5

1 Refer to Exceptional items for further information.

Non-IFRS financial information derived from Consolidated Cash Flow Statement

Capital and exploration expenditure

Half year ended 31 December	2023 US$M	2022 US$M
Capital expenditure (purchases of property, plant and equipment)	**4,545**	2,871
Add: Exploration and evaluation expenditure	**199**	156
Capital and exploration expenditure (cash basis)	**4,744**	3,027

Free cash flow

Half year ended 31 December	2023 US$M	2022 US$M
Net operating cash flows	**8,884**	6,770
Net investing cash flows	**(5,079)**	(3,289)
Free cash flow	**3,805**	3,481

Non-IFRS financial information derived from Consolidated Balance Sheet

Net debt and gearing ratio

	31 Dec 2023 US$M	31 Dec 2022 US$M	30 June 2023 US$M
Interest bearing liabilities – Current	**2,839**	2,015	7,173
Interest bearing liabilities – Non-current	**19,565**	12,686	15,172
Total interest bearing liabilities	**22,404**	14,701	22,345
Comprising:			
Borrowing	**18,826**	12,007	19,326
Lease liabilities	**3,578**	2,694	3,019
Less: Lease liability associated with index-linked freight contracts	**840**	247	287
Less: Cash and cash equivalents	**10,319**	9,605	12,428
Less: Net debt management related instruments[1]	**(1,183)**	(2,063)	(1,572)
Less: Net cash management related instruments[2]	**(220)**	2	36
Less: Total derivatives included in net debt	**(1,403)**	(2,061)	(1,536)
Net debt	**12,648**	6,910	11,166
Net assets	**45,591**	46,552	48,530
Gearing	**21.7%**	12.9%	18.7%

1 Represents the net cross currency and interest rate swaps included within current and non-current other financial assets and liabilities.

2 Represents the net forward exchange contracts related to cash management included within current and non-current other financial assets and liabilities.

Net debt waterfall

	31 Dec 2023 US$M	31 Dec 2022 US$M
Net debt at the beginning of the period	**(11,166)**	(333)
Net operating cash flows	**8,884**	6,770
Net investing cash flows	**(5,079)**	(3,289)
Net financing cash flows	**(5,983)**	(10,911)
Net (decrease)/increase in cash and cash equivalents	**(2,178)**	(7,430)
Carrying value of interest bearing liability net repayments/(proceeds)	**1,324**	1,340
Carrying value of debt related instruments settlements/(proceeds)	**–**	383
Carrying value of cash management related instruments (proceeds)/settlements	**(311)**	(274)
Fair value change on hedged loans	**(345)**	754
Fair value change on hedging derivatives	**323**	(659)
Foreign currency exchange rate changes on cash and cash equivalents	**74**	(201)
Lease additions (excluding leases associated with index-linked freight contracts)	**(298)**	(320)
Transfer to liability directly associated with assets held for sale	**69**	–
Other	**(140)**	(170)
Non-cash movements	**(317)**	(596)
Net debt at the end of the period	**(12,648)**	(6,910)

Net operating assets

	31 Dec 2023 US$M	31 Dec 2022 US$M
Net assets	**45,591**	46,552
Less: Non-operating assets		
Cash and cash equivalents	**(10,319)**	(9,605)
Trade and other receivables[1]	**(12)**	(22)
Other financial assets[2]	**(1,197)**	(1,219)
Current tax assets	**(446)**	(444)
Deferred tax assets	**(76)**	(54)
Assets held for sale[3]	**(1,570)**	–
Add: Non-operating liabilities		
Trade and other payables[4]	**272**	149
Interest bearing liabilities	**22,404**	14,701
Other financial liabilities[5]	**1,761**	2,276
Current tax payable	**290**	469
Non-current tax payable	**39**	65
Deferred tax liabilities	**3,325**	3,367
Liabilities directly associated with the assets held for sale[3]	**752**	–
Net operating assets	**60,814**	56,235

1 Represents external finance receivable and accrued interest receivable included within other receivables.

2 Represents cross currency and interest rate swaps, forward exchange contracts related to cash management and investment in shares, other investments and receivables contingent on outcome of future events relating to mining and regulatory approvals.

3 Represents Blackwater and Daunia coal assets and liabilities classified as held for sale as at 31 December 2023.

4 Represents accrued interest payable included within other payables.

5 Represents cross currency and interest rate swaps and forward exchange contracts related to cash management.

Other non-IFRS financial information

Principal factors that affect Revenue, Profit from operations and Underlying EBITDA

The following table describes the impact of the principal factors that affected Revenue, Profit from operations and Underlying EBITDA for half year ended 31 December 2023 and relates them back to our Consolidated Income Statement.

	Revenue US$M	Total expenses, Other income and Profit/(loss) from equity accounted investments US$M	Profit from operations US$M	Depreciation, amortisation and impairments and Exceptional Items US$M	Underlying EBITDA US$M
Half year ended 31 December 2022					
Revenue	**25,713**				
Other income		**269**			
Expenses excluding net finance costs		**(15,429)**			
Profit/(loss) from equity accounted investments, related impairments and expenses		**280**			
Total other income, expenses excluding net finance costs and Profit/(loss) from equity accounted investments, related impairments and expenses		**(14,880)**			
Profit from operations			**10,833**		
Depreciation, amortisation and impairments				**2,477**	
Exceptional item included in Depreciation, amortisation and impairments				**–**	
Exceptional items				**(80)**	
Underlying EBITDA					**13,230**
Change in sales prices	629	–	**629**	–	**629**
Price-linked costs	–	147	**147**	–	**147**
Net price impact	629	147	**776**	–	**776**
Change in volumes	228	32	**260**	–	**260**
Operating cash costs	–	(372)	**(372)**	–	**(372)**
Exploration and business development	–	(64)	**(64)**	–	**(64)**
Change in controllable cash costs[1]	–	(436)	**(436)**	–	**(436)**
Exchange rates	(1)	81	**80**	–	**80**
Inflation on costs	–	(507)	**(507)**	–	**(507)**
Fuel, energy, and consumable price movements	–	340	**340**	–	**340**
Non-cash	–	(119)	**(119)**	–	**(119)**
One-off items	–	–	**–**	–	**–**
Change in other costs	(1)	(205)	**(206)**	–	**(206)**
Asset sales	–	38	**38**	–	**38**
Ceased and sold operations	(98)	75	**(23)**	–	**(23)**
New and acquired operations	610	(370)	**240**	–	**240**
Other	151	(155)	**(4)**	–	**(4)**
Depreciation, amortisation and impairments	–	(165)	**(165)**	165	–
Exceptional items	–	(6,510)	**(6,510)**	6,510	–
Half year ended 31 December 2023					
Revenue	**27,232**				
Other income		**261**			
Expenses excluding net finance costs		**(19,982)**			
(Loss)/profit from equity accounted investments, related impairments and expenses		**(2,708)**			
Total other income, expenses excluding net finance costs and Profit/(loss) from equity accounted investments, related impairments and expenses		**(22,429)**			
Profit from operations			**4,803**		
Depreciation, amortisation and impairments				**6,142**	
Exceptional item included in Depreciation, amortisation and impairments				**(3,500)**	
Exceptional items				**6,430**	
Underlying EBITDA					**13,875**

1 Collectively, we refer to the change in operating cash costs and change in exploration and business development as Change in controllable cash costs. Operating cash costs by definition do not include non-cash costs. The change in operating cash costs also excludes the impact of exchange rates and inflation, changes in fuel, energy costs and consumable costs, changes in exploration and evaluation and business development costs and one-off items. These items are excluded so as to provide a consistent measurement of changes in costs across all segments, based on the factors that are within the control and responsibility of the segment.

Underlying return on capital employed (ROCE)

	31 Dec 2023 US$M	31 Dec 2022 US$M
Profit after taxation	**1,706**	7,126
Exceptional items[1]	**5,642**	140
Subtotal	**7,348**	7,266
Adjusted for:		
Net finance costs	**821**	652
Exceptional items included within net finance costs[1]	**(190)**	(222)
Income tax expense on net finance costs	**(187)**	(166)
Profit after taxation excluding net finance costs and exceptional items	**7,792**	7,530
Annualised Profit after taxation excluding net finance costs and exceptional items	**15,584**	15,060
Net assets at the beginning of the period	**48,530**	48,766
Net debt at the beginning of the period	**11,166**	333
Capital employed at the beginning of the period	**59,696**	49,099
Net assets at the end of the period	**45,591**	46,552
Net debt at the end of the period	**12,648**	6,910
Capital employed at the end of the period	**58,239**	53,462
Average capital employed	**58,968**	51,281
Underlying Return on Capital Employed	**26.4%**	29.4%

1 Refer to Exceptional items for further information.

Underlying return on capital employed (ROCE) by segment

Half year ended 31 December 2023 US$M	Copper	Iron Ore	Coal	Group and unallocated items/ eliminations[1]	Total Group
Annualised profit after taxation excluding net finance costs and exceptional items	**3,242**	**12,180**	**1,032**	**(870)**	**15,584**
Average capital employed	**31,029**	**14,406**	**6,743**	**6,790**	**58,968**
Underlying Return on Capital Employed	**10%**	**85%**	**15%**	**–**	**26.4%**

Half year ended 31 December 2022 US$M	Copper	Iron Ore	Coal	Group and unallocated items/ eliminations[1]	Total Group
Annualised profit after taxation excluding net finance costs and exceptional items	2,776	9,352	3,250	(318)	15,060
Average capital employed	25,184	15,273	5,934	4,890	51,281
Underlying Return on Capital Employed	11%	61%	55%	–	29.4%

1 Group and unallocated items includes functions, other unallocated operations including Potash, Western Australia Nickel (which is made up of Nickel West and West Musgrave which was acquired on 2 May 2023 as part of the acquisition of OZ Minerals Ltd), legacy assets and consolidation adjustments.

Underlying return on capital employed (ROCE) by asset

Half year ended 31 December 2023 US$M	Western Australia Iron Ore	Antamina	Escondida	BHP Mitsubishi Alliance	Pampa Norte	Copper South Australia[1]	Western Australia Nickel[2]	Potash[3]	New South Wales Energy Coal[4]	Other	Total Group
Annualised profit after taxation excluding net finance costs and exceptional items	12,184	426	2,484	822	258	372	(514)	(258)	300	(490)	15,584
Average capital employed	19,718	1,382	10,693	6,903	4,221	14,462	1,648	4,859	(338)	(4,580)	58,968
Underlying Return on Capital Employed	62%	31%	23%	12%	6%	3%	(31%)	–	–	–	26.4%

Half year ended 31 December 2022 US$M	Western Australia Iron Ore	Antamina	Escondida	BHP Mitsubishi Alliance	Pampa Norte	Copper South Australia[1]	Western Australia Nickel[2]	Potash[3]	New South Wales Energy Coal[4]	Other	Total Group
Annualised profit after taxation excluding net finance costs and exceptional items	9,362	346	2,464	1,626	86	98	(2)	(114)	1,706	(512)	15,060
Average capital employed	19,123	1,308	10,209	6,250	4,498	9,189	1,089	3,789	(546)	(3,628)	51,281
Underlying Return on Capital Employed	49%	26%	24%	26%	2%	1%	(0%)	–	–	–	29.4%

1 Includes Olympic Dam as well as Prominent Hill and Carrapateena which were acquired on 2 May 2023 as part of the acquisition of OZ Minerals Ltd.

2 Western Australia Nickel includes Nickel West and West Musgrave (acquired on 2 May 2023 as part of the acquisition of OZ Minerals Ltd).

3 Potash ROCE has not been shown because it is distorted as the asset is non-producing and in its development phase.

4 NSWEC ROCE has not been shown as it is distorted by negative capital employed due to the rehabilitation provision being the primary balance remaining on Balance Sheet following previous impairments.

Unit costs

Unit costs do not include the re-allocation to Assets in FY2024 of the costs associated with the employee entitlements and allowances review conducted in FY2023, which were reported in Group and Unallocated in that period.

The calculation of Escondida and Spence unit costs is set out in the table below.

	Escondida unit costs		Spence unit costs	
US$M	H1 FY2024	H1 FY2023	H1 FY2024	H1 FY2023
Revenue	**4,427**	4,089	**1,029**	892
Underlying EBITDA	**2,347**	2,160	**428**	295
Gross costs	**2,080**	1,929	**601**	597
Less: by-product credits	**248**	190	**49**	40
Less: freight	**89**	110	**22**	23
Net costs	**1,743**	1,629	**530**	534
Sales (kt)	**522.6**	512.1	**121.4**	110.5
Sales (Mlb)	**1,152.1**	1,129.0	**267.6**	243.6
Cost per pound (US$)[1]	**1.51**	1.44	**1.98**	2.19

1 H1 FY24 based on average realised exchange rates of USD/CLP 874 (H1 FY23 USD/CLP 920).

The calculation of WAIO unit costs is set out in the table below.

	WAIO unit costs	
US$M	H1 FY2024	H1 FY2023
Revenue	**13,991**	11,756
Underlying EBITDA	**9,646**	7,623
Gross costs	**4,345**	4,133
Less: freight	**979**	1,020
Less: re-allocation of costs associated with the employee entitlements and allowances review	**33**	–
Less: royalties	**992**	793
Net costs	**2,341**	2,320
Sales (kt, equity share)	**126,786**	126,753
Cost per tonne (US$)[1]	**18.46**	18.30

1 H1 FY24 based on an average realised exchange rate of AUD/USD 0.65 (H1 FY23 AUD/USD 0.67).

The calculation of BMA unit costs is set out in the table below.

	BMA unit costs	
US$M	H1 FY2024	H1 FY2023
Revenue	**2,882**	3,598
Underlying EBITDA	**810**	1,426
Gross costs	**2,072**	2,172
Less: freight	**14**	19
Less: re-allocation of costs associated with the employee entitlements and allowances review	**4**	–
Less: royalties	**631**	799
Net costs	**1,423**	1,354
Sales (kt, equity share)	**11,031**	13,509
Cost per tonne (US$)[1]	**129.00**	100.23

1 H1 FY24 based on an average realised exchange rate of AUD/USD 0.65 (H1 FY23 AUD/USD 0.67).

Definition and calculation of Non-IFRS financial information

Non-IFRS financial information	Reasons why we believe the non-IFRS financial information are useful	Calculation methodology
Underlying attributable profit	Allows the comparability of underlying financial performance by excluding the impacts of exceptional items and is also the basis on which our dividend payout ratio policy is applied.	Profit after taxation attributable to BHP shareholders excluding any exceptional items attributable to BHP shareholders.
Underlying basic earnings per share	On a per share basis, allows the comparability of underlying financial performance by excluding the impacts of exceptional items.	Underlying attributable profit divided by the weighted basic average number of shares.
Underlying EBITDA	Used to help assess current operational profitability excluding the impacts of sunk costs (i.e. depreciation from initial investment). Each is a measure that management uses internally to assess the performance of the Group's segments and make decisions on the allocation of resources.	Earnings before net finance costs, depreciation, amortisation and impairments, taxation expense, Discontinued operations and exceptional items. Underlying EBITDA includes BHP's share of profit/(loss) from investments accounted for using the equity method including net finance costs, depreciation, amortisation and impairments and taxation expense/(benefit).
Underlying EBITDA margin		Underlying EBITDA excluding third party product EBITDA, divided by revenue excluding third party product revenue.
Underlying EBIT	Used to help assess current operational profitability excluding net finance costs and taxation expense (each of which are managed at the Group level) as well as Discontinued operations and any exceptional items.	Earnings before net finance costs, taxation expense, Discontinued operations and any exceptional items. Underlying EBIT includes BHP's share of profit/(loss) from investments accounted for using the equity method including net finance costs and taxation expense/(benefit).
Profit from operations		Earnings before net finance costs, taxation expense and Discontinued operations. Profit from operations includes Revenue, Other income, Expenses excluding net finance costs and BHP's share of profit/(loss) from investments accounted for using the equity method including net finance costs and taxation expense/(benefit).
Capital and exploration expenditure	Used as part of our Capital Allocation Framework to assess efficient deployment of capital. Represents the total outflows of our operational investing expenditure.	Purchases of property, plant and equipment and exploration and evaluation expenditure.
Free cash flow	It is a key measure used as part of our Capital Allocation Framework. Reflects our operational cash performance inclusive of investment expenditure, which helps to highlight how much cash was generated in the period to be available for the servicing of debt and distribution to shareholders.	Net operating cash flows less net investing cash flows.
Net debt	Net debt shows the position of gross debt less index-linked freight contracts offset by cash immediately available to pay debt if required and any associated derivative financial instruments. Liability associated with index-linked freight contracts, which are required to be remeasured to the prevailing freight index at each reporting date, are excluded from the net debt calculation due to the short-term volatility of the index they relate to not aligning with how the Group uses net debt for decision making in relation to the Capital Allocation Framework. Net debt includes the fair value of derivative financial instruments used to hedge cash and borrowings to reflect the Group's risk	Interest bearing liabilities less liability associated with index-linked freight contracts less cash and cash equivalents less net cross currency and interest rate swaps less net cash management related instruments for the Group at the reporting date.
Gearing ratio		Ratio of Net debt to Net debt plus Net assets.

Non-IFRS financial information	Reasons why we believe the non-IFRS financial information are useful	Calculation methodology
	management strategy of reducing the volatility of net debt caused by fluctuations in foreign exchange and interest rates. Net debt, along with the gearing ratio, is used to monitor the Group's capital management by relating net debt relative to equity from shareholders.	
Net operating assets	Enables a clearer view of the assets deployed to generate earnings by highlighting the net operating assets of the business separate from the financing and tax balances. This measure helps provide an indicator of the underlying performance of our assets and enhances comparability between them.	Operating assets net of operating liabilities, including the carrying value of equity accounted investments and predominantly excludes cash balances, loans to associates, interest bearing liabilities, derivatives hedging our net debt, assets held for sale, liabilities directly associated with assets held for sale and tax balances.
Underlying return on capital employed (ROCE)	Indicator of the Group's capital efficiency and is provided on an underlying basis to allow comparability of underlying financial performance by excluding the impacts of exceptional items.	Profit after taxation excluding exceptional items and net finance costs (after taxation) divided by average capital employed. Profit after taxation excluding exceptional items and net finance costs (after taxation) is profit after taxation excluding exceptional items, net finance costs and the estimated taxation impact of net finance costs. These are annualised for a half year end reporting period. The estimated tax impact is calculated using a prima facie taxation rate on net finance costs (excluding any foreign exchange impact). Average capital employed is calculated as the average of net assets less net debt for the last two reporting periods.
Adjusted effective tax rate	Provides an underlying tax basis to allow comparability of underlying financial performance by excluding the impacts of exceptional items.	Total taxation expense/(benefit) excluding exceptional items and exchange rate movements included in taxation expense/(benefit) divided by Profit before taxation excluding exceptional items.
Unit cost	Used to assess the controllable financial performance of the Group's assets for each unit of production. Unit costs are adjusted for site specific non-controllable factors to enhance comparability between the Group's assets.	Ratio of net costs of the assets to the equity share of sales tonnage. Net costs is defined as revenue less Underlying EBITDA and excludes freight, re-allocation of the costs associated with the employee entitlements and allowance review in FY2023, and other costs, depending on the nature of each asset. Freight is excluded as the Group believes it provides a similar basis of comparison to our peer group. The re-allocation to Assets in FY2024 of the costs associated with the employee entitlements and allowances review in FY2023 are excluded in Asset unit costs as these costs were already recognised in Group and Unallocated in FY2023. Escondida and Spence unit costs exclude: • by-product credits being the favourable impact of by-products (such as gold or silver) to determine the directly attributable costs of copper production. WAIO and BMA unit costs exclude: • royalties as these are costs that are not deemed to be under the Group's control, and the Group believes exclusion provides a similar basis of comparison to our peer group.

Definition and calculation of principal factors

The method of calculation of the principal factors that affect the period on period movements of Revenue, Profit from operations and Underlying EBITDA are as follows:

Principal factor	Method of calculation
Change in sales prices	Change in average realised price for each operation from the prior period to the current period, multiplied by current period sales volumes.
Price-linked costs	Change in price-linked costs (mainly royalties) for each operation from the prior period to the current period, multiplied by current period sales volumes.
Change in volumes	Change in sales volumes for each operation multiplied by the prior year average realised price less variable unit cost.
Controllable cash costs	Total of operating cash costs and exploration and business development costs.
Operating cash costs	Change in total costs, other than price-linked costs, exchange rates, inflation on costs, fuel, energy, and consumable price movements, non-cash costs and one-off items as defined below for each operation from the prior period to the current period.
Exploration and evaluation and business development	Exploration and evaluation and business development expense in the current period minus exploration and business development expense in the prior period.
Exchange rates	Change in exchange rate multiplied by current period local currency revenue and expenses.
Inflation on costs	Change in inflation rate applied to expenses, other than depreciation and amortisation, price-linked costs, exploration and business development expenses, expenses in ceased and sold operations and expenses in new and acquired operations.
Fuel, energy, and consumable price movements	Fuel and energy expense and price differences above inflation on consumables in the current period minus fuel and energy expense in the prior period.
Non-cash	Change in net impact of capitalisation and depletion of deferred stripping from the prior period to the current period.
One-off items	Change in costs exceeding a pre-determined threshold associated with an unexpected event that had not occurred in the last two years and is not reasonably likely to occur within the next two years.
Asset sales	Profit/(loss) on the sale of assets or operations in the current period minus profit/(loss) on sale of assets or operations in the prior period.
Ceased and sold operations	Underlying EBITDA for operations that ceased or were sold in the current period minus Underlying EBITDA for operations that ceased or were sold in the prior period.
New and acquired operations	Underlying EBITDA for operations that were acquired in the current period minus Underlying EBITDA for operations that were acquired in the prior period.
Share of profit/(loss) from equity accounted investments	Share of profit/(loss) from equity accounted investments for the current period minus share of profit/(loss) from equity accounted investments in the prior period.
Other	Variances not explained by the above factors.

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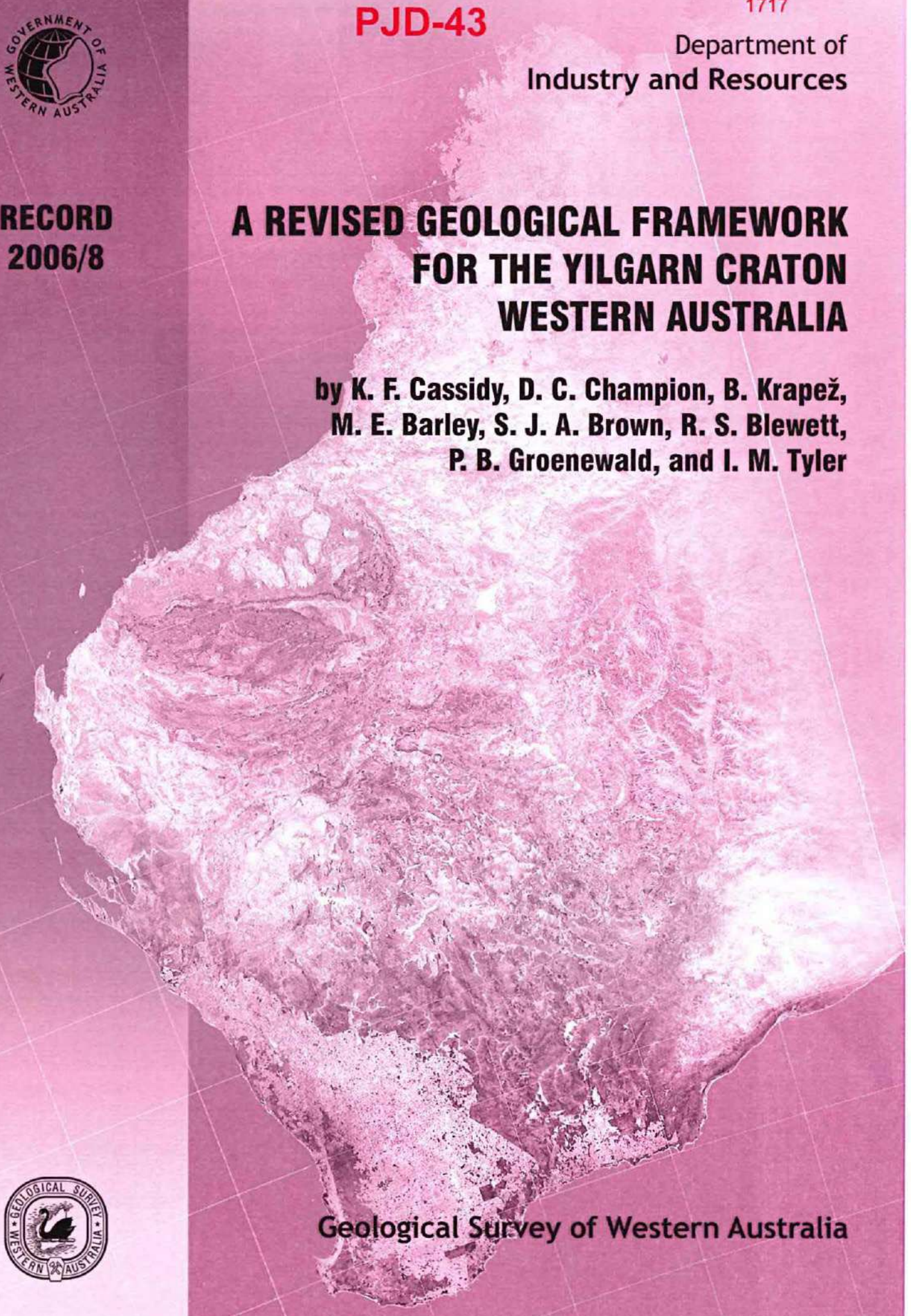
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RECORD
2006/8

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by K. F. Cassidy, D. C. Champion, B. Krapež,
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Geological Survey of Western Australia

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GEOLOGICAL SURVEY OF WESTERN AUSTRALIA

Record 2006/8

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K. F. Cassidy[1], D. C. Champion[1], B. Krapež[2], M. E. Barley[2], S. J. A. Brown[2], R. S. Blewett[1], P. B. Groenewald[3], and I. M. Tyler[3]

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Perth 2006

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A revised geological framework for the Yilgarn Craton, Western Australia

by

K. F. Cassidy[1], D. C. Champion[1], B. Krapež[2], M. E. Barley[2], S. J. A. Brown[2], R. S. Blewett[1], P. B. Groenewald[3] and I. M. Tyler[3]

Abstract

New geological and geochronological data have been used to revise the geological framework of the Archean Yilgarn Craton of Western Australia. The Yilgarn Craton is subdivided into six terranes, three of which constitute a superterrane. Along the western margin the Narryer Terrane and South West Terrane are dominated by granite and granitic gneiss, whereas the Youanmi Terrane and Eastern Goldfields Superterrane are composed of north-trending greenstone belts separated by extensive granite and granitic gneiss. The Youanmi Terrane is the amalgamation of the Murchison and Southern Cross Domains and possibly represents the nucleus, or protocraton, onto which the Narryer Terrane and Eastern Goldfields Superterrane were accreted.

The Eastern Goldfields Superterrane comprises three tectono-stratigraphic terranes, defined on the basis of distinct volcanic facies, geochemistry, and age of volcanism. From southwest to northeast, these are the Kalgoorlie, Kurnalpi, and Burtville Terranes, with each terrane containing a number of domains. These terranes and domains are bounded by interconnected fault systems. From west to east, the terranes are bounded by the Ida, Ockerburry and Hootanui Fault Systems. Ongoing mapping, geochronology, geochemistry, and interpretation of regional geophysical data will continue to refine the character and boundaries of tectonic units of the Yilgarn Craton.

KEYWORDS: Archean, Yilgarn Craton, Western Australia, terranes, tectonic units.

Introduction

This Record presents a revised geological framework for the Archean Yilgarn Craton. This framework is based on remapping and re-evaluation of geological data at various scales, combined with new geophysical, geochemical, isotopic, and geochronological data. The data have been largely acquired since 1996 during a series of AMIRA International Limited (AMIRA) and Minerals and Energy Research Institute of Western Australia (MERIWA) projects undertaken by Geoscience Australia (GA), the University of Western Australia (UWA) and Monash University, Victoria, in collaboration with the Geological Survey of Western Australia (GSWA). The identification of major terranes and their boundaries in the eastern part of the Yilgarn Craton, and an improved understanding of its tectonic evolution, is based on the results of AMIRA projects P437A (Barley et al., 2002), P482/M281 (Cassidy et al., 2002) and P624 (Barley et al., 2003), in conjunction with the Predictive Mineral Discovery Cooperative Research Centre (pmd*CRC) Y2 project (Blewett and Hitchman, 2004).

The nomenclature and boundaries of the tectonic units of the Yilgarn Craton have been modified from those of Tyler and Hocking (2001) to accommodate our better understanding of their tectonic evolution. The four-fold tectonic unit hierarchy scheme applied herein consists of Craton > Superterrane > Terrane > Domain. Terranes are defined as fault-bounded bodies of rock of regional extent, characterized by a geological history different from those adjacent to them (Neuendorf et al., 2005). A superterrane is defined as a grouping of related, adjacent terranes. Domains are defined as fault-bounded, geologically contiguous blocks within terranes. The amalgam of all of these elements is the craton.

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Terranes of the Yilgarn Craton

The Yilgarn Craton is one of three areas of crust in Western Australia that stabilized before 2.4 Ga. It consists of metavolcanic and metasedimentary rocks, granites, and granitic gneiss that formed principally between c. 3.05 and 2.62 Ga, with a minor older component (to >3.7 Ga; e.g. Nelson, 1997; Pidgeon and Hallberg, 2000; Cassidy et al., 2002; Barley et al., 2002, 2003). Voluminous granite intrusion between 2.76 and 2.62 Ga was coincident with Neoarchean orogeny, resulting in amalgamation and assembly of several tectonic units to form the Yilgarn Craton (Myers, 1993, 1995; Barley et al., 2003).

The Yilgarn Craton has traditionally been subdivided into four geologically distinct provinces — the Eastern Goldfields, Southern Cross, and Murchison Provinces and the Western Gneiss Terrain (Gee et al., 1981). The criterion used to distinguish the Western Gneiss Terrain was its predominant gneiss content compared with the predominantly granite–greenstone provinces, each of which was described as having 'a unified lithology, structural history, and perhaps even stratigraphy' (Gee et al., 1981, p. 44). The relationship between these provinces, however, was enigmatic, and the boundaries largely hypothetical and not based on observed geological features. Myers (1993, 1995), Wilde et al. (1996), Myers and Swager (1997), and Witt et al. (1998) subdivided parts of the Yilgarn Craton into superterranes and terranes based on greenstone belt shapes and trends, rock associations, and ages of greenstone deposition, granite intrusion, and deformation. This subdivision was not incorporated into the tectonic units of Western Australia map by Tyler and Hocking (2001), which largely followed the framework of Myers and Hocking (1998), because the status of some of the 'terranes' was regarded as controversial (e.g. Groenewald et al., 2000).

Swager et al. (1992) and Swager (1995, 1997) subdivided the southeastern Yilgarn Craton into a number of terranes (such as the Kalgoorlie and Kurnalpi Terranes), and further subdivided those terranes into domains. Subsequent studies by Barley et al. (2002, 2003) and Cassidy et al. (2005) resulted in modifications to the original terrane and domain definitions and the introduction of additional terranes and domains. Further refinement of the terranes and domains by Blewett and Hitchman (2004) resulted in rationalization, including amalgamation, of some the terranes and the recognition of new terranes and domains in the very easternmost part of the craton. Terranes and domains are now separated from one another by regional fault systems with the original boundaries obscured by considerable post-amalgamation deformation and the intrusion of later granites.

The Yilgarn Craton is subdivided here into six terranes, three of which constitute a superterrane (Fig. 1). In the west the Narryer Terrane and the South West Terrane are dominated by granite and granitic gneiss, whereas the central Youanmi Terrane and the Eastern Goldfields Superterrane are composed of north-trending greenstone belts separated by extensive granite and granitic gneiss. Table 1 summarizes the features of each of the terranes. The new framework differs from that of Tyler and Hocking (2001), as well as the subdivisions of Myers and Swager (1997) and Witt et al. (1998), with the recognition that what were the Murchison and Southern Cross Granite–Greenstone Terranes are domains of the new Youanmi Terrane, and the modification of terranes and domains within the Eastern Goldfields Superterrane. No modifications have been made to the Narryer and South West Terranes. Boundaries of the terranes and superterrane presented here will be more completely validated by ongoing mapping, geochronology, geochemistry, and interpretation of regional geophysical data.

In the discussion that follows, reference is made to greenstone belts, metamorphic belts, and structural elements that have been described in some detail in previous reviews of the geology of the Yilgarn Craton (e.g., Geological Survey of Western Australia, 1990; Groenewald et al., 2000; Groenewald and Riganti, 2004). The reader is referred to those publications and the references therein.

Narryer and South West Terranes

The Narryer Terrane and the South West Terrane form the northwestern and southwestern parts of the Yilgarn Craton respectively. They are retained as separate terranes in the current framework. Additional work is required on the boundaries and geological components of each terrane to establish their relationship with the adjacent Youanmi Terrane.

The Narryer Terrane is the oldest known component of the craton and includes several ages of gneisses derived from early to middle Archean granitic rocks and interleaved metasedimentary and mafic–anorthositic meta-igneous rocks. It is of considerable geological significance because it contains the oldest known rocks in Australia (c. 3.73 Ga Manfred Complex: Kinny et al., 1988) and metasedimentary rocks with detrital zircons as old as 4.4 Ga (Froude et al., 1983; Wilde et al., 2001). Rocks of the Narryer Terrane were metamorphosed at high grade and intruded by granite and pegmatite between 3.3 and 3.05 Ga, and were multiply deformed, metamorphosed, and intruded by granite between c. 2.75 and 2.62 Ga (Myers, 1988; Nutman et al., 1993). The boundary between the Narryer and Youanmi Terranes is marked by the Balbalinga Fault in the southeast and the Yalgar Fault in the northeast (Myers, 1993).

The South West Terrane consists of high-grade granitic gneisses and metasedimentary and meta-igneous rocks that have experienced multiple phases of deformation and granite and pegmatite intrusion between c. 2.75 and 2.62 Ga (Myers, 1993; Wilde et al., 1996). Although the South West Terrane has been informally divided into three domains on the basis of the interpretation of geophysical and geochronological data (Wilde et al., 1996), the exact nature of the domains, as well as the location of the boundaries between them, is poorly constrained, and this subdivision is not incorporated into the present overview. Supracrustal sequences in the South West Terrane range in age from c. 3.2–2.8 Ga in the Chittering, Jimperding, and Balingup metamorphic belts to c. 2.72–2.67 Ga in the Saddleback greenstone belt (Wilde et al., 1996).

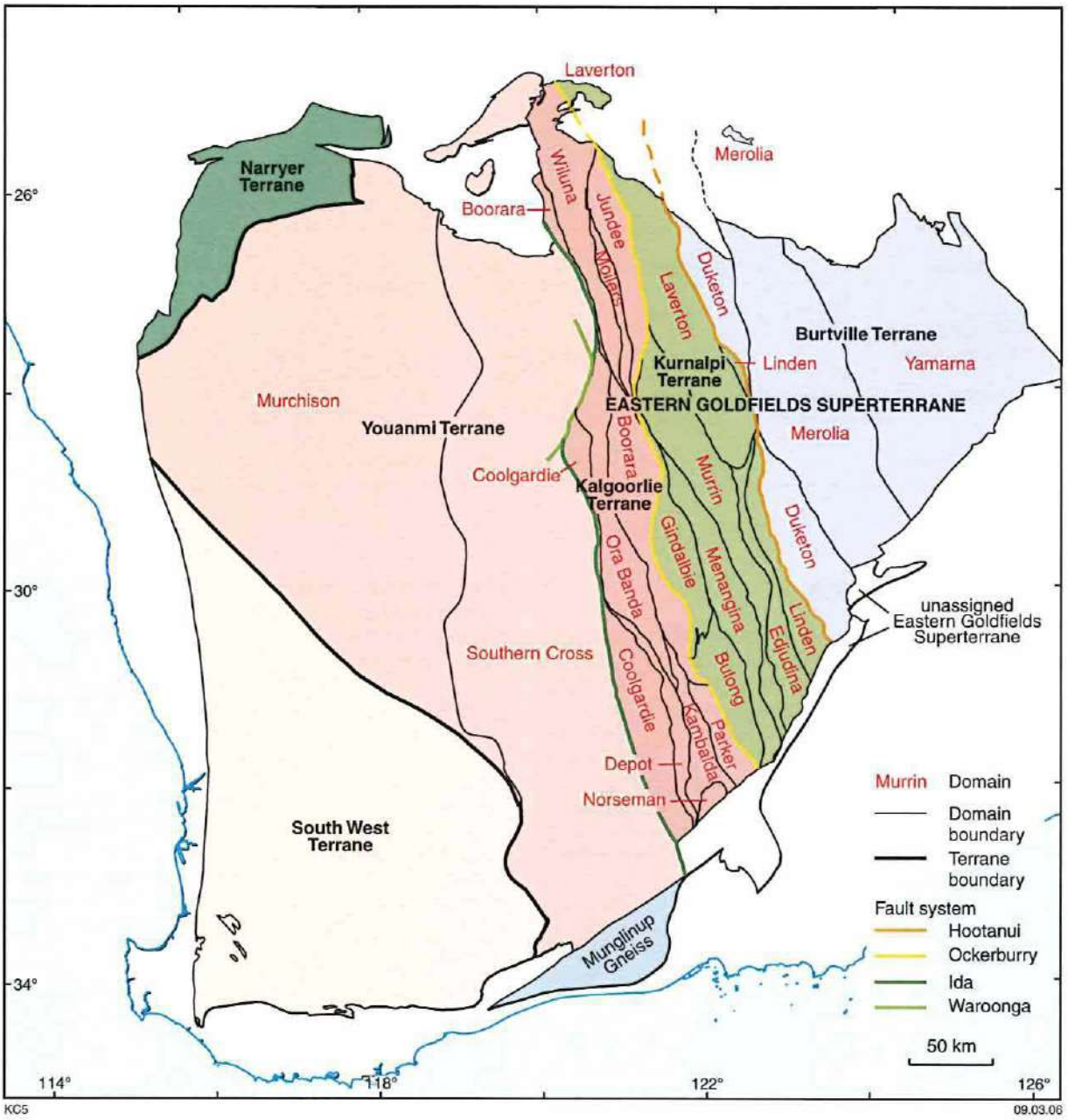


Figure 1. Tectonic division of the Yilgarn Craton, showing subdivision into terranes and domains

Quartzite in the Jimperding metamorphic belt contains detrital zircons with a spectrum of ages from older than 3.73 to 3.17 Ga (Wilde et al., 1996), indicating a complex and variably old provenance. Granites in the South West Terrane have emplacement ages from c. 2.75 to 2.62 Ga, with the majority younger than 2.69 Ga (Nemchin and Pidgeon, 1997). Granulite-facies metamorphism was synchronous with emplacement of c. 2.64–2.62 Ga charnockitic granites (Nemchin et al., 1994). Intrusive and volcanic rocks at Boddington were metamorphosed to upper greenschist to lower amphibolite facies at c. 2640 Ma. The boundary between the South West Terrane and the Youanmi Terrane is poorly defined.

The older crustal components of the Narryer and South West Terranes are thought to have accreted to the Youanmi Terrane after 2.8 Ga (Myers, 1993; Nutman et al., 1993; Qiu and Groves, 1999; Cassidy et al., 2005), with collision of the Narryer Terrane at c. 2.75 Ga, and continued deformation and metamorphism and granite intrusion between 2.75 and 2.65 Ga (Myers, 1993). Nutman et al. (1993) suggested that rocks of the Narryer Terrane were emplaced over the Youanmi Terrane at c. 2.75 Ga. Collision of the South West Terrane with the Youanmi Terrane is inferred to have resulted in the emplacement of volumetrically abundant granites at 2.64–2.62 Ga and coincident high-grade metamorphism (Wilde et al., 1996; Qiu and Groves, 1999).

Table 1. Summary of features of the terranes of the Archean Yilgarn Craton (modified from Cassidy et al., 2005)

Terrane	*Age of initial crust formation*	*Depositional ages of greenstones*	*Emplacement ages of granites and gneisses (Ga)*	*Age of deformation and metamorphism*
Narryer Terrane	3.8–3.4	?3.73	3.73–3.6, 3.48, 3.3 3.0, 2.75, 2.68–2.62	c. 3.3, ?2.75, 2.68–2.62
South West Terrane	?3.5–3.0	?3.0, ?2.8, 2.7	?3.2, 2.8, 2.68–2.62	?2.68–2.62
Youanmi Terrane	?3.4–2.9	3.01–2.92, 2.81, 2.76–2.72	3.01–2.92, 2.81, 2.76–2.68, 2.66–2.62	>2.74–2.68, 2.66–2.63
Eastern Goldfields Superterrane				
Kalgoorlie Terrane	?3.0–2.8	?2.94, ?2.81, 2.74–2.66	2.81, 2.75–2.74, 2.68–2.66, 2.65–2.63	2.67–2.63
Kurnalpi Terrane (western domains)	2.9–2.8	2.71–2.68	2.7, 2.68–2.66, 2.65–2.63	2.67–2.63
(eastern domains)	?3.1–2.9	>2.8, 2.72–2.68	?2.95, 2.71, 2.68–2.66, 2.65–2.63	2.67–2.63
Burtville Terrane	?3.0–2.8	2.81, 2.77–2.66	?2.95, 2.8–2.77, 2.69–2.63	?2.67–2.63

Youanmi Terrane

The Youanmi Terrane combines the Murchison and Southern Cross Granite–Greenstone Terranes of Tyler and Hocking (2001). It is named after the historical mining locality of Youanmi in the central part of the Youanmi Terrane. The Murchison and Southern Cross 'Granite–Greenstone Terranes' are here regarded as domains and are the subject of ongoing mapping, geochronology, and interpretation of regional geophysical data. They have a broadly similar lithostratigraphy and tectonic history (e.g., Chen et al., 2003) implying that they are related. The Youanmi Fault, invoked by Myers (1993, 1997) and Myers and Swager (1997) as the boundary fault between their West Yilgarn and Southern Cross Superterranes, is probably related to D_3 deformation throughout the central Yilgarn Craton (Chen et al., 2004) and is considered herein to represent a domain-bounding structure.

Geochronological data indicate periods of volcanism and sediment deposition within the Youanmi Terrane at c. 3.05–2.93, 2.81, and 2.76–2.72 Ga (Pidgeon and Wilde, 1990; Schiøtte and Campbell, 1996; Wang et al., 1998; Pidgeon and Hallberg, 2000; Geological Survey of Western Australia, 2005). A c. 3.05–2.93 Ga lower succession is characterized by mafic and ultramafic volcanic rocks and banded iron-formation, with minor felsic volcanic rocks and local basal quartzite. Quartzites in the Illaara and Maynard Hills greenstone belts in the central Southern Cross Domain have maximum depositional ages of c. 3.13 Ga, with detrital zircons up to 4.35 Ga (Wyche et al., 2004). In the Marda–Diemals greenstone belt in the central Southern Cross Domain, a c. 2.73 Ga upper succession composed of felsic–intermediate volcanic rocks and clastic sedimentary rocks unconformably overlies the lower succession. This succession has some common features with other felsic volcanic rocks in the Southern Cross and Murchison Domains (e.g. Watkins and Hickman, 1990; Pidgeon and Hallberg, 2000), although limited geochronological data suggest that they have been deposited at different times. Along the eastern margin of the Murchison Domain are a series of c. 2.81 Ga layered mafic intrusions (e.g. Windimurra and Narndee), associated with minor 2.81 Ga felsic volcanism (Ahmat, 1990; Geological Survey of Western Australia, 2005).

Granite and granitic gneiss, dominated by monzogranite, were emplaced at c. 2.76–2.68 and 2.66–2.62 Ga (Cassidy et al., 2002; Geological Survey of Western Australia, 2005). A minor amount of sanukitoid-like granodiorite was emplaced at c. 2.76–2.75 Ga in the Meekatharra–Mount Magnet district in the northeastern Murchison Domain (Cassidy et al., 2002). The 2.76–2.73 Ga felsic magmatism in the Youanmi Terrane, as well as 2.75 Ga granites in the Narryer Terrane, may be related to collision between those terranes (Nutman et al., 1993; Cassidy et al., 2005).

The Youanmi Terrane is isotopically distinct from other terranes in the Yilgarn Craton and possibly represents the nucleus, or protocraton, onto which younger terranes (Narryer Terrane, Eastern Goldfields Superterrane) were accreted. A pre-3.05 Ga age for the initial formation of the Youanmi Terrane is based on neodymium and hafnium isotope depleted-mantle model ages (Nd and Hf T_{DM}) for granites and felsic volcanic rocks that indicate a 3.4–3.1 Ga felsic crustal component (Barley et al., 2003; Griffin et al., 2004; Cassidy et al., 2005). Detrital and xenocrystic zircon ages that suggest sources of c. 4.3, 3.8, and 3.4–3.1 Ga age (Myers, 1995; Pidgeon and Hallberg, 2000; Wyche et al., 2004; Geological Survey of Western Australia, 2005) may provide evidence of older components in the Murchison and Southern Cross Domains. Future work may further subdivide the Youanmi Terrane.

Eastern Goldfields Superterrane

The Eastern Goldfields Superterrane (EGST) contains at least three tectono-stratigraphic terranes, defined on the basis of distinct volcanic facies, geochemistry, and age

of volcanism that ranges from older than 2.81 to 2.66 Ga. Tectonic juxtaposition occurred after c. 2.66 Ga (e.g. Swager, 1997; Barley et al., 2002, 2003). From southwest to northeast, these are the Kalgoorlie, Kurnalpi and Burtville Terranes. Each terrane is divided into structurally bound domains (Fig. 1) that preserve dismembered, thrust-repeated parts of the succession and locally have distinct volcanic facies relationships (Swager et al., 1992; Swager, 1997; Barley et al., 2002). Proterozoic overprinting of Archean rocks along the southeastern margin of the EGST (Fig. 1) precludes division of this area into terranes and domains.

The terranes and domains in the EGST are bound by interconnected systems of faults (Swager et al., 1992, Swager, 1995, 1997; Liu et al., 2001) that have been defined as part of the pmd*CRC Y2 project (see Champion, 2004). From west to east, these terrane-bounding fault systems are the Ida, Ockerburry and Hootanui Fault Systems (Fig. 1). The Ida Fault System is not clearly established north and south of the Davyhurst–Mount Ida district. To the north the Waroonga Fault System truncates the Ida Fault System (Fig. 1; Champion, 2004), whereas to the south it has been extensively intruded by late granites. Domain-bounding fault systems have been established within each terrane. For example, the eastern boundary of the Ora Banda Domain is the Bardoc Fault System. Further refinement of the terrane- and domain-bounding fault systems will continue as new data becomes available. For example, Jones (in prep.) has demonstrated that the eastern margin of the Bulong Domain of the Kurnalpi Terrane is probably the Roe Hills Fault and the current framework and nomenclature has been changed to accommodate this new information.

Kalgoorlie Terrane

The Kalgoorlie Terrane comprises predominantly young (2.71–2.66 Ga) and minor old (>2.73 Ga) lithostratigraphic sequences. Older (>2.73 Ga) greenstone successions in the Norseman, Boorara, and Wiluna Domains (Nelson, 1997; Barley et al., 2003) are overlain by younger greenstones. Further research is required on these older successions to determine whether they represent autochthonous basement to the younger greenstones. Clearly defined lithostratigraphic sequences are restricted to areas in the Kambalda and Ora Banda Domains in the southern Kalgoorlie Terrane. In these areas the 2.71–2.66 Ga greenstone successions are divided into the c. 2.71–2.69 Ga tholeiitic and komatiitic mafic–ultramafic Kambalda Sequence and c. 2.69–2.66 Ga felsic volcaniclastic Kalgoorlie Sequence (Swager et al., 1992; Krapež et al., 2000; Barley et al., 2002, 2003). Similar lithostratigraphic sequences are present in the Boorara and Jundee Domains in the northern part of the Kalgoorlie Terrane (Swager, 1995, 1997; Messenger, 2000; Barley et al., 2003).

The Kalgoorlie Sequence, incorporating the Black Flag Group (Swager et al., 1992), comprises felsic volcaniclastic and epiclastic rocks of trondhjemite–tonalite–granodiorite (TTG) type, with subordinate lavas that were deposited between 2.69 and 2.66 Ga (Krapež et al., 2000; Brown et al., 2001; Barley et al., 2002). Several depositional and magmatic sequences in the Kalgoorlie Sequence can be grouped into unconformity-bound lithostratigraphic packages that record deposition in a series of transtensional, deep-marine intra-arc basins (Krapež et al., 2000; Barley et al., 2003).

Kurnalpi Terrane

The Kurnalpi Terrane includes c. 2.72–2.70 Ga mafic volcanic rocks, calc-alkaline complexes, feldspathic sedimentary rocks, and mafic intrusive rocks, and c. 2.69–2.68 Ga bimodal rhyolite–basalt and felsic calc-alkaline complexes that extend along a linear belt at the western edge of the terrane, principally in the Gindalbie Domain (Swager, 1995, 1997; Brown et al., 2001; Barley et al., 2003). The c. 2.72–2.71 Ga andesitic volcaniclastic rocks and fine-grained sandstone–shale units ('banded iron-formation') in the eastern part of the Kurnalpi Terrane are separated out as the Edjudina Domain (Barley et al., 2002). The Bulong Domain at the southern part of the Kurnalpi Terrane comprises rocks from the Kurnalpi and Kalgoorlie Terranes that are inferred to have been tectonically interleaved along a complex terrane boundary (Barley et al., 2003).

Rocks of the c. 2.72–2.68 Ga Edjudina, Murrin, and Menangina Domains are interpreted, on the basis of geochemistry, as representing an arc basin, whereas geochemical similarity between intermediate rocks in these domains and the Gindalbie Domain, combined with the trends in age, epsilon Nd, and the distinct volcanic facies, suggest that the Gindalbie Domain represents an assemblage developed during rifting of the Kurnalpi Terrane (or similar) arc. In combination the data are interpreted to suggest that distinct arc-related terranes formed over about 60 m.y., and possibly formed along the same convergent margin, but were probably not adjacent during volcanism (Barley et al., 2002, 2003).

The c. 2.81 Ga Laverton Domain includes mafic and ultramafic volcanic rocks, banded iron-formation, fine-grained volcanogenic sedimentary rocks, and possibly younger-than-2.87 Ga mafic and ultramafic volcanic rocks and banded iron-formation of the Dingo Range greenstone belt (Barley et al., 2003). The Laverton Domain sequence may represent autochthonous basement to the younger rock units in the Edjudina and Linden Domains.

Burtville Terrane

The Burtville Terrane consists of three poorly defined domains. The Hootanui Fault System (Fig. 1) forms the boundary between the Kurnalpi Terrane and the Duketon Domain of the Burtville Terrane. The Duketon Domain includes c. 2.81 Ga intermediate and felsic volcanic rocks and associated mafic(–ultramafic) rocks in the central and eastern parts of the Duketon greenstone belt as well as greenstone assemblages dominated by mafic and ultramafic volcanic and fine-grained sedimentary rocks (Barley et al., 2003). The Merolia and Yamarna Domains contain variably deformed and metamorphosed mafic and felsic volcanic and sedimentary sequences of undetermined age. Further work is required to validate

the character and boundaries of the current divisions in the Burtville Terrane.

Ages of volcanism

SHRIMP U–Pb zircon data show minor peaks in the age of the volcano-sedimentary sequences across the EGST at c. 2.95 and c. 2.81 Ga, with a major peak at 2.72–2.65 Ga (Nelson, 1997; Barley et al., 2002, 2003; Geological Survey of Western Australia, 2005). The older ages are restricted to the minor old sequences in the Kalgoorlie Terrane as well as the Laverton Domain in the Kurnalpi Terrane and Duketon Domain in the Burtville Terrane. Neodymium isotope data have been interpreted to indicate that the volcano-sedimentary sequences across the EGST represent marginal arcs rather than purely oceanic arc systems, and recycling of crust (>3.0–2.8 Ga) along a complex convergent margin, with the youngest crust represented by rocks of the Gindalbie Domain and some intermediate volcanic rocks from the Menangina and Murrin Domains of the Kurnalpi Terrane (Barley et al., 2003). Hafnium isotope data suggest that there were periods of addition of newly generated crust–lithosphere (positive epsilon Hf) and that these episodes also reworked crust that originally formed at c. 3.05 Ga or earlier (Barley et al., 2003). Overall, there is a dominance of continental-margin signatures as well as evidence of magmatic recycling of older arc-related crust. The terranes are interpreted to have been part of the same arc – back-arc system, dismembered, and then reassembled by accretionary tectonics (Krapež et al., 2000; Barley et al., 2003).

Granitic magmatism

The history of granite magmatism, with the exception of the high-high field strength element (high-HFSE) granites, is broadly similar across the EGST (Champion and Sheraton, 1997; Cassidy et al., 2002). High-Ca, mafic and high-HFSE granites correspond to specific volcanic associations in the greenstone belts in terms of timing as well as chemistry. Low-Ca and syenitic granites have no known extrusive equivalents in the EGST. High-HFSE and syenitic granites are largely, but not exclusively, concentrated within a north-northwesterly trending zone (characterized by young Nd T_{DM} ages) that approximately corresponds to domains (Gindalbie, Menangina, Murrin) in the western part of the Kurnalpi Terrane (Barley et al., 2003; Cassidy et al., 2005).

Most magmatism occurred between c. 2.72 Ga and 2.63 Ga, with minor older (>2.73 Ga) granites in the Kalgoorlie Terrane, Laverton and Linden Domains in the Kurnalpi Terrane, and Duketon Domain in the Burtville Terrane (Hill et al., 1989; Nelson, 1997; Cassidy et al., 2002; Dunphy et al., 2003; Geological Survey of Western Australia, 2005). Intrusion of granites post-dates greenstone deposition in the Kurnalpi and Burtville Terranes, but is contemporaneous with TTG-related felsic volcanic and sedimentary rocks in the Kalgoorlie Terrane (Barley et al., 1998, 2002). Although each granite group was long-lived, there are distinct periods where particular groups are most common: c. 2.72–2.68 Ga was dominated by emplacement of high-HFSE and high-Al TTG-type high-Ca granites, whereas c. 2.67–2.65 Ga was dominated by emplacement of transitional TTG-type high-Ca granites, with minor mafic and syenitic granites (Champion and Sheraton, 1997; Champion and Cassidy, 2002). Between 2.65 and 2.63 Ga, low-Ca and minor syenitic granites were emplaced across the EGST (Champion and Sheraton, 1997; Cassidy et al., 2002) and, in places, act as stitching plutons across terrane- and domain-bounding structures.

The high-Ca, low-Ca, and high-HFSE granites have a clear crustal component involved partly, or solely, in their genesis (Champion and Sheraton, 1997; Champion and Cassidy, 2002). Many granites contain older inherited zircons, consistent with Nd-isotopic data, that clearly indicate that extensive older crust must have been present (Hill et al., 1989; Champion and Sheraton, 1997; Cassidy et al., 2005). There is a distinct Nd T_{DM} age change across the boundary between the Kalgoorlie Terrane and the Youanmi Terrane, consistent with the interpretation that the EGST represents younger crustal growth onto the pre-existing Youanmi Terrane (Champion and Sheraton, 1997; Cassidy et al., 2005).

Synorogenic sedimentary rocks

Synorogenic siliciclastic sequences are preserved as erosional synclinal remnants generally adjacent to terrane- or domain-bounding fault systems in the Kalgoorlie and Kurnalpi Terranes. Detrital zircon ages indicate deposition after c. 2.66 Ga (Nelson, 1997; Krapež et al., 2000; Fletcher et al., 2001; Barley et al., 2003). Detrital zircon ages also indicate multiple older sources with ages of c. 2.73, 2.81, 3.0–2.95, and, less commonly, older than 3.40 Ga. These ages correspond to ages of greenstones and granites, but some (c. 3.0 and >3.40 Ga) have no recognized source in the EGST (Barley et al., 2002, 2003). Whole-rock Nd and detrital-zircon Hf isotopic data are consistent with a complicated history involving both mantle-derived magmatism and recycling of older crust since c. 3.1 Ga (Barley et al., 2003). The sequences were deposited after terrane amalgamation, and are probably synorogenic to collision between the EGST and the Youanmi Terrane (Yilgarn protocraton; Krapež et al., 2000; Barley et al., 2003).

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A revised geological framework for the Yilgarn Craton, Western Australia

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The Geology of the Kambalda Nickel Field, Western Australia

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Abstract

The Kambalda nickel field contains the major concentration of volcanic peridotite-associated nickel ores in the Archean of Australia. It occupies an area of 400 km^2 underlain by two sequences of ultramafic, mafic, and felsic volcanics, and sedimentary rocks, which can be broadly correlated with previously erected regional stratigraphic sections. Most ores occur in the lower (Kambalda) sequence at the base of the Kambalda ultramafic rocks; the ultramafic rocks of the upper (Bluebush) sequence contain only minor mineralization. Four phases of deformation occurred over an interval of more than 300 m.y., resulting in a dominant north-northwest structural trend. Faulting was active during original volcanism and continued intermittently until after mafic-felsic dike intrusion. The ultramafic rocks have been serpentinized, and all rocks have exerienced low strain upper greenschist to lower amphibolite facies metamorphism and variable carbonation and potash metasomatism.

The Kambalda ultramafic rocks are of variable thickness and in places exceed 1,000 m. The lower member of the ultramafic rocks, occupying approximately one-third of the sequence, comprises thicker high magnesium flows (i.e., generally >36% volatile-free MgO) commonly separated by sulfidic quartz-albite sediments. Thinner, lower magnesium flow units characterize much of the upper member. The ores occur in 24 shoot complexes each containing multiple ore surfaces caused by structural dislocation of a few, grouped, ribbonlike original orebodies. Eighty percent is contact ore occurring at the base of the lowermost ultramafic flow and generally occupying elongate troughs in the footwall basalt-ultramafic contact; these troughs are thought to be original volcanic depressions which focused later deformation. The remainder is hanging-wall ore usually occurring directly above contact ore at the base of the second or third ultramafic flow. Hanging-wall orebodies occasionally grade into stratigraphically equivalent interflow sediments.

The ores contain varying proportions of massive, matrix, and disseminated sulfides. The hypogene mineral assemblage is pyrrhotite-pentlandite-pyrite-(chalcopyrite), but millerite is locally significant. The massive ores in particular are commonly severely deformed and some have been remobilized tens of meters from their original contact position. Metamorphism has caused recrystallization of the ores and some redistribution of elements. There is commonly marked variation of mineralogy and S/Ni ratios between contact and overlying hanging-wall ores, and within and between original ore surfaces within shoots, but the relative contributions of primary composition and metamorphism to this variation are uncertain.

In ore environments, the footwall basalt-ultramafic contact is more deformed than in non-ore environments, the basal ultramafic flows are thicker and richer in magnesium, the trend toward lower magnesium flows upward through the ultramafic formation is less well developed, and there is better textural development and compositional differentiation within flows. Non-ore environments usually contain interflow sediments, whereas there is generally a sediment-free prism of ultramafic rocks above ore environments.

Most ores are interpreted to have originated by either simultaneous or sequential extrusion of sulfide melts and ultramafic lavas rich in olivine phenocrysts along active linear fissures in a basaltic ocean floor. Volcanic exhalations probably contributed to minor ore in sulfidic sediments.

Introduction

THE Kambalda nickel sulfide deposits occur in the Norseman-Wiluna belt of the Yilgarn Block (Gee et al., 1980), a major Archean craton covering approximately 635,000 km^2 of southwestern Western Australia (see Frontispiece). Nickel sulfides were discovered in January 1966 at Lunnon shoot, and production from this orebody commenced in March 1967 (Woodall and Travis, 1969). As of 24 June 1980, 12.97 million metric tons (t) of ore at a grade of 3.15 percent Ni had been mined from nine separate production centers. Ore reserves at this date were 21.6 million t at 3.28 percent Ni. Annual production is approximately 1.3 million t, and the nickel metal produced

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represents about 5 percent of world production (Ross and Travis, 1981).

The geology of the Kambalda deposits was first described by Woodall and Travis (1969), and the updated description by Ross and Hopkins (1975) included the detailed work by Ross (1974). The literature on more specific aspects is extensive, with the ore mineralogy of Lunnon and Otter shoots being described by Hudson (1972) and Keele and Nickel (1974), respectively, and the supergene alteration of the primary sulfide ores by Nickel et al. (1974) and Thornber (1975a and b). Travis et al. (1976) documented the geochemistry of gossans, and aspects of metamorphism have been discussed by Binns et al. (1976), Barrett et al. (1977), Groves et al. (1977), and Bavinton (1979). An extensive study of the interflow sediments at Kambalda was made by Bavinton (1979), and specific aspects of this work were described by Bavinton and Keays (1978), Donnelly et al. (1978), Bavinton and Taylor (1980), and Bavinton (1981). Precious metal contents of the ores were discussed by Keays and Davison (1976), Ross and Keays (1979), and Keays et al. (1981). The geology and genesis of the Juan complex has been discussed by Marston and Kay (1980), and genetic aspects of the ores have been considered by Groves et al. (1979). A review of volcanic-associated nickel sulfide deposits was presented by Groves and Hudson (1981), and detailed documentation of the various ore shoots of Kambalda has been prepared by Marston (1981).

Numerous internal reports have been prepared by the mine and exploration geologists at Kambalda, and they form an invaluable record of the evolving understanding of the orebodies and the ore environments. These internal reports are referred to in the text and in Table 8, below, and are listed separately in the Appendix.

One objective of this paper is to describe the regional and detailed stratigraphic and tectonic setting of the deposits. Emphasis is placed on describing the variability and complexity of the numerous ore shoots and ore types that constitute the Kambalda nickel field and the differences between ore and nonore environments. Consideration is given to genetic models for the Kambalda deposits and to interpretation of the geologic history.

Regional Geology

Stratigraphy

The Norseman-Wiluna belt of the Yilgarn Block is approximately 800 km long and 200 km wide and contains considerable thicknesses of Archean sedimentary and volcanic rocks with abundant granitoid intrusions. These rocks are broken up by north-northwest trending faults into blocks, within which continuity of lithological sequences can be recognized over distances of more than 100 km. Between blocks it is often difficult, or impossible, to correlate lithologies. In the Kalgoorlie-Norseman area there is sufficient continuity between blocks for a broad correlation of sequences (Gemuts and Theron, 1975), and the interpreted relationship of the stratigraphy in the area encompassing the Kambalda nickel field (Fig. 1) with these sequences is shown in Table 1. The definition of the stratigraphy shown in Figure 1 is based on detailed mapping and drilling carried out over a period of 15 years; this work supports the interpretation by Gemuts and Theron (1975) of two sequences of ultramafic rocks and highlights the concentration of nickel sulfide deposits in the lower (Kambalda) sequence rather than the upper (Bluebush) sequence. It also highlights the difficulty of detailed correlation within the felsic volcanics and sedimentary rocks (sequences 4 and 5 of Gemuts and Theron, 1975). A major problem is the stratigraphic position of the Merougil, Lake Dam, and Triangle Island beds (McCall, 1969), which contain abundant fragments of some underlying rocks and may postdate some deformation. Pending further work, these beds are assigned to the interval between the Kambalda and the Bluebush sequences (Fig. 1).

The original distribution of the Kambalda ultramafic rocks may have been restricted to the Kambalda block and possibly the East Kambalda block (Fig. 1). On the western side they lens out at St. Ives and thin markedly at the Kambalda dome. Within the Kambalda sequence, stratigraphic thicknesses have been established and are shown in Figure 6 (below), but elsewhere in the Kambalda area thicknesses are uncertain because of poor definition of structure caused by sparse outcrop and scarcity of facing evidence. Some formations may be thickened by the poorly understood F_1 isoclinal folding, as suggested by Archibald et al. (1978), on the basis of facing reversals within F_2 folds in the Widgiemooltha-Norseman area.

The volcanic and sedimentary rocks of the Kambalda area are intruded by Archean granites, felsic and intermediate porphyries, mafic dikes and sills, and Proterozoic dolerite dikes. Semiconcordant mafic sills intrude the Bluebush sequence, whereas the mafic to felsic dike suite is mainly restricted to the Kambalda sequence, as noted by Gemuts and Theron (1975) for their equivalent sequence 3. The intrusive rocks are discussed in more detail in the description of the stratigraphy of the Kambalda sequence.

Structure

It is thought that four deformational events accompanied by fold development have occurred in the Kambalda area. The earliest phase of folding, F_1, is

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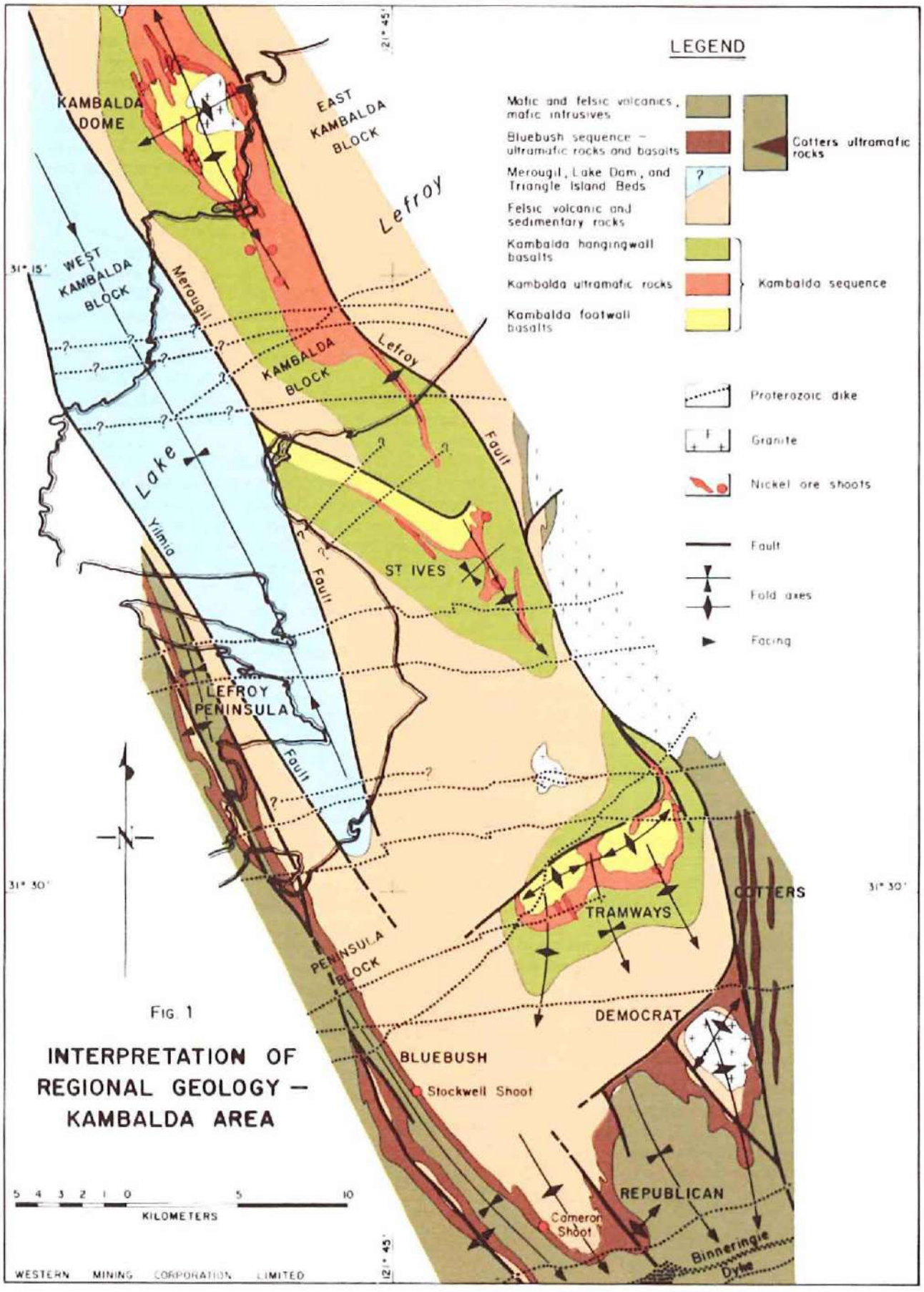


FIG. 1

INTERPRETATION OF REGIONAL GEOLOGY – KAMBALDA AREA

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TABLE 1. Correlation of the Stratigraphy of the Kambalda and Peninsula Blocks (Fig. 1) with Regional Archean Stratigraphy

This paper, Kambalda and Peninsula blocks	Woodall (1965), Kalgoorlie	Gemuts and Theron (1975), Coolgardie-Norseman area
		Sequence 7
Mafic and felsic volcanics, graywackes, mafic intrusives		Acid tuffaceous rocks and acid volcanic breccia; some acid extrusive rocks
Bluebush sequence		Sequence 6
Bluebush hanging-wall basalt Bluebush ultramafic rocks Bluebush footwall basalt		High Mg basalt, ultramafic rocks, minor chert, black slate, and tholeiitic basalt.
		Sequence 5
Felsic volcanics, graywackes, argillites	Black Flag Beds	Conglomerate; arkosic graywacke and argillite with minor tholeiitic basalt; basal conglomerate
		Sequence 4
		Top of sequence gradational to sequence 5; graywacke, minor chert, and black slate; acid extrusive rocks and feldspar-porphyry intrusives; minor tholeiitic basalt
Kambalda sequence		Sequence 3
Kambalda hanging-wall basalt upper member	Golden Mile Dolerite Paringa Basalt Williamstown Dolerite	Black chert marker, tholeiitic basalt, and 4 horizons of ultramafic rocks; intercalated slate and chert bands; minor high Mg basalt.
Sediment-intrusive complex	Kapai Slate	
Kambalda hanging-wall basalt lower member	Devon Consols Basalt	
Kambalda ultramafic rocks	Hannan's Lake Serpentinite	
Kambalda footwall basalt		

characterized by localized, tight to isoclinal, recumbent and napped folds which are evident at a variety of scales within certain sedimentary layers and at the footwall basalt-ultramafic contact. S_1 is weakly developed and trends between north-northeast and north-northwest. F_1 has been refolded by the second, dominant folding event, F_2, which is characterized by relatively open, upright folding and north-northwest trending axial surfaces. These folds are seen to tighten to the north of the Kambalda area and become isoclinal with subvertical axial surfaces and shallow plunges (R. Perriam, pers. commun., 1981).

In the Kambalda area, F_3, with east-northeast to northeast trending axial surfaces, is seen to refold F_2, and this interference effect is considered chiefly responsible for the domal structures of the Kambalda dome, St. Ives, Tramways, and Democrat areas. The F_3 folds have imposed asymmetric plunges on F_2 structures. For example northerly plunges of F_2 caused by F_3 are steeper than the southerly plunges at the Kambalda dome, and the northerly terminations have been sheared out on the St. Ives (Fig. 4, below), Tramways (Fig. 5, below), and Democrat (Fig. 1) structures. F_4 structures are poorly developed, open, upright folds with south-plunging axes which refold F_3.

Archibald et al. (1978), working to the west of the Kambalda area, also defined four deformational events. Their D_1 involved localized recumbent folding and appears similar in style to the F_1 of the Kambalda area. Their D_2 event produced north- to north-northwest trending fold axes and is similar in style to the Kambalda F_2. Their north-northwest-trending F_3 and F_4 have not been observed in the Kambalda area. Platt et al. (1978), studying the Agnew area, defined D_1 as producing isoclinal folds with gently dipping axial surfaces which were refolded by north-northwest-trending upright folds generated by D_2. The asymmetry of F_2 fold plunges in the Kambalda area is consistent with structures in the Kalgoorlie area (Travis et al., 1971).

Fold styles differ markedly between rock types and

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scales of observation. Much of the open folding appears to have been concentric and controlled by flexural slip and possibly flexural flow, especially along less competent horizons such as massive sulfides, sedimentary bands, and other rock unit contacts. Structures at the footwall and hanging-wall contacts of the ultramafic rocks of the Kambalda sequence are commonly not related (Figs. 2, 3, 4, 5). This is caused both by the different physical characteristics of the ultramafic rocks with respect to the enclosing basalts and by the fact that some of the footwall contact deformation is postulated to predate hanging-wall basalt emplacement, as discussed below.

O'Driscoll (1981) has observed that the Kambalda dome and St. Ives areas occur within crosscutting west-northwesterly linear features that are visible in regional photo lineament and aeromagnetic data. Similar features characterize the Perseverance, Mount Keith, and Mount Windarra deposits. O'Driscoll considered these linear features to represent the traces of deep, fundamental fracture zones that exercise a structural control over the formation of the deposits.

As noted by previous workers (Horwitz and Sofoulis, 1965; Williams, 1970; Gee, 1979), north-northwest faults have been active throughout the Archean, forming basin margins, controlling lines of volcanicity and intrusion, and variably offsetting different lithological sequences. In addition, they commonly show a spatial relationship with nickel mineralization: the deposits of the Widgiemooltha area mostly lie immediately to the west of the Higginsville fault (Gemuts and Theron, 1975); the deposits of the Wiluna belt lie to the west of the Perseverance fault (Martin and Allchurch, 1975) or Kilkenny lineament (Gee, 1979); and the Kambalda deposits lie to the west of the Lefroy fault. The major north-northwest-trending faults of the Kambalda area (Fig. 1) have subsidiary branches curving off into northeast and west-northwest directions, as in the Norseman-Kalgoorlie area (Gemuts and Theron, 1975). The Lefroy fault is interpreted to deflect around both the St. Ives and Tramways structures and was therefore probably deformed by D_3. The Lefroy fault is a zone of intense deformation and on its western margin the stratigraphy has been steepened and in places overturned at the Kambalda dome and St. Ives and Tramways structures.

Stratigraphy of the Kambalda Sequence

The Kambalda sequence consists of the footwall basalt, the ultramafic rocks, the hanging-wall basalts, and associated sedimentary and intrusive rock types. Although all rock types are metamorphosed, igneous textures and structures are so well preserved that igneous terminology is used. Ninety-nine percent of Kambalda's premined nickel metal ore reserve is associated with the Kambalda sequence, and although the thickness and characteristics of the rocks within this sequence are variable, it is possible to generalize the stratigraphy (Fig. 6). The stratigraphy of the Bluebush sequence is not discussed in detail, but it contains basaltic, ultramafic, and sedimentary rocks with chemical and textural characteristics similar to those of the Kambalda sequence.

Kambalda footwall basalt

The footwall basalt is greater than 2,000 m thick and is dominated by fine- to medium-grained dark green massive basalt comprising amphibole, chlorite, calcite, and albite with minor epidote and biotite and rare almandine. Pillowed zones are common, normally displaying some degree of stratigraphic continuity. Ellipsoidal pillows ranging in length from 20 cm to 1 m display well-defined rims of chlorite and minor acicular amphibole with epidote, carbonates, and rarely almandine occurring in the interpillow spaces. Vesicles are rare.

In places such as Juan shoot (Harley, 1980) and Lunnon shoot (Middleton, 1980) (Fig. 2), it is possible to map individual basalt flows of both pillowed and massive varieties. Massive flows up to 15 m thick are commonly capped with flow top breccias that comprise angular basalt fragments in a predominantly carbonate matrix. The flow top breccia zones generally parallel overlying ultramafic-basalt contacts. Chemically (Table 2), the footwall basalt is similar to other Archean basalts in the Kalgoorlie-Norseman area (Hallberg, 1970), and although chemical data are limited, it is considered to be of tholeiitic affinity. A thin discontinous, pale albitic sedimentary horizon characteristically occurs 100 to 200 m below the ultramafic-basalt contact.

Kambalda ultramafic rocks

The Kambalda ultramafic rocks are considered to consist of a sequence of extrusive flows spanning an MgO range of 15 to 48 percent (volatile free); the possibility of high-level sills cannot be completely discounted but is unlikely from present evidence. These rocks have been classified into three types based on their volatile-free MgO content (Ross and Hopkins, 1975): picrites contain 15 to 28 percent MgO peridotite, 28 to 36 percent MgO, and olivine peridotites more than 36 percent MgO—these terms are used throughout this paper. Total thickness of the ultramafic rocks is variable, from zero northwest of Foster shoot in the St. Ives area (Fig. 4) to over 1,000 m in the north and east of the Kambalda dome (Fig. 2).

The sequence can be divided into an upper and lower member. The lower member occupies approximately one-third of the sequence and comprises one or more thick high magnesium (generally >36%

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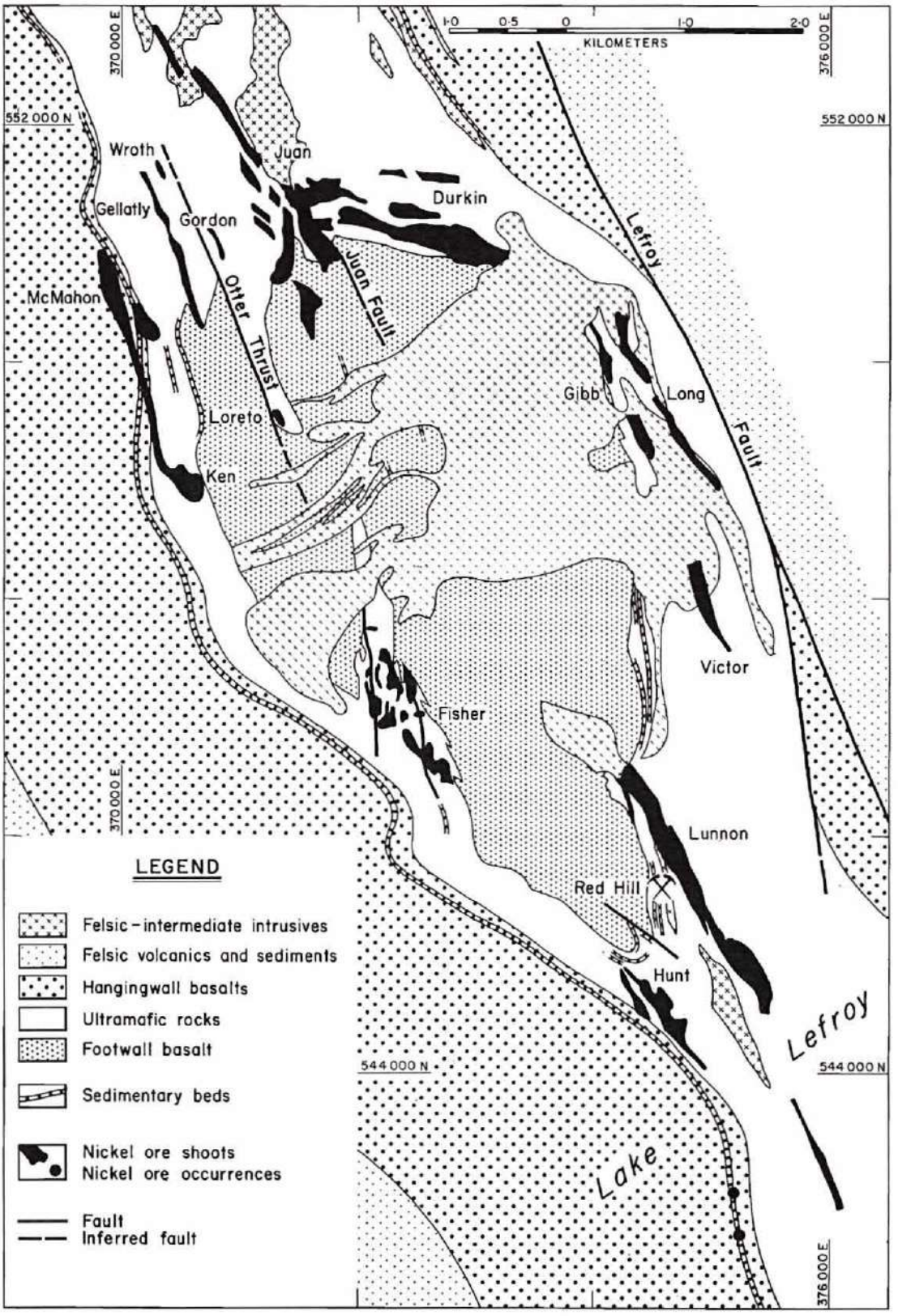


FIG. 2. Geologic plan of the Kambalda dome showing the nickel shoots in plan projection, and the position of the Red Hill gold mine.

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MgO) units with or without intercalated sedimentary horizons. Individual units associated with ore may exceed 100 m in thickness (Fig. 7A) and have thin (2 to 10 m) but well-developed spinifex-textured flow tops (A zones of Arndt et al., 1977). At Lunnon shoot the basal unit is capped by an unusually thick (up to 25 m) spinifex-textured picrite. Ross and Hopkins (1975), referring to this picritic zone as the Lunnon metapicrite, considered it to be a separate flow unit extruded after deposition of the basal unit. In places there is evidence for such a phenomenon, but elsewhere in the mine this picritic zone does appear to be the flow top to the basal unit. The lower member, where it is remote from ore, generally comprises a series of moderately thick (15 to 20 m), high magnesium units with interflow sediments.

The upper member generally consists of thin (1 to 10 m) flow units of peridotitic and picritic composition, although in some instances thicker, high magnesium units may occur well up in the succession. Sediments are generally absent from this part of the sequence. Individual flow units display variable textural development and compositional differentiation but typically (Fig. 7) comprise a thin flow top breccia; a fine-grained polyhedrally jointed zone below this; random, sheaf, and skeletal spinifex development; a cumulate-rich B zone; and a magnesium-depleted basal chilled zone (cf. Arndt et al., 1977). In individual thin units the A zone forms a greater percentage of the total flow unit than does the A zone of the thicker basal units. Thin picrites and peridotites typical of the upper member (Figs. 6 and 7) show all the characteristics attributed to komatiitic lava flows (Arndt et al., 1977). The most phenocryst-rich thick units of the lower member (Figs. 6 and 7) do not display the same classic textural development, but there is a gradation between the two types, and their stratigraphic relationships imply an evolutionary magmatic sequence. There is an overall trend of diminishing Mg content upward through the ultramafic pile, although many reversals occur. Chemical data for different rock types from varying stratigraphic positions are shown in Table 3 and plotted in Figure 8.

The Kambalda ultramafic rocks have undergone varying degrees of serpentinization, talc-carbonate alteration, and late-stage potassium metasomatism; the only relic igneous minerals are chromites, and rare cumulate olivines observed in the Durkin and Victor shoot environments. However, from this and other textural evidence—and from comparison with similar but less altered rocks elsewhere (e.g., Arndt et al., 1977)—it is evident that the rocks originally comprised varying proportions of cumulate olivines and magnesium-rich liquid. Chemical variation within individual flow units is shown in Figure 7. Ni displays a strong positive correlation with MgO, and the MgO content largely reflects the relative amounts of original cumulate olivine and ultramafic liquid. Thus Ni and MgO are enriched in the cumulate B zone of the flows and this corresponds to lower CaO, Al_2O_3, Na_2O, TiO_2, SiO_2, and FeO contents. Cu, Cr, S, and Zn are usually concentrated in the upper spinifex-textured portion of the flows, with Cr, S, and Zn also tending to concentrate toward the base.

Sediments associated with the Kambalda ultramafic rocks

Thin (generally <5 m) sedimentary horizons occur on the footwall basalt-ultramafic contact and in interflow positions within the lower member. Sediments are rare in the upper member, but a few thin discontinuous bands have been recorded. The lithology, mineralogy, chemistry, and spatial distribution patterns of the three main types of sediments at Kambalda have been described in detail by Bavinton (1979, 1981). Pale cherty and dark carbonaceous varieties predominate, with poorly defined chloritic or amphibole-rich varieties occurring less commonly. The cherty and carbonaceous varieties are composed essentially of quartz and albite with minor tremolite, chlorite, calcite, and talc occurring with pyrrhotite, pyrite, and minor sphalerite and chalcopyrite. Sulfides average 15 to 20 percent of the rock. Alternating bands of sulfidic and cherty material occur, with the sulfide content commonly decreasing toward the base. The chloritic sediments have a low quartz and albite content and are dominated by chlorite with lesser amounts of biotite, tremolite, pyrrhotite, and pyrite with minor talc and carbonate. They tend to be more common toward the ore environments but could be stratigraphically equivalent to the cherty varieties. At the base of some interflow sediments a massive siliceous-albitic spheroidal or ocellous zone occurs, and albite-rich, spinifex-textured rocks occur associated with some sediments. The sediments are considered to comprise a mixed detrital component derived from both felsic and mafic-ultramafic sources and an exhalative component contributing sulfur and chalcophile elements (Bavinton, 1981). Chemical data for sediments and associated rock type from the Kambalda sequence are shown in Table 4.

Kambalda hanging-wall basalts

The Kambalda hanging-wall basalt formation comprises an upper and lower member separated by a thin zone of sulfidic sediments and associated intrusives. Ross and Hopkins (1975) ascribed formation status to these members, but in this paper all hanging-wall basaltic rocks have been grouped as a formation that is equivalent in status to the footwall basalts and ultramafic rocks. The contact between the ultramafic

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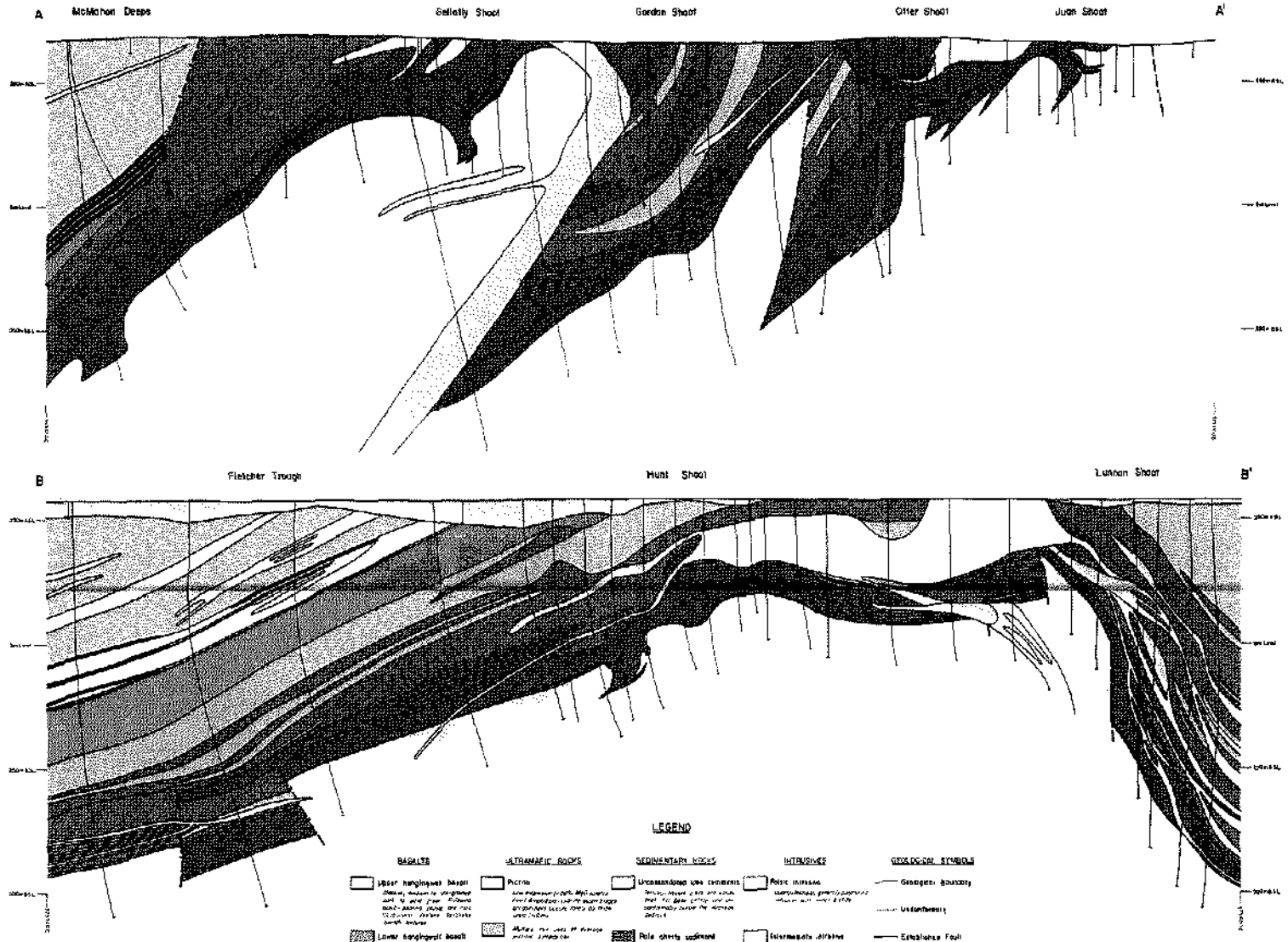


FIG 3 GEOLOGICAL CROSS-SECTIONS - KAMBALDA DOME

KAMBALDA, WESTERN AUSTRALIA

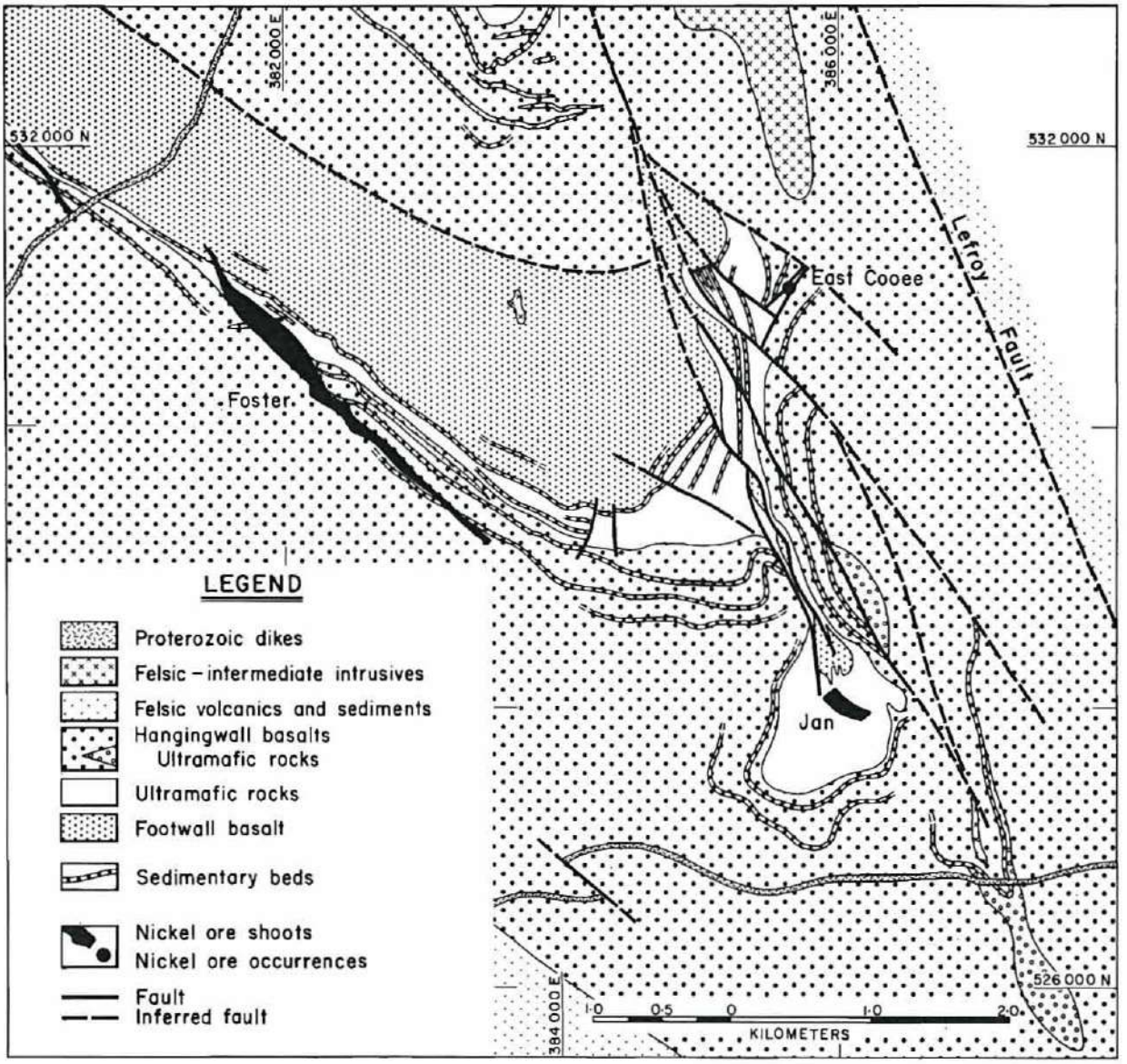


FIG. 4. Geologic plan of St. Ives showing the nickel shoots in plan projection.

rocks and the hanging-wall basalt is extremely sharp at the Kambalda dome, but at St. Ives and Tramways it is more gradational, with interfingering of ultramafic units and basalt occuring over considerable thicknesses. At St. Ives a separate lens of ultramafic rocks occurs well up in the hanging-wall basalt (Figs. 1 and 4). The lower member, 60 to 100 m thick, comprises two major lithologies: zones of well-developed pillowed basalt, with felsic ocelli (Ferguson and Currie, 1972) dispersed in more mafic material around the felsic pillow cores, occur in units up to 30 m thick; flows of more massive basalt, in places also pillowed, form the remainder of the member. The basalts are composed of amphibole, chlorite, biotite, albite, calcite, and quartz. Levy (1979) has observed possible skeletal textures representing original quenched pyroxenes. Because of the highly variable nature of the basalt, average compositions are difficult to determine, but both high magnesian and tholeiitic varieties probably occur. Analyses for the mafic and felsic phases are presented in Table 2.

The upper and lower members of the hanging-wall basalt are separated by a strongly deformed zone of sediments which are intruded, particularly at the Kambalda dome, by felsic and intermediate sills. At St. Ives (Fig. 4) and Tramways (Fig. 5) the zone is

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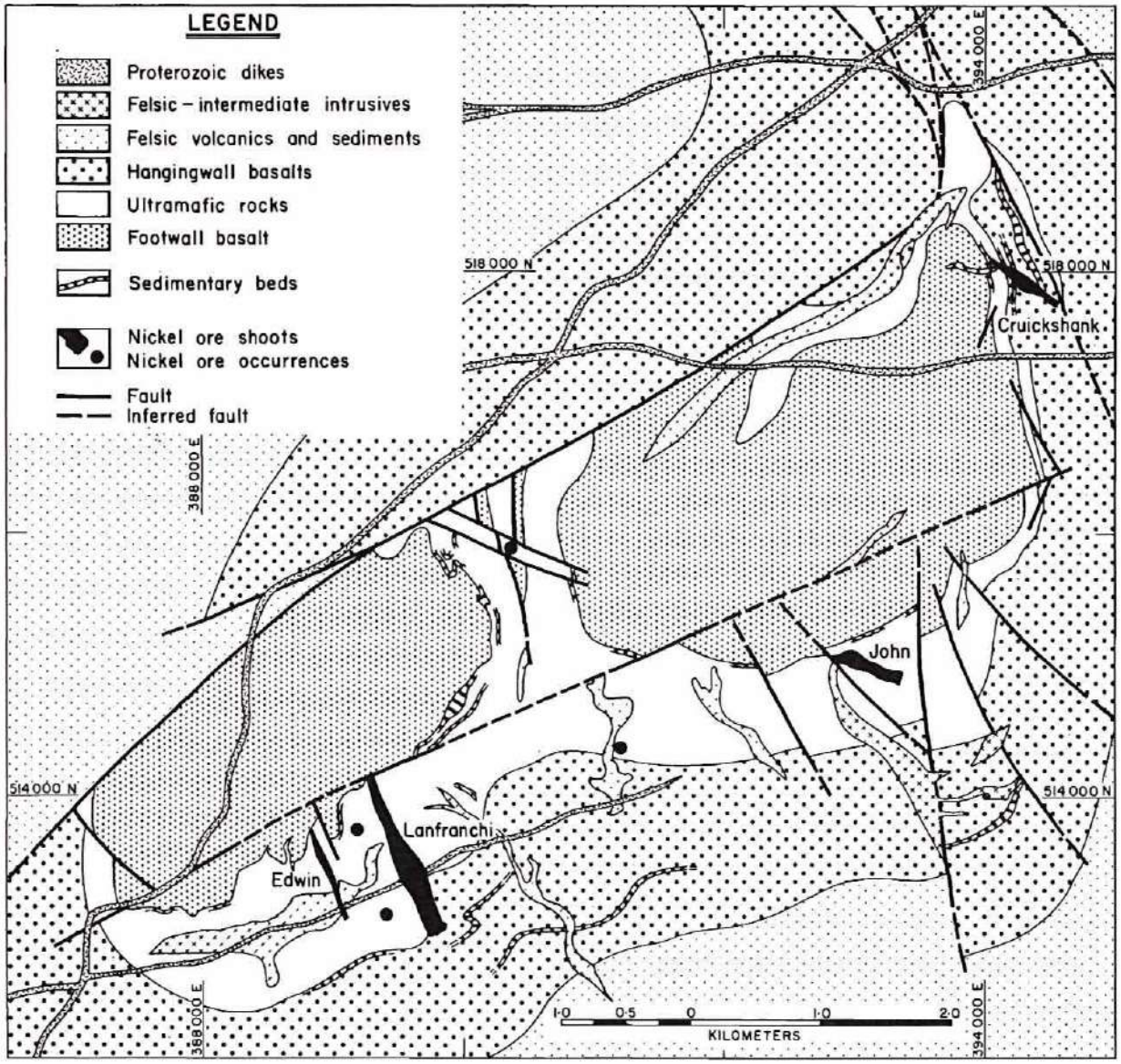


FIG. 5. Geologic plan of Tramways showing the nickel shoots in plan projection.

less well defined, the sediment layers being distributed more widely through the basalt. Individual sediment layers seldom exceed 10 m, and the total thickness of the sediment-intrusive complex is generally less than 40 m. The sediments are dominantly carbonaceous varieties (up to 7% C; Bavinton, 1979) although pale cherty sediments occur. Pyrite and pyrrhotite are the dominant sulfides with chalcopyrite and sphalerite being minor. Quartz and albite and minor tremolite and calcite are the predominant nonsulfide minerals.

Massive basalts with rare pillowed and ocelli-bearing zones occur above the sediment-intrusive complex, and this upper member of the Kambalda hanging-wall basalt formation may be more than 500 m thick. The basalt has experienced pervasive potassium metasomatism and contains abundant biotite. Chemical data (Table 2 and Fig. 8) indicate that the upper member has a higher MgO content and more komatiitic affinities than the lower member and the Kambalda footwall basalt.

Intrusive rocks

Archean granitoids, felsic and intermediate porphyries, minor mafic dikes, and Proterozoic dolerite dikes intrude the mafic and ultramafic rocks of the

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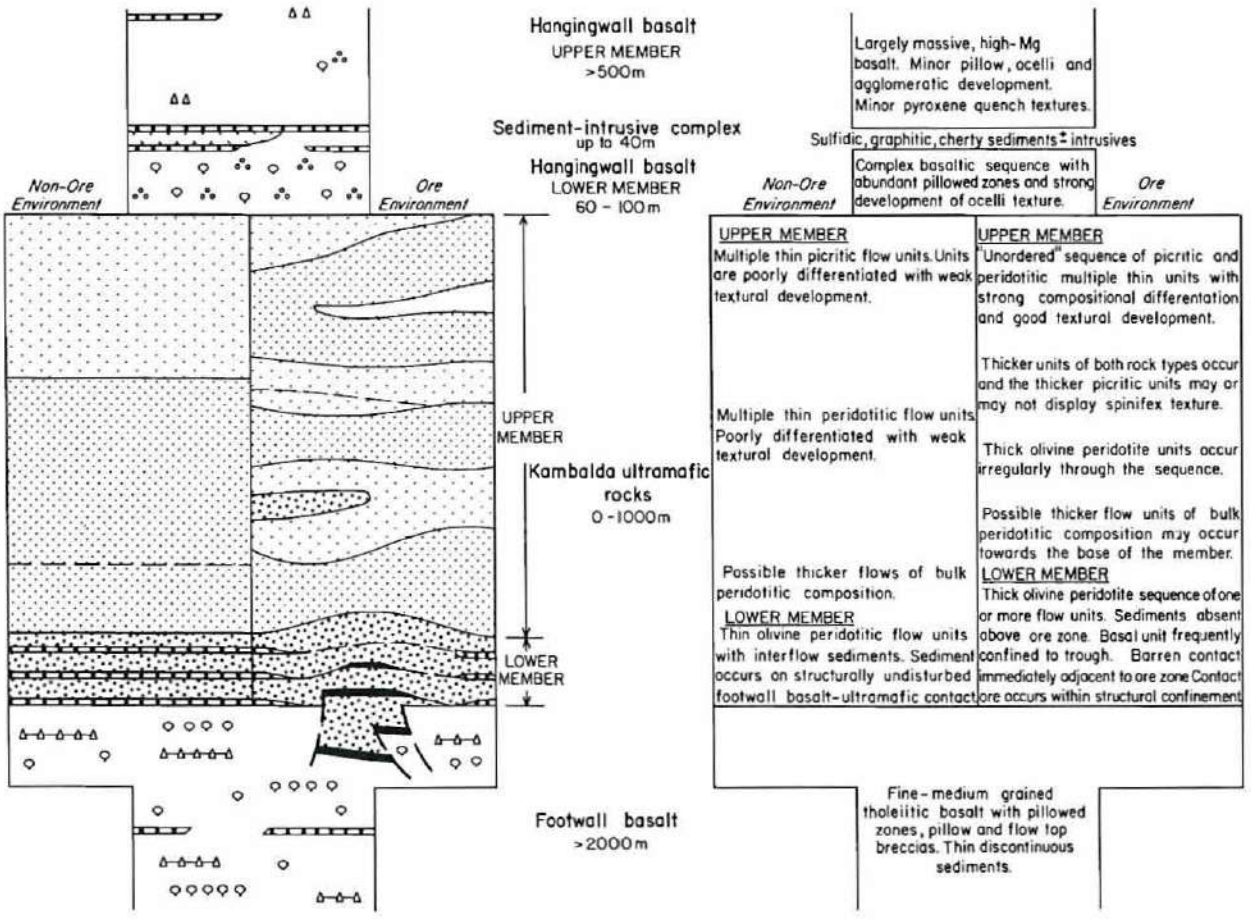


FIG. 6. Stratigraphic column of the Kambalda sequence at the Kambalda dome, with most intrusives removed for clarity. Features of ore and non-ore environments are portrayed and described.

Kambalda sequence. The coarse-grained weakly porphyritic sodic granite in the core of the Kambalda dome has a radiometric age of 2,760 ± 70 m.y. (Oversby, 1975) or 2,820 ± 15 m.y. (unpublished results of R. T. Pidgeon and W. Compston, 1980). This is little different from the most reliable age of the Kambalda footwall basalt, although the best dates on these rocks still have a wide error (2,880 ± 180 to 60 m.y., Compston, 1980). This age is currently being refined using Sm/Nd techniques. The other granitoids shown in Figure 1 are interpreted to be equivalent to the syn- or postkinematic granitoids of Archibald et al. (1978), but no radiometric age data are available.

Felsic porphyries of sodic rhyolite composition are common as dikes and sills throughout the Kambalda sequence, and these are rich in quartz and commonly strongly porphyritic. Examples of these dikes have been dated (zircon and total rock U-Pb) at 2,560 ± 30 m.y. (Compston, 1980). Intermediate intrusives of dacitic composition are also common, although variable in texture and composition. Strongly porphyritic varieties predominate, with large (up to 1 cm) feldspar phenocrysts occurring in a biotite- and amphibole-rich matrix. Mafic intrusives of basaltic composition are less common but have been recorded in Durkin, Gibb, and Lunnon shoots. Typical distribution of the felsic, intermediate, and mafic intrusives is shown in Figure 3 and chemical data are presented in Table 5.

The age relationships between the various Archean intrusives and the timing of their emplacement relative to structural and metamorphic events is a complex problem. In Durkin shoot, mafic dikes predate the felsic intrusives (Hayden, 1976a), and at Lunnon shoot the intermediate intrusives predate the felsic intrusives and are comagmatic with the mafic intrusives (Ross and Hopkins, 1975; Middleton, 1980). Only felsic intrusives have been observed cutting the sodic granite. Intrusives transect major structural features (e.g., the Juan fault and Lunnon thrusts) without obvious offset, but all dikes display varying degrees of offset and dislocation on the ultramafic-basalt con-

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TABLE 2. Chemical Analyses of Kambalda Footwall Basalt, Hanging-Wall Basalts, and Comparative Basalts

	1	2	3	4	5	6	7	8	9	10
SiO_2	52.10	50.77	51.30	51.10	51.64	50.08	56.20	51.47	51.60	52.20
TiO_2	0.68	0.75	0.96	1.70	1.61	0.69	0.71	0.60	0.65	0.80
Al_2O_3	14.70	14.41	14.80	13.30	13.70	14.64	15.68	11.85	13.30	14.83
Fe_2O_3							0.80	1.39		
FeO	10.70*	11.86*	10.50*	15.00*	14.90*	10.14*	5.90	8.94	11.70*	10.36*
MnO	0.07	0.20		0.28	0.30	0.20	0.18	0.19	0.19	0.20
MgO	8.20	8.13	6.70	5.92	4.09	8.59	4.12	11.02	10.00	7.95
CaO	11.00	11.60	10.80	8.60	8.96	9.56	8.46	9.23	10.40	10.17
Na_2O	2.10	1.81	2.70	3.90	2.37	1.80	2.64	1.48	2.16	2.44
K_2O	0.36	0.13	0.18	0.25	0.28	0.16	0.50	0.44	0.11	0.28
P_2O_5			0.12			0.07	0.07	0.07		
S						0.03	0.17	0.02		
CO_2						2.38	2.06	2.78		
H_2O^+						0.06	0.09	0.12		
H_2O^-		1.20				1.44	2.35	0.20		
S≡O						0.22	0.24	0.28		
Rest							0.10	0.03		
Total	99.91	100.86	98.06	100.05	97.85	100.06	100.27	100.10	100.11	99.23

1. Average of 12 footwall basalts from KD1029, Kambalda dome (Roddick, 1974)
2. Average of 6 footwall basalts from KD1029, Kambalda dome (Unpub. data, B. Redman, Univ. of Melbourne, Australia)
3. Averaged composition for 337 basalts from Eastern Goldfields. Total Fe as FeO (Hallberg and Williams, 1972)
4. Average of 9 tholeiitic basalts from Munro Township region, Ontario, Canada; analyses recalculated volatile free (Arndt et al., 1977)
5. Average of 20 Fe-rich tholeiites from Lower Greenstones, Lawlers, and Mt. Goode (Naldrett and Turner, 1977)
6. Averaged composition of five samples of mafic fraction of lower member hanging-wall basalt. Anlayst: B. W. Chappell, Australian Natl. Univ. (Levy, 1979)
7. Averaged composition of four samples of felsic, ocelli-rich fraction of lower member hanging-wall basalt. Analyst: B. W. Chappell, Australian Natl. Univ. (Levy, 1979)
8. Averaged composition of two samples of massive, mafic, upper member hanging-wall basalt. Analyst: B. W. Chappell, Australian Natl. Univ. (Levy, 1979)
9. Averaged composition of 18 komatiitic basalts from Munro Township region, Ontario, Canada; analyses recalculated volatile free (Arndt et al., 1977)
10. Averaged composition of 9 komatiitic basalts from Yakabindie Upper Greenstones (Naldrett and Turner, 1977)

Note. For other basalt analyses see Ross and Hopkins (1975), tables 2 and 3

* Combined Fe_2O_3 and FeO as FeO

tact. Most felsic and intermediate intrusives have well-developed alteration halos of biotite, chlorite, and amphibole where they cut the ultramafic rocks. Ross and Hopkins (1975) consider these halos to be contact metamorphic effects that postdated regional metamorphism, whereas Hayden (1976a) interpreted all the intrusives at Durkin shoot to predate the major metamorphic events. Barrett et al. (1977) and Marston and Kay (1980) also consider the alteration halos to have formed during regional metamorphism.

Unmetamorphosed Proterozoic dolerite dikes occupy, or are displaced along, fractures trending 85°, 45°, 65°, and 165° (Figs. 1, 4, and 5), and these directions coincide with photogeological fracture sets defined at the Kambalda dome by R. Mason (White, 1979). Some of the fractures occupied by dikes are interpreted to be faults with minor movement (Fig. 5). These dikes have Rb-Sr isotopic ages of approximately 2,400 m.y., and younger (Turek, 1966; Roddick, 1974).

Metamorphism and Metasomatism

The rocks of the Kambalda sequence have been metamorphosed to upper greenschist–lower amphibolite facies. Excellent preservation of original igneous textures within the basalts and ultramafic rocks, and limited development of mineral lineations and penetrative structures indicate that metamorphism took place under lower strain conditions than for many other nickel sulfide deposits in the Eastern Goldfields (e.g., Nepean, Widgiemooltha deposits, Windarra). Barrett et al. (1977) considered metamorphic temperatures to be 500° to 600°C with low pressures, and Bavinton (1979), on the basis of more detailed studies of mineral assemblages and compositions of coexisting mineral phases, postulated peak metamorphic conditions of 510° ± 20°C and 2.5 ± 1 kb and considered that equilibrium was achieved at or near the peak of metamorphism. Barrett et al. (1977) and Marston and Kay (1980) postulated two thermal peaks during re-

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gional metamorphism, the second, probably lower grade, postdating intrusion of the felsic dikes.

All rocks in the Kambalda area have been subjected to varying amounts of hydration, carbonation, and potassium metasomatism. The ultramafic rocks have been most susceptible to these alteration processes, with olivines only rarely preserved in some cumulate zones at Durkin and Victor shoots. Textural evidence indicates that carbonation was later than serpentinization, and oxygen isotope data (A. Wilson, pers. commun., 1980) suggest that it was possibly related to the metamorphic events. Bavinton (1979) concluded that carbonation occurred after the peak of metamorphism.

Within the ultramafic rocks, glass and pyroxenes were hydrated to form tremolite and chlorite, and olivines altered to serpentine. Antigorite is the dominant serpentine mineral, suggesting an origin from either prograde metamorphism of preexisting lizardite assemblages or direct serpentinization of olivine at metamorphic temperatures. Antigorite-bearing assemblages occur throughout the Kambalda dome area, although they are more common on the north and east flanks. They are less common in the St. Ives and Tramways areas and are preserved in places in the Bluebush ultramafic rocks, particularly in the Republican area (Fig. 1).

Subsequent carbonation of serpentinites resulted in breakdown of tremolite and antigorite to talc-dolomite and talc-magnesite assemblages. Chlorite appears to have been relatively stable under carbonating conditions, and the lower chlorite content in higher magnesium rocks is a function of original mineralogy. During hydration and carbonation, rock compositions, especially of the high magnesium ultramafic rocks, were changed by the addition of up to 20 percent volatiles and the loss of CaO (Ross, 1974). Ratios of the other major elements remained relatively constant.

The occurrence of biotite replacing chlorite in many mafic and ultramafic rocks is evidence of a widespread, late phase of potassium metasomatism, possibly concomitant with the felsic-intermediate intrusive event. Within the ultramafic rocks biotite occurs mainly in the spinifex-textured flow tops of some units (Fig. 7B). In some areas of more intense alteration, vermiculite and montmorillonite are considered to have replaced biotite (Summons, 1977). Biotitic alteration within the Kambalda hanging-wall basalts is more pervasive than in the ultramafic rocks, and within the footwall basalt biotite-rich zones commonly occur associated with late-stage faults, close to the ultramafic-basalt contact, and on the margins of some quartz veins. More general biotitic alteration has affected the footwall basalt at Tramways.

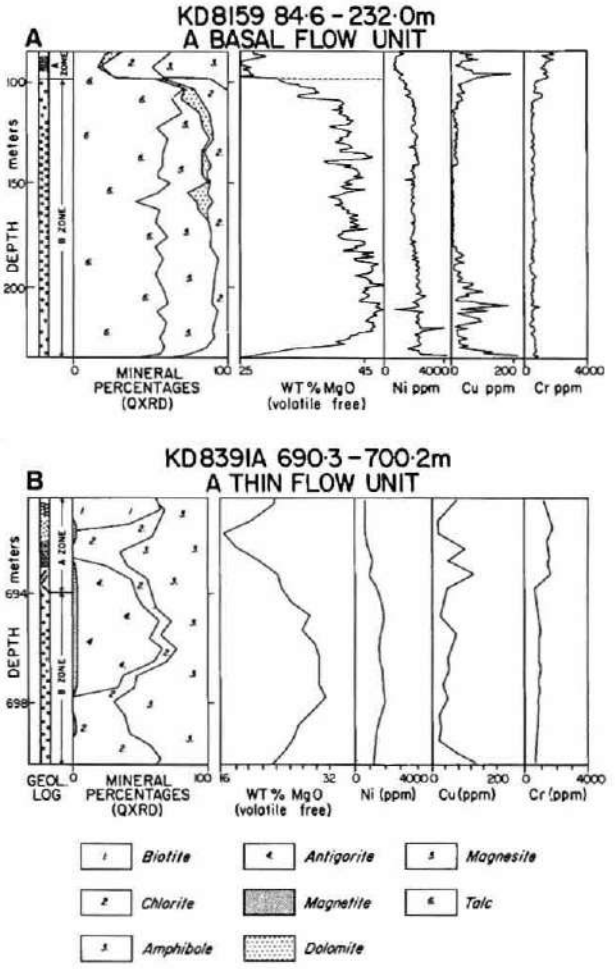


FIG. 7. A. KD8159, chemical and mineralogical variations within a basal flow unit (Bavinton and Scott, 1975). B. KD8391A, chemical and mineralogical variations within a thin flow unit from the upper member. Both examples are from the Juan shoot area.

Ore Distribution

Ores of the Kambalda nickel field are classified into three broad categories (Table 6): (1) contact ore, occurring at the base of the ultramafic rocks, where it usually lies directly on the underlying footwall basalt or, as in a few cases, on or within a sedimentary layer occupying this contact; (2) hanging-wall ore, which occurs within the lower member of the ultramafic rocks and is usually stratiform; and (3) offset ore, which is ore structurally displaced from either contact or hanging-wall positions.

Contact ores and trough structures

Contact ores generally occur within embayments or depressions in the top of the footwall basalt, which are termed trough structures (Figs. 3, 6, and 9). Only rarely do contact ores occur without clearly defined

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TABLE 3. Chemical Analyses of Kambalda Ultramafic Rocks

	1	2	3	4	5	6	7	8	9	10
SiO_2	47.7	46.1	45.1	44.26	45.74	42.69	42.79	49.15	49.31	46.54
TiO_2	0.34	0.29	0.14	0.14	0.33	0.36	0.17	0.40	0.20	0.39
Al_2O_3	7.87	6.22	3.23	2.81	8.01	4.90	4.02	9.40	4.92	8.85
Fe_2O_3					2.64	4.27	3.76	2.15	2.06	
FeO	11.5*	10.8*	8.80*	9.20*	8.59	5.50	5.93	10.11	6.65	11.73*
MnO	0.20	0.18	0.12		0.14	0.15		0.15	0.15	
MgO	23.9	31.0	41.10	41.68	26.93	35.49	37.91	26.46	31.77	22.15
CaO	7.79	5.05	1.41	1.69	5.95	3.91	2.68	6.81	3.89	7.03
Na_2O	0.15	0.06	0.06	0.03	0.12	0.06	0.06	0.22	0.11	1.14
K_2O	0.06	0.01	0.02	<0.01	0.06	0.01			0.07	0.01
P_2O_5	0.08	0.05	0.05	<0.01	0.02	0.01	0.01	0.03	0.02	0.02
S	0.39	0.24	0.30		0.08		0.55	0.72	0.65	
CO_2	0.65	6.59	10.71		1.5	12.32	13.67	5.44	7.47	13.70
H_2O^+	5.77	5.78	6.07		6.34	4.10	3.37	5.44	4.59	
H_2O^-	0.14	0.07	0.07		0.95	0.57	0.37	0.72	1.15	
NiO					0.15	0.25	0.39	0.17	0.26	

1. Averaged composition of 18 drill hole samples from the Lunnon metapicrite (possible picritic flow top to the basal flow unit, Lunnon shoot) (Ross, 1974; Binns et al., 1981)
2. Averaged composition of 10 drill hole samples of peridotitic material from the basal flow unit, Lunnon shoot (Ross, 1974; Binns et al., 1981)
3. Averaged composition of 36 drill hole samples of olivine peridotite from the basal flow unit, Lunnon shoot (Ross, 1974; Binns et al., 1981)
4. Averaged composition of 41 olivine peridotite samples from the basal units of Hunt, Fisher, Gibb, Long, Otter, Juan, and Durkin shoots (Woolrich and Giorgetta, 1978; Binns et al., 1981)
5. Averaged composition of seven drill hole samples from the picritic flow top of the second flow unit, Lunnon shoot (Ross, 1974: samples KR7, 8, 11, 119, 120, 121, W10)
6. Averaged composition of eight drill hole samples of periodotite from the second flow unit, Lunnon shoot (Ross, 1974: samples KR9, 10A, 10B, 12, 13, 123, 124, W12)
7. Averaged composition of three drill hole samples of olivine-peridotite from the second flow unit, Lunnon shoot (Ross, 1974: samples W13, 14, 15)
8. Averaged composition of five drill hole samples of picrites from thin flow units stratigraphically above Lunnon shoot (Ross, 1974: samples W1, 2, 3, 6, 7)
9. Averaged composition of eight drill hole samples of peridotite from thin flow units stratigraphically above Lunnon shoot (Ross, 1974: samples W8, 9, KR68, 69, 160, 161, 162, 163)
10. Averaged composition of three plagioclase-bearing ultramafics from the upper member of the Kambalda ultramafic rocks, south and west of the Kambalda dome (Levy, 1979)

Notes: (1) Major elements recalculated volatile-free, (2) For other analyses of ultramafic rocks see Ross and Hopkins (1975), table 7
* Combined Fe_2O_3 and FeO as FeO

structural confinement. The troughs are narrow, elongate, complex structures, which, apart from Durkin shoot at the north end of the Kambalda dome (Fig. 2), trend between 310° and 345°. Their length, width, and depth are highly variable. Major continuous troughs like those at Durkin, Long, Victor, Lunnon, and Foster shoots vary from 1,000 m to more than 2,300 m in length, but their width seldom exceeds 300 m and is generally 150 to 250 m. Other trough structures such as Ken lower trough and Gellatly and Wroth shoots are only 50 to 100 m wide. The depth of trough floors below the basalt flanks ranges from a few meters to as much as 100 m at the northwest end of Foster shoot (Fig. 4).

The terminology of the bounding structures is related to the ultramafic-basalt contact, not the horizontal. These structures are variable, with Durkin, Gibb, Long, and Hunt shoots being confined up- and downdip by reverse thrusts, and Lunnon and Victor shoots confined updip by normal faults. Where low-angle reverse thrusts confine the ore, and the basalt margins of these structures overlie the mineralization, the ore is said to occupy a basalt-basalt pinch-out. Some of these faults appear to have been active during deposition because they offset the contacts more than the overlying hanging-wall ores and lithological boundaries. On the other hand, diamond drill hole information indicates that the deep trough at the

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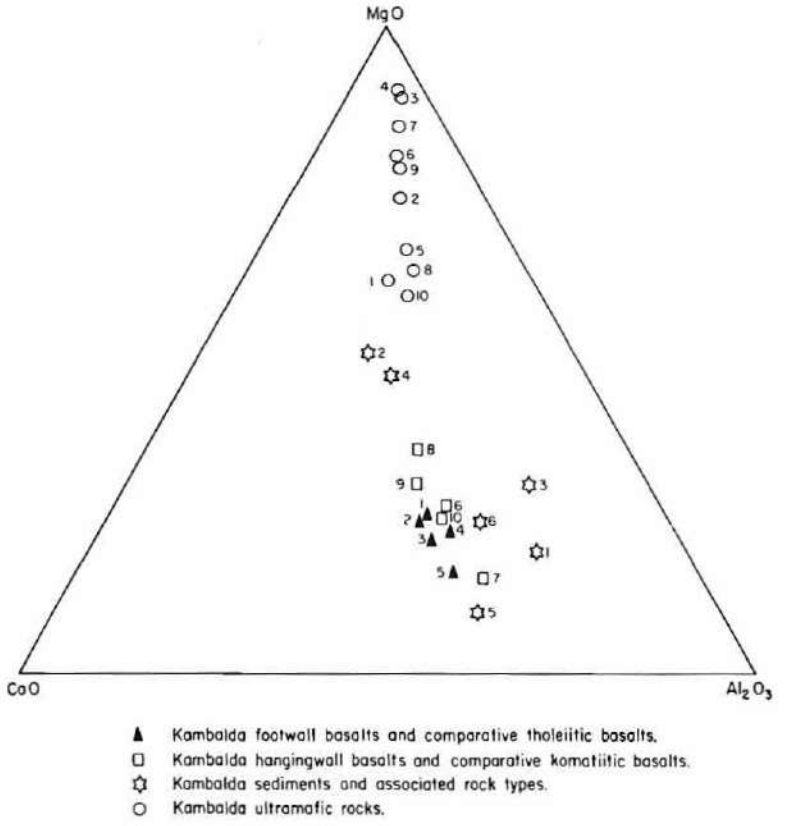


FIG. 8. Al_2O_3-CaO-MgO diagram summarizing the chemistry of major Kambalda rock types and comparative rocks. The numbers refer to averaged analyses given in Tables 2 to 5.

northwestern end of Foster shoot is underlain by a relatively undeformed, homoclinal sedimentary layer, indicating that this trough may have been a preexisting topographic feature within the footwall basalts. In the narrower troughs of Ken lower trough and Gellatly and Wroth shoots, the structural style is dominated by folding with minor faulting (Table 8). Some shoots such as Fisher, Juan, and Jan have undergone such intense postdepositional deformation that the nature of many of the original troughs is difficult to discern.

Mineralization within the major troughs—i.e., Durkin, Long, Victor, Lunnon, and Foster—is essentially continuous even though the ores have been disrupted by postdepositional faults. Within the smaller, narrower troughs mineralization is not continuous, with irregular patches of ore on the trough floor generally confined by clearly secondary structural features. At both Ken shoot and an area of Juan shoot immediately east of the Juan fault, small depressions occur within a relatively planar, patchily mineralized basalt surface. These depressions contain high-grade ore, are generally elliptical, and vary between 20 and 130 m in length; the ore may be almost totally enclosed around its perimeter by a basalt-basalt pinch-out. The relief of these structures with respect to the surrounding basalt surface is generally not greater than 10 m.

Edwin shoot appears to sit in a shallow depression within the basalt, and the ore is typically attenuated at its margins rather than structurally terminated. The central section of Foster shoot also occurs in a shallow depression in the contact with the ore zone lapping onto contact sediments both up- and down-dip. Cruickshank shoot, and part of the margin of Lanfranchi shoot, lack well-defined structural terminations, and a large proportion of the Cruickshank ore overlies planar beds of sediments.

Hanging-wall ore

Hanging-wall ores generally occur within 100 m of the ultramafic-basalt contact and display a close spatial relationship to contact ores. They can be subdivided into two types. Stratiform ores occurring at unit boundaries form the greater proportion of hanging-wall ores at Kambalda, but zones of blebby sulfides located within the basal flow unit form signif-

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TABLE 4. Chemical Analyses of Kambalda Sediments and Associated Rock Types

	1	2	3	4	5	6
SiO_2	53.69	38.33	57.32	50.77	54.46	46.54
TiO_2	0.50	0.35	0.47	0.40	0.21	0.39
Al_2O_3	12.06	8.30	12.12	8.40	6.52	9.07
Fe_2O_3			3.68	5.17		
FeO	12.43*	18.50*	5.46	4.70	16.33*	22.24*
MnO	0.08	0.54	0.06	0.14	0.11	0.11
MgO	3.67	8.45	6.53	14.27	1.10	4.13
CaO	4.07	10.18	3.76	8.34	3.81	4.64
Na_2O	2.82	0.50	6.58	2.58	0.77	1.31
K_2O	1.99	0.63	0.17	0.68	1.02	1.22
P_2O_5	0.14	0.08	0.09	0.05	0.09	0.08
S	4.84	3.38	2.79	3.23	10.01	8.53
CO_2			0.63	0.03		
H_2O^+	3.10	10.99	1.16	1.81	10.19	4.94
H_2O^-			0.08	0.14		
S=O	2.42	1.69	1.39	1.61	5.00	4.26
Rest	0.41	0.46	0.32	0.72	0.47	0.48
Total	97.38	99.00	99.83	100.71	100.19	99.41

1. Averaged composition of 64 siliceous and carbonaceous interflow sediments from Kambalda dome (Bavinton, 1979)
2. Averaged composition of 12 mafic chloritic interflow sediments from Kambalda dome (Bavinton, 1979)
3. Averaged composition of 3 spheroidal rocks found at base of interflow sediments, Kambalda dome (Bavinton, 1979)
4. Averaged composition of 2 albite-rich rocks with spinifex texture associated with interflow sediments, Kambalda dome (Bavinton, 1979)
5. Averaged composition of 4 sediments from sediment-intrusive complex of the Kambalda hanging-wall basalts (Bavinton and Taylor, 1980)
6. Averaged composition of 3 sediments from Kambalda footwall basalts, Kambalda dome (Bavinton and Taylor, 1980)

* Combined Fe_2O_3 and FeO as FeO

icant hanging-wall ore concentrations at Otter, Durkin, and Fisher shoots.

The major hanging-wall ores of Lunnon, Hunt, and McMahon shoots occur at the base of the second olivine peridotite flow unit and are generally in contact with the underlying low magnesium spinifex-textured flow top of the basal unit. All except McMahon are associated with major contact ore zones and are generally closely spatially related to them. These stratiform ores commonly grade laterally into stratigraphically equivalent sediments, with the sediments themselves in places carrying ore-grade mineralization. At Lunnon shoot mineralization is also associated with the third olivine peridotite flow unit, and this is low-grade disseminated ore. The original stratigraphic position of hanging-wall ore at Fisher shoot is obscured by structural complexity.

At Jan shoot, where most ore is in the hanging-wall, the main ore horizon occurs at the base of the third flow unit, and at least two additional ore zones occur associated with units higher in the sequence. A sediment occurring at the base of the second flow unit develops into a low-grade ore position downplunge.

Offset ore

Offset or fault ore forms a small, but locally significant, proportion of the metal reserve. Offset ore may consist of massive, matrix, or disseminated sulfides, but the massive ores are much more intensely brecciated and deformed. At Juan and Hunt shoots, major ore surfaces within the ultramafic rocks are formed by ore that has been sheared off adjacent ore-bearing contacts. One ore surface at Hunt shoot has been sheared out above a normal contact ore surface (see Table 8, below, Hunt cross section 43), and a dislocated lobe of basalt partly overlies this ore. In many places in Juan, Durkin, and Lunnon shoots late-stage faults have offset ore-bearing contacts and ore has been remobilized and sheared out along the fault planes.

Ore Types and Mineralogy

Contact and hanging-wall ores at Kambalda generally comprise varying amounts of massive, matrix, and disseminated sulfides, although sequential layering of massive, matrix, then disseminated ores is generally rare. Individual ore types can occur without the others; massive ores can overlie disseminated and matrix ores; and matrix ores may be separated from the basalt contact by up to 1 m of barren ultramafic rock. Ore zones are generally 1 to 3 m thick and seldom exceed 10 m.

Massive ores are more common in contact orebodies and generally display an irregular distribution varying in thickness from a few centimeters to rarely more than 5 m. The average thickness of massive ores in most orebodies would seldom exceed 1 m. They are defined as containing 80 percent or more sulfides and generally comprise pyrrhotite, pentlandite, and pyrite with minor chalcopyrite and spinels (ferrochromite and magnetite). Spinel concentrations may occur at the base, at the upper contact, or throughout the massive ore. At Gibb shoot and in the hanging-wall ores south of Lunnon shoot, there are assemblages of pyrite-millerite-magnetite and pentlandite-pyrite with minor chalcopyrite and variable amounts of magnetite. Primary millerite also occurs in the supergene ores of Otter shoot (Keele and Nickel, 1974). Minor primary violarite has been recorded in some localities.

Massive ore is often banded with alternating pyrrhotite and pentlandite-rich bands, although pyrite-pentlandite, pyrite-chalcopyrite, and spinel assem-

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TABLE 5. Chemical Analyses of Felsic and Intermediate Intrusive Rocks from Kambalda and Comparative Granitoid Rocks

	1	2	3	4	5
SiO_2	72.53	72.03	60.86	72.90 (2.4)	73.17 (1.9)
TiO_2	0.27	0.31	0.66	0.26 (0.16)	0.25 (0.12)
Al_2O_3	15.70	15.71	15.22	14.47 (0.84)	14.37 (0.70)
Fe_2O_3	0.26	0.64	1.29	0.82 (0.46)	0.76 (0.33)
FeO	1.04	0.86	3.22	1.10 (0.66)	1.04 (0.52)
MnO	0.00	n.d.	0.07	0.04 (0.03)	0.04 (0.02)
MgO	1.54	0.62	4.44	0.52 (0.39)	0.47 (0.35)
CaO	0.48	1.13	2.98	1.80 (0.73)	1.70 (0.62)
Na_2O	5.43	5.56	5.44	4.30 (0.63)	4.34 (0.49)
K_2O	2.19	2.28	2.13	3.72 (1.1)	3.78 (0.77)
P_2O_5	n.d.	n.d.	n.d.	0.10 (0.09)	0.09 (0.05)
H_2O^+	0.44	0.37	0.88		
H_2O^-	0.14	0.09	0.11		
CO_2	n.d.	0.05	2.32		
Total	100.02	99.65	99.62	100.03	100.01

1. Sodic granite from Kambalda analyzed by O'Beirne (1968) (from Ross and Hopkins, 1975)
2. Average of 7 sodic rhyolite (felsic) porphyries from Kambalda analyzed by O'Beirne (1968) (from Ross and Hopkins, 1975)
3. Average of 4 dacitic (intermediate) intrusives from Kambalda analyzed by O'Beirne (1968) (from Ross and Hopkins, 1975)
4. Average (and standard deviation) of 188 Archean granitoids (Archibald et al., 1978)
5. Average (and standard deviation) of 26 synkinematic granitoids (Archibald et al., 1978)

n.d. = not detected

blages can also form layering (Marston and Kay, 1980). The banded texture is caused by recrystallization under directed stress (Ewers and Hudson, 1972). It generally parallels adjacent wall-rock contacts, although in places it is transgressive and can be strongly folded independently of the wall rocks. Irregular, pyrite-rich lenses occur throughout, but they tend to be more common toward the upper and lower contacts. Coarse-grained pyrite occurs associated with fine-grained pyritic lenses and in zones of strongly remobilized massive ore. Individual pyrite megacrysts range up to 5 cm in diameter.

Massive ore displays varying degrees of deformation and remobilization. It thickens and thins over short distances along contacts and is commonly thickened in fold hinges, in basalt-basalt pinch-outs, and along the margin of intrusive rocks. It is often remobilized from its original position on contacts into the footwall basalt or the overlying ultramafic rocks, but maximum movement rarely exceeds a few tens of meters. Fragments of basalt, ultramafic rock, and rarely of intrusives can be incorporated into the remobilized sulfides, and in more intensely deformed areas true breccia ores can occur. This plastic deformation of massive ore relative to more brittle enclosing rocks, including matrix and disseminated ores, is consistent with the deformation experiments on ore and wall rocks carried out by McDonald and Paterson (1980).

Matrix ores occur in both contact and hanging-wall orebodies and generally display a more consistent distribution than massive ores. Thicknesses are variable but seldom exceed 5 m and generally average approximately 1 m. They are defined as containing 40 to 80 percent sulfides with interstitial silicates of moderate to high magnesium composition. The sulfide-oxide mineralogy of matrix ore is similar to that in associated massive ores, and where matrix ore overlies massive ore the contact is usually extremely sharp and defined by a concentration of spinels.

Disseminated ores also occur in contact and hanging-wall orebodies and generally contain 10 to 40 percent sulfides that are interstitial in a moderate to high magnesium ultramafic host. Where disseminated ores occur in contact with matrix ores, the contact is

TABLE 6. Distribution of Ore Types—Kambalda Nickel Field*

	% metal reserve as contact ore	% metal reserve as hanging-wall ore	% metal reserve as offset ore
Kambalda sequence:			
Kambalda dome	83	14	3
St. Ives	69	31	minor
Tramways	80	20	minor
Bluebush sequence	85	15	0
Total Kambalda	80	17	3

* Based on premined ore reserve data, January 1980

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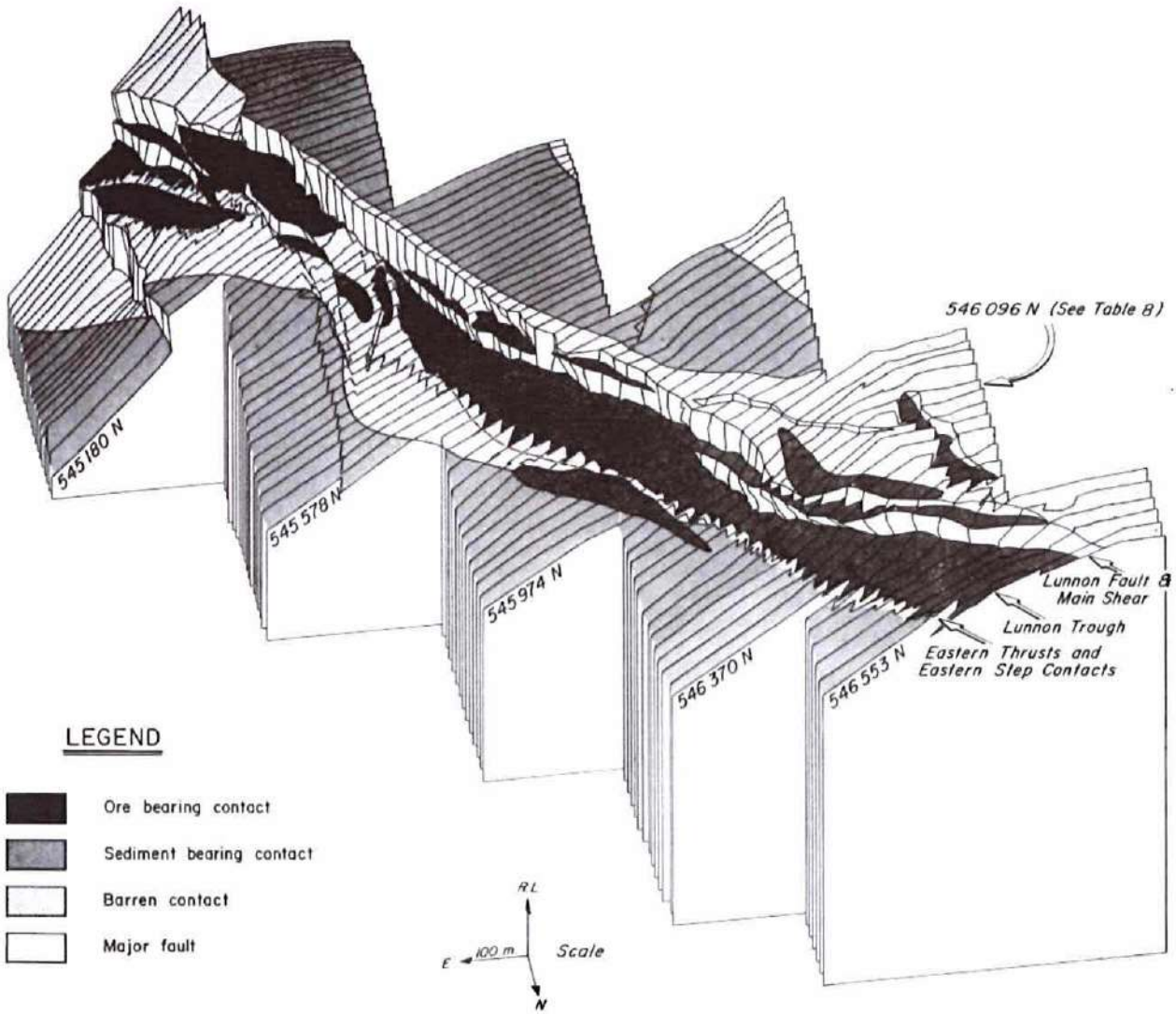


FIG. 9. Isometric projection of the ultramafic-footwall basalt contact at Lunnon shoot looking south from 375 000 E, 547 000 N, +1000 reduced level. Interpretation after Middleton (1980).

generally sharp but is not as well defined as that between massive ore and matrix ore. The thick, low grade disseminated zone containing 5 to 10 percent sulfides (i.e., nickel grades approximately 0.5 to 1.0%) described at Lunnon shoot by Ross and Hopkins (1975) is not common at other Kambalda ore shoots, except for considerable thicknesses of similar material within the basal unit above ore at Foster shoot and associated with some hanging-wall ores at Jan shoot.

Matrix and disseminated ores show much less evidence of deformation (McDonald and Paterson, 1980) and metamorphic effects (Barrett et al., 1977) than massive ores. It is considered that metamorphism did not generate massive ore from more disseminated ores (Barrett et al., 1977; Marston and Kay, 1980), but that minor amounts of massive ores may have been generated during deformation of matrix and disseminated ores (Goss, 1979; Hayden, 1976a).

Blebby ores (Keele and Nickel, 1974; Hayden, 1976b) represent less than 1 percent of the premined nickel metal reserve and occur within the basal ultramafic unit at Otter, Durkin, Gibb, and Fisher shoots and in an isolated mineralized area of Victor shoot (drill hole KD 6068A). Individual ore zones may occasionally exceed 10 m in thickness but are generally 3 to 5 m thick. Hosted in either antigoritic or carbonated high magnesium ultramafics, the blebs may comprise up to 30 percent of the rock mass and average 0.5 to 1.0 cm in diameter. They contain varying proportions of silicates and sulfides, but the sul-

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TABLE 7. Variation in Massive Ore Grades and S/Ni Ratios of Kambalda Ores

Shoot	Ore types	Typical range of massive ore grades (% Ni)	Average S/Ni ratio of ores
Juan	High tenor surfaces	12–30	2.2[1]
	Low tenor surfaces	10–12	3.0[1]
Otter	Contact	15–35	1.6[1]
Durkin	Contact	14–16	2.1[1]
Gibb	Contact	15–45	1.6[1]
Long	Contact	10–14	2.6[2]
Victor	Contact	14–16	2.0[3]
Lunnon	Contact	7–12	3.9[1]
	Hanging wall	10–20	1.8[1]
Hunt	A zone contact	8–10	3.6[1]
	D zone contact	8–12	3.0[1]
	Hanging wall	7–10	3.8[1]
Fisher	Low tenor contact	8–12	2.8[2]
	High tenor contact	12–16	n.d.
	Hanging wall	12–16	2.2[2]
Ken	Contact	12–20	2.2[1]
McMahon	Hanging wall	8–12	3.7[1]
Gellatly	Contact	9–11	3.2[1]
Wroth	Hanging wall	6–8	4.4[1]
Gordon-Loreto	Hanging wall	6–8	n.d.
Jan	Hanging wall	14–20	1.9[2]
	Contact	10–12	2.6[2]
Foster	Contact	8–12	4.2[3]
Edwin-Lanfranchi	Contact	10–15	2.0–2.4[2]
Cruickshank	Contact	n.d.	4.6[3]

[1] Based on drill hole, production, and detailing sampling data
[2] Based on drill hole and production data
[3] Based on drill hole data only
n.d. = no data

fides are rich in pentlandite and millerite with very minor pyrrhotite and the blebs are commonly rimmed by spinels.

Spinifex ores are locally important in the hanging-wall ores of Lunnon and Fisher shoots. In places, massive sulfides occur in contact with the spinifex-textured flow top of the underlying unit, which in turn can contain sulfides interstitial to the plates (now amphibole-chlorite) of the spinifex zones.

As discussed below there is a clear antithetic relationship between sediments and ore. However, in some places they are juxtaposed, and there can be a development of sedimentary ores, with well-banded sulfidic-albitic sediments forming nickel ore. In most places where these occur (McMahon, Ken, Juan, Fisher, Hunt, Lunnon, and Jan shoots), there is evidence for varying degrees of structural deformation and remobilization of the nickel sulfides into the sediments, but in some examples nickel sulfide layers are intimately interbedded with otherwise normal sediments (Bavinton, 1981).

Basalt breccia ores occur locally (Juan and Lunnon shoots) and comprise basalt fragments in a matrix of sulfides. These ores may underlie associated massive or matrix ores, or an otherwise barren ultramafic-basalt contact.

Interpillow sulfide ores are not common and may occur up to 15 m below the ultramafic-basalt contact (Ken shoot) and comprise well-developed pillow zones with the interpillow spaces partially filled with massive sulfides.

Variation in mineralogy and S/Ni ratios

Variation in the Ni tenor of the sulfide component and consequently the S/Ni ratios, between different ore shoots, between ore surfaces within individual shoots, and even within individual ore surfaces of the shoots is a marked feature of the Kambalda nickel field (Ross and Keays, 1979; Woolrich et al., 1981) and is summarized in Table 7. S/Ni ratios and massive ore grades are averages based on sampling of the ores, and grade data have not been recalculated to 100 percent sulfides. The variation in S/Ni ratios is a reflection of the relative abundances of pentlandite, pyrrhotite, and pyrite in the ores.

The contact orebodies of the Kambalda dome can be classified as high tenor (i.e., S/Ni >2.5) or low tenor ores (i.e., S/Ni <2.5), apart from Juan and

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TABLE 8. Description

Shoot dimensions are measured in the plane of the shoot and relate to the overall shoot complex, not individual ore surfaces, unless otherwise specified. The categories of premined ore reserves relate to June 1980 estimates and refer to metric tons (t) of contained nickel metal as follows: A, greater than 100,000 t; B, 50,000 to 100,000 t; C, 20,000 to 50,000 t; D, 10,000 to 20,000 t; E, less than 10,000 t.

Location, geometry, mining, and references	Stratigraphy and structure
JUAN Juan shoot is located at the NW corner of the dome and is the single largest concentration of nickel sulfides at Kambalda. It comprises two structural domains separated by the Juan fault. The ore complex crops out to the S and plunges at 335° over a known distance of 3 km to where it is 1,200 m below the surface. The width of the mineralized zone west of the Juan fault in places exceeds 400 m. The orebody is mined by a decline to the 15L and a raise-bored haulage shaft to the 9L. Premined ore reserve: A. Production: A. Percent metal reserve—contact: 91; hanging wall: 6; offset: 3. Average Cu:Ni = 1:15; Co:Ni = 1:55. References: Kay (1977), Anderson (1978), Harley (1980), Marston and Kay (1980).	The basal unit in the ore environment can be extremely thick (up to 100 to 150 m). Talc-magnesite-chlorite assemblages predominate with antigorite being very rare. Elsewhere the lower member may comprise a series of thinner high magnesium units with interflow albitic sediments. The upper member is unordered and frequently flow units display strong textural development and compositional differentiation. The ultramafic sequence thickens to more than 1,000 m to the N. The footwall basalt is frequently well pillowed and in places excellent flow top breccias occur. W of the Juan fault, ore surfaces plunge NW on W-dipping thrust slices caused by high-angle reverse faulting with indeterminate vertical and horizontal movement. E of the fault the Durkin Deeps area is characterized by strong folding with associated overthrusting. Extensive areas of ore are entirely enclosed within basalt. The outside surface is characterized by a highly irregular contact with lobes and depressions in the basalt. Some ores are located within these depressions.
OTTER Otter shoot is a small ore position associated with a basalt "lip" or "tongue," in part isolated within the ultramafic sequence above Juan shoot. The orebody crops out and plunges to the NNW for 200 m at approximately 20°. The width of the main contact surface is 150 m. The orebody is accessed via the Otter-Juan decline and mined from the 2L to the 5L. Premined ore reserve: D. Production: D. Percent metal reserve—contact: 88; hanging wall: 10; offset: 2. Average Cu:Ni = 1:17; Co:Ni = 1:81. References: Woodhouse (1971), Keele and Nickel (1974).	All ultramafic rocks above the contact ore comprise olivine peridotites with both antigorite and talc-chlorite-magnesite assemblages, although within a meter of the contact development of lower magnesium amphibole-chlorite rocks with strong amphibole recrystallization is common. On the underside of the Otter Basalt there are some contact sediments and below this is a sequence of predominantly peridotitic flow units. The Otter Basalt is connected to the main basalt to the S, but the erosional surface has severed this connection to the N and created a tongue of basalt enclosed within ultramafic rocks. The origin of the structure is uncertain, but strong deformation is evident on the underside contact and ore has been sheared off the western edge into the ultramafic. The main ore surface is spoon shaped with a steep E margin and a flat N plunge.
DURKIN Durkin shoot occurs at the N end of the dome and to the E of and essentially continuous with the Juan East/Durkin Deeps ore surfaces of Juan shoot. The orebody crops out and extends over 1,500 m at 285° and is terminated to the E by granite. Individual ore surfaces display very low plunge angles to the E and W and the orebody attains a maximum width of 300 m. Durkin has been extensively mined by a vertical shaft to the 8 level. A crosscut on the 7 level connects to Gibb shoot. Premined ore reserve: A. Production: B. Percent metal reserve—contact: 96; hanging wall: 3; offset: 1. Average Cu:Ni = 1:16; Co:Ni = 1:63. References: Hayden (1976a and b), Hancock (1977).	The basal unit which is commonly also the lower member is 25 to 60 m thick. Minor original olivine occurs at the E end of the shoot but overall the basal unit is antigorite rich with carbonation increasing to the W. The upper member comprises flow units with excellent textural development and compositional differentiation, and although these are predominantly peridotitic and picritic in composition, thick high magnesium units do occur high in the sequence. The shoot contains three structural domains. In the E a well-defined trough is formed by thrusts. The central portion of the orebody is dominated by thrusts with NE-plunging pinch-outs and pronounced scissor movement along the faults. In the W the structures are dominated by thrusts and significant subvertical normal fault movement; pronounced folds of the basalt contact with flat NW-plunging axes are common. Thrusting took place early, with later movement on normal faults and the contact and folding of some thrusts.

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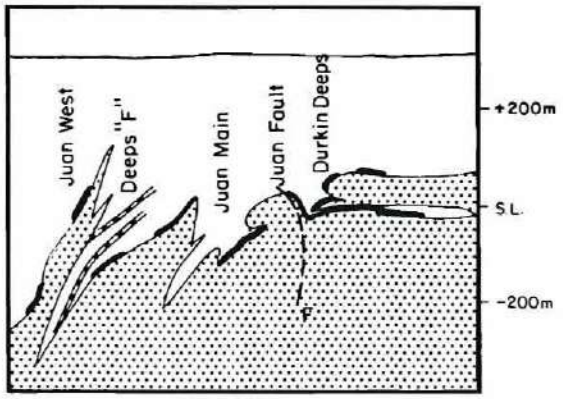
of Kambalda Ore Shoots

Production figures to June 1980 also refer to contained nickel metal, as follows: A, greater than 100,000 t; B, 50,000 to 100,000 t; C, 20,000 to 50,000 t; D, 10,000 to 20,000 t; E, 5,000 to 10,000 t; F, less than 5,000 t. For abbreviations, see the end of the table.

Ore distribution, mineralogy, and sediment relationships	Typical cross-section

Almost all ore is contact ore in embayments within the basalt and confined by pinch-outs. Ore is typically primary pn-po-py assemblages with massive and disseminated zones and minor matrix ores.Pronounced banding is common in massive ores with distinct lamellae mainly of pn-po. Juan contains ores of low (10–12% Ni) and high (12–30% Ni) tenor. It probably originally comprised two ore belts of different tenor. These were widely separated in the S but converged to the N where in the Juan West area they are separated by a 50 m horst structure. Breccia ore within the footwall basalt is common and all contacts show varying degrees of disturbance.

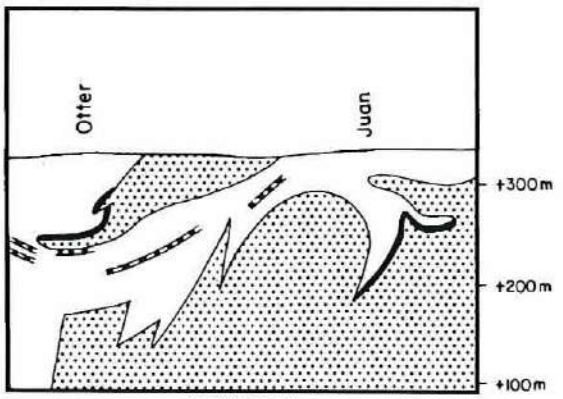
Contact sediments are generally absent within the ore environment, with minor sediments occurring on the Deeps "F" contact and developing on the surfaces W of the Juan West environment. Within the ultramafic sequence there is a well defined break in the internal sediments above the major ore positions (Juan Main–Durkin Deeps), but sediments do occur within the ultramafic sequence on the underside of the Juan West structure and W of Juan West. Ni ore associated with sediments occurs in only two localities—on the 6L intimately associated with the Juan fault and on the 12L on the underside of the Juan West structure.



551 420 N
JUAN

Contact ore comprises supergene assemblages of vl-py-ml although some vl is considered to be primary. Ores have very low S/Ni ratios and contain massive zones of varying thickness (0.3–1.0 m) and disseminated ores with minor matrix ores. Blebby hanging-wall ores occur within the basal unit (cf. Durkin) in zones 2–10 m thick. Spherical blebs of ml–pn±py±mg occur associated with silicate blebs.

No sediments occur above the ore zone.

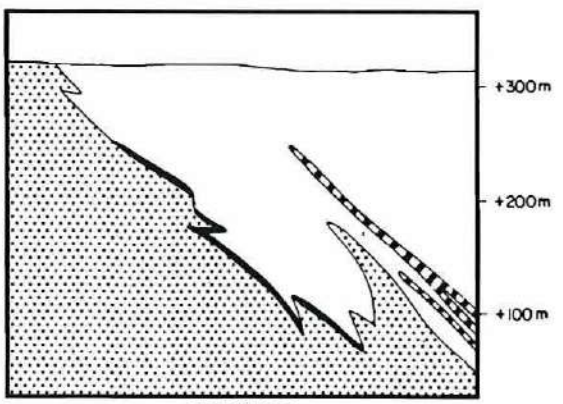


550 850 N
OTTER

More than 90 percent of the ore is contact ore and the style of mineralization changes from E to W as the structural style changes. The E area is dominated by massive ore approximately 0.5 m thick, the central area by matrix ore with massive ore in the pinch-outs of the diagonal thrusts, and the W area by matrix ore with massive ore occurring only in major pinch-outs. Massive ore tenor is variable but averages 14–16 percent. Ni with S/Ni ratios 2.0–2.2. Above the 2L all ore is supergene vl+py±cpy with transition ore occurring down to the 7L. Blebby hanging-wall ore occurs within the serpentinized basal unit. Sulfide blebs consisting essentially of pentlandite occur associated with silicate blebs.

Contact sediments are absent within the ore environment. They occur immediately downdip of the orebody at the E end and further downdip in the central section.

Within the ultramafic sequence, interflow sediments are abundant on the downdip flanks of the orebody but die out above the ore zone. There are no known occurrences of ore associated with sediments.



373 049 E
DURKIN

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TABLE 8.—(*Continued*)

Location, geometry, mining, and references	Stratigraphy and structure
GIBB Gibb shoot is a small orebody located on the E flank of the dome and updip from the much larger Long shoot. The shoot is 1,300 m long, terminated to the north by granite and separated into two sections by a large porphyry mass. The orebody trends at 335°, achieves a maximum width of 150 m and both gentle northerly and southerly plunges have been observed. The orebody is extensively invaded by felsic intrusives. The northern section has been mined and developed on 5 levels and ore is hoisted through Durkin shaft. The orebody is serviced by a small raise-bored shaft. Premined ore reserve: D. Production: F. Percent metal reserve—contact: 91; hanging wall: 2; offset: 7. Average Cu:Ni = 1:13; Co:Ni = 1:65. References: Quick (1978, 1979).	The lower member is approximately 50 m thick and may comprise one or more units. Both antigorite and talc-chlorite-magnesite assemblages are common and the lower part of the basal unit is strongly sheared and generally talc-chlorite-dolomite in composition. Little of the upper member is recorded, but where seen individual units are well developed. The footwall basalt is intensely deformed and partially biotitized with abundant epidote and quartz-calcite veinlets. Gibb occupies a complex trough structure with an inner ore-bearing surface almost completely enclosed by basalt-basalt pinch-outs. Numerous outside "flank" or "shoulder" surfaces have been delineated, only some of which are mineralized. This complex is cut off downdip by a major normal fault (with reference to the contact) which separates the trough structure from a series of undulating contacts with highly variable dips. Some of these surfaces carry ore-grade mineralization.
LONG Long shoot is a major ore position on the E flank of the dome downdip of Gibb shoot. The orebody has a plunge length of at least 2,300 m and a maximum width of 300 m. The orebody plunges gently to the north where it is terminated by granite. Dips are steep to subvertical, but appear to flatten to the S as southerly plunges become more pronounced. The shoot is extensively invaded by felsic intrusives. The orebody is accessed by the deepest vertical shaft at Kambalda (936 m) and is currently being developed on the 7 to 12 levels. Premined ore reserve: A. Production: F. Percent metal reserve—contact: 98; hanging wall: 1; offset: 1. Average Cu:Ni = 1:13; Co:Ni = 1:53. References: Middleton and Levy (1977), Reeve (1977).	The lower member, comprising one or more units, is 50 to 100 m thick, and is composed dominantly of antigorite assemblages, although talc-magnesite-chlorite rocks do occur. Basal units commonly display excellent spinifex-textured picritic flow tops. The upper member is characterized by thin flow units with strong textural development and compositional differentiation. The sequence has no defined upward-diminishing Mg trend and thicker high magnesium units occur well up in the sequence. The footwall basalt is moderatly deformed with biotitic alteration common. Pillow zones and flow top breccias have been observed. The ore appears to be confined up- and downdip by low-angle thrusts defining a low-relief trough structure 200 to 300 m wide. Normal faults at high angles to the contact have been delineated and these may have appreciable scissor movement along strike. Occurring in the hanging-wall of the ore are pods or lenses of basalt that in places form basalt-basalt pinchouts.
VICTOR Victor shoot is a contact ore position located on the E flank of the dome S of Gibb and Long shoots but separated from them by a major body of felsic intrusives. The orebody occurs in a well-defined trough structure and has a plunge length of 850 m. The trough structure is defined to the S for at least another 700 m but is apparently unmineralized. The orebody plunges to 160° at approximately 10° and has a width of 200 m. Victor shoot has not been mined and is defined entirely by surface diamond drilling. Premined ore reserve: C. Production: nil. Percent metal reserve—contact: 99; hanging wall: 1; offset: 0. Average Cu:Ni = 1:14; Co:Ni = 1:69. References: Middleton and Levy (1977).	The ultramafic succession is at least 750 m thick in the Victor area and comprises well-defined lower and upper members. The basal unit above the orebody can be in excess of 75 m thick and is dominantly antigorite with rare zones of relic primary olivine. It may be overlain by 2 or 3 additional olivine peridotite units, making total thicknesses of 200 m for the lower member not uncommon. Indivdual units have very thin (2 to 4 m) spinifex-textured flow tops. Updip from the ore environment the lower member comprises a number of relatively thin (10 to 15 m) high magnesium flow units with interflow sediments. The upper member is an unordered sequence of well-developed peridotitic and picritic thin units with occasional thicker high magnesium units. Carbonation is common throughout. The orebody is confined downdip by low-angle reverse faulting and updip by high-angle normal faults, the throw on which increases to the S. 200 m updip from the orebody another trough with similar confining structures has been defined but no mineralization has been recorded within it.

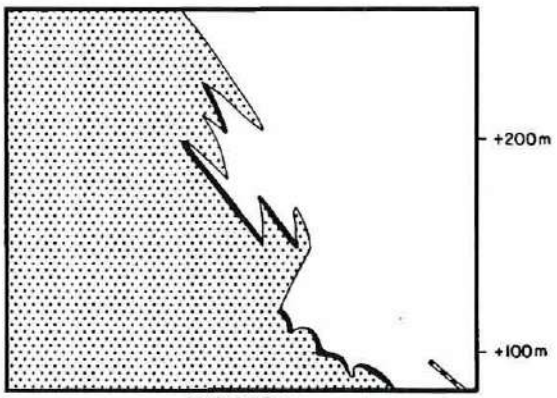
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Ore distribution, mineralogy, and sediment relationships	Typical cross-section

The inner surface of the trough is the most extensively mineralized ore surface but massive ore is rarely more than 50 cm thick and occurs predominantly in pinch-outs and irregularities in the contact. Matrix ores are also rare. Most ore occurs as disseminated sulfides and frequently as veinlets in the basalt. Two dominant ore types occur—pn–py±cpy and ml–py–mg±cpy—as well as their supergene equivalents. Gibb is a very low sulfur ore with massive ores varying between 15 to 45 percent Ni depending on mineralogy. S/Ni ratios range from 1.2 to 2.0.

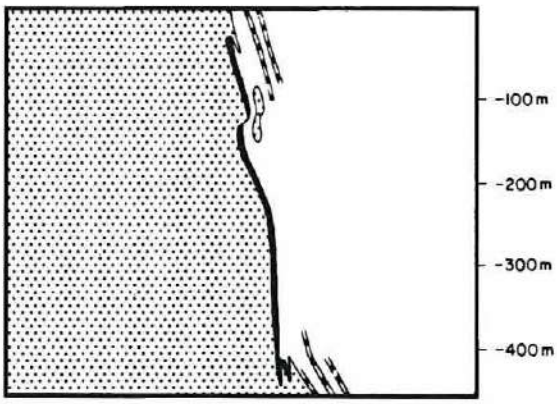
Both contact and hanging-wall sediments are absent from the ore environment, although the latter occur on and above the barren section of contact separating Long and Gibb shoots.



550 000 N
GIBB

Ore is almost entirely contact, and structural modification has resulted in significant concentrations of massive ore. Relatively undisturbed contact may contain 0.5 m massive ore with 1 to 2 m of matrix ore and very minor disseminated ore. All ore is primary and is dominantly pn–po–py±cpy, although ml-py assemblages have been recorded. In strongly disturbed areas large (up to 4 cm) megacrysts of py develop. Massive ore commonly grades 10 to 14 percent Ni and matrix ores 3.5–4.5% Ni. S/Ni ratios average 2.6.

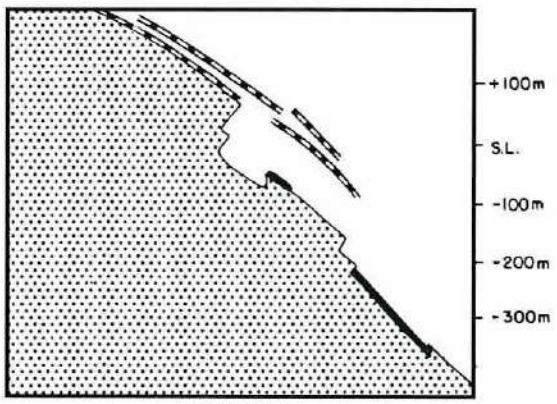
No sediments occur within the ore trough, although chloritic, possibly tuffaceous zones have been noted on the outside contact of the basalt pods in the hanging-wall of the ore. Contact sediments are developed on both sides of the trough structure, and hanging-wall sediments overlap the trough on both sides.



549 880 N
LONG

Almost all the ore occurs as contact ore and primary assemblages of po–pn–py are dominant. Massive, matrix, and disseminated ores occur, with massive ore grades being highly variable but averaging 14 to 16 percent Ni, with S/Ni ratios approximately 2.0.

No contact sediments occur within the trough structures and there is generally a well-defined zone of barren contact away from the trough before sediments develop. Interflow sediments display a clear antithetic relationship to ore, with no internal sediments observed directly above the ore zone, but numerous sediments develop on the W margin of the orebody. To the S where the trough structure continues but is barren, internal sediments overlap the structure.



547 925 N
VICTOR

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TABLE 8.—(*Continued*)

Location, geometry, mining, and references	Stratigraphy and structure
LUNNON Lunnon shoot is a major ore position located at the S end of the dome on the E flank. The ore complex crops out and has a plunge length of over 2,400 m, and although plunge reversals do occur it has an overall plunge at 150° of 15°. The maximum width of the ore is 280 m. The main Lunnon trough has a plunge extent of 1,850 m. Hanging-wall ore occurs at interflow positions and generally overlies the contact mineralization. The orebody is accessed by a vertical shaft to the 11L and development extends 2,500 m south of the shaft. Premined ore reserve: A. Production: A. Percent metal reserve—contact: 61; hanging wall: 33; offset: 6. Average Cu:Ni = 1:11; Co:Ni = 1:44. References: Ross (1974), Shugg (1975), Middleton (1980).	The lower member comprises a series of thick high magnesium flow units with carbonated assemblages dominant. It may be up to 140 m thick over the trough but thins both to the E and W, and interflow sediments become more prevalent. The basal unit in places is overlain by a thick (up to 25 m) picritic horizon. The upper member is dominated by flow units of peridotitic composition with minor thin picritic flows. The footwall basalt is strongly pillowed in places and flow top breccias have been observed. The W margin of the trough structure is defined by a steeply dipping normal fault that has been active over a period of time displacing the contact ores considerably more than the hanging-wall ores. Low to moderate angle reverse thrusts define the E limit of the trough, and these structures also displace the hanging-wall ores. At the S end of the trough the dominant structures are high-angle normal faults that progressively displace the contact down to the E (Fig. 9).
HUNT Hunt shoot is located at the S end of the dome on the W flank. The orebody comprises two major contact ore surfaces as well as significant hanging-wall and offset ore. The plunge length exceeds 1,000 m and the orebody is open to the S. The ore zone plunges at 10° to 20° to the SE and attains a maximum width of 250 m, although the individual ore surfaces are only 40 to 60 m wide. The orebody was initially accessed by a major crosscut from Lunnon 7L but is currently mined from a decline and has been developed to the 13L. Premined ore reserve: B. Production: D. Percent metal reserve—contact: 54; hanging wall: 27; offset: 19. Average Cu:Ni = 1:12; Co:Ni = 1:43. References: Sharpe (1978), Goss (1979).	Thickness of the basal unit associated with the ore environment is 30 to 50 m thinning to 10 to 15 m to the W. Both hydrated and carbonated assemblages are common. E of the contact ores a second thick (30 to 40 m) olivine peridotite unit with associated hanging-wall ore occurs and is dominantly antigorite bearing. To the E of this the lower member comprises a series of thin high magnesium flow units with numerous interflow sediments. The lower member is overlain by a series of peridotitic and picritic thin units displaying a general decreasing magnesium content up through the pile. The total ultramafic sequence is 200 to 300 m thick in the Hunt shoot environment. Hunt comprises two well-defined trough structures (A zone to the W; D zone to the E) which have been strongly modified by later thrusting and faulting. Thrusting appears to have been dominantly from the W and has displaced ore thought to occur previously as contact ore to the W of D zone into an offset hanging-wall position above the trough. NW-trending subvertical to steeply W-dipping faults offset the contact ore positions by varying amounts, with S block down-left lateral movements predominating.
FISHER Fisher shoot is a structurally complex ore zone located in a deep syncline on the SW flank of the dome. The orebody crops out and comprises two major linear ore belts that trend approximately NW-SE with variable plunge angles. The width of the complex is up to 400 m, but the individual ore zones seldom exceed 100 m. Major hanging-wall ores, both within the basal unit and at interflow positions, occur associated with the sourthern ore belt. The orebody is accessed by a decline and has been mined to the 8L. Premined ore reserve: B. Production: D. Percent metal reserve—contact: 71; hanging wall: 28; offset: 1. Average Cu:Ni = 1:15; Co:Ni = 1:49. Reference: Hancock (1978).	Owing to structural complexity it is difficult to determine accurately the true thickness of the basal unit and lower member. In places the basal unit is greater than 50 m thick and is generally composed of talc-magnesite-chlorite assemblages, although antigorite is locally important. Interflow sediments, generally of the pale cherty variety, are common within the lower member away from the ore environment. The upper member comprises poorly developed peridotitic and picritic thin units. The orebodies occur within the Fisher syncline, which plunges gently S and becomes more open in that direction. The synclinal structure is possibly related to the Otter syncline to the N. Overprinting the folding is complex thrusting and faulting with movements from the W and NW predominating.

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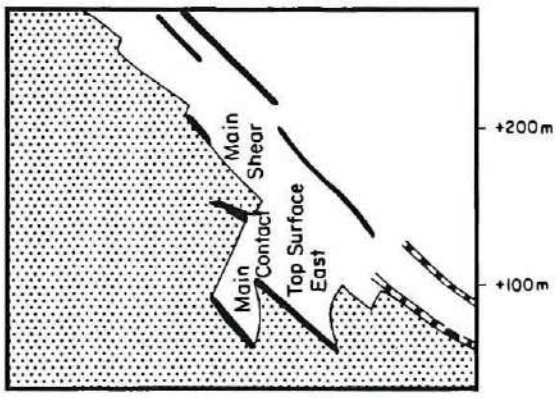
1756

Ore distribution, mineralogy, and sediment relationships

Typical cross-section

Contact ores display physical and chemical changes from N to S. To the N on Main contact massive sulfides predominate with matrix sulfides being more dominant on the southern Top Surface East contacts, with thin massive matrix-disseminated zones occurring on the Lunnon East contacts. Average grades of massive ore on Main contact are 7 to 8 percent Ni with 10 to 12 percent common on Lunnon East contacts. Metal concentrations are greater close to the Main Shear fault sytem. Hanging-wall ores are higher tenor with massive ore grades up to 20 percent Ni. Most hanging-wall ore is associated with the second flow unit, but minor ore occurs at the base of the third unit.

Contact sediments display a well-defined antithetic relationship to ore, with a section of barren contact occurring each side of the trough structure before sediments develop. The only contact ore associated with sediments occurs on the outside E contact, where massive ore occurs interlayered between two different types of sediment. Hanging-wall ores frequently display stratigraphic equivalence to sediments, and in some locations on the E margin of the main hanging-wall ore zones the ore and sediments interfinger.

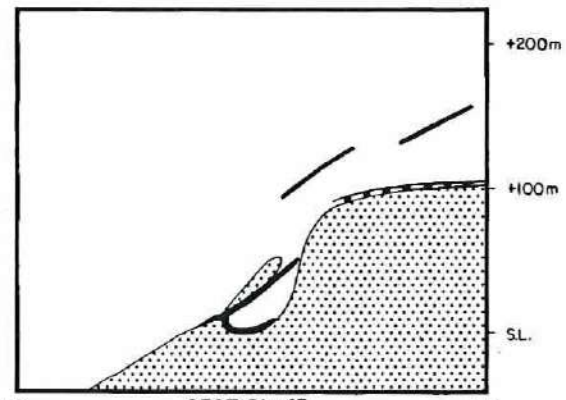


546 096 N

LUNNON

Contact ores occur within well-defined trough structures and hanging-wall ores at the base of the second flow unit. Strongly deformed offset ore also comprises a significant proportion of the ore reserve. Primary pn–po–py assemblages predominate, but Ni tenor and S/Ni ratios vary between all major ore surfaces. The hanging-wall ores have the highest S/Ni ratio (approximately 3.8, massive ores 7 to 10 percent Ni) with contact and offset ores displaying a general decrease in S/Ni ratios from W to E.

Contact sediments are absent from within the ore environment but are recorded on the overthrust contact of A zone and updip of D zone. The hanging-wall ores display stratigraphic equivalence to sediments, and ore-bearing sediments occur at the NE margin of the main hanging-wall zone.

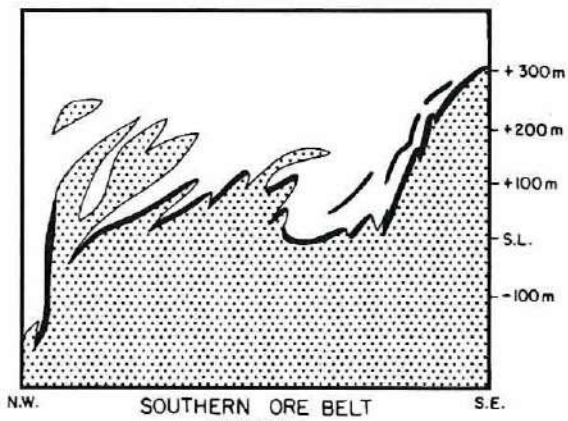


SECTION 43

HUNT

The bulk of the ore occurs in two NW-trending linear ore belts of different tenor. In the S ore belt contact ore is lower tenor with massive ores 8 to 12 percent Ni and S/Ni ratios 2.6 to 3.0. Hanging-wall ore occurs associated with this belt at unit boundaries, frequently overlying spinifex flow tops and displaying stratigraphic equivalence to sedimentary horizons. This ore is generally of higher tenor. Hanging-wall ore also occurs as blebs and disseminations within the basal unit with ml–mg–py assemblages and S/Ni ratios may be less than 1. The N ore belt comprises a series of dislocated ore surfaces generally of higher Ni tenor (massive ore 12 to 16 percent Ni) and with very little associated hanging-wall ore.

The two major belts of contact ore appear to be separated by and bounded to the N and S by sediment-bearing contact. Many of the hanging-wall ores at Fisher display stratigraphic equivalence to interflow sediments, and in places the sediments contain ore-grade nickel mineralization. Sedimentary ore is generally po rich with low Ni tenor.



SOUTHERN ORE BELT

FISHER

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TABLE 8.—(*Continued*)

Location, geometry, mining, and references	Stratigraphy and structure
KEN Ken shoot is located on the W flank of the dome close to the crossfold (F_3) axis. Most surfaces plunge to the N, but there is a plunge reversal at the S end of the shoot. The ores are predominantly contact and generally occur as small high-grade pods embayed in larger, geologically continuous surfaces. The lower trough structure is continuous over at least 1,600 m plunging at 10° to 15° on an azimuth of 340°. Mineralization within the trough appears to be patchy. Mineralization on the east flank appears to plunge at approximately 310°. The maximum width of the ore zone is 300 m. The orebody is mined via the McMahon decline and has been developed from 2L to 10L. Premined ore reserve: C. Production; D. Percent metal reserve—contact: 98; hanging wall: 2. Average Cu:Ni = 1:13; Co:Ni = 1:55. References: Reeve (1975a), Anderson (1977), Nevill (1980).	The olivine peridotite lower member, comprising talc-chlorite-magnesite assemblages and minor antigorite, is up to 50 m thick over the upper Ken main surface but thickens to 100 to 150 m over the lower trough, thinning again on the W flank. Poorly developed multiple thin units of lower magnesium ultramafic rocks form an upper member 150 to 200 m thick. Sediments occur within the lower member and on the contacts of both surfaces that flank the lower trough but are absent in the trough itself and from the ultramafic sequence immediately above the trough. The shoot can be divided into three structural domains. (1) The upper east flank which varies in dip from 70° to the W to subhorizontal in the upper levels. Within this surface there is a series of irregularly distributed depressions within which concentrations of high-grade ore occur. These high-grade pods vary in length from 20 to 150 m, are elongate in a NNW direction, and may be enclosed by basalt-basalt pinch-out on most of their margin. (2) The lower trough is of variable morphology and irregular mineralization. The structures confining the ore are poorly defined, although folding and low-angle thrusting have been recognized. (3) The west flank, from limited information, appears to be a relatively undisturbed sediment-bearing contact, with ore sporadically overlying sediments.
McMAHON McMahon shoot is located on the W flank of the dome 1 km N of Ken shoot. Nearly 70 percent of the ore occurs as hanging-wall ore, the remainder as contact ore on the E and W flanks of the northern extension of the Ken lower trough. The hanging-wall ore surface crops out and has a width and downdip extent of approximately 200 m. Plunges are poorly defined, but a 310° trend predominates. The hanging-wall ores are developed via the McMahon decline from the 2L to 6L. Premined ore reserve: D. Production: E. Percent metal reserve—contact: 27; hanging wall: 69; offset: 4. Average Cu:Ni = 1:11; Co:Ni = 1:40. References: Nevill and Randell (1975), Anderson (1977), Nevill (1980).	Total ultramafic thickness in the McMahon hanging-wall environment is 120 to 150 m. The lower member (talc-magnesite-chlorite) is up to 70 m thick and may comprise several units with interflow sediments. The total sequence and the lower member thicken in the Ken lower trough area. The hanging-wall ore zone occurs 20 to 30 m above the basalt contact, although this distance diminishes downdip and there is the possibility that the hanging-wall ore laps onto the contact. The ore occurs at the base of the second flow unit. The upper member of the ultramafic sequence consists of poorly defined peridotitic and picritic thin units. Generally there is a well-defined trend of diminishing magnesium up through the pile. The footwall basalt is strongly pillowed close to the contact. The hanging-wall ore and associated sediments display intense structural deformation. Four phases of folding have been tentatively identified. High-angle reverse faults offset the ore on the upper levels of the mine. The N extension of the Ken lower trough is a deeply incised embayment in the footwall basalt and may be a refolded fold.
GELLATLY Gellatly shoot is located on the NW corner of the dome. The main contact orebody is located in a narrow trough which varies in width from 50 to 100 m and extends from outcrop in the S at least 1,300 m downplunge on an azimuth of 340° at plunge angles of approximately 20°. The contact of the trough is patchily mineralized. In places a second parallel trough is developed. Minor low-grade hanging-wall ore occurs in various locations. The orebody is mined via the McMahon decline and has been developed to the 8L with mineralization being defined to the 11L. Premined ore reserve: E. Production: F. Percent metal reserve—contact: 82; hanging wall: 17; offset: 1. Average Cu:Ni = 1:13; Co:Ni = 1:44. References: Anderson (1977), Evans (1977a, b), Nevill (1980).	The trough is filled with olivine-peridotite (talc-magnesite-chlorite) units with no distinct boundaries, which in places are 120 to 130 m thick. Hanging-wall sediments occur above the troughs but generally toward the flanks. These sediments are of both the pale cherty and carbonaceous varieties and occur within moderate to high magnesium ultramafic rocks. Sediments are absent directly over the trough. Above the lower member are 60 to 100 m of strongly brecciated peridotites which are in turn overlain by weakly textured peridotites and picritic thin units. The basalt is strongly pillowed in places and the pillows appear to form definable stratigraphic horizons. The trough is considered to be a synclinal structure overprinting an original volcanic depression. The discontinuous paralleling troughs are also folds and the overhanging footwall basalt is possibly a synformal anticline. Within the trough, minor overthrusting from the W is seen from the 3L to 5L. The trend of the mineralization within the trough is due N.

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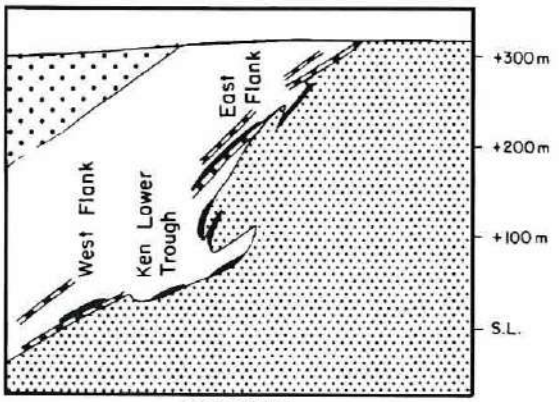
1758

Ore distribution, mineralogy, and sediment relationships

Typical cross-section

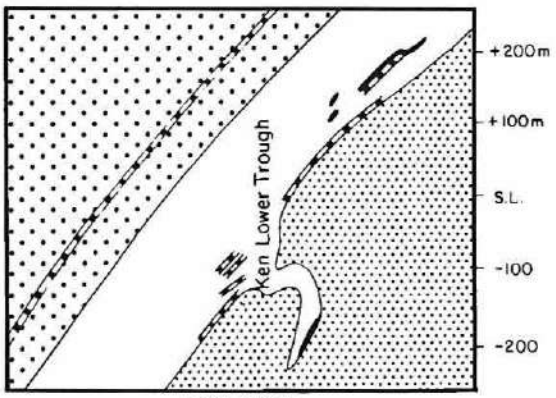
Typical contact ore is high-tenor, low sulfur (S/Ni = 2.2) with massive ores varying from 10 to 20 percent Ni. Lower tenor ores are associated with sediments in both contact and hanging-wall positions. Occurrences of ore as interpillow sulfides within the footwall basalt occur 10 to 15 m below the contact (massive sulfides 8 to 10 percent Ni).

Ore occurs intimately associated with sediments on sections of the upper east flank surface and on the 4 level low-tenor ore (8 percent Ni massive) occurs above, below, and within a sediment in a hanging-wall position which comes onto the contact at the 3 level. Associated sediments are predominantly black carbonaceous varieties with erratic distribution of As. On the west flank massive sulfides (8 percent Ni) occur above a pale cherty sediment, and in places the sediment carries 1.0 to 1.5 percent Ni.



549 083 N
KEN

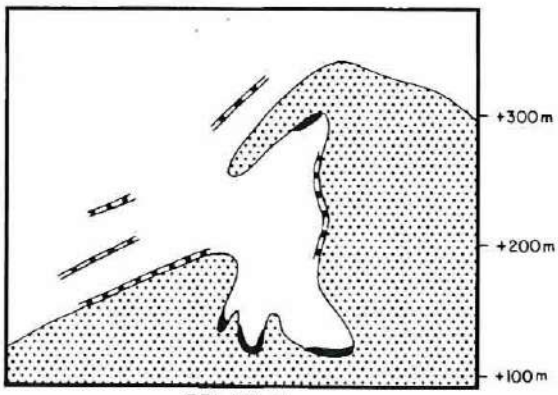
Strongly brecciated massive sulfides overlying sediments are common in the main hanging-wall positions. The ore contains fragments of both sediment and ultramafic and the massive ore grades 8 to 10 percent Ni. Overlying this are highly sheared talc-chlorite-dolomite rocks with minor disseminated sulfides. In places the sediments contain ore-grade Ni sulfides. Py–vl assemblages change to typical primary po–pn–py assemblages around the 4L. Arsenic content of these ores can be quite high. A second type of ore not associated with sediments is massive and matrix ore overlying high-magnesium ultramafic rocks. This forms small discontinous bodies and the massive ore grades 10 to 12 percent Ni. On the E flank of the trough massive ores grade 7 to 14 percent Ni, with the lower tenor material associated with sediments. Ore on the W flank of the trough is also associated with sediments.



550 318 N
McMAHON

Apart from the minor supergene ore above the 1L contact, ore is primary po–pn–py±cpy±mg. The contact ore is characterized by a thin (generally 30 cm or less) massive ore zone overlain by a slightly thicker zone of heavily disseminated to matrix ore. Massive ore grades 9 to 11 percent Ni with S/Ni ratios 2.5 to 3.5. Hanging-wall ores can be of high tenor (16 percent Ni massive), or low tenor where associated with sediments.

No sediments occur on the trough floor but contact sediments occur on the E wall of the trough and on the W flank. There is therefore a well defined zone of barren sediment-free contact between the ore zone and the contact sediments. Some of the hanging-wall ore is closely associated with and stratigraphically equivalent to the hanging-wall sediments.



551 156 N
GELLATLY

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TABLE 8.—(*Continued*)

Location, geometry, mining, and references	Stratigraphy and structure
WROTH Wroth shoot is a small hanging-wall orebody located in a trough structure approximately 100 m E of Gellatly and higher up the W flank of the dome. The trough trends at 340° and has a plunge length of at least 400 m. The plunge angle is approximately 40°, which is steeper than that of Gellatly. The orebody comprises three elliptical ore surfaces whose dips vary 20° to 85° W. The orebody is mined via the McMahon decline and has been developed on the 5 and 6 levels. Premined ore reserve: E. Production: F. Percent metal reserve—hanging wall: 100. Average Cu:Ni = 1:13; Co:Ni = 1:39. References: Evans (1977a), Nevill (1980).	The ultramafic within the trough is poorly defined but appears to comprise a series of flow units of moderate to high magnesium content. Textural evidence within flows indicates a W facing. Associated with much of the ore and almost enveloping it in places is a coarsely crystalline amphibole-chlorite rock. The Wroth trough has a similar gross morphology to the Gellatly trough, which it parallels. The E margin of the trough is formed by the fine-grained felsic intrusive that occupies the Otter thrust.
LORETO-GORDON Loreto and Gordon shoots are small ore positions located in the Otter syncline in the NW corner of the dome. Loreto comprises two hanging-wall positions and contact ore as well as ore occurring as interpillow sulfides. The upper hanging-wall mineralization crops out and occurs within a steeply southerly plunging zone over a plunge length of 150 m. The width of the mineralized zone is 50 m. Gordon, 1,400 m to the north of Loreto, is a small hanging-wall orebody in a stratigraphic and structural environment similar to Loreto. The ore zone is approximately 50 m wide, but plunge lengths and azimuths are uncertain. The upper Loreto hanging wall has been explored by a 34 m surface winze, and Gordon shoot has been explored via the McMahon decline. Premined ore reserve: E. Production: F. Percent metal reserve—contact: 23; hanging wall: 77. Average Cu:Ni = 1:13; Co:Ni = 1:44. References: Reeve (1975b), Nevill (1980).	The ultramafic sequence within the Otter syncline is predominantly picritic and peridotitic in composition, although there are some high magnesium flow units (both serpentinized and carbonated varieties) and in places numerous associated interflow sedimentary horizons. Individual flow units are well defined in places with good textural development particularly associated with the hanging-wall ore. The basalt is strongly pillowed. The Otter syncline is an overturned structure, the upper limb of which has been partially sheared out and is now occupied by a fine-grained felsic intrusive. At the S end of the syncline the structure is more complex, with possibly overturned folds creating unconnected basalt segments within the ultramafic.
FOSTER Foster shoot is a major ore position 5 km NW of Jan shoot on the SW flank of the St. Ives structure. It has a plunge length of 2,700 m at an azimuth of 130°. The shoot plunges to the SE at 15°, and the width is variable but seldom exceeds 250 m. It is entirely defined by surface drilling and will be mined via a decline and a haulage shaft. Premined ore reserve: A. Production: nil. Percent metal reserve—contact: 91; hanging wall: 9. Average Cu:Ni = 1:13; Co:Ni = 1:44. Reference: McDonald et al. (1976).	The total ultramafic thickness in the Foster area rarely exceeds 150 m in the ore environment and is less than 100 m on the flanks. Above the ore, most of the sequence consists of a few high magnesium flow units. On the flanks the basal unit appears to be thinner, and sediments occur at interflow positions. The contact with the hanging-wall basalt is commonly gradational with alternations of basaltic and ultramafic rocks. The orebody comprises three structural domains. To the NW the ore occurs within a deeply incised trough structure. The central portion is an open shallowly depressed contact, and the SE end is within a clearly defined trough structure of variable morphology. The relationship between these domains is not yet understood, but mineralization appears to be discontinuous.

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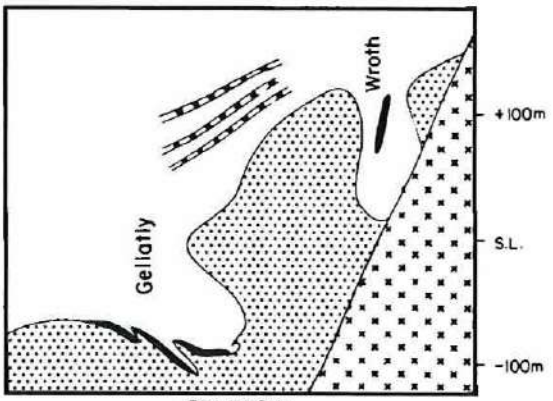
1760

Ore distribution, mineralogy, and sediment relationships

Typical cross-section

One surface abuts the basalt; the other two occur within the ultramafic sequence. The ore is very sulfur rich with massive ore grades 6 to 8 percent Ni and matrix sulfides 3 to 5 percent Ni. All ores are primary po–pn–py assemblages. The massive ore is occasionally strongly brecciated and contains fragments of ultramafic rock.

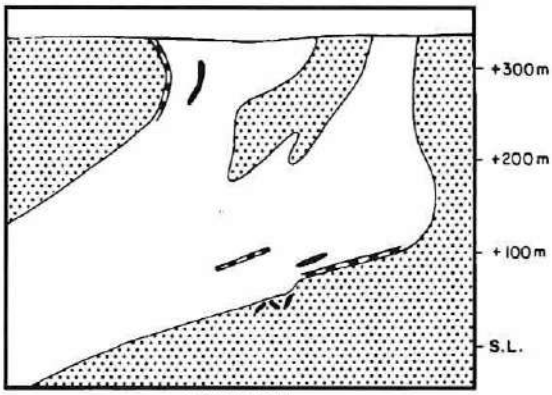
No sediments occur within the Wroth trough, but both contact and hanging-wall sediments occur between the Wroth and Gellatly environments.



551 520 N
WROTH

The upper hanging-wall ore at Loreto comprises remobilized supergene ore within an antigorite-bearing olivine peridotite. At Gordon primary po–pn–py occurs as irregular stringers and disseminations within talc-chlorite-dolomite host rocks. The ore in places is underlain by spinifex-textured rocks which indicate a facing. The lower hanging-wall ore is low tenor. Primary po–pn–py occurs between footwall basalt pillows up to 10 m below a barren ultramafic-basalt contact.

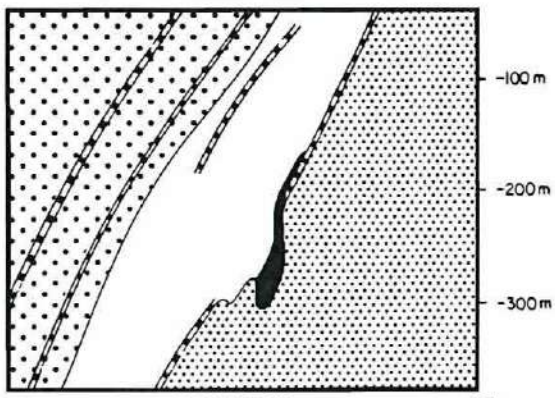
No sediments are associated with Gordon or the upper hanging-wall ore of Loreto. There are numerous hanging-wall sediments in the lower part of the syncline, and the lower hanging-wall horizon may be stratigraphically equivalent to one of these horizons.



549 480 N
LORETO

Most of the ore occurs as contact ore in well-defined trough structures, on open contact, or associated with sediments. Ni tenor of contact mineralization is variable downplunge and across the shoot. Massive ores on open and sediment-bearing contact are 8 to 10 percent Ni with even lower grade massive ores associated with sediments on the N margin. To the SE, where the major contact ore occurs within a well-defined trough structure, massive ores average 10 to 12 percent Ni.

Contact sediments occur on both flanks of the ore zone. On the N flank the ore is commonly seen to overlie the contact sediment, which is usually a complex of carbonaceous and pale cherty varieties. Only minor ore is seen to occur with sediments on the S flank. Hanging-wall ores are associated with and appear to be stratigraphically equivalent to sediments, and grade into sediments updip.



FOSTER

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TABLE 8.—(*Continued*)

Location, geometry, mining, and references	Stratigraphy and structure
JAN Jan shoot is located at the S end of the St. Ives structure 24 km SSE of Kambalda. It is a series of hanging-wall ore horizons, with minor contact ore in small structurally complex positions. The main hanging-wall ore surfaces crop out and plunge at 130° to 140°, have plunge lengths exceeding 500 m, and are open downplunge. Plunge angles average approximately 60°, and the maximum width of the ore zones is 150 m with dips varying 45° to 85°. The orebody is accessed by a vertical shaft to the 13L. Premined ore reserve: B. Production: D. Percent metal reserve—contact: 11; hanging wall: 89. Average Cu:Ni = 1:12; Co:Ni = 1:59. References: Hudson, G. (1973), McDonald et al. (1976), Duggan (1977a).	The lower member is 100 to 150 m thick and in the ore environment contains 3 to 4 units. Generally either ore or sediments occur at interflow positions. The sequence is entirely carbonated with coarse-grained talc-magnesite-chlorite rocks predominating. The upper member is composed of predominantly peridotitic thin units with minor picrites. Tight isoclinal folds with steeply E-dipping axial surfaces characterize the Jan area. The basalt appears to be the core of an anticlinorial structure which plunges very steeply SE to NE.
LANFRANCHI-EDWIN Lanfranchi and Edwin shoots are two subparallel contact orebodies located on the SE flank of the W Tramways dome, approximately 14 km S of Jan shoot. Both ore shoots crop out. Lanfranchi has a defined plunge length of 1,000 m and Edwin 600 m, and both have plunge angles of 25° to 30° and azimuths of 160°. Lanfranchi is a least 250 m wide in some sections, whereas Edwin seldom exceeds 50 m. Lanfranchi and Edwin are separated by 600 m and midway between them a small low-tenor ore position has been defined. Edwin is accessed by an inclined shaft to the 4L and is essentially mined out above this level. Lanfrachi is entirely defined by surface diamond and percussion drilling. Premined ore reserve: B. Production: F. Percent metal reserve—contact: 98; hanging wall: 2. Average Cu:Ni = 1:15; Co:Ni = 1:59. References: McDonald et al. (1976), Duggan (1977b).	The ultramafic succession is 400 to 450 m thick, and the olivine peridotite basal unit 40 to 50 m thick in the ore environment, thinning to almost zero between the two shoots. Numerous interflow sediments occur immediately above the basal unit, and the remainder of the sequence is thin peridotitic and picritic flow units. Carbonate assemblages predominate, although minor antigorite-bearing units occur in the upper member. Although Edwin occurs in a shallow depression in the basalt, ore-confining structures are poorly defined, with ore frequently grading to subgrade then barren contact. At Lanfranchi the N section of the orebody occurs on open contact, with little apparent structural disturbance. The central section is deformed by a major basalt overthrust or fold from the E, both upper and lower contacts being mineralized. To the S the contact again becomes open with no known ore-confining structures.
CRUICKSHANK Cruickshank shoot is a low-grade contact position located at the NE corner of the E Tramways dome approximately 13 km SE of Jan shoot. Located on the upper limb of a recumbent anticline, the orebody crops out and has a plunge extent of at least 700 m at an azimuth of 130° and a plunge angle of approximately 30°. Width of the ore zone varies 50 to 150 m. The orebody is defined entirely by surface diamond and percussion drilling. Premined ore reserve: D. Production: nil. Percent metal reserve—contact: 99; hanging wall: 1. Average Cu:Ni = 1:15; Co:Ni = 1:57. References: McDonald et al. (1976).	The ultramafic rocks are less than 100 m thick and dominated by poorly defined peridotitic and picritic units with minor high magnesium varieties. Carbonation is intense throughout. The lower portion of the ultramafic sequence containing the ore is a low magnesium quartz-dolomite-chlorite rock, commonly strongly compositionally banded. The contact with the hanging-wall basalt is gradational with interfingering of basalt and ultramafic rocks. The orebody occurs on the upper limb of a recumbent anticline that plunges SE at 35°. No obvious structures limit the margins of the ore, which apparently grades out to the N and S. A high-angle NW to SE reverse fault cuts the ore zone on the upper levels.

Abbreviations: as = arsenopyrite; cpy = chalcopyrite; mg = magnetite; ml = millerite; pn = pentlandite; po = pyrrhotite; py = pyrite; vl = violarite; L = level.

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Ore distribution, mineralogy, and sediment relationships

Nearly 90 percent of the ore occurs as hanging-wall ore and a large percentage of this occurs in two closely associated ore horizons, Jan Main and Hangingwall 1, separated by up to 15 m of barren ultramafic. Both horizons appear to be related to the third flow unit. Jan Main comprises varying thicknesses of massive, matrix, and disseminated ore, with Hangingwall 1 being predominantly disseminated ore. Massive ores grade 14 to 20 percent Ni with low S/Ni ratios (~2.0). Other hanging-wall horizons generally comprise low-grade disseminated ores, although irregular stringers occur in places. Contact ore positions are seldom wider than 50 m, and ores are generally of a lower tenor with massive ore grading 10 to 12 percent Ni.

The hanging-wall ores, particularly Jan Main, commonly grade laterally into nickeliferous sediments on both margins. These sediments can be either black carbonaceous or pale cherty varieties, and strike lengths of 20 m of ore-grade sediments have been developed. The sediment horizon overlying the first flow unit becomes ore grade downplunge. The complexly folded basalt is sediment covered in part, but a section of barren contact usually surrounds contact ore positions.

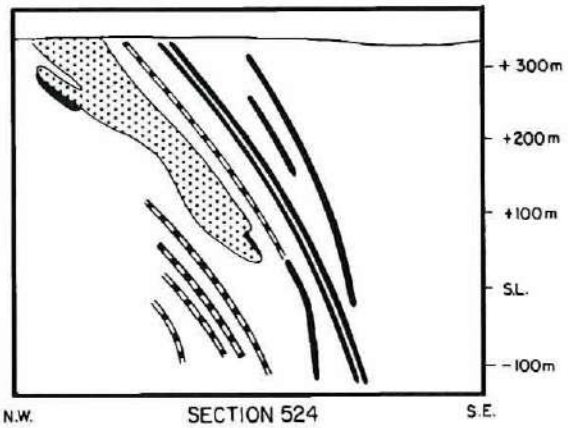
Almost all ore occurs as contact ore and comprises varying amounts of massive and matrix sulfides. At Edwin primary po–pn–py assemblages occur below 120 m, and massive ore grades 10 to 15 percent Ni with matrix ores 4 to 6 percent Ni. An intersection at the S end of the shoot contains a thin zone of pn–ml. Lanfranchi ore is of similar tenor to Edwin with massive ore grading 10 to 13 percent Ni.

Both ore positions display antithetic relationships to contact sediments, with well-defined zones of barren contact occurring at the margins of the orebodies before contact sediments are recorded. Interflow sediments are common and in many places directly overlie the ore zones, there being no clearly defined sediment-free prism above ore.

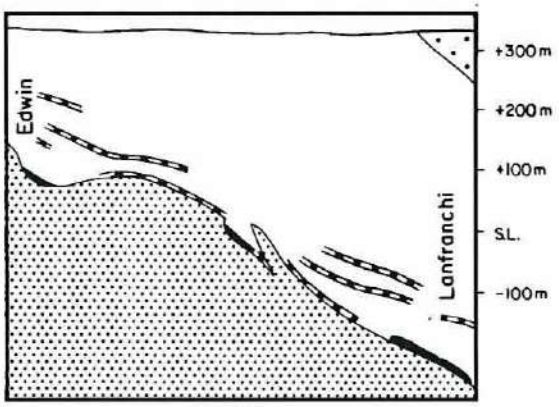
The entire orebody is composed of disseminated and, rarely, matrix sulfides, and the main ore zone is more than 5 m thick.

Contact sediments occur throughout the Cruickshank area, and the disseminated ores lie directly above them. The sediments themselves seldom carry nickel mineralization.

Typical cross-section

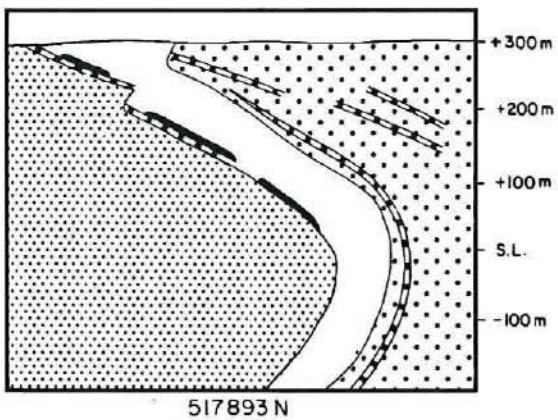


JAN



513 080 N

LANFRANCHI-EDWIN



517893 N

CRUICKSHANK

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Fisher shoots where separate high and low tenor linear orebodies occur within the same shoot (Marston and Kay, 1980; Hancock, 1978). Otter, Durkin, Gibb, Victor, and Ken shoots can be classified as high tenor orebodies, whereas Long, Lunnon, Hunt and Gellatly shoots are low tenor ones. Foster shoot is a major low tenor orebody, whereas other shoots in the St. Ives and Tramways areas are of higher tenor.

At Hunt shoot the two major low tenor contact ore positions are seen to have significantly different S/Ni ratios. Within the Lunnon environment nickel grades of massive ores appear to increase to the south within the Lunnon trough, representing a progressive drop in S/Ni ratio (Middleton, 1980). Detailed sampling by Harley (1980) on the 13-level Juan West high tenor surface indicated significant variations in massive ore grades across the surface, with the highest grades (lower S/Ni values) being recorded in the central section.

Hanging-wall ores generally display quite different mineralogical characteristics from associated contact ores. Lunnon, Fisher, and Jan hanging-wall ores have significantly lower S/Ni ratios than their associated contact ores, whereas the Hunt hanging-wall ore has an average S/Ni ratio of 3.8, which is higher than the associated contact ores. McMahon and Wroth hanging-wall ores are also both high sulfur ores.

Ore Shoots

The nickel ore reserve of the Kambalda nickel field occurs within 24 ore shoots and some additional poorly defined isolated bodies of mineralization. In turn, the shoots contain more than 350 ore surfaces, which are geologically distinct, essentially continuous zones of ore-grade mineralization. Each shoot is different in size, morphology, structure, stratigraphic relationships, mineralization, and ore-sediment relationships. The significant features of the major ore shoots are summarized in Table 8, in which a typical cross section of each shoot is presented to highlight the morphology, ore distribution, and ore-sediment relationships. Although it is difficult to categorize the shoots, the following groupings emphasize gross similarities.

Ore shoots of the Kambalda dome

At Juan, Otter, Durkin, Gibb, Long, and Victor shoots (Fig. 2), the ore is predominantly contact ore with well-defined structural control; the only hanging-wall ores are blebby ore zones within the basal unit and there is no stratiform hanging-wall ore. There are clearly defined antithetic ore-sediment relationships both on ultramafic-basalt contacts and within ultramafic sequences above ore zones. The only ores associated with sediments occur at Juan. Arsenic values of the ores are extremely low (generally less than 20 ppm).

Contact orebodies with associated stratiform hanging-wall ores include Lunnon, Hunt, and Fisher shoots (Fig. 2). Contact ores display clear structural control and antithetic ore-sediment relationships, apart from minor sediment-associated mineralization at Lunnon shoot. Significant tonnages of ore occur as hanging-wall ore at the base of the second high magnesium flow unit, and at their margins these ores are commonly associated with sediments. Arsenic values of contact ores are generally less than 20 ppm, although values in hanging-wall ores are more erratic, especially toward their margins.

Orebodies with more variable ore types occur on the northwest flank of the Kambalda dome and include Ken, McMahon, Gellatly, Wroth, Gordon, and Loreto shoots (Fig. 2). Ken shoot is predominantly a contact orebody, but some of the ores are associated with sediments. McMahon shoot is essentially a hanging-wall ore position, the greater proportion of which is associated with sediments and with some very high arsenic concentrations. Gellatly shoot is primarily a contact ore position but has weakly developed stratiform hanging-wall ore. Wroth shoot is composed entirely of hanging-wall ore, although strong structural modification has masked original relationships. Gordon and Loreto shoots are small, complex ore positions, and the greater proportion of their ore reserve is contained in strongly remobilized hanging-wall ore high in the ultramafic succession.

Ore shoots of the St. Ives area

Foster shoot (Fig. 4) is a major contact orebody that is largely similar to the contact orebodies of the Kambalda dome, but in the central section the contact ore is not strongly confined by structures and laps onto marginal contact sediments. There are appreciable tonnages of hanging-wall ore, both within the basal unit and at the base of the second flow. Arsenic values are generally less than 20 ppm. Predominantly hanging-wall orebodies occur at Jan and East Cooee shoots (Fig. 4). These ore positions are dominated by stratiform hanging-wall ores with erratic arsenic values and are commonly associated with sediments at their margins. Small, structurally confined contact ore surfaces also occur.

Ore shoots of the Tramways area

Contact orebodies occur at Lanfranchi and Edwin shoots (Fig. 5). These are mainly contact ores with varying degrees of structural confinement and arsenic values of less than 20 ppm. There are clearly defined antithetic relationships between ore and contact sed-

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iments, but some hanging-wall sediments overlap the ore zone. Cruickshank and John shoots (Fig. 5) both comprise thick zones of disseminated sulfides which at Cruickshank shoot occur in a low magnesium, quartz-rich rock in contact with barren contact sediments, and at John shoot in high magnesium ultramafic rocks 9 to 10 m above the base of the basal ultramafic unit, in part overlapping a contact sediment.

Orebodies associated with the Bluebush sequence

Two definable ore shoots and several other minor, isolated bodies of mineralization have been delineated, but none has been mined. Cameron and Stockwell shoots are both small ore positions containing less than 5,000 t of nickel metal. The ore at Stockwell shoot occurs on an open, steeply dipping basalt contact, whereas at Cameron shoot it occurs both as contact ore overlying a mafic intrusive and sediment, and as hanging-wall ore.

Summary Criteria for Ore vs. Non-Ore Environments

Although the orebodies and associated host rocks have undergone pronounced metamorphic, metasomatic, and structural modification, there are structural, stratigraphic, and lithological features within the Kambalda sequence which characterize ore environments. These features are displayed within prisms of rock that are elongate along the length of a shoot and extend from the immediate footwall upward through the ultramafic sequence stratigraphically above an ore shoot. The features are shown schematically in Figure 6 and can be recognized in the cross sections of Figure 3. The clearest indicators to an ore environment are those spatially closest to the ore, such as the structural form and nature of the footwall basalt-ultramafic contact and the thickness and nature of the basal ultramafic unit. However, not all environments with the features discussed below contain economic concentrations of nickel sulfides, and not all ore environments include all of the features.

Structural complexity and the nature of the footwall basalt-ultramafic contact

Trough structures, whatever their origin, are closely correlated with nickel sulfide mineralization. In unmineralized areas, the ultramafic-basalt contact appears to be a relatively undisturbed and planar, sediment-covered surface.

The ultramafic footwall basalt contact can be classified into one of the following categories: (1) ore-bearing (defined at Kambalda as greater than 1 percent Ni over a specified minimum mining thickness), (2) subgrade (i.e., less than 1 percent Ni but greater than 0.35 percent Ni), (3) barren (i.e., less than 0.35 percent Ni and essentially devoid of sulfides), (4) sediment-bearing, and (5) obscured by intrusive rocks.

As the contact ores almost always occur within troughs within the footwall basalt, the cutoff from ore-bearing to barren contact is generally abrupt, with the subgrade zone only marginally enlarging the areas of the orebodies. However, most ore zones are surrounded by a zone of completely barren contact (Figs. 3 and 9). On the barren contacts, 5 to 30 cm of almost pure chlorite commonly occurs, possibly representing a metasomatic reaction zone. Farther from the contact ores, cherty or carbonaceous sediments develop on what is generally a planar contact. In some instances sediments may occur to the edge of the major confining structures—as, for example, with the reverse thrusts defining the eastern limit to Lunnon shoot (Fig. 9) and the western limit to the A zone of Hunt shoot (Fig. 3)—but it is extremely rare to find contact sediments within the troughs.

This antithetic ore-sediment relationship is best displayed by the major contact orebodies (e.g., Long, Victor, Durkin, and Juan shoots); the smaller orebodies occur in a broader, more diffuse zone of sediment-free contact. Overall, approximately 60 to 70 percent of the ultramafic-basalt contact at the Kambalda dome is sediment bearing, and the majority of sediment-free contact areas contain ore.

Nature and thickness of the basal unit

The thickness of the basal unit of the ultramafic sequence varies from a few meters to more than 100 m and, apart from rare preservation of olivine, comprises antigorite or talc-magnesite assemblages. In most cases the basal unit contains a spinifex-textured flow top, which varies in thickness from the very thin (1 to 2 m) flow top associated with the thick (50 to 100 m) basal unit in the Victor shoot area up to a possible maximum in the 25-m-thick picrite at Lunnon shoot. An example of a basal unit associated with ore is shown in Figure 7A.

It has been clearly established that the orebodies are associated with thick (greater than 30 m) basal flow units. Thicknesses of up to 150 m have been recorded at Juan shoot (Marston and Kay, 1980), more than 100 m at Victor and Foster shoots, and 50 to 70 m at Durkin and Lunnon shoots. In areas such as Lunnon, Victor, and parts of Juan shoots, the basal unit appears to be a separate unit of the ultramafic sequence which is largely confined to the trough structure. At the well-documented Lunnon shoot, the basal unit displays an overall thinning from north (50 to 70 m) to south (30 to 40 m) associated with

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diminishing intensity of mineralization. Conversely, at Victor shoot, although only defined by surface diamond drilling, the basal unit appears to thicken to the south in the direction of diminishing nickel sulfide mineralization. Thus there is no direct correlation between the size or grade of an orebody and basal unit thickness. Remote from ore environments, basal units are commonly only 10 to 15 m thick, can apparently be correlated over considerable distances (see Fig. 3, east of Hunt shoot), and maintain a relatively constant thickness.

The thick ore-associated basal units are almost always olivine peridotites with 40 to 45 percent MgO (volatile free) and represent extremely olivine rich flows. They are low in Cr and Zn and rich in Ni relative to basal units not associated with ore. Another feature of the ore-associated basal units, although not entirely restricted to them, is the development of sulfide and silicate blebs. The silicate blebs are preserved as ovoid clots of chlorite, amphibole, or talc and/or carbonate and are commonly associated with sulfide blebs of millerite or pentlandite with minor pyrrhotite that in places constitute ore (e.g., at Durkin, Otter, Gibb, and Fisher shoots).

Nature and thickness of the lower member

The lower member is defined as the contiguous sequence of flow units of similar bulk composition—usually olivine peridotite—at the base of the ultramafic rocks. In some places it may comprise only one unit; in others, a series of units with or without intercalated sediments. In general, lower member thickness trends reflect those of the basal unit, with the thickest development of the lower member being associated with ore occurrences. In ore environments, the lower member generally consists of 2 to 4 units and may exceed 250 m in thickness (e.g., Victor, Ken, and Fisher shoots), whereas in nonore environments it may comprise up to six units and is generally less than 100 m thick.

Sediment distribution within the ultramafic rocks

Above almost all major orebodies at Kambalda there is a sediment free prism of ultramafic rocks (Fig. 3), although in some instances there is partial overlap of sediments across the ore zone. Remote from ore the lower member may contain up to six interflow horizons of sediment, but their number diminishes toward the ore (Fig. 3). The change from areas of multiple sedimentary horizons to an ore environment may be abrupt.

Ordering of the upper member of the ultramafic rocks

The upper member is dominated by thin flow units, and less commonly contains thicker units of picritic and peridotitic composition and rare thicker olivine peridotite units. The distribution of these units differs between ore and nonore environments. Remote from ore, particularly on the west flank of the Kambalda dome, the upper member displays a regular trend of diminishing magnesium content upward through the pile. Overlying the lower member is a series of thin peridotitic flow units followed by a series of thin picritic flow units. Reversals of the trend to lower magnesium unit compositions upward through the pile are rare, and there are few thicker flow units. Above major ore positions the upper member is much less ordered. There is no distinct magnesium trend within the pile, peridotitic and multiple thin picritic unit sequences appear to interfinger, and correlation between even closely spaced drill holes is rare. In some places a series of thick (10 to 30 m) peridotitic flow units lies above the lower member and below the lowermost thin units. Thick olivine peridotitic units may occur high in the sequence, and thick picritic sequences—sometimes strongly textured, and apparently single flows—are also recorded.

Textural and compositional development within thin flow units of the upper member

The textural development and compositional differentiation within individual flow units has been well documented at Kambalda (Ross and Hopkins, 1975) and elsewhere (Arndt et al., 1977). Individual thin flow units at Kambalda display some or all of the following features: (1) flow top breccia; (2) upper chilled, jointed zone; (3) spinifex textures, both random and sheaf; (4) bladed, skeletal, or tabular textures; (5) a distinct break between the upper spinifex-textured portion of the unit (usually picritic in composition), and the basal portion (usually peridotitic) which commonly displays cumulate texture; (6) the highest magnesium content of the flow unit generally occurring just below the central section of the cumulate zone; and (7) diminishing magnesium content toward the base of the unit with a basal chilled zone.

Flow units associated with ore generally display better development of these textural features and a strong degree of compositional differentiation, and the contact between the textured flow top and cumulate basal section is well defined. The unit from drill hole KD 8391A (Fig. 7B) is an excellent example of a well-developed thin flow unit. Conversely, units remote from ore commonly display weak compositional differentiation and poorly developed textures.

Origin of the Nickel Ores

Kambalda is considered to be the type example of Archean stratabound volcanic peridotite-associated nickel sulfide deposits. Groves and Hudson (1981)

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have reviewed the genetic models for these deposits, and Table 9 combines their summary with two additional models. Ross (1974), Barrett et al. (1977), and Marston and Kay (1980) have demonstrated that the Kambalda ores were present prior to metamorphism, and there is abundant evidence that they were present prior to deformation. Ross and Hopkins (1975) postulated a volcanic origin for the ores at Lunnon shoot and elsewhere at Kambalda; and the following field evidence supports such an origin.

1. The ores are associated with ultramafic rocks that, based on field and petrological evidence, are considered to have a volcanic origin. There is a consistent pattern of ore distribution within the lower member of the ultramafic rocks, and an association of ore with the high magnesium ultramafic rocks.

2. Ore zones occupy prisms of stratigraphic discontinuity. Although some of this discontinuity is due to later deformation, careful removal of this deformation clearly reveals an original discontinuous stratigraphy (Hancock, 1978; Marston and Kay, 1980; Middleton, 1980; Ross and Hopkins, 1975). This spatial coincidence of ore zones with original stratigraphic discontinuities in the lava pile, and in particular the association with high-magnesian ultramafic rocks, is strong support for an association between ore and volcanism.

3. There is interfingering at the margins of some hanging-wall ores with interflow sedimentary layers.

4. Massive sulfides occur between pillows in the footwall basalt at Ken, Loreto, Juan, and Lunnon shoots. These occurrences are generally beneath contact ore zones and appear to be primary, implying either that sulfide melt was present during the last stages of basalt extrusion or that interpillow interstices were filled by sulfide liquid percolating down from initial ultramafic flows.

Groves et al. (1976), Naldrett (1979), and Bavinton (1979) have argued cogently that most of the sulfides were magmatic, not exhalative. The total evidence therefore supports a genetic model involving immiscible sulfide liquids associated with ultramafic lava flows.

Interpretation of ore zones

The ore-confining troughs are interpreted as original linear depressions, probably in part fault bounded, which developed along lines of eruptive fissures in the basaltic ocean floor. These depressions are analogous in shape and dimensions to structures present on modern mid-oceanic ridges, where vents occur in linear belts 200 to 300 m wide over several kilometers (Spiess et al., 1980). Some depressions (at Ken, Juan East) are elliptical rather than linear but are considered to be primary volcanic features. The same explanation appears likely for the deep trough underlain by homoclinal sediments at the northwestern end of Foster shoot. Contact ores within troughs are associated with the first ultramafic lava flow at that point, and contact ores outside troughs may be associated with overflows from the troughs onto the flanks, separate simultaneous flows, or later flows which may override the trough flow(s) and emplace hanging-wall ore above the trough contact ore. The stacking of hanging-wall ore surfaces above contact ore surfaces is probably due to troughs being reformed by movement on bounding structures immediately prior to further extrusion. Studying Lunnon shoot, Ross (1974) and Middleton (1980) have proposed the continued reactivation of trough structures during emplacement of the lowest ultramafic units; they based their conclusion on the observation of diminishing amounts of displacement on these structures upward through the ultramafic rocks. The thickening in the vicinity of the ore zone of both the lower member and the basal unit, especially where the latter is confined within a trough, is regarded as a natural consequence of proximity to the extrusion site, and the process was enhanced by the higher viscosity of phenocryst-rich lavas. The lack of sediments within, above, and adjacent to many ore environments would result from both positive topography along the line of eruption (Bavinton, 1979) and convective turbulence in the overlying seawater (Ross and Hopkins, 1975), although theoretical evidence presented by Bavinton (1979) indicated that the latter effect was not a significant factor in inhibiting sediment development.

Similarly, relative disorder in the ultramafic sequence above an ore zone could be due to an irregular and constantly changing topography which caused ponding and channeling of lava flows, whereas farther from the vent, relatively flat terrain would allow more uniform lava distribution and more regular interflow sedimentary interlayering. Flows farthest from the vent would be cooler and flowing more slowly, which would explain the poorer textural development and compositional differentiation in those areas compared to flows in proximity to the ore zone (Ross, 1974; Arndt et al., 1977).

The more obvious the discontinuity within the ore zone prisms, the more likely it is that they represent original extrusion sites. It is more difficult to define possible extrusion sites for those orebodies where development of the discontinuity is less obvious, particularly where they are entirely in hanging-wall positions and/or lying on sediments.

Interpretation of ore types

Most massive ore is interpreted to have been originally emplaced as a sulfide melt that crystallized to

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TABLE 9. Summary of Genetic Models of the Formation of Volcanic Peridotite-Associated Ni-Fe-Cu Deposits

Volcanic models	References
Ores formed by flow and gravity separation of immiscible oxysulfide liquids from komatiitic lava flows	Ewers and Hudson (1972); Hudson, D. R. (1972; 1973); Naldrett (1973); Naldrett and Cabri (1976); Green (1978); Groves et al. (1978)
Separation of oxysulfide liquid before, or during, eruption	Ross and Hopkins (1975); Groves et al. (1979); Muir and Comba (1979); McQueen (1979); Marston and Kay (1980)
Gravity separation of magmatic sulfide liquid in situ	Naldrett (1973, 1979); Usselman et al. (1979)
Massive ores formed by volcanic exhalations; disseminated ores by incomplete melting of massive ores by the komatiitic lava extruded over them	Lusk (1976)
Intrusive models	
Ores formed during multiple intrusions of ultramafic sills concomitant with deformation	
Separation of magmatic sulfides	O'Driscoll (1971a and b)
Sulfur derived from contamination of the melt by sulfidic sediments; massive ores formed by reaction with other sulfidic sediment wall rocks; disseminated ores by segregation within the melt	Hopwood (1981)
Metamorphic models	
Ores formed by sulfurization or metal diffusion	
Sulfur released during alteration or metamorphism reacted with Fe, Ni, and Cu in ultramafic units to produce ores	Naldrett (1966, 1973); Prider (1970)
Ni and Cu migrated down high chemical potential gradients from Ni-rich ultramafic rocks to S-rich sediments during metamorphism	Seccombe et al. (1978); McQueen (1979)
Ores formed by metamorphic or tectonic concentration of preexisting, disseminated magmatic sulfides	Woodall and Travis (1969); Hudson, D. R. (1973); Barrett et al. (1976, 1977); Binns et al. (1977); Groves et al. (1979); McCall (1972); McQueen (1979); O'Driscoll (1971a)

form a layer of sulfide with only minor gangue. These sulfides have undergone substantial remobilization, both along contacts, sometimes forming breccia ore, and into offset positions. Disseminated and matrix ores are considered to have consisted originally of sulfides dispersed in silicate rock, or vice versa, although all textures have been modified by metamorphism.

The genetic relationship between massive ores and disseminated and matrix ores has been a contentious issue (Hudson, 1972; Ewers and Hudson, 1972; Ross, 1974; Ross and Hopkins, 1975; Barrett et al., 1977). Groves et al. (1979) have argued against the single stage-settling model—the "billiard-ball" model of Naldrett (1973) and Usselman et al. (1979)—and further evidence against this model is provided in the chemistry of the silicate gangue fraction of the matrix and disseminated ores. Ross and Hopkins (1975) and R. A. Binns et al. (written commun.) provide chemical data indicating that this gangue material has a composition similar to the overlying basal unit, particularly with respect to Al_2O_3 and CaO contents. In contrast, if silicate liquid had been displaced by sulfides in the matrix ore, as suggested in the billiard-ball model, a composition approaching that of olivine would be expected. These data imply that some segregation of the sulfide-silicate phases took place during extrusion-emplacement of the flows. There are marked viscosity differences between the components of a mixture of molten sulfide, olivine crystals, and picritic liquid. Ross (1974) and Ross and Hopkins (1975) considered that segregation of these components took place during vertical flow in the volcanic conduit, with the result that sulfide melt preceded a mixed section of sulfide and silicate melt, and this was followed by an olivine-rich section. Marston and Kay (1980), on the basis of the close spatial and chemical affinities of the massive and disseminated ore at Juan, proposed that the peridotitic magma and sulfide liquid were both erupted at the same time and that segregation took place during transport on the ocean floor, with the main mass of sulfide liquid being em-

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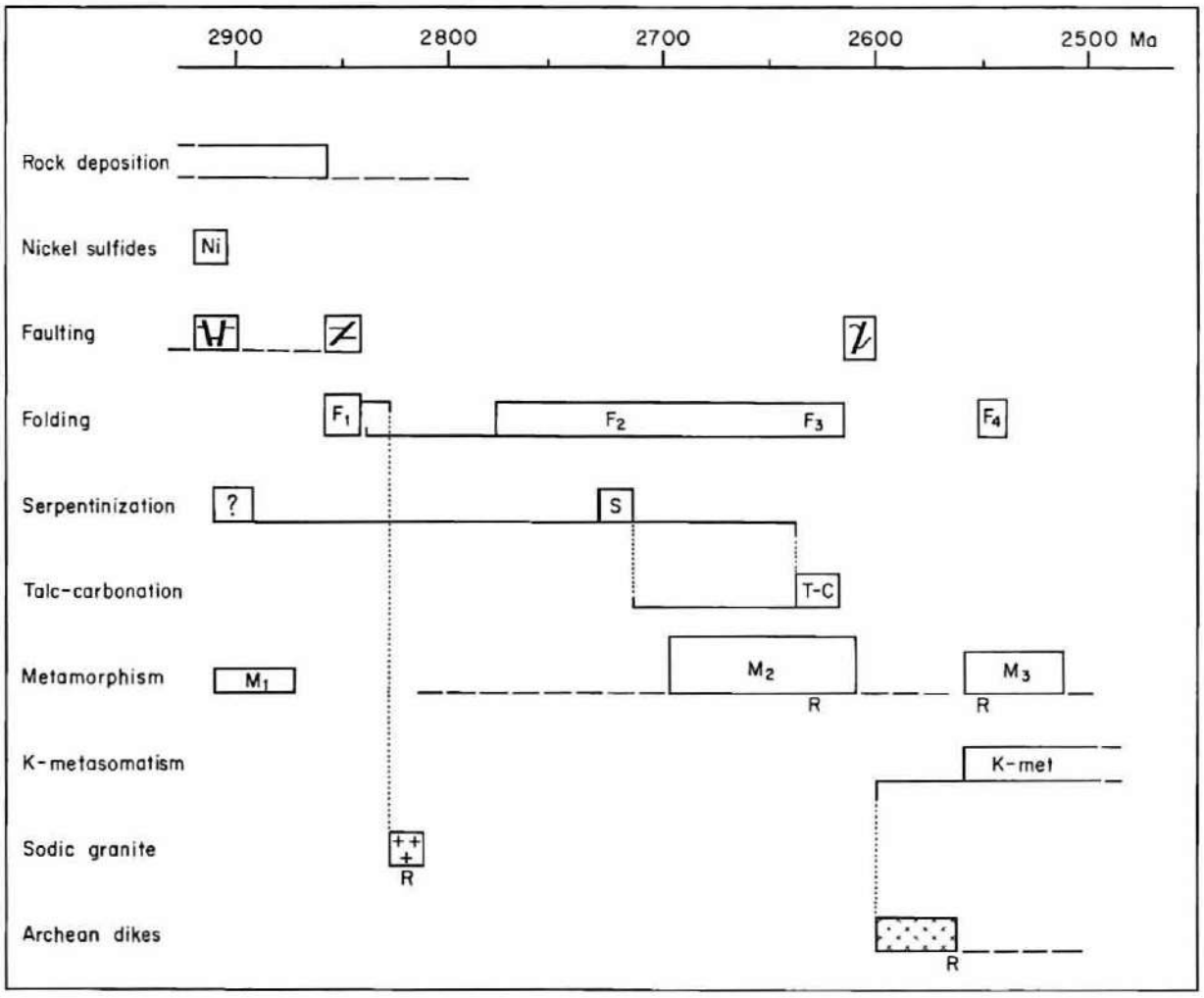


FIG. 10. Interpreted Archean geologic history of the Kambalda area. The boxes indicate the probable timing of geologic events; the solid lines = the time span within which the event could have occurred; and the dashed lines = time spans over which events may have been continuous. R = event interpreted to be recorded by a radiometric date. All other age correlations are hypothetical.

placed slightly in advance of olivine-rich magma containing sulfide droplets.

We conclude that there is insufficient evidence to rule out either of these models, and it is probable that varying degrees of segregation took place during both vertical and horizontal transport. Detailed studies of the silicate gangue of the disseminated and matrix ores and the chemical characteristics of the lower sections of the basal unit would be required to determine which model predominated.

Blebby ores are regarded as residual sulfide droplets which did not settle, although Willett (1975) considered they could be amygdaloidal sulfides. Spinifex ores are of uncertain origin; they could represent replacement of the original matrix by sulfides from the overlying ore, or percolation of the original sulfide melt into a honeycomb of spinifex plates which had somehow been drained of its liquid matrix. Sedimentary ores are interpreted to result from either remobilization of Ni sulfides into the sedimentary beds or, less commonly, syngenetic precipitation during original sediment deposition (Bavinton, 1981). Basalt breccia ores, like interpillow sulfides, are considered to be a primary volcanic feature.

Problems

One difficulty with the described volcanic model is the lack of feeder dikes of ultramafic rock in the ore environment, within either the footwall basalt or the ultramafic pile. Few feeders have been identified with any of the Archean lavas in the Yilgarn Block; however, they have been identified at similar deposits

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at Langmuir (Green, 1978) and in Rhodesia (Williams, 1979). The feeders beneath troughs were possibly obliterated by repeated reactivation of the associated faults and later felsic-intermediate dike intrusion.

A second enigma is the variable Fe/Ni and S/Ni ratios of the sulfide fraction of the ores between adjacent contact shoots and between contact ore and overlying hanging-wall ore. The distribution of precious metals in the ores and their host rocks (Ross and Keays, 1979; Keays et al., 1981) indicates that these variations are probably original magmatic differences. Woolrich et al. (1981) suggest oxidation prior to, or during, extrusion to account for the variation in nickel tenor between shoots, while Keays et al. (1981) suggest that variation in f_{S_2} and f_{O_2} of the original magmas may have caused these differences. However, some modification by later metamorphic events cannot be precluded, because much of the pyrite in the ore is thought to have formed as a result of a late-stage, relatively low-temperature addition of sulfur to the ores (Seccombe et al., 1981).

A third problem is that many of the structures that confine ore are approximately parallel to the dominant north-northwest fold axes and faults, implying a genetic relationship. We suggest that these structures paralleled the major structures controlling the original volcanism and that some of the later deformation was localized by original inhomogeneities such as the troughs.

Geologic History

Although we have assigned emplacement of the nickel sulfides to the original volcanic episode, there remain considerable uncertainties in interpreting the geologic history of the Kambalda area and correlating it with the history suggested for adjacent areas. The major difficulty concerns the sodic granite at the Kambalda dome, which truncates the deformed ore-bearing ultramafic footwall basalt contact and shows little evidence of having been severely deformed. However, the best date available, 2,820 ± 15 m.y. (U-Pb zircon results) is significantly earlier than the 2,700 to 2,550 m.y. radiometric ages assigned to metamorphism at widely separated localities in the southern Yilgarn Block (Archibald et al., 1978; Compston, 1980). Total rock Rb-Sr results for Kambalda basalts give an age of 2,615 ± 30 m.y., which is thought to be close to the metamorphic peak and is similar to the 2,610 ± 25 m.y. age derived by Turek (1966) in Kalgoorlie. Rb-Sr biotite ages from the basalts and felsic porphyries give an age of 2,550 ± 10 m.y. and probably reflect cooling to approximately 300°C after the second metamorphic peak. At Kambalda, all workers have interpreted metamorphism and major deformation to be essentially simultaneous events, but if we assume the granite date is correct, either the "metamorphic" dates are recording some other event, or metamorphism and major deformation were not simultaneous.

Ross (1974), who believed that neither the granite nor the dikes that cut it were metamorphosed, resolved the problem by assigning the "metamorphic" dates at Kambalda to late K-metasomatism, and he placed both metamorphism and major deformation prior to granite emplacement. These conclusions were based on the field relationships of alteration envelopes of the dikes to the ultramafic host rocks and the mineral assemblages within the intrusives. Barrett et al. (1977) and Marston and Kay (1980) regarded the dikes as being metamorphosed but did not relate their geologic histories to geochronology. Groves and Hudson (1981) assign deformation of the Western Australian nickel deposits in general to a 2,700- to 2,600-m.y. time interval of metamorphism, which also ignores the problem. Our preferred explanation is that not all deformation events accompanied the metamorphism, an interpretation already derived from the area west of Kambalda by Archibald et al. (1978). They surmised that D_1 produced no microfabrics and involved horizontal tectonics—precisely the deformation needed at Kambalda to be consistent with the structures known to be pregranite. This interpretation is consistent with the fact that the early deformation has not been obvious in stope scale observations; it is also consistent with some recent observations of recumbent pre-D_2 structures in McMahon shoot (T. Richards, pers. commun., 1981). Under this interpretation, the Kambalda granite provides the opportunity to date D_1 at Kambalda as older than 2,820 ± 15 m.y. A difficulty with this interpretation is that some basalt ultramafic wall-rock alteration has been shown to precede contact deformation (Barrett et al., 1977; Marston and Kay, 1980). This can be resolved by assigning this alteration to the original ocean-floor metamorphism, a possibility previously considered by Ross (1974).

The sequence of Archean geological events at Kambalda based on this interpretation is listed below and is shown in Figure 10.

1. Emplacement of the Kambalda sequence, including the nickel sulfides, controlled by major faults such as the Lefroy fault and accompanied by minor faulting. This was immediately followed by ocean-floor metamorphic effects (M_1) and possibly serpentinization. Further volcanic and sedimentary activity occurred.

2. Deformation (D_1) and development of F_1 recumbent folds and thrusting.

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3. Intrusion of the sodic granite. Archibald et al. (1978) have suggested that this is the subvolcanic expression of the felsic volcanics overlying the Kambalda sequence.

4. Possible deposition of the Merougil, Lake Dam, and Triangle Island beds in a separate, late basin.

5. Commencement of north-northwest folding (F_2), followed by the "first" peak of metamorphism as defined by Barrett et al. (1977), here called M_2, and talc-carbonate alteration.

6. East-northeast–northeast folding (F_3) forming domal structures, followed by major north-northwest-trending faults such as the Otter thrust and Juan fault, although the latter may have experienced previous movement associated with granite emplacement and the rotation of Durkin shoot into its unusual trend, and with F_2. The late movement on the Lefroy fault interpreted to be responsible for the faulting, steepening, and overturning of the eastern end of the Kambalda dome, St. Ives, and Tramways structures may have occurred at this time.

7. Emplacement of mafic, intermediate, and felsic intrusives.

8. The onset of the "second" phase of metamorphism, here called M_3.

9. Dislocation of metamorphosed intrusives and folding (F_4).

10. Possible continued dike emplacement.

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APPENDIX

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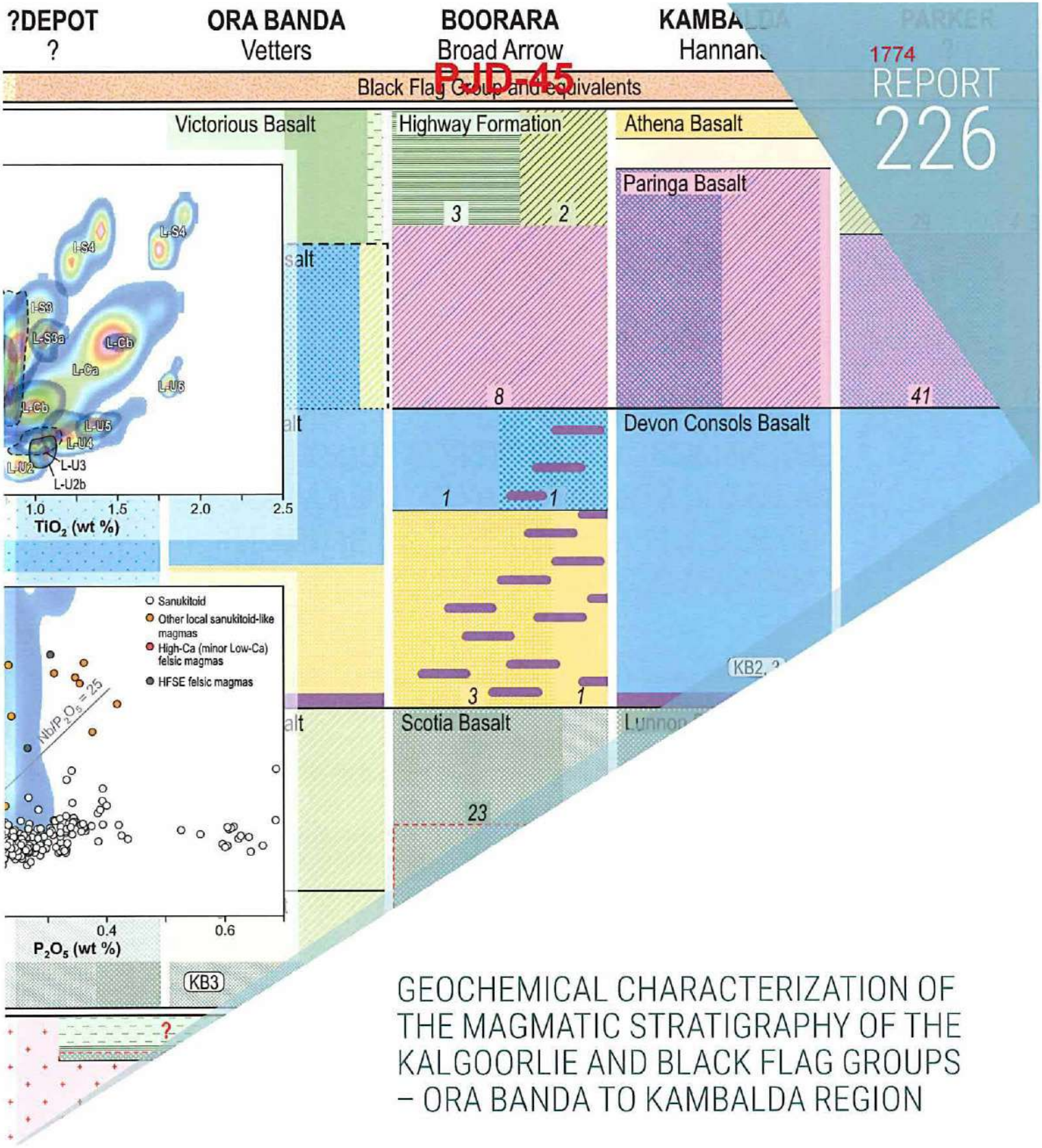
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GEOCHEMICAL CHARACTERIZATION OF THE MAGMATIC STRATIGRAPHY OF THE KALGOORLIE AND BLACK FLAG GROUPS – ORA BANDA TO KAMBALDA REGION

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Geochemical characterization of the magmatic stratigraphy of the Kalgoorlie and Black Flag Groups – Ora Banda to Kambalda region

RH Smithies, JR Lowrey, J Sapkota, MC De Paoli, P Hayman[1], SJ Barnes[2], DC Champion[3], Q Masurel[4], N Thébaud[4], LL Grech, M Drummond and R Maas[5]

Summary

A new dataset of ~2800 high-quality whole-rock geochemical analyses of subvolcanic, volcanic, and volcaniclastic rocks from the Kalgoorlie Group is used to construct chemostratigraphic barcodes within the broader Ora Banda – Kambalda region, encompassing most of the previously defined domains of the Kalgoorlie Terrane as well as the Bulong complex area of the Kurnalpi Terrane. Twenty-six distinct mafic compositional units are identified based on combinations of major and trace element variation diagrams, and in particular, on bivariate plots utilizing strongly incompatible trace elements. These distinctions highlight the controls that various fundamental igneous processes and source compositional variations exerted on evolving magma compositions. There are broad trends in compositional change common to the chemostratigraphy in most areas throughout the region. As described in numerous previous studies, the three commonly identified regional stratigraphic levels (basaltic lower, komatiitic/basaltic middle, and basaltic upper) correspond to changes in tectonomagmatic processes, and in particular, a general increase in the degree of mantle partial melting, an increase in the amount and/or effect of crustal contamination, and an increase in overall magmatic compositional diversity.

According to the new barcode data, individual established lithostratigraphic units, typically of formation level, seldom comprise a single geochemical unit. Most stratigraphic formations comprise at least two distinct geochemical units. Most of these units occur at multiple stratigraphic levels, and in most regional stratigraphic columns. Thus, the individual greenstone stratigraphic formations comprise overlapping flow fields, each representing the products of a discrete eruptive event and/or eruptive centres that tapped genetically unrelated magma sources. This potentially makes some stratigraphic formation boundaries somewhat arbitrary reflections of changing source eruptions. Although complex, this diversity ensures that, in most cases, the position of stratigraphically unknown geochemical samples can be uniquely established as long as enough samples are taken over a continuous stratigraphic interval that also incorporates the overlying and underlying units.

However, there are rare cases where local mafic geochemical associations appear anomalous with respect to the surrounding associations. For example, in the Kundana region to the west of the Zuleika Shear Zone and in an area close to Londonderry Siding, a stratigraphy characteristic of the Ora Banda Domain, rather than of the Coolgardie Domain, is intersected in drillcore and in outcrop. The reasons for this remain elusive.

Because of the greater continuity of low-viscosity flows that characterize its basalt-dominated stratigraphy, the application of chemostratigraphic barcoding is more reliable or appropriate for the Kalgoorlie Group than for the overlying Black Flag Group. Geochemical characteristics and relationships within the felsic-dominated Black Flag Group, nevertheless, provide broad stratigraphic information and significant insight into crustal architecture and regional geological evolution.

The regional distribution of the mafic chemostratigraphic associations is distinctly asymmetric, in many cases ignoring previously established domain boundaries. However, a broadly north-trending structure incorporating segments of (from south to north) the Zuleika Shear Zone, Abattoir Fault, and the Bardoc Shear Zone (ZAB) does appear to separate the 'western domains' (Coolgardie and Ora Banda) from the 'eastern domains' (Kambalda, Boorara, Parker, and the Bulong complex). These domain groups have distinctive chemostratigraphic variations that probably reflect a fundamental crustal architectural control and potentially provide some basis for reassessing the present domain boundaries. In particular, the mafic units in the western domains probably reflect melting of a more depleted source in thicker lithosphere. The Ora Banda Domain is clearly a part of the western domains throughout evolution of the lower and middle greenstone stratigraphy, but it also appears transitional to the eastern domains in terms of its diversity of fractionated layered sill-derived magmatism in its upper stratigraphy. The chemostratigraphic differences between the western and eastern domains extend to the felsic units of the Black Flag Group. The subvolcanic, volcanic, and volcaniclastic rocks of the Black Flag Group have the composition of sanukitoids, which are likely derived from a metasomatically enriched lithospheric mantle source. They show a wide range in La/Th ratios, which broadly correlates with crystallization age, source composition, and location. Low La/Th sanukitoid is typically older (>c. 2678 Ma), derived from a source with less radiogenic Nd (i.e. is more crustal) and is more common west of the ZAB, whereas high La/Th sanukitoid is typically younger (<c. 2675 Ma), has more mantle-like isotopic compositions and is virtually restricted to east of the ZAB. The simplest explanation for the regional compositional variations in the mafic and felsic rocks is that the ZAB is the exposed trace of a structure separating compositionally contrasting translithospheric domains. Northern and southern extensions of that trace should be entirely identifiable based on geochemical and isotopic data.

Keywords: Crustal evolution, geochemistry, greenstone, Neoarchean, stratigraphy, Yilgarn Craton

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Introduction

This Report discusses the use of a large amount of new, high-quality, geochemical data from drillcore and outcrop samples collected from the Eastern Goldfields Superterrane in the Archean Yilgarn Craton of Western Australia, to establish unique lithogeochemical characteristics (barcodes) applicable to local- to regional-scale greenstone stratigraphy. Although large regional lithogeochemical datasets can be extremely useful in identifying petrogenetic processes (e.g. Barnes et al., 2021; Halley, 2021), they typically include analyses of a range of individual lithological units that have no geological reasons to be genetically related. Inferring stratigraphic or lithological correlations between such geochemical data without significant additional geological constraints is fraught with danger. However, by generating a substantial amount of data on specific rock units within a more geologically and geographically constrained area, and taking into account their stratigraphic context, chemically distinctive groups that relate to lithostratigraphy can be identified.

There are numerous benefits in employing a geochemical approach to stratigraphy within an established geological context. This is particularly the case in many Archean greenstone sequences, where the typically basalt-dominated stratigraphy offers very few mineralogical or textural clues that can be used to uniquely identify a specific unit. This problem is significantly compounded in regions such as the Yilgarn Craton, where typically poor outcrop and locally high structural complexity severely limit the geological context required to confidently assign stratigraphy.

The barcodes we produced, are based on the better-known and sampled stratigraphic sections and provide templates against which geochemical data sourced from other parts of a specific greenstone belt can be evaluated for possible lithostratigraphic or chemostratigraphic equivalence. Some of the established stratigraphic units we examined are geochemically unique. In these cases, inferring the stratigraphic position of geochemically similar 'unknowns' within that general region can be done with a high degree of confidence. More typically, individual lithostratigraphic units have a composition or compositional range similar to several other units from the same or other stratigraphic intervals, resulting in non-unique stratigraphic allocation options. In these cases, a single or low number of analyses will not uniquely identify a lithostratigraphic position. However, more comprehensive sampling, particularly if it incorporates the underlying and overlying stratigraphic units, almost certainly will.

The geochemical barcoding approach is particularly applicable to basalt-dominated stratigraphy, since low-viscosity mafic (and ultramafic) lavas can travel long distances and blanket the underlying surface. This may particularly be the case where magmas rapidly erupted on to subdued topography, as suggested for mafic flows of at least the lower parts of the stratigraphy in the region between Ora Banda and Kambalda (Morris, 1993). However, the approach is less effective in the case of felsic volcanic and volcaniclastic accumulations, although broad but significant compositional variations can still be observed and utilized.

It is hoped that the geochemical barcodes that we have constructed will provide a useful stratigraphic tool from the mine site to the regional scale. An additional, substantial benefit is perhaps the most comprehensive, regional, high-quality, and (analytically) internally consistent, major and trace element geochemical dataset from any Archean greenstone terrane globally.

The Ora Banda – Kambalda region (Fig. 1) provides the ideal choice for an initial attempt at establishing a greenstone chemostratigraphy. This region is extremely well-studied geologically and the focus of many geochemical studies. Perhaps the most detailed study is by Morris (1993), and many of its conclusions are confirmed and repeated here. The study region lies mainly within the Kalgoorlie Terrane but crosses the boundary with the Kurnalpi Terrane to the east (Fig. 1). The present stratigraphic interpretations show that each domain of the Kalgoorlie Terrane has its own, slightly different, greenstone lithostratigraphic variation (Fig. 2) within a broadly similar framework based on the classic Kalgoorlie–Kambalda stratigraphy of the Kambalda Domain. The latter comprises basalt (Lunnon Basalt)-komatiite (Kambalda Komatiite)-basalt (Devon Consols Basalt)-basalt (Paringa Basalt) magmatic stratigraphy of the 2720–2690 Ma Hannans Subgroup overlain by the 2690–2665 Ma felsic volcaniclastic-dominated Black Flag Group.

Within the study region, the geochemical barcoding project has so far produced ~2300 new high-quality geochemical samples of greenstones from drillcore and outcrop, to supplement the ~500 existing, publicly available high-quality analyses. We largely restrict our analysis to data from volcanic rocks covering the mafic to felsic compositional range (SiO_2 from ~45–77 wt% anhydrous) and MgO up to 18 wt% (except for some partial cumulate and cumulate layers that have MgO up to 25 wt%). We have omitted any interpretation of komatiite compositional variation, although our data sampling regime has routinely included ultramafic rocks; those data are presented in Appendix 1. Likewise, we have routinely sampled mafic dykes and mineralogically layered sills within the stratigraphy (data presented in Appendix 1); however, we only refer to these data where appropriate, in terms of comparisons with basaltic stratigraphy. A detailed account of these mafic intrusions is given by Hayman et al., (2021).

We compare our chemostratigraphy against the current lithostratigraphic interpretations and assess our results in the light of existing domain boundaries, and more recent studies suggesting alternative geographical arrangements of lithostratigraphically coherent blocks (e.g. Hayman et al., 2015). We also discuss compositional changes within and between regional stratigraphic successions that could add to our understanding of greenstone-sequence evolution. In addition, we assess what mafic and felsic magma evolutionary trends might indicate in terms of lithospheric architectural evolution and the identification of those transcrustal to translithospheric structures that played the most fundamental roles in fluid and melt transfer.

A short summary of the more functional aspects of the chemostratigraphy and its potential field application, without the accompanying geochemical, petrological or geodynamic discussions or implications, is provided as Appendix 6.

Regional geological introduction

The Yilgarn Craton has been subdivided into seven terranes based on distinct stratigraphic characteristics, geochemistry, and age (Fig. 1). The western part includes the Narryer, Youanmi and South West Terranes. The eastern part of the craton includes the Kalgoorlie, Kurnalpi, Burtville and Yamarna Terranes, which together comprise the Eastern Goldfields Superterrane (Cassidy et al., 2006; Pawley et al., 2012). The terrane boundaries in the eastern half of the craton are widely believed to correlate with large-scale shear zones active during late Archean tectonic events. Geophysical data show at least some of them to be east-dipping listric structures that penetrate the deep crust (Drummond et al., 2000; Goleby et al., 2006; Blewett et al., 2010).

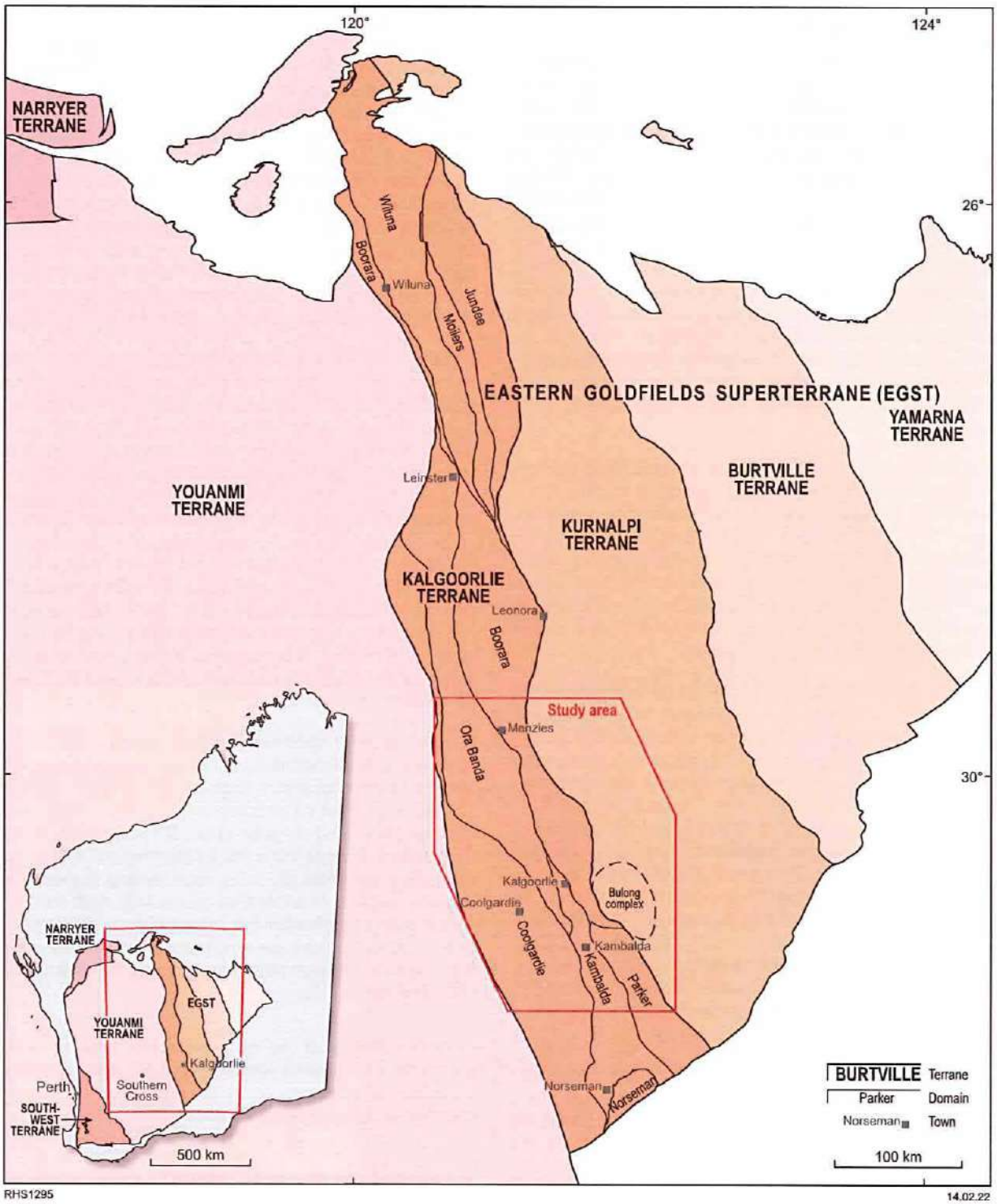


Figure 1. Regional map of the Eastern Goldfields region of the Yilgarn Craton showing the outline of the various proposed terrane and domain boundaries (from Cassidy et al., 2006), and the location of the study area, mainly within the Kalgoorlie Terrane

The Kalgoorlie Terrane includes several domains (Fig. 1) representing structurally bound regions each showing unique variations within an otherwise broadly common stratigraphic succession and structural history (Swager et al., 1990). These include the Coolgardie, Bullabulling, Kambalda, Parker, Ora Banda and Boorara Domains. Cassidy et al., (2006) subdivide the Coolgardie Domain into the Depot Domain, separating the remainder of the Coolgardie Domain, to the west, from the Kambalda and Ora Banda Domains to the southeast and northeast, respectively. The study region incorporates parts of all domains of the Kalgoorlie Terrane, except Bullabulling Domain. To the east, the Kalgoorlie Terrane is separated from the Kurnalpi Terrane by the Mount Monger Fault. Within the Kurnalpi Terrane, the western part of the Bulong Domain, comprising the Bulong complex, is also included within the study region. In general, although the Bulong Domain shows significant stratigraphic, structural, and geochronological differences with the Kalgoorlie Terrane, it has been suggested that the area around the Bulong Anticline (i.e. the Bulong complex) shows closer similarities with the Kalgoorlie Terrane (e.g. Nelson, 1997; Hollis et al., 2017).

The Kambalda Domain contains the classic Kalgoorlie–Kambalda stratigraphy developed and modified by Woodall (1965), Gresham and Loftus-Hills (1981), and Roberts (1988), which was subsequently used as the template for greenstone stratigraphy throughout the eastern Yilgarn Craton. Current GSWA nomenclature includes the lower (2720–2690 Ma) mafic–ultramafic package of this model stratigraphy in the Hannans Subgroup (Kalgoorlie Group) (Fig. 2), with the basal Lunnon Basalt overlying unknown basement. In turn, this is overlain by the Kambalda Komatiite, Devon Consols Basalt, Kapai Slate, and the Paringa Basalt (Fig. 2). A detailed description of the magmatic stratigraphy, its lithological variations, physical volcanological features, and geochemical variations has previously been provided by Morris (1993). The Hannans Subgroup is conformably to locally unconformably overlain by the Black Flag Group (2690–2665 Ma), which mainly comprises felsic volcanic and volcaniclastic rocks. These felsic rocks and their stratigraphy have been studied in detail by Tripp (2013, 2019). All parts of this stratigraphy have been extensively intruded by mafic- or mafic–ultramafic sills and granitic rocks.

Variations in detailed stratigraphy across major structures and between greenstone belts has formed a basis for the recognition of domains. However, the significance of the various domains, at least as presently defined – if not, the concept in general – is increasingly questioned (e.g. Hayman et al., 2015) as our understanding of the stratigraphy and structural evolution increases. The recent GSWA Eastern Goldfields Superterrane seamless geological mapping program attempted to avoid domain-based lithostratigraphic nomenclature altogether. According to the current GSWA nomenclature (Sapkota, 2020), the Kalgoorlie Group comprises most of the mafic–ultramafic package in greenstone belts between Norseman and Leinster (Figs 1 and 2) and is overlain by the felsic volcanosedimentary succession of the Black Flag Group or its regional equivalent. The various rock successions that form the Kalgoorlie Group, although linked in terms of broadly similar stratigraphy and age range, are not physically continuous throughout the Kalgoorlie Terrane. However, it is probable that they are products of the same geological events and at least some successions were originally deposited in distinct depocentres.

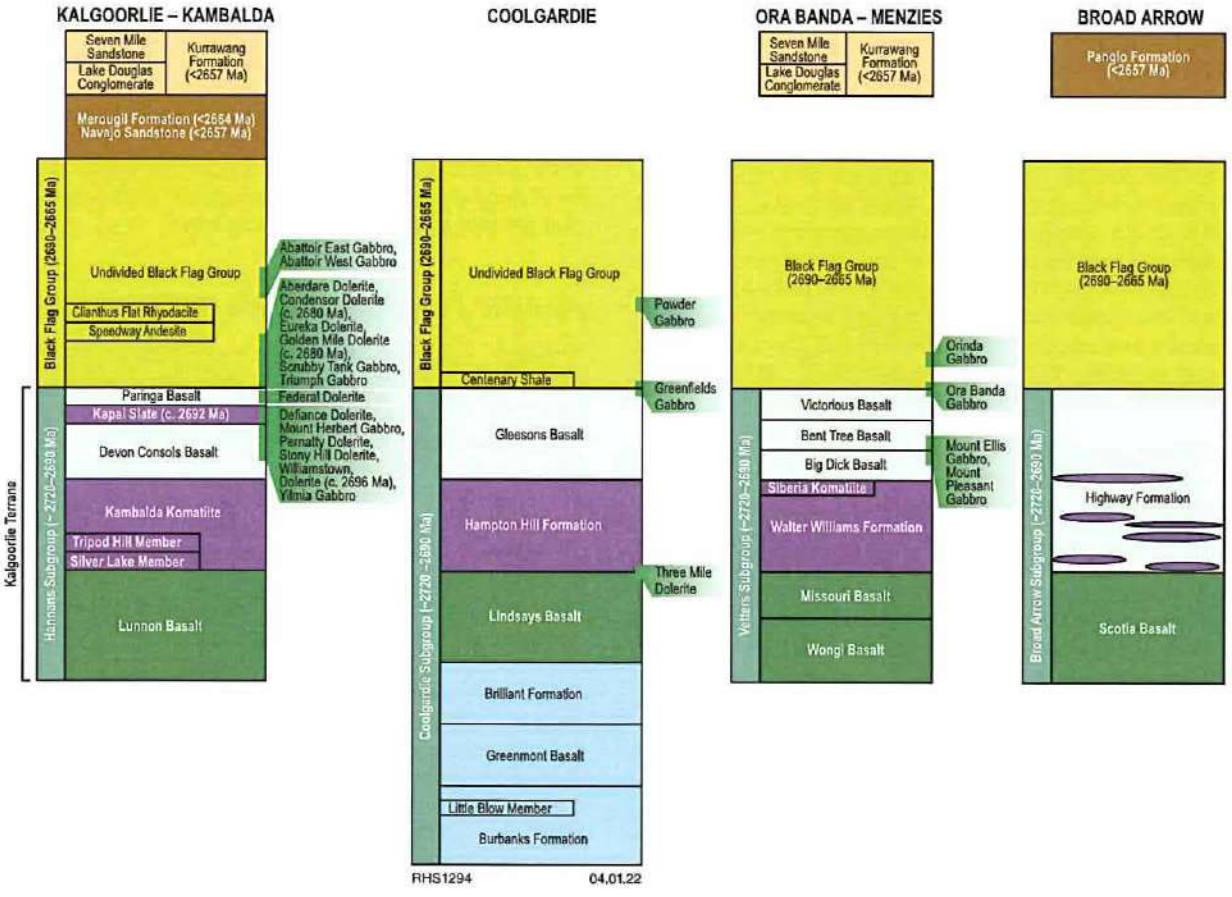


Figure 2. Present stratigraphic interpretations of the central part of the Kalgoorlie Terrane (Sapkota, 2020)

Where these structurally or geographically isolated sequences form stratigraphically continuous successions of the Kalgoorlie Group, they have each been distinguished as a distinct subgroup (Sapkota, 2020; Fig. 2). However, this new interpretation uses much the same dataset used to establish the original domains and the spatial distribution of the subgroups does not differ markedly from that of the original domains. Hence:

- In the Kalgoorlie–Kambalda region, this part of the stratigraphy is assigned to the Hannans Subgroup, and lies entirely within the Kambalda Domain
- Greenstone successions around and to the north and west of Coolgardie are placed in the Coolgardie Subgroup, which is entirely within the Coolgardie Domain. However, isolated greenstone outcrops between the Coolgardie Subgroup and the Hannans Subgroup are currently placed within the Coolgardie or the Hannans Subgroups or have been left stratigraphically unassigned
- Greenstones in the Ora Banda region and those to the north, in the area west of Menzies, are placed in the Vetters Subgroup, which lies entirely within the Ora Banda Domain
- Greenstones in the area immediately east of Broad Arrow and Menzies are placed in the Broad Arrow Subgroup, which lies entirely within the Boorara Domain.

Hayman et al., (2015) integrated detailed stratigraphic logging with previously available and new whole-rock geochemistry and suggested modifications to the structural arrangement apparently separating stratigraphically coherent blocks throughout much the same region studied in this Report. This interpretation identifies an enriched basaltic component contributing to the lower basaltic units in the Coolgardie and Ora Banda Domains that appears absent from domains further east (Kambalda, Boorara, Parker). If valid, this interpretation has significant implications in terms of regional stratigraphic correlations at least as far north as the Agnew region.

Additionally, basement greenstone stratigraphy older than 2730 Ma and in some cases, older than 2750 Ma, has been confirmed stratigraphically or structurally beneath several successions of the Kalgoorlie Group (e.g. Hill et al., 1992; Hayman et al., 2015; Jones et al., 2020). These include greenstone sequences and felsic units with close geochronological similarities with units of the Youanmi Terrane to the west and the Burtville Terrane to the east. This has led to the suggestion that the Eastern Goldfields Superterrane was continuously underlain by basement of that age, and formed an incomplete rift of the Youanmi Terrane crust at c. 2710 Ma (Pawley et al., 2012, Smithies et al., 2018a, Mole et al., 2019).

There has been debate about whether the complex basalt-komatiite stratigraphy underlying the Lindsays Basalt in the Coolgardie Subgroup (Fig. 2) is a thrust repetition of the Lindsays Basalt and overlying Hampton Hill Formation or an older (lower) and hitherto undated sequence (perhaps one of the >2730 Ma basement remnants). This lower basaltic sequence is portrayed in most existing stratigraphic columns to include a lower basaltic Burbanks Formation overlain by the Greenmont Basalt and capped by ultramafic flows and sills of the Brilliant Formation, and is currently assigned to the Coolgardie Subgroup.

In this study, we use domain names as convenient, familiar, regional geographic markers, and subgroup names in reference to the regional stratigraphy. Our sampling of the Coolgardie Subgroup included units from below, within, and above the Lindsays Basalt and does not support the suggestion of major stratigraphic repetition (see later). Where a distinction is required, the terms 'lower Coolgardie succession' and 'upper Coolgardie succession' are used to denote the stratigraphy below and above the boundary between the Brilliant Formation and Lindsays Basalt, respectively.

Sample selection and analytical techniques

Dataset content

From an initial dataset of 2843 whole-rock major and trace element analyses, we used a subset of ~2100 analyses covering nearly all the supracrustal igneous (or meta-igneous) lithologies, including volcaniclastic rocks [omitting komatiites (n = 279), dolerites (n = 243), and altered felsic rocks (n = 184); analyses of altered mafic rocks are not reported]. The initial dataset incorporates 2272 new analyses (see analytical techniques below) performed specifically for this project, of which 1733 are from diamond drillcores (Appendix 2) and the remainder from outcrop samples. For samples that were not specifically analysed (or re-analysed in the case of many samples from Morris, 1993) for this project, only analyses that incorporated a wide range of trace element determinations by ICP-MS [including the full suite of rare earth elements (REE), Nb and Th], were included in our dataset. New GSWA data, and older ICP-MS data from GSWA and Geoscience Australia are presented in Appendix 1 and their distribution is shown in Figure 3. Appendix 1 includes suitable ICP-MS data from literature sources but does not include company data or other otherwise unpublished data. Table 1 shows the number of samples collected within each domain. Relatively low sample numbers from the Boorara Domain (n = 219), Parker Domain (178), and the Bulong complex (n = 244) mean chemostratigraphic interpretations from these regions are significantly less robust than those from the regions to the west. Sampling in the Parker Domain was mainly limited to several drillholes to the west of Logans Find on the northeast edge of Lake Lefroy, hence, the regional stratigraphic context was limited. In the case of the Boorara Domain, the majority of the samples are from the stratigraphically lower regions of the Broad Arrow Group (Scotia Basalt) and should reflect a reasonable representation of that part of the stratigraphy.

Analytical procedure for new GSWA samples

All samples specifically collected for the greenstone geochemical barcoding project were analysed at BV Minerals, Canning Vale, Perth. The drillcore and surface samples were visibly inspected and any weathering or excessive vein material was removed. Each sample was crushed in-house or by BV Minerals in a plate jaw crusher and low-Cr steel mill to produce a pulp with a nominal particle size of 90% <75 µm. A representative pulp aliquot was analysed for 13 elements as major components, ignition loss, and 54 elements as trace elements (ppm or ppb). The major elements were determined by X-ray fluorescence spectrometry on a fused glass disc. A fragment of each disc was then laser ablated and analysed by ICP-MS for 51 of the 54 minor elements. Gold, Pd, and Pt were analysed on a separate pulp aliquot by lead-collection fire assay and ICP-MS. Data quality was monitored by blind insertion of sample duplicates (a second pulp aliquot), GSWA internal reference materials, and the certified reference material OREAS 24b (www.ore.com.au). BV Minerals also included duplicate samples (including OREAS 24b), variably certified reference materials, and blanks. The accuracy and precision was assessed using data from 17 analyses of OREAS 24b, determined during the analysis of greenstones.

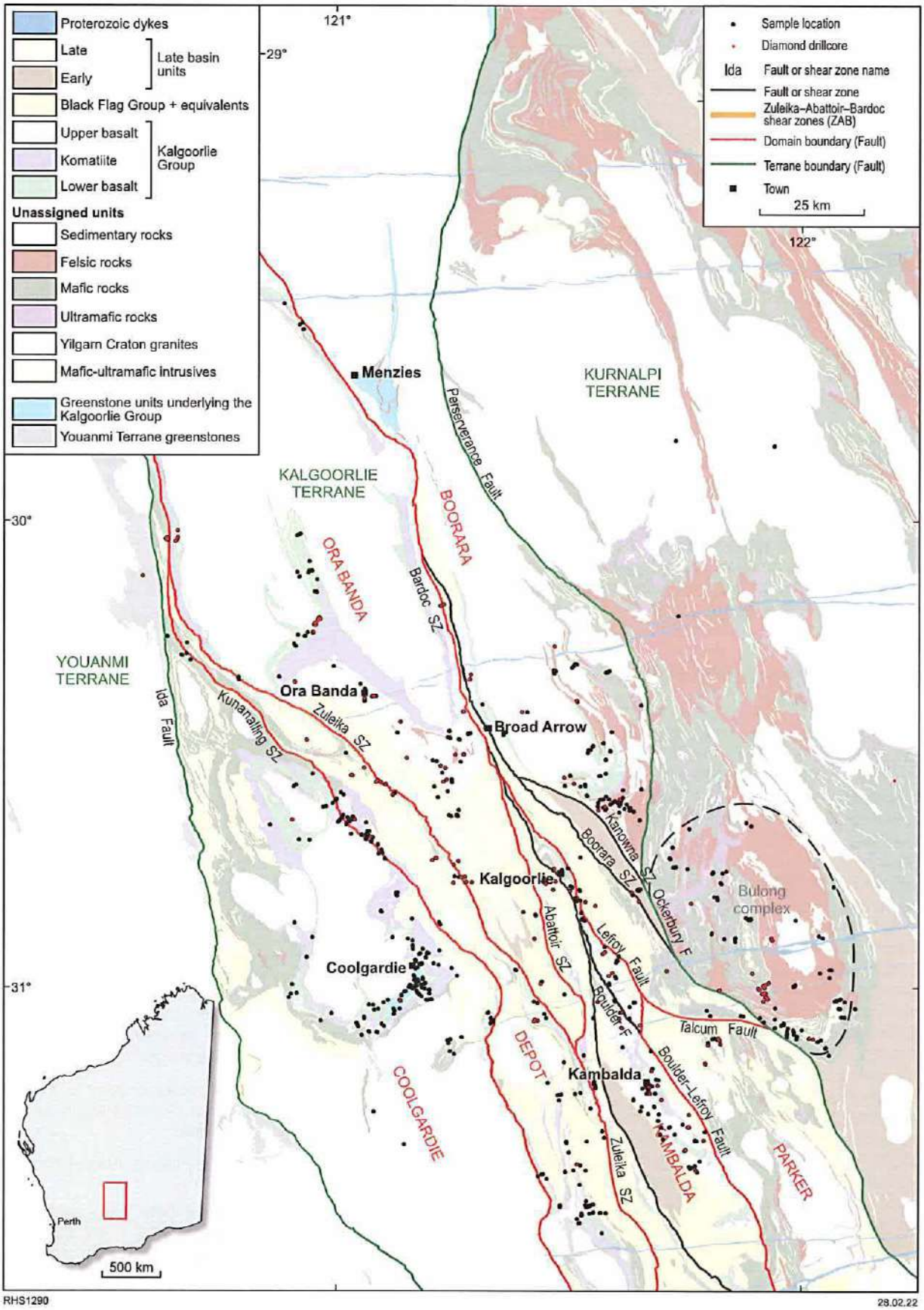


Figure 3. Geological map of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology, including the major faults and shear zones. Also shown is the distribution of sites where greenstones have been sampled for whole-rock geochemical data, including the locations for diamond drillcore. Diamond drillcore locations are typically the sites of multiple (up to 55) individual samples

Table 1. Number of samples from various regions

Domain	Coolgardie	Depot	Coolgardie + Depot	Ora Banda	Boorara	Kambalda	Parker	Bulong (complex)	Totals
Mafic volcanic samples									
Core	137	88	225	263	41	406	105	27	1067
Outcrop	127	110	237	69	57	28	12	63	466
Dolerite	19	41	60	25		143	1	27	256
Komatiite	35	71	106	18	11	114	16	41	306
Felsic samples									
Core	12	77	89	121	59	114	60	139	582
Outcrop	6	12	18	28	62	42	1	15	166
Total samples	336	399	735	524	230	847	195	312	2843

For analytes where the concentration is at least ten times the lower level of detection, a measure of accuracy is provided by the agreement between the average determined value and the certified value (Stanley and Lawie, 2007), which is <0.05 for all analytes, except Be and Cu. In terms of precision, the percent relative standard deviation or covariance for the analysis of OREAS 24b is <10 for all analytes, except As, Cu, Ni, Sc, and Zn. Similar levels of agreement were found for parent-duplicate pairs. All blank values were <3 times the lower level of detection.

Primary lithological classification and sample screening

For the classification and screening of rocks with broadly mafic to ultramafic compositions, we slightly modified the approach by Barnes et al. (2012). In particular, the maximum SiO_2 content used to identify rocks of broadly basaltic composition has been raised from 56 wt% to 57 wt%. This modification recognizes that even diamond drillcore samples from the Eastern Goldfields Superterrane, particularly in regions close to known mineralization, are commonly hydrothermally altered, often with significant effects on the concentrations of important major elements, such as SiO_2, CaO, Na_2O, and K_2O. Often, this alteration is not obvious during visual inspection of samples prior to selection for analyses. All samples taken as part of the greenstone geochemical barcoding project represent the visually least-altered examples of lithologically, texturally, and mineralogically homogeneous intervals of core. Nevertheless, it is not uncommon that one or more of a group of samples taken from within a lithologically uniform segment of drillcore has anomalous concentrations of SiO_2 (e.g. above the 56 wt% maximum value of Barnes et al., 2012) or other major elements, despite being visually indistinguishable from the associated samples. The approach used by Barnes et al. (2012) allows for the expansion of such parameters for classification of various basaltic rock types, as it primarily uses variations in ratios of incompatible trace elements (Ti, Th, La, Nb, Zr) that are relatively immobile during hydrothermal alteration and low- to medium-grade metamorphism, and are insensitive to small variations in major elements related to common igneous processes (e.g. fractional crystallization).

Since classification of the mafic rocks was based primarily on variations in ratios of fluid immobile trace elements, high loss on ignition (LOI) values were not used as a filter for strongly altered rocks. Nevertheless, >80% of our samples had LOI values <5 wt%.

All major element concentrations used and quoted throughout this Report have been recalculated on a volatile-free basis, although the differences with non-normalized data are typically insignificant.

The following screens and caveats were applied to the data:

- Intrusive rocks with SiO_2 <57 wt% and MgO <18 wt% were classified as dolerite or gabbro, depending upon their grain size. Those samples with MgO >18 wt% were classified as peridotite
- Non-intrusive rocks with SiO_2 <57 wt% were classified as basaltic, if MgO was <18 wt% and >3 wt% and komatiite, where MgO >18 wt%
- Many samples with MgO between 16 wt% and 18 wt% (and a few with MgO <16 wt%) have Al_2O_3/TiO_2 ratios ~20 (Al/Ti of ~1.0), a criteria Barnes et al. (2012) used to identify komatiitic basalts (KB). In nearly every case, these were directly associated, in drillcore, with true komatiites (olivine spinifex-textured rocks or rocks intimately associated with olivine spinifex-textured rocks) and have been classified as komatiitic basalt. In some cases, these formed a compositionally distinct series that extended to much lower MgO contents
- Intrusive rocks with SiO_2 >57 wt% were almost invariably plagioclase-porphyritic and were simply classified as felsic dykes [andesitic/dioritic (SiO_2 up to 63 wt%), dacitic/granodioritic (SiO_2 between 63 wt% and 68 wt%) or rhyolitic/granitic (SiO_2 between 68 wt% and 77 wt%)
- Rocks with SiO_2 >57 wt% and strong evidence for a volcanic and volcaniclastic origin were classified as felsic volcanic or volcaniclastic (andesitic, dacitic or rhyolitic) rocks
- Rocks with SiO_2 >57 wt% and no clear evidence for a volcanic and volcaniclastic origin were classified simply as felsic (andesitic, dacitic or rhyolitic) rocks
- Rocks with SiO_2 >57 wt% were initially excluded from the dataset if:
  - The aluminium saturation index [ASI = molar $Al_2O_3/(CaO+Na_2O+K_2O)$] exceeded 1.1 (i.e. the sample was peraluminous in many cases indicative of feldspar-destructive alteration)
  - SiO_2 exceeded 77.5 wt% (silica concentrations higher than values achievable through normal igneous processes; i.e. silicified)
  - Na_2O was <1 wt% or Na_2O+K_2O <3.5 wt% (values suggestive of alkali loss).

Mafic rocks

Identifying primary mafic geochemical groups

Having applied the screens described above, samples identified as basaltic were further divided into primary lithochemical groups, including low-Th basalt [LTB, a proxy for any light rare earth element (LREE)-, high field strength element (HFSE)-, and Th-poor basalts], intermediate-Th basalt (ITB), and high-Th siliceous basalt (HTSB) groups, as defined by Barnes et al. (2012) (Fig. 4). LTB, ITB, and HTSB groups, by definition, are typically distinguished on a plot of Th vs TiO_2. Although Th concentrations are generally more sensitive to contamination and fractionation, the fluid immobile and only slightly less incompatible elements, Zr, Nb, and La, can also be used to discriminate broadly similar low, intermediate, and high groups. Instances where the variation in one or more of these elements is slightly decoupled from the others provide potential means of further subdivision. Hence, some LTB groups identified because of their low Th, Nb, and La concentrations can have slightly high Zr concentrations that result in Zr/TiO_2 ratios within the range typical of ITB, rather than LTB. These would be separated as a Zr-enriched LTB group. Similarly, if anomalism is identified with Th, the result might be a Th-enriched LTB. If this is a persistent characteristic within a specific geographical region and/or at a specific stratigraphic level, then we define a LTB group with high-Th. Similar arguments apply for the identification of ITB. HTSB should be identified based on Th and La vs TiO_2 alone. We used all four trace elements (each plotted against TiO_2) to identify our primary mafic LTB and ITB groups. In cases where a sample plots in one field based on two elements and in another based on the other two elements, geological judgement is required to classify the sample. For example, the degree of alteration (visually or inferred from high LOI) compared to that of closely associated but less ambiguously classifiable samples might need to be considered. Likewise, whether a feature was an unusual or a persistent feature within a spatially or stratigraphically related series of samples might need to be considered. Although it is clearly feasible that a group of related samples could truly have perfectly transitional characteristics between LTB and ITB or ITB and HTSB, we have not yet identified any such cases.

Some samples classified earlier as komatiite based on having MgO >18 wt%, fall into the HTSB field in plots of TiO_2 vs Th, Nb, La, and Zr. These are clearly not komatiites based on their trace element patterns. Some could be fine-grained sills, although many show clear pillow structures, are closely associated in core with HTSB, and are interpreted here to reflect primitive HTSB lavas containing variably high proportions of olivine phenocrysts.

Some basalts with MgO generally between 10 wt% and 18 wt% and $Mg^{\#}$ [molar ratio of Mg/(Mg+Fe) where all Fe is calculated as Fe^{2+}] from 60 to 78 (higher than typical LTB), have Al_2O_3/TiO_2 ratios of ~20 but have concentrations of Th, Zr, Nb, and La considerably lower than that of HTSB (and mostly lower than LTB). These are typically directly associated with komatiites and are classified as komatiitic basalt.

Further subdivision of mafic geochemical groups

Primary mafic geochemical groups were further manually subdivided into smaller geologically plausible geochemical groups. This was an iterative process undertaken by simultaneously viewing the position of data points with respect to natural data clusters on plots of:

- TiO_2 vs Th, Nb, Zr, and La
- Th vs Nb and Zr
- Nb/Yb vs Th/Yb (Pearce, 2014)
- Al_2O_3/TiO_2 vs Th/Nb
- Mantle-normalized incompatible trace element diagrams
- $Mg^{\#}$ vs all major elements, Cr, Ni, and Al_2O_3/TiO_2
- Longitude vs latitude vs position in drillcore (a map).

Given the need to draw comparisons between numerous individually large data populations, many of the geochemical variation diagrams shown in this Report appear extremely cluttered (data rich), particularly at the presented scale. For this reason, most geochemical variation diagrams are also reproduced in Appendix 3 in a layered PDF format, which allows a user to enlarge the figure on screen and select specific geochemical unit(s) (each as separate layers) they wish to display.

Compositional variations in the plots listed (above) highlight the controls that various fundamental igneous processes and source compositional variations might have exerted on an evolving magma, and in the case of TiO_2, P_2O_5, and most of the selected trace elements, are not strongly influenced by post-magmatic alteration. Throughout this process, there is continuous consideration of critical factors, including whether:

- an observed geochemical trend or cluster plausibly reflects a rational geochemical/igneous process
- the identified trend- or cluster-related groups make sense in terms of spatial relationships (map patterns), clear and established geological features (textural, mineralogical, structural characteristics), and currently understood stratigraphic relationships. This does not mean that all members must be restricted to a single stratigraphic horizon in a single region. A geochemical grouping simply reflects a specific set of petrogenetic conditions, which might be reproduced at several stages in time and at several localities.

Mafic rock trends and units

The groups or units defined in this way, commonly show a degree of compositional overlap with other groups or units on one or many of the compositional variation diagrams, although each has either a compositional feature or a geographical distribution (or both) that allows it to be uniquely distinguished. A total of 26 units were identified (Table 2) and most of these were either LTB or ITB. These cannot be given stratigraphic names because in most cases they occur at several stratigraphic positions, even within a single subgroup stratigraphy. They also commonly form one of several distinct geochemical units that together form an established stratigraphic unit. For example, the Big Dick Basalt in the Vetters Subgroup comprises at least two distinct geochemical units and the Lindsays Basalt in the upper Coolgardie succession comprises at least three units.

Most of the basalt units were named primarily using plots of Th against TiO_2 and Th against Zr. The naming convention was constructed to emphasize the magmatic or petrogenetic history of the unit without any necessary implications in terms of direct

genetic relationships to other units. It is clear from the systematic data spread in bivariate plots utilizing incompatible trace elements that varying degrees of crustal contamination probably exerted the greatest control over compositional differences between the units identified here. The primary classification itself is based on increasing amounts of crustal contamination from LTB to ITB to HTSB; however, we added a secondary code that reflects the relative degrees of contamination of a specific unit within a given primary classification (Fig. 5). An additional code is added to indicate the relative extent of fractionation of a specific unit. Thus:

- For the primary classification descriptor:
  - LTB is denoted as 'L'
  - ITB is denoted as 'I'
  - HTSB is denoted as 'H'.
- The secondary measure of the amount of crustal material a mafic parental magma had incorporated was based broadly on the position of a unit in plots of Th vs TiO_2 and Th vs Zr, (with Th/TiO_2 and Th/Zr ratios increasing with increasing contamination) relative to the other units of the same primary classification, such that:
  - U = uncontaminated or nearly uncontaminated
  - C = contaminated
  - S = strongly contaminated
  - VS = very strongly contaminated.
- The degree of fractionation was a qualitative measure within each primary classification group for each specific secondary measure of contamination (e.g. L-S for strongly contaminated LTB) and was based on the concentration range of Th and Nb in each unit. For example, L-S1 had lower Th concentrations (was less fractionated) than L-S2, which was less fractionated than L-S3, and so on.

Hence, L-U5 would be an uncontaminated but strongly fractionated LTB whereas I-S2 would be a strongly contaminated but weakly fractionated ITB.

It is important to note that 10 of the 26 identified units could be combined to produce three distinct geochemical trends, each characterized by relatively constant incompatible trace element ratios. Variation between these various trends ranges from ratios between normal mid-oceanic ridge basalt (N-MORB) and primitive mantle values, reflecting very little or no crustal contamination, to ratios reflecting significant amounts of crustal contamination of the parental magmas. However, if we assume that the consistency of the ratios defining each of the three distinct geochemical trends is not fortuitous, then variation along the trends (element concentration) reflects closed-system fractionation.

Mafic groups, units and trends identified using the procedure outlined above (Figs 6–8) include:

- LTB
  - L-U1 and L-U2: two units of uncontaminated (or nearly uncontaminated) and relatively unfractionated primitive LTB, including one minor variant (L-U2*)
  - L-U3–7: five units of uncontaminated (or nearly uncontaminated) but moderately to strongly fractionated LTB, forming the L-U3–7 trend
  - L-Ca: a coherent unit of contaminated LTB showing a continuous fractionation trend that [except for a possible Zr-enriched variant (L-Cb)], cannot plausibly be divided into discrete units. L-Ca also forms the L-C* trend

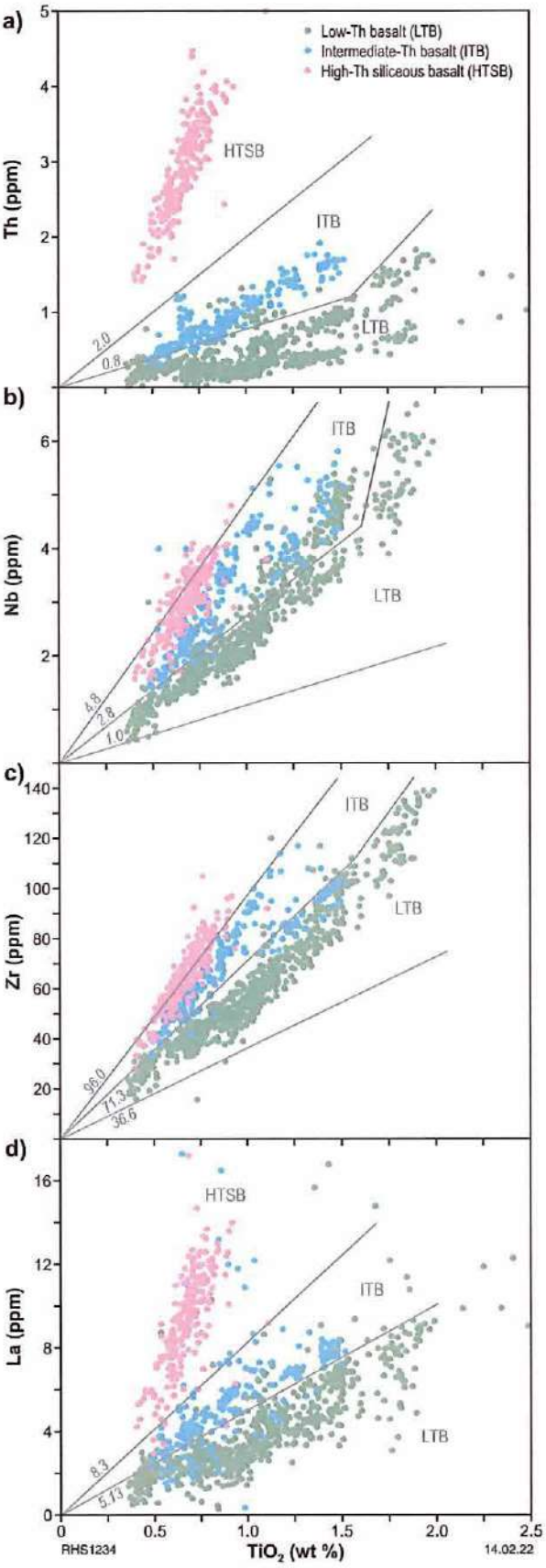


Figure 4. Variation in Th, Nb, Zr, and La with TiO_2 used for the primary classification of basalts into LTB, ITB, and HTSB (after Barnes et al., 2012). Numbers indicate the x/TiO_2 ratio of the lines chosen here to separate the various groups

  - L-S1 and L-S2: two units of strongly contaminated but relatively unfractionated LTB
  - L-S3a, L-S3b, and L-S4: three groups of strongly contaminated and strongly fractionated LTB.
- ITB
  - I-S1–3: three units of strongly contaminated and weakly to moderately fractionated ITB, forming the I-S1–3 trend
  - I-S4: a strongly contaminated and strongly fractionated ITB, but compositionally distinct from units of the I-S1–3 trend
  - I-VS: a very strongly contaminated but relatively weakly fractionated ITB.
- HTSB: By definition, all HTSB are strongly contaminated with crustal material. Three groups are recognized:
  - H-HMg#: HTSB with high Mg#
  - H-LMg#: HTSB with lower Mg#
  - H-LTh: HTSB with lower Th concentration compared with other HTSB.
- KB (Not shown in Figs 6–8)
  - Three units: KB1–3.

Table 2. List of mafic geochemical units identified and the corresponding symbols used in this Report

Mafic group	Unit	Symbol
Low-Th Basalt (LTB)		
	L-U1	
	L-U2	
	L-U2*	
	L-U3	L-U3–7 trend
	L-U4	
	L-U5	
	L-U6	
	L-U7	
	L-Ca	L-C trend
	L-Cb	
	L-S1	
	L-S2	
	L-S3a	
	L-S3b	
	L-S4	
Intermediate-Th Basalt		
	I-S1	I-S1–3 trend
	I-S2	
	I-S3	
	I-S4	
	I-VS	
High-Th Siliceous Basalt		
	H-HMg#	
	H-LMg#	
	H-LTh	
Komatiitic Basalt		
	KB1	
	KB2	
	KB3	
	Komatiite	

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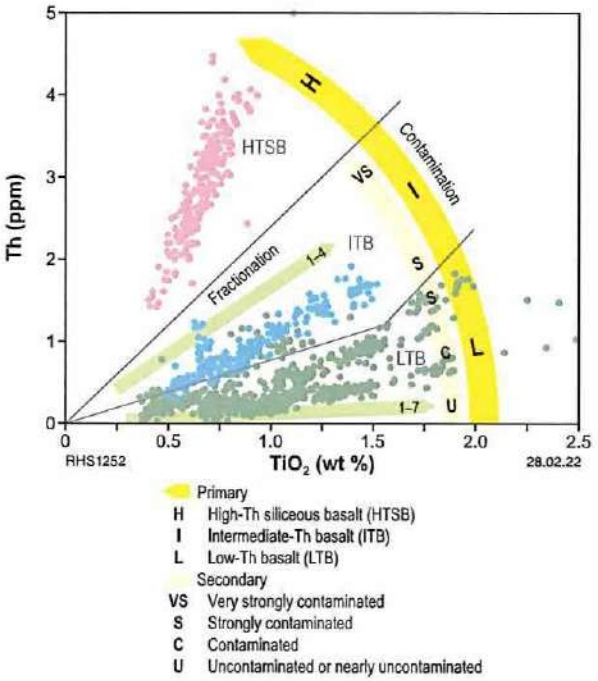


Figure 5. Variation in Th with TiO_2 used for classification of basalts (after Barnes et al., 2012) showing the rationale behind the system used to give unique codes to each specific mafic geochemical unit (see text for details)

A degree of caution should be exercised regarding discrimination procedures, such as the one outlined above, which utilizes trace elements typically present at concentrations approaching analytical detection limits. This is particularly the case for Th. Under these circumstances, it can be difficult to be certain that some of the typically more incompatible trace element-depleted groups are not simply artefacts of variation within the limits of analytical precision. In Figure 6, we show the estimated precision of analyses for Th, Nb, Zr, and La at very low concentrations, based on historical analytical results from BV Minerals (Champion et al., 2021). These uncertainties are generally small but clearly, at least in the case of Th, do potentially provide a plausible explanation for some of the compositional groups. However, we are confident that these groups are geologically meaningful for several reasons:

- The precision estimates are (only) indicative values based on analysis of a wide range of rock types and standards mostly with significantly higher concentrations of Th and Nb (for example). These estimates would almost certainly be improved, if calculated based only on samples and standards with very low concentrations.
- Mantle normalized incompatible trace element patterns (see later) are typically smooth between Th, which has a relatively high analytical uncertainty, and La and other LREE, which have a significantly lower analytical uncertainty; strongly suggesting that the analytical precision for Th is much better than the estimated values.
- The same groups are consistently identified and often well defined and on a wide range of compositional variation diagrams, including those using major element data at concentrations not subject to the same limitation in precision.
- Most of the groupings make geological sense in terms of their geographic and/or stratigraphic distribution.
- The compositional groups typically comprise samples from several different analytical batches.

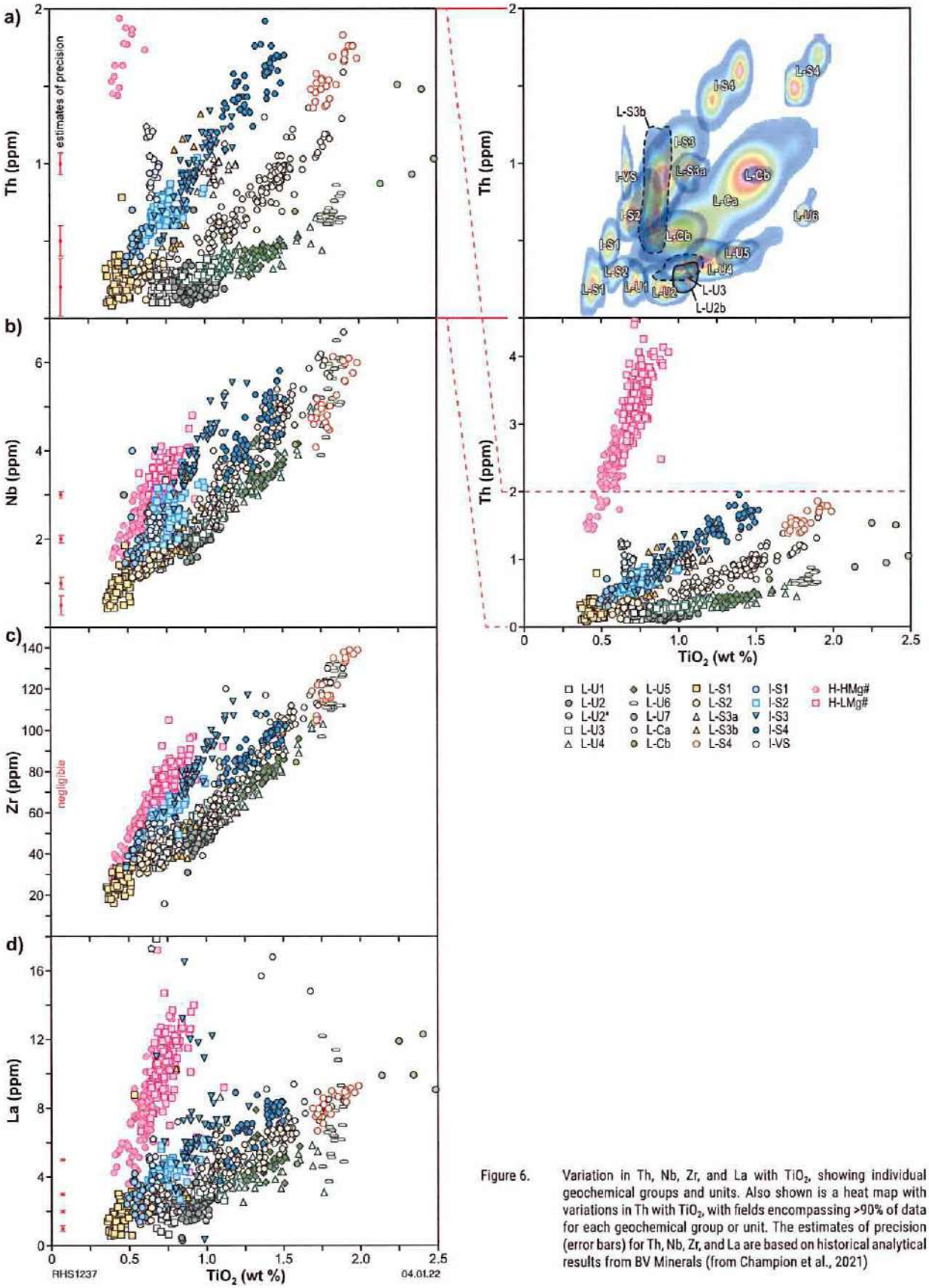


Figure 6. Variation in Th, Nb, Zr, and La with TiO_2, showing individual geochemical groups and units. Also shown is a heat map with variations in Th with TiO_2, with fields encompassing >90% of data for each geochemical group or unit. The estimates of precision (error bars) for Th, Nb, Zr, and La are based on historical analytical results from BV Minerals (from Champion et al., 2021)

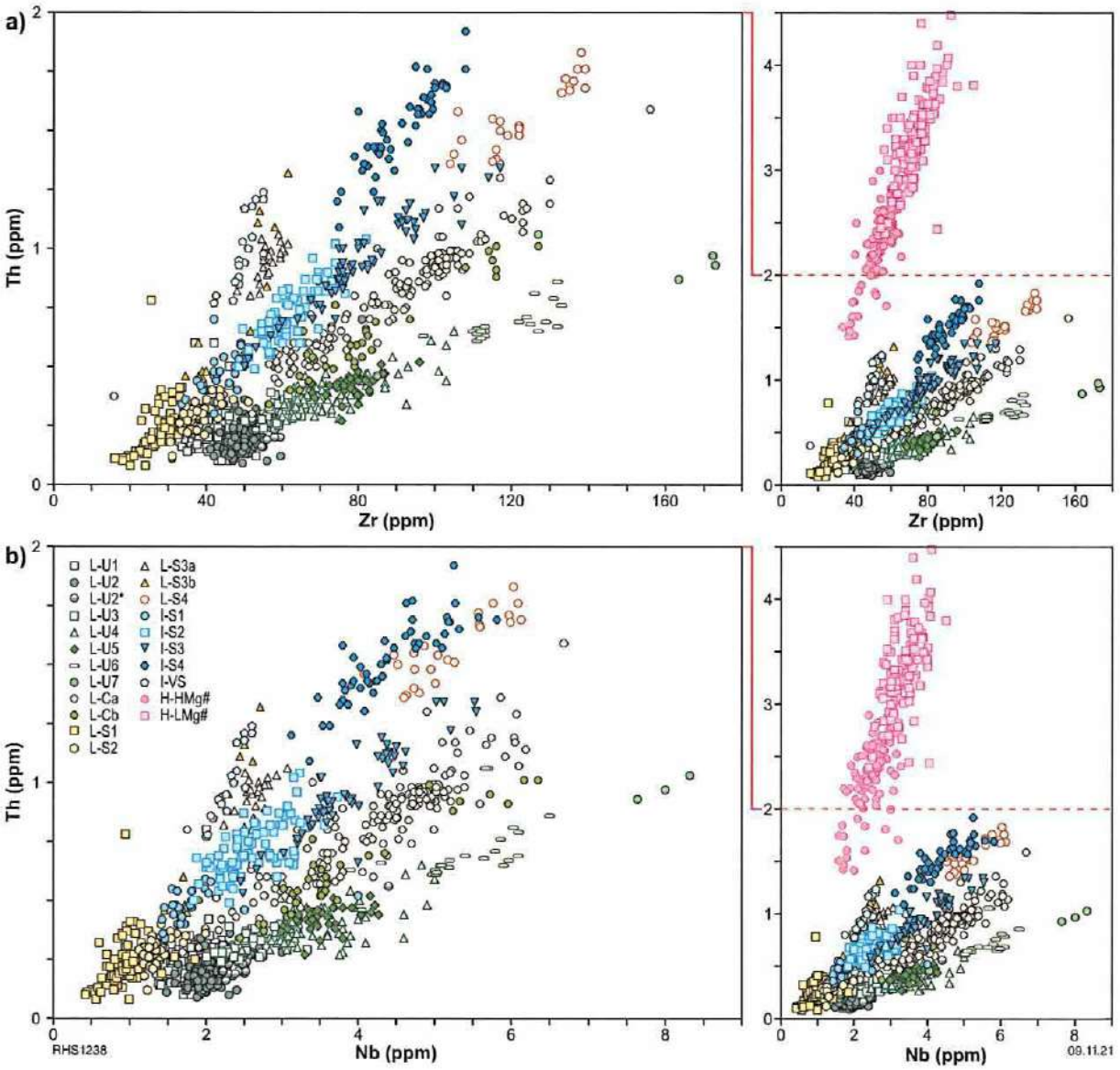


Figure 7. Variation in Th with Zr and Nb showing individual geochemical groups and units (right-hand panels are expanded to include data for the HTSB)

There is an additional problem, unrelated to the analytical issues outlined above, inherent in initially grouping basaltic magmas using the TiO_2 vs Th, Nb, Zr, and La diagrams. This is that for primitive (high $Mg^{\#}$, high MgO) magmas with very low concentrations of incompatible trace elements, the fields for LTB and ITB converge to the point where the classification becomes difficult to apply. Such compositions typically represent high degrees of mantle partial melting and are probably transitional to komatiitic basalts, if not actually komatiitic basalt. Two such groups encountered in this study are designated as L-S1 and L-S2 (Fig. 6).

Using the available stratigraphic information, we identified various geochemical components comprising the basaltic formations of the stratigraphic columns for regions broadly corresponding to each of the domains within the study region (Fig. 9). Within some of these regions, in particular, those corresponding with the Depot Domain, several areas show a local stratigraphy significantly different to the wider domain stratigraphy. These stratigraphic segments are discussed further in a following section. In general, however, the regional stratigraphies are divided into three levels (lower, middle, and upper) according to previous lithostratigraphic interpretations (e.g. Swager et al., 1990; Morris, 1993), which clearly correspond to broad but significant changes in the composition and style of magmatism (see later). Komatiite provides a convenient marker at the base of the middle level. We provide no information on the relative stratigraphic thicknesses of levels within each column. The dashed line shown in the upper part of the Coolgardie, Depot, and Ora Banda columns (Fig. 9) indicates that an upper stratigraphic level does not appear to have been developed or preserved (e.g. in the case of the Coolgardie Domain: e.g. Swager et al., 1990) or that the boundary between the middle and upper levels is unclear (e.g. for the Ora Banda Domain). Relatively low sample numbers mean that the chemostratigraphy for some parts of the Boorara and Parker Domains and the Bulong complex is less robust than for the regions to the west. For units where this might be of concern, the actual number of samples collected is indicated in Figure 9.

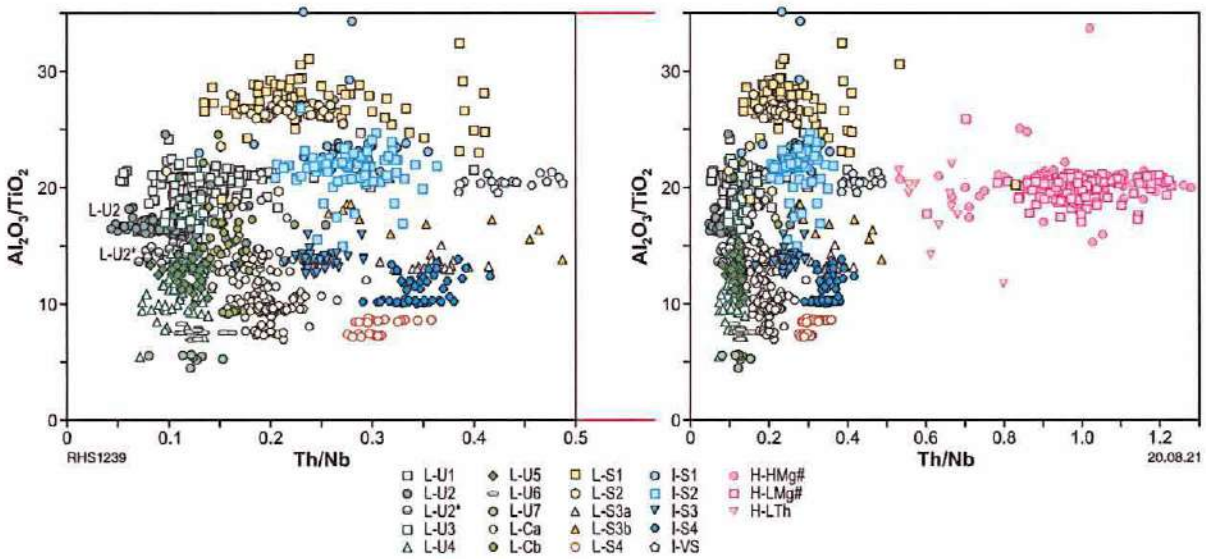


Figure 8. Variation in Al_2O_3/TiO_2 with Th/Nb showing individual geochemical groups and units (right-hand panel is expanded to include data for the HTSB)

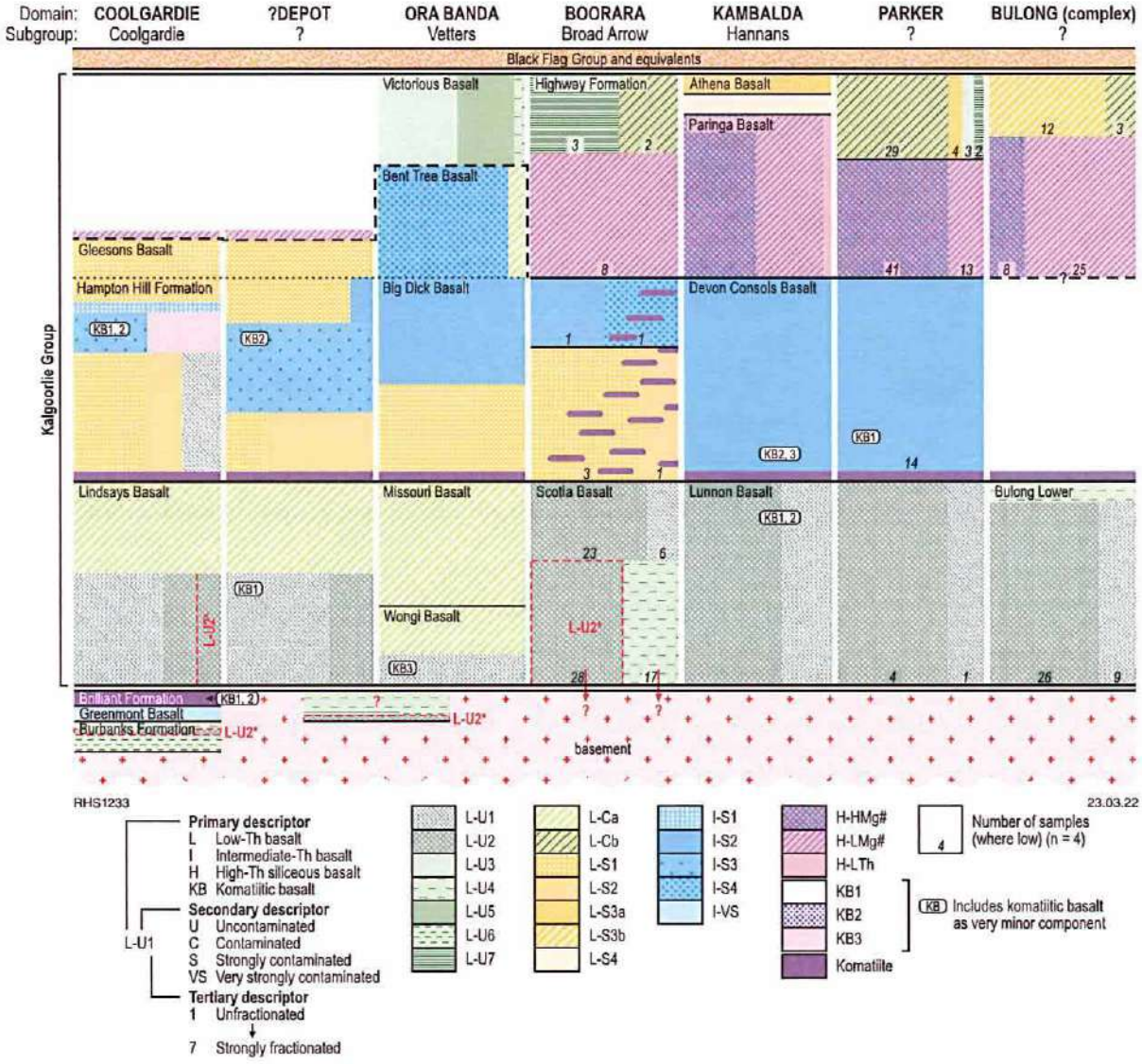


Figure 9. Chemostratigraphic columns representing each of the domain regions within the area studied. Also included is a brief graphical explanation of the coding system used. See text for further details and explanation

Otherwise, for each stratigraphic formation, the volume of each specific geochemical unit is proportional to the number of samples analysed. Whilst this might be broadly representative of true relative abundances in most cases, it is clearly open to sample bias. Where the relative position of geochemical units within a stratigraphic formation is known or reasonably inferred, those units are organized vertically. Otherwise, geochemical units within the stratigraphic formations are organized horizontally.

Low-Th basalts (LTB)

By definition, most of the LTB units have undergone relatively little contamination by compositionally evolved crustal material. Therefore, most have Th/Nb and La/Nb ratios close to primitive mantle values (Fig. 10). Even the most strongly contaminated LTB are typically less contaminated than most ITB units. The LTBs are tholeiitic basalts and the more primitive examples (less contaminated and less fractionated) represent the most common class of Archean mafic rock globally (Barnes et al., 2021). They can occur at all levels within a typical greenstone stratigraphy but in general, more contaminated and fractionated units occur at higher stratigraphic levels, dominantly as intrusions rather than volcanic rocks. Although they are classified based mainly on where they lie on plots of Th, Nb, Zr, and La against TiO_2, we found that the individual LTB units and trends are also well discriminated based on their Th/Zr ratios (Fig. 7a).

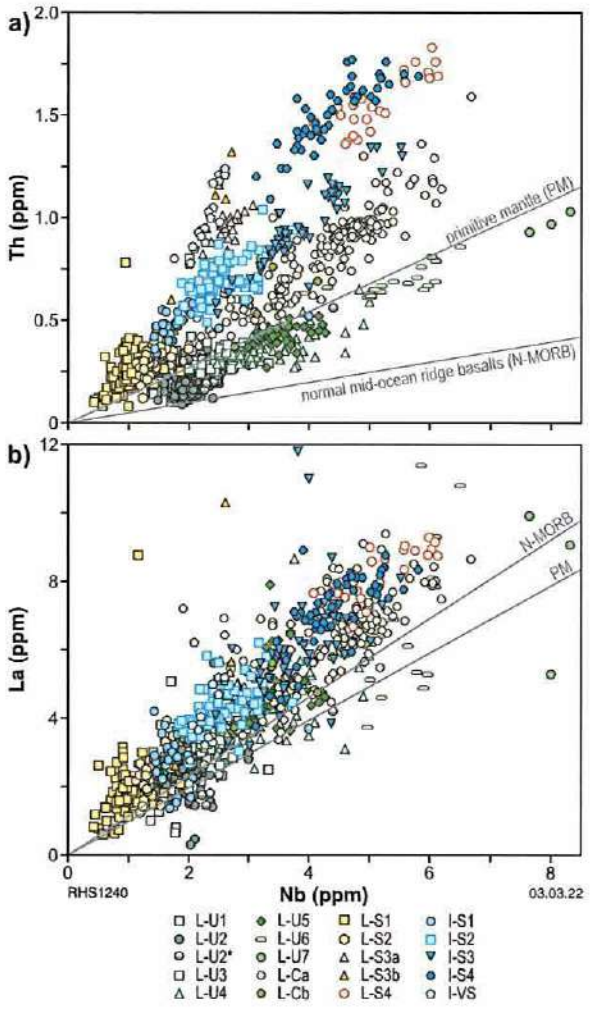


Figure 10. Variation in Th and La with Nb for individual LTB and ITB geochemical groups and units. Ratios for primitive mantle and mid-oceanic basalt (N-MORB) are from Sun and McDonough (1989)

An important finding from this work is that there are at least 12 major geochemically discrete LTB units within the study area. This represents a significant problem or source of serious misinterpretation when LTB compositions are erroneously regarded as synonymous with the Lunnon Basalt (the stratigraphic unit) or more broadly referred to as Lunnon-like. Similar problems arise regarding the Devon Consols Basalt and the ITB groups. Of the 12 LTB units, seven are L-U units but only two (L-U1 and L-U2) occur within the Lunnon Basalt. Hence:

- **LTB is not synonymous with Lunnon Basalt:** Only a basalt pile within the region between Kalgoorlie and Kambalda, comprising combinations of primitive L-U1 < L-U2 alone, overlain by komatiite and then by I-S2 (i.e. the Devon Consols Basalt), is Lunnon Basalt. Likewise, ITB is not synonymous with Devon Consols Basalt.

Primitive LTB (L-U1 and L-U2)

Primitive L-U1 and L-U2 typically include a large proportion of samples with higher $Mg^{\#}$ (>50) and correspondingly high MgO contents and Cr concentrations (>200 ppm) than other LTB units (Fig. 11). They typically show very restricted ranges in incompatible trace element concentrations and at a given $Mg^{\#}$, have lower incompatible trace element concentrations than all other magmas, except for the L-S1 and L-S2 units. As with the other L-U units, they have low Th/TiO_2 and high Zr/Th ratios reflecting least-contaminated compositions. Some examples have compositions consistent with derivation from a mantle source slightly more depleted than primitive mantle, approaching that of modern N-MORB source. Many samples also show only a minimal or no perceptible mantle-normalized Nb anomaly. L-U2 have slightly lower $Mg^{\#}$ and slightly lower Th/Nb, Th/La, and Th/Zr ratios than L-U1 (Figs 7, 11; Appendix 4), indicating that neither fractional crystallization nor crustal contamination (alone or together) can directly genetically relate the two units.

Discriminating primitive L-U1 and L-U2

On most compositional variation diagrams, L-U1 shows significant compositional overlap only with L-U2 but can be uniquely identified by its lower TiO_2 (<0.8 wt%) (Fig. 11) and unique position with respect to all other LTB units on a plot of Al_2O_3/TiO_2 vs Al_2O_3 (L-U1 = 18.3–22.5) (Fig. 12). L-U2 can be separated on a plot of Al_2O_3/TiO_2 vs Al_2O_3 (L-U2 mostly 13–18) from other LTB units except L-Ca and L-S3 (a and b), which show a minor overlap but can all be uniquely separated from L-U2 on a plot of Th vs TiO_2. (Fig. 6a).

L-U2 can be subdivided based on slight differences in Al_2O_3/TiO_2 ratio (Figs 8, 12a). A main group has Al_2O_3/TiO_2 ratios from 16 to ~18, whilst a minor group (denoted L-U2* where a distinction is required) has ratios from 13 to 16. No other systematic difference is noted between the two populations, although L-U2* has a specific geographical coverage (see below).

Like L-U1 and L-U2, L-S1 and L-S2 have very low incompatible trace element concentrations but can be clearly identified by their unusually high Al_2O_3/TiO_2 (mainly >25) (Fig. 8).

Distribution of primitive L-U1 and L-U2

Primitive L-U1 and L-U2 are geochemically distinctive and widespread, forming a significant component of the basalt stratigraphy (Figs 9, 13). They appear mainly restricted to the basal basalt units of each stratigraphic sequence, including the lower and upper Coolgardie successions, lying immediately below the main komatiite layer. Exceptions are L-U1 samples taken from the lower

levels of the komatiitic Hampton Hill Formation, although these could be from tectonic slices of Lindsays Basalt. The subsidiary L-U2* population includes all L-U2 samples from the Burbanks Formation in the lower Coolgardie succession. It also includes >50% of L-U2 sample of the Scotia Basalt at the base of the Broad Arrow Subgroup, forming what appears to be the stratigraphically lowest samples around the southern flank of the Nine Mile Monzogranite in the Scotia–Kanowna Anticline. Samples of L-U2* form scattered occurrences in the Ora Banda Domain near Davyhurst, immediately east of the Zuleika Shear Zone, and these are potentially age equivalents of the Burbanks Formation.

Fine-grained L-U2 also occurs stratigraphy above (Bent Tree Basalt) and below (Big Dick Basalt) the Mount Pleasant Gabbro (sill) in the Vetters Subgroup, but in these cases are almost certainly quenched marginal intrusions related to the Mount Pleasant Gabbro itself, which comprises 76% L-U2 compositions (Table 3). L-U2 forms a significant component of other major mafic sills (e.g. Condenser and Golden Mile Dolerites) (Table 3) emplaced at the end of or immediately after mafic greenstone deposition.

While both L-U1 and L-U2 occur throughout the study region, there is a clear tendency for L-U1 to be the dominant primitive L-U unit in the western domains (Coolgardie/Depot and Ora Banda Domains) while L-U2 is slightly more common to the east. L-U2 is almost twice as common as L-U1 in the Lunnon Basalt (Hannans Subgroup), locally forming the entire formation. However, where accompanied by L-U1, neither is confined to any specific stratigraphic interval within the formation (L-U1 and L-U2 can be interlayered). Similarly, L-U1 is twice as abundant as L-U2 in the Lindsays Basalt at the base of the upper Coolgardie succession.

L-U3–L-U7 and L-U3–7 trend

L-U3–L-U7 are more evolved (fractionated) than L-U1 and L-U2, although they show a similar, negligible, degree of crustal contamination. Near-constant incompatible trace element ratios (Th/Zr, Th/Nb, Nb/Zr, La/Sm; Fig. 14) that extrapolate to or close to the origin (x=0, y=0) in compositional variation diagrams, define a common bulk parental composition with most trace element ratios around or slightly lower than primitive mantle values. The constant incompatible trace element ratios also indicate that all subsequent compositional evolution of these magmas was essentially by crystal fractionation alone, in a system closed to further contamination. The concentrations of most strongly incompatible trace elements overlap between many of the units forming the L-U3–7 trend. However, the degree of enrichment in incompatible trace elements broadly follows the order L-U3 to L-U7, reflecting a general increase in the degree of crystal fractionation. On bivariate plots employing TiO_2, the straight incompatible trace element trends break down at TiO_2 concentrations above ~1.7 wt% (Figs 4, 6), as FeTi oxides become saturated and join the fractionating mineral assemblages. Only L-U6 and L-U7 reach such high TiO_2 contents.

It is important to note that the consistent incompatible trace element ratios shared by units of the L-U3–7 trend only indicate that each individual unit was derived from a compositionally similar source and suffered no later crustal contamination, not that they were necessarily directly genetically related to each other or to a common magma batch. The wide overlap in $Mg^{\#}$ between the units combined with significant variations in Al_2O_3 and Al_2O_3/TiO_2 (Figs 11, 12) suggests that plagioclase fractionation (removal or accumulation) was a major control on trace element enrichment or depletion. Commonly, the upper volcanic L-U3–L-U5 units contain conspicuous plagioclase phenocrysts, megacrysts, and glomerophenocrysts.

Discriminating units of the L-U3–7 trend

There are significant compositional overlaps between the various units forming the L-U3–7 trend, although combinations of various plots uniquely distinguish each unit. All of the units are completely distinct from all ITB units in a plot of Th vs TiO_2 (Fig. 6a).

L-U3 has low incompatible trace element concentrations that overlap those of L-U2 on most plots. However, L-U3 has a higher Al_2O_3 content (>17.5 wt%, e.g. Fig. 12). In any case, primitive L-U2 has not been found within the same stratigraphy as L-U3. Higher Al_2O_3 and lower TiO_2 (hence higher Al_2O_3/TiO_2) also distinguishes L-U3 from other units in the L-U3–7 trend (Fig. 12).

L-U4 has the broadest range of incompatible trace element concentrations within the L-U3–7 trend, with extensive overlap with L-U5 but occupies a unique field from all other L-U units on plots of Al_2O_3/TiO_2 vs Al_2O_3 or TiO_2 (Fig. 12). An exception includes very minor overlap with L-U2*, although the latter can be separated based on its slightly lower Nb concentrations (L-U2* <2.75 ppm <L-U4).

L-U5 shows extensive overlap only with L-U4 in terms of incompatible trace element concentrations, but also occupies a unique field on plots of Al_2O_3/TiO_2 vs Al_2O_3 or TiO_2 (Fig. 12).

L-U6 and L-U7 are effectively discriminated from the other units of the L-U3–7 trend units by their high (L-U6 = 1.75–1.9 wt%) and very high (L-U7 = >2.0 wt%) TiO_2 contents (e.g. Fig. 12).

Distribution of units of the L-U3–7 trend

Samples from various units of the L-U3–7 trend are considerably less abundant overall than those of the primitive L-U1 and L-U2 units (Fig. 15). They are not found within the upper Coolgardie succession or the Hannans Subgroup, although quenched intrusive equivalents of L-U4 and L-U5 related to the intrusion of the Golden Mile Dolerite form a minor component of the Black Flag Group overlying the Hannans Subgroup.

Units of the L-U3–7 trend occur in the Burbanks Formation of the lower Coolgardie succession (L-U4 and L-U6) and, more commonly, in the uppermost part of the basalt-dominated greenstone stratigraphy of the Vetters Subgroup (Victorious Basalt: L-U3, 4, and 5) and the Highway Formation (L-U7).

The more incompatible trace element-enriched and plagioclase-poor (lower Al_2O_3) L-U6 unit occurs only in the Burbanks Formation of the lower Coolgardie succession. However, strongly fractionated L-U7 occurs in several major mafic sills (Condenser Dolerite, Golden Mile Dolerite, Mount Pleasant Gabbro; Table 3) emplaced at the end of mafic greenstone deposition. The less incompatible trace element-enriched but plagioclase-rich (higher Al_2O_3) units (L-U3–5) mostly lie in the upper parts of the stratigraphy and form a significant component of late gabbro sills. However, L-U4 is associated with L-U6 in the Burbanks Formation and is found in the Scotia Basalt at the base of the Broad Arrow Subgroup (Fig. 9).

Contaminated LTB–L-C

This contaminated LTB unit covers a broadly similar major and compatible trace element compositional range to the units forming the L-U3–7 trend (Fig. 11), although it is characterized by higher values of incompatible trace element ratios such as Th/Nb, Th/Zr, La/Sm, and La/Yb (Fig. 16). As inferred from a plot of Th vs TiO_2 (Fig. 6a), ranges in these ratios reflect the significant effect of crustal contamination (assimilation) on the composition of primitive L-C basalts.

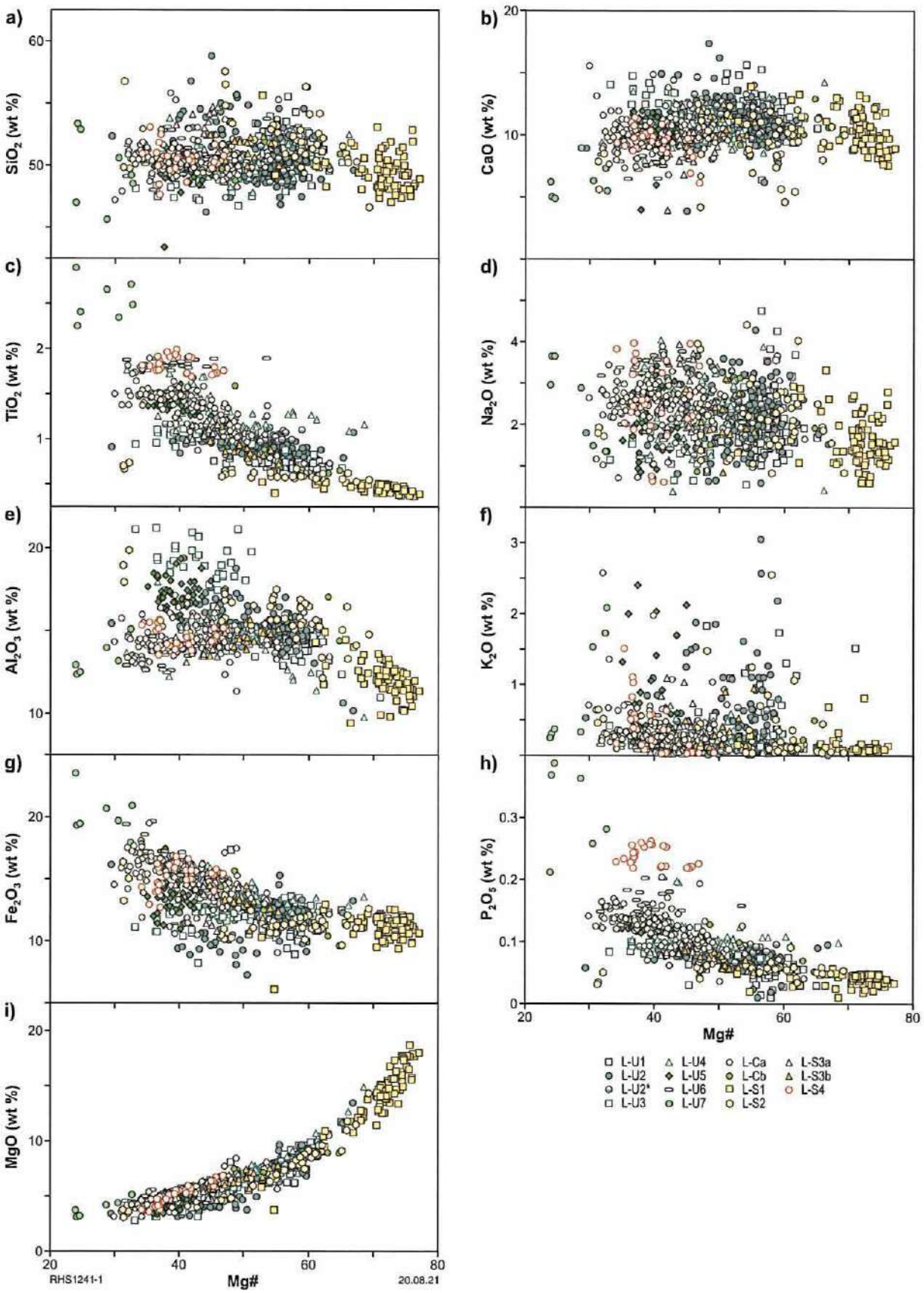


Figure 11. Variation in major and selected trace elements with Mg# for individual LTB geochemical groups and units

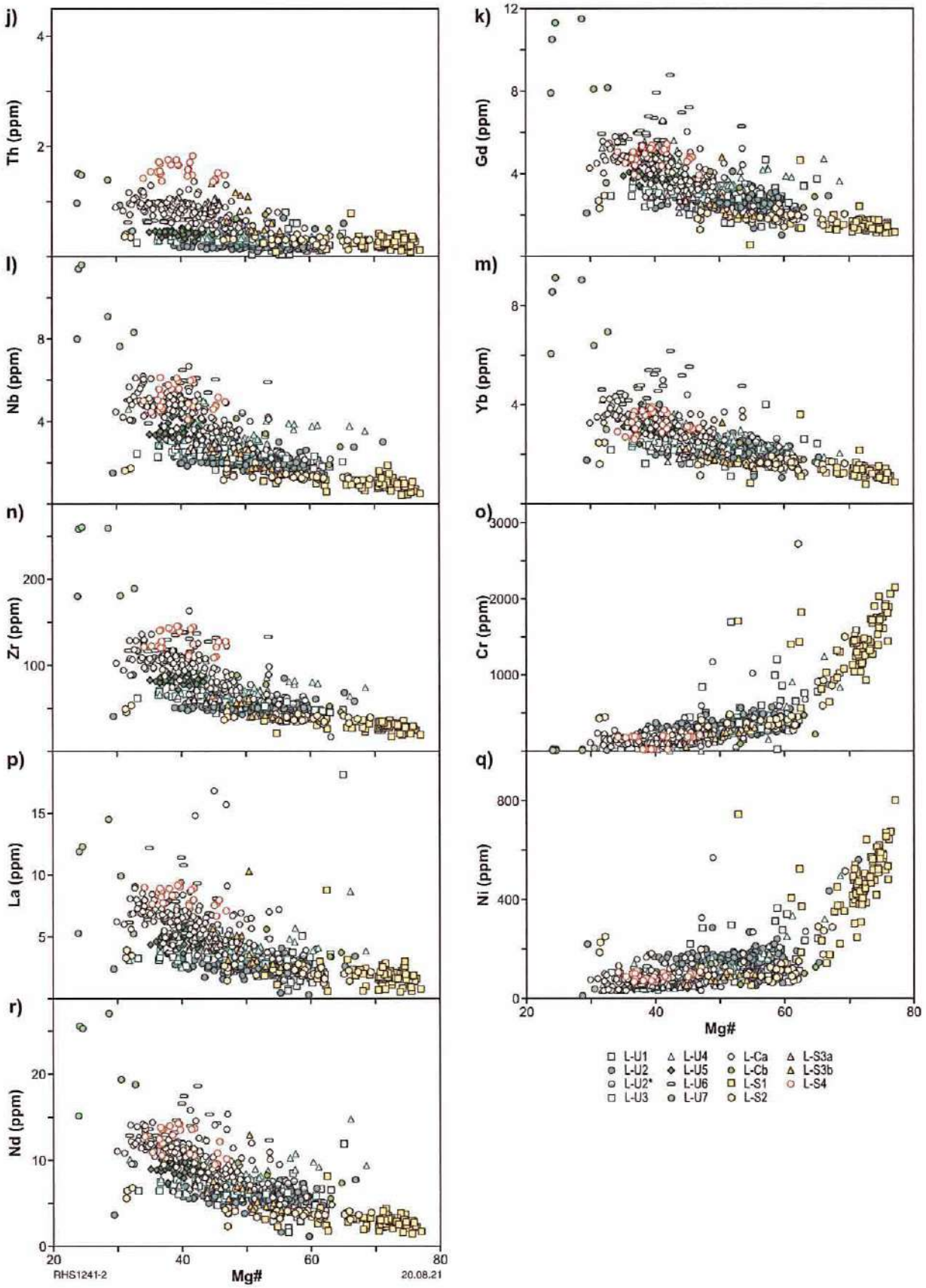


Figure 11. continued

A potential subset of the main unit (L-Cb) is identified based on low Th concentrations and consequently low Th/Nb and Th/Zr ratios, consistently within (or below) the lowest end of the range for L-Ca. L-Cb also has La/Sm ratios that scatter at higher values than those of L-Ca and shows a much wider and correlated range to lower Mg# (65 to 21) and higher SiO_2 (49.6 to 57.2 wt%) (Fig. 11).

Discriminating L-C

Most diagrams plotting co-variation of two strongly incompatible trace elements effectively discriminate L-C units from other LTB units (Fig. 16). Plots of Th vs TiO_2, Nb, and Zr appear most effective. L-Ca and L-Cb are probably best distinguished from each other on a plot of SiO_2 vs Mg# or La vs Sm.

Distribution of L-C

Although L-C is common and widespread throughout much of the western part of the study region, it does not occur within the Kambalda Domain (Hannans Subgroup) (Fig. 17). L-Ca broadly corresponds to an enriched basalt unit first identified by Hayman et al. (2015), who noted that it was primarily restricted to the region to the west and north of the Kambalda Domain. There seems to be a strong association with primitive L-U1 and together, these units dominate the lower part of the upper Coolgardie succession and the lower Vetters Subgroup (Wongi and Missouri Basalts) (Fig. 9). Although less common, L-Ca also forms a component in the upper part of the Vetters Subgroup (Bent Tree Basalt). L-Cb forms a minor component in the upper parts of the Broad Arrow Subgroup and in the upper parts of the Bulong complex and Parker Domain stratigraphy. It is restricted to the regions east of the Coolgardie/Depot and Ora Banda Domains (referred here as the 'eastern domains').

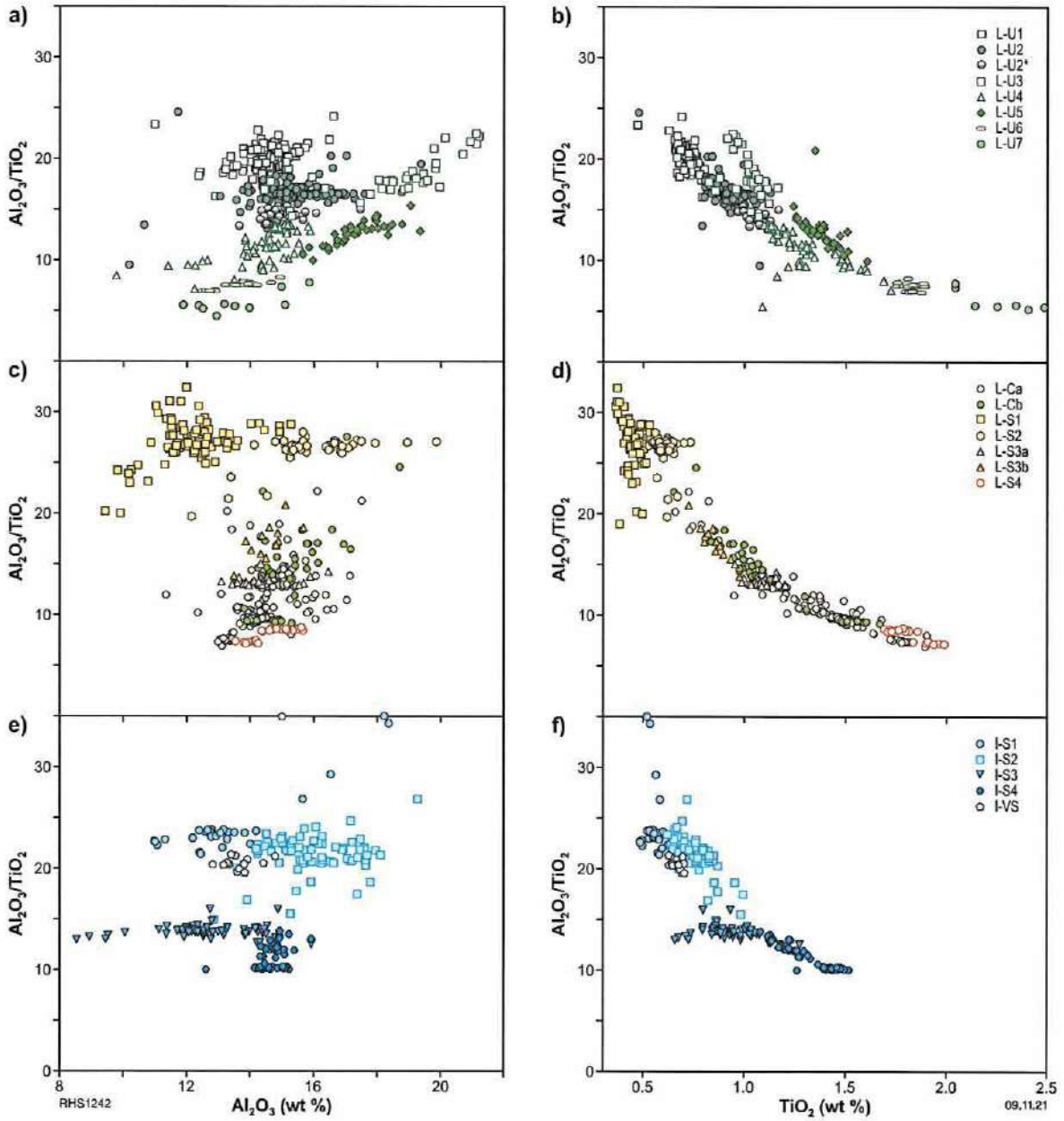


Figure 12. Variation in Al_2O_3/TiO_2 with Al_2O_3 (left-hand column) and Al_2O_3/TiO_2 with TiO_2 (right-hand column) for individual LTB and ITB geochemical groups and units

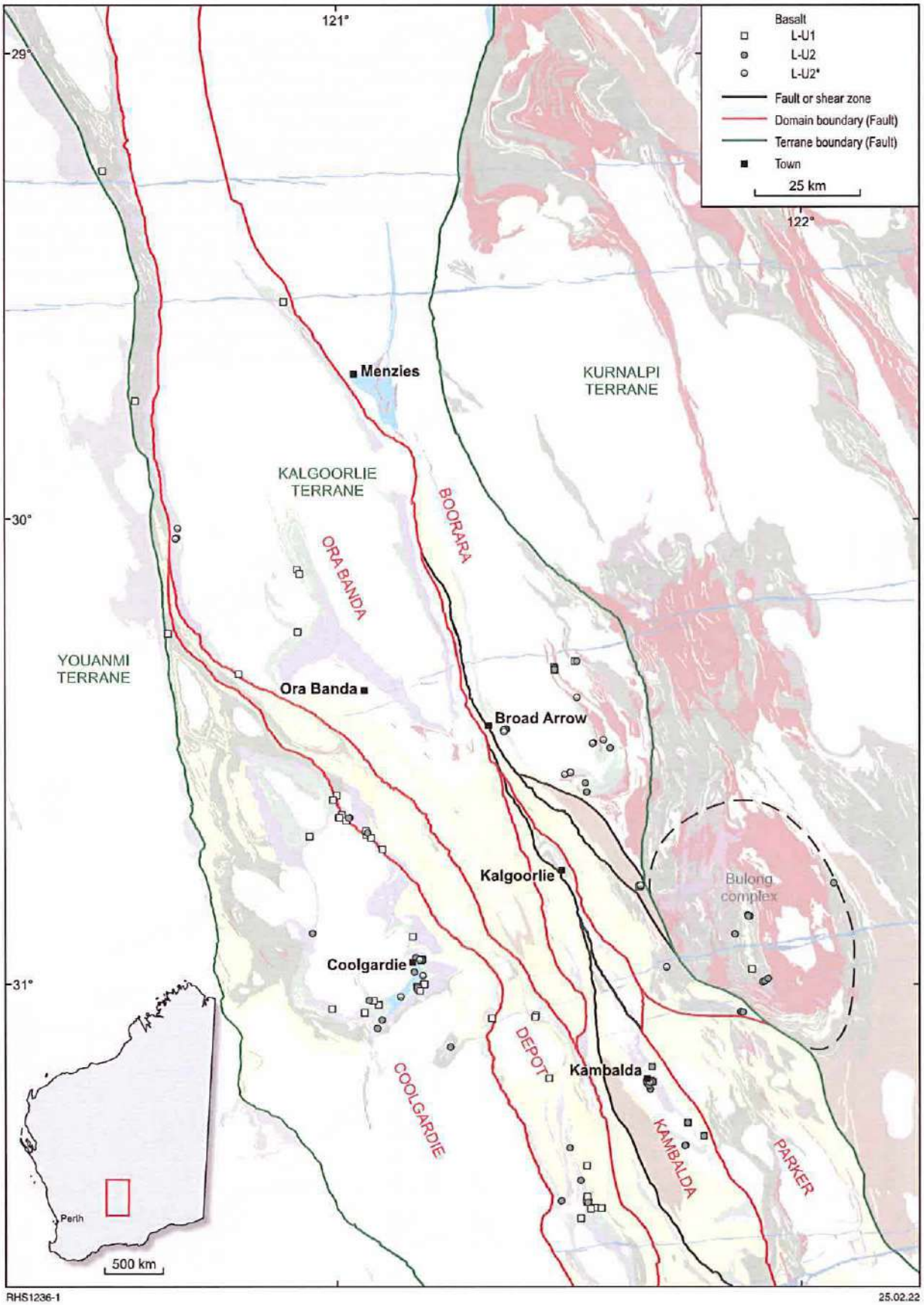


Figure 13. Geological map of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology and the distribution of primitive L-U geochemical units (see Figure 3 for fault names)

Table 3. Geochemical units and relative abundance within selected major mafic intrusions

Intrusion		L-U1	L-U2	L-U4	L-U5	L-U7	L-C	L-S3	L-S4	H-HMg#	H-LMg#
Condenser Dolerite	(n=28)	7%	11%	57%		21%	4%				
Golden Mile Dolerite (GMD)	(n=100)		32%	26%	37%	2%	2%		2%		
Gabbroic GMD	(n=85)		31%	27%	40%	2%					
Basaltic GMD in the Black Flag Group	(n=21)		42%	29%	29%						
Mount Pleasant Gabbro	(n=40)	3%	71%	5%	18%		3%				
Oak Hill Gabbro	(n=17)									41%	59%
Powder Gabbro	(n=23)									43%	57%
Stony Hill Dolerite	(n=19)	16%	52%	5%	11%			16%			
Three Mile Dolerite	(n=17)						100%				
Triumph Gabbro	(n=6)			33%	17%	50%					

Strongly contaminated LTB–L-S

L-S units are tentatively grouped as LTB magmas, but compared with other LTB magmas, show higher Th/TiO_2, Th/Nb, and Th/Zr ratios (Figs 6a, 16), interpreted to reflect very high degrees of crustal contamination. As discussed earlier, L-S1 and L-S2 are so depleted in incompatible trace elements that it is difficult to determine, if they relate to either LTB or ITB bulk sources. Although the most primitive members do appear to originate in the LTB field, they extend to higher $Mg^{\#}$ (77 and 69 respectively) and Ni and Cr concentrations than all other LTB (Fig. 11) and most ITB, with I-S1 being the only ITB unit having similarly primitive compositions. In most bivariate incompatible trace element diagrams, L-S1 appears to lie at the primitive extension of the ITB I-S1–3 trend and the L-C trend (Figs 6, 16), suggesting it might be a primitive member of either trend, and in this respect possibly an ITB. The common stratigraphic association of L-S1 and L-S2 with ITB units and L-Ca (Fig. 9) supports either relationship. Plots of various incompatible trace elements against $Mg^{\#}$ (Fig. 18) show trends suggesting that L-S2 is simply fractionated L-S1 magma.

L-S3 (a and b) and L-S4 are minor units that are also transitional to ITB in terms of Th/TiO_2 ratio but are more clearly LTB in terms of TiO_2 vs Zr and Nb (Figs 6a, 16). They are all restricted to the upper basaltic parts of the stratigraphy, although there is no indication that they are petrogenetically related to each other, and probably derive their high Th/Zr character through different processes. L-S4 shows very strong enrichments in P_2O_5, TiO_2, Fe_2O_3, and in all incompatible trace elements (Fig. 11) and has compositional features resembling strongly fractionated tholeiitic magmas that locally form granophyric portions of intrusions. It is possible that this unit represents the local eruption of highly evolved magmas from a fractionating subvolcanic mafic sill similar to, for example, the Golden Mile Dolerite, although the extreme enrichments in Th are difficult to explain without an additional process. Such a process might include the assimilation of high-Th country rock or mixing with high-Th magmas.

L-S3a, which is entirely equivalent to the Athena Basalt, is less evolved than L-S4 and shows closer similarities with L-U and L-Ca units on all plots that do not involve Th. L-S3b shares very similar compositions with L-S3a but also shows considerably greater scatter in data.

Discriminating L-S

A plot of Al_2O_3/TiO_2 vs Th/Nb (Fig. 8) collectively distinguishes L-S1 and L-S2 from all other basaltic units, with L-S1 and L-S2 effectively separated from each other based on TiO_2 concentration (L-S1 <0.54 wt% TiO_2 <L-S2). The Al_2O_3/TiO_2 vs Th/Nb plot also uniquely identifies L-S3a, L-S3b, and L-S4 (based on Al_2O_3/TiO_2 range) from all basalt units except for I-S4, which can be uniquely separated from the L-S units on a plot of Th vs Zr (Fig. 16b).

Distribution of L-S

L-S1 forms the main component of the lower part of the Big Dick Basalt in the Vetters Subgroup. However, the main concentration of L-S1 and L-S2 is in the Coolgardie and Depot Domains. L-S2 is almost exclusively restricted to a small area in the eastern Depot Domain (A in Fig. 17), where it appears to be part of the lower basaltic component interleaved with komatiite (e.g. in BDDD0017). In the Coolgardie Domain (upper Coolgardie succession), L-S1 forms part of the upper basaltic section of the Hampton Hill Formation and is the sole component of the Gleesons Basalt, and appears to occupy a similar stratigraphic position in the Depot Domain (Fig. 9).

L-S3 (a and b) and L-S4 are minor units found only in the eastern part of the studied region in the Hannans Subgroup (L-S3a and L-S4) and Bulong complex stratigraphy (L-S3b) (Figs 9, 17), where they occur in the uppermost parts of the basalt-dominated stratigraphy. In the Hannans Subgroup, L-S3a overlies L-S4 and both are localized between the Paringa Basalt and the Black Flag Group.

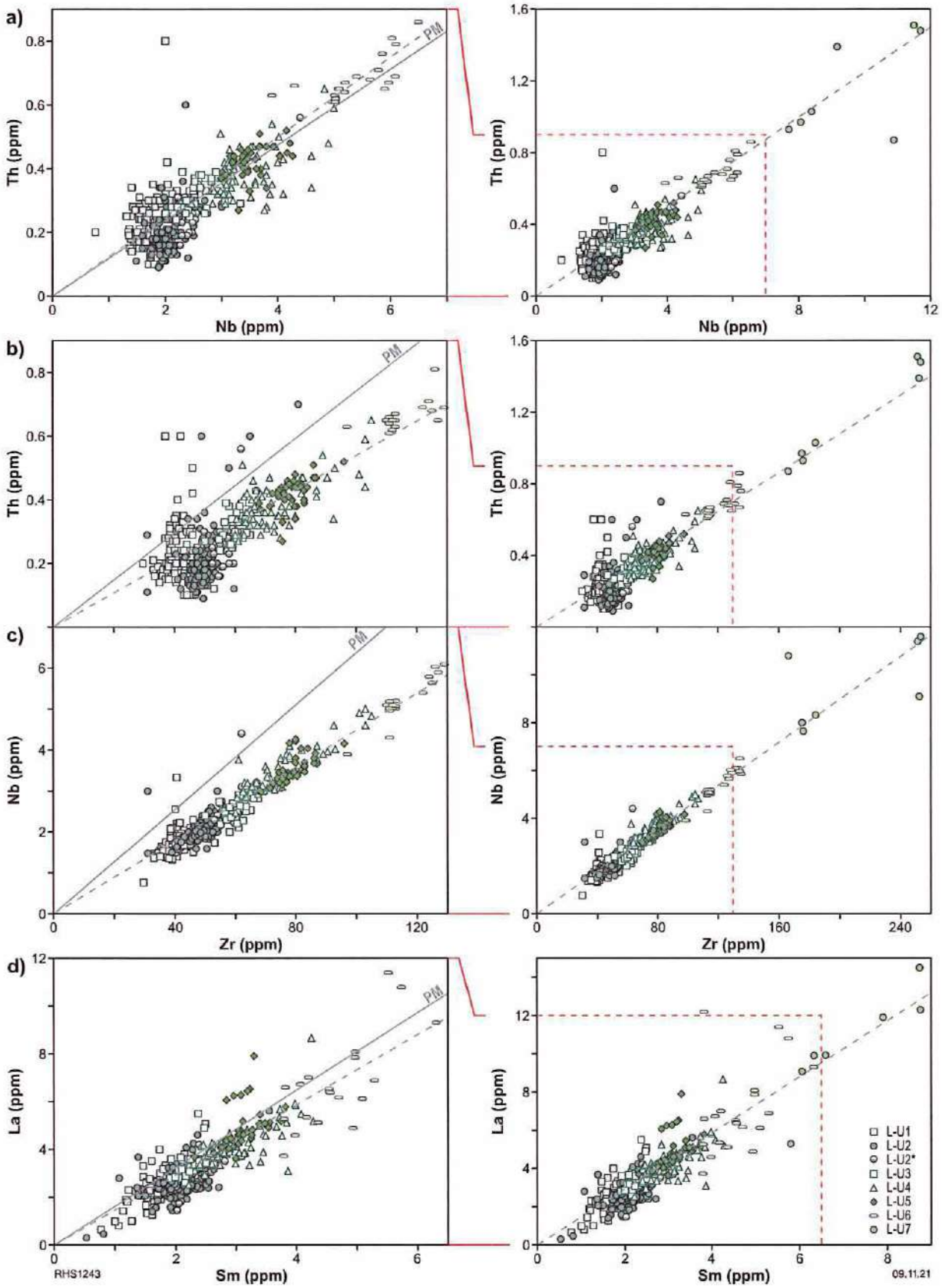


Figure 14. Variation in Th with Nb, Th and Nb with Zr and La with Sm for individual LTB geochemical groups and units (left-hand columns are expanded views)

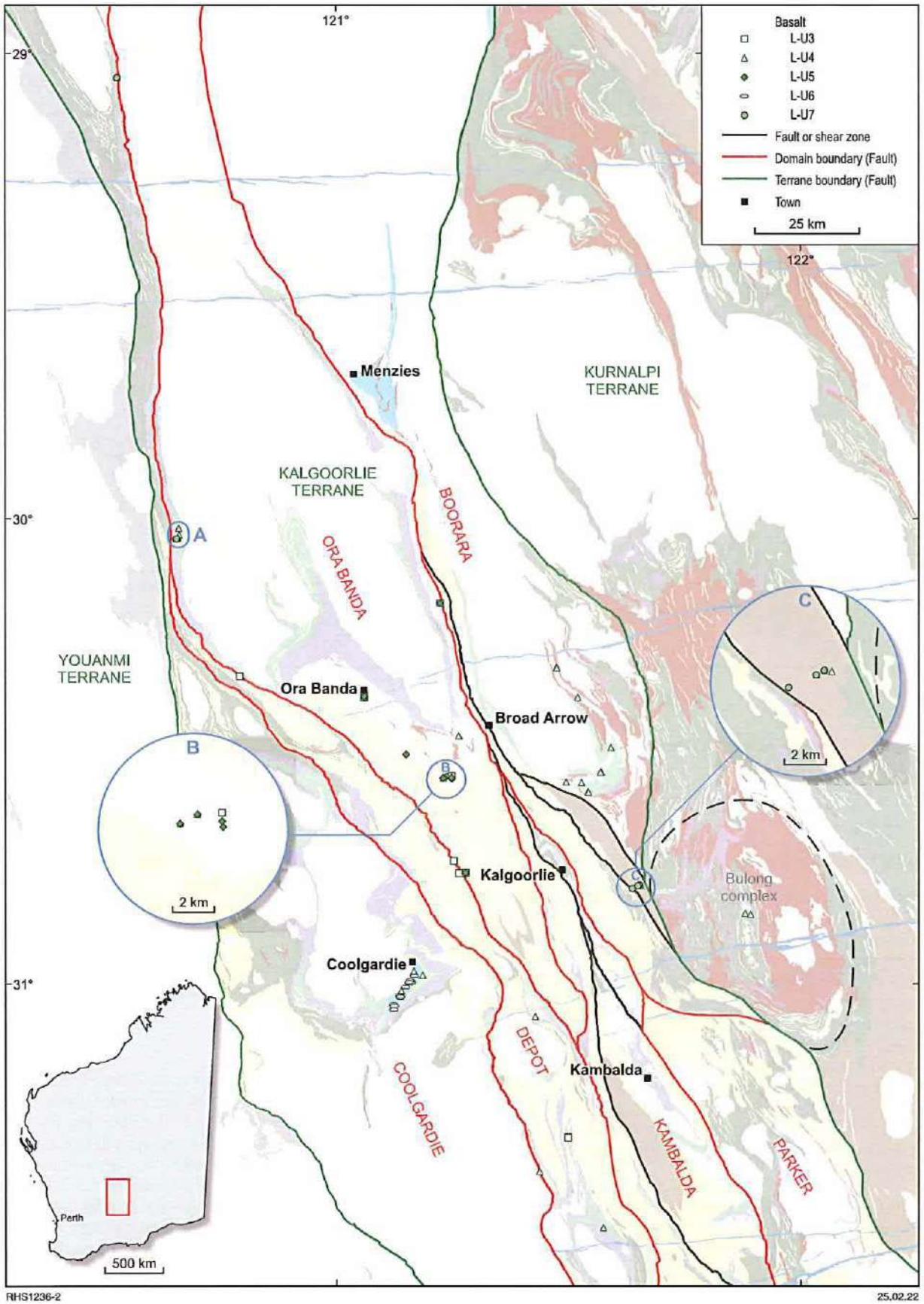


Figure 15. Geological map of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology and the distribution of L-U3 to L-U7 geochemical units (see Figure 3 for fault names)

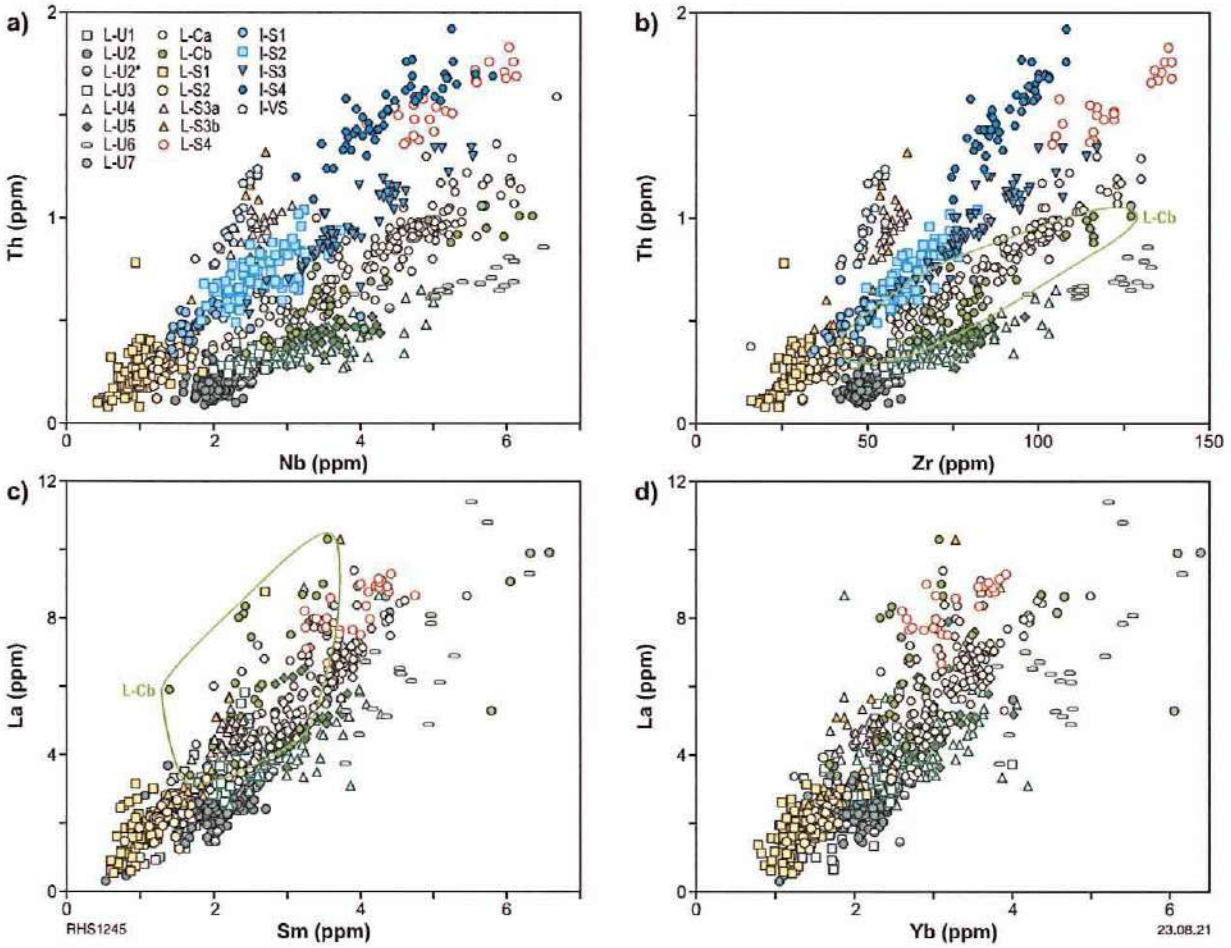


Figure 16. Variation in Th with Nb, Th with Zr, La with Sm, and La with Yb for individual LTB and ITB geochemical groups and units. Green line in b) and c) outlines the fields for L-Cb

Intermediate-Th basalts (ITB)

ITB cover a broadly similar major element compositional range as LTB, except in the case of P_2O_5, which at a given $Mg^{\#}$ is generally higher in abundance in ITB than in most LTB (Fig. 19). However, ITB groups seldom include samples with $Mg^{\#}$ <45, whereas several LTB units are dominated by such evolved samples. Conversely, samples of LTB with $Mg^{\#}$ >~62 (and MgO >10 wt%) are very rare, whereas the range for ITB extends to high-Mg basalts with $Mg^{\#}$ ~71 (MgO ~14.7 wt%). L-S1 has $Mg^{\#}$ as high as 77 (MgO to 17.9 wt% although some of these are probably partial olivine cumulates), although it is unclear whether these are LTB or ITB. Nevertheless, this highlights the possibility that some or most ITB might be derived through the fractionation of ultramafic (komatiitic) magmas (i.e. are really komatiitic basalts) rather than being direct basaltic melts of the mantle.

ITB by definition are typically significantly more enriched in incompatible trace elements, and have higher Th/Nb, La/Nb, and La/Sm ratios than most LTB at a given $Mg^{\#}$ (Fig. 20), suggesting the ITB magmas have incorporated a slightly greater amount of crustal material. As with the LTB, although the concentrations of incompatible trace elements increases with decreasing $Mg^{\#}$ (Fig. 19), the range in incompatible trace element ratios (La/Sm, La/Yb, Th/Nb, La/Nb) remains relatively constant (Fig. 20). This indicates that the addition of any crustal material to parental magmas has occurred before significant magmatic differentiation, which itself is controlled by minerals with very low incompatible trace element concentrations (e.g. olivine, pyroxenes, and plagioclase). Incompatible trace element plots show that the ITB form trends that extrapolate to primitive compositions more depleted in incompatible trace elements than the most primitive LTB units (L-U1 and L-U2), and similar to L-S1 (Fig. 7).

Discriminating ITB units

ITB units occur in the middle parts of all well-characterized stratigraphies (Fig. 9), although they have compositionally unique characteristics in each case. A plot of Th vs TiO_2 (Fig. 6a) serves to discriminate ITB units from LTB units and a plot of Al_2O_3/TiO_2 vs Th/Nb (Fig. 8) discriminates between most of the individual ITB units. However, I-S1, I-S2, and I-S3 consistently form continuous incompatible trace element trends (I-S1–3 trend). As with the L-U3–7 trend, it is possible that these relationships reflect the varying degrees of fractionation (I-S1 least fractionated, I-S3 most fractionated) of magmas sharing very similar bulk source compositions. They can be broadly separated based on their increasingly enriched character. However, I-S1 has a lower TiO_2 content (<0.6 wt%) than I-S2 (0.6–1.0 wt%), whereas I-S3 has a distinctly lower Al_2O_3/TiO_2 (<15 cf. >20) than both I-S1 and I-S2, as a result of plagioclase fractionation (Fig. 12).

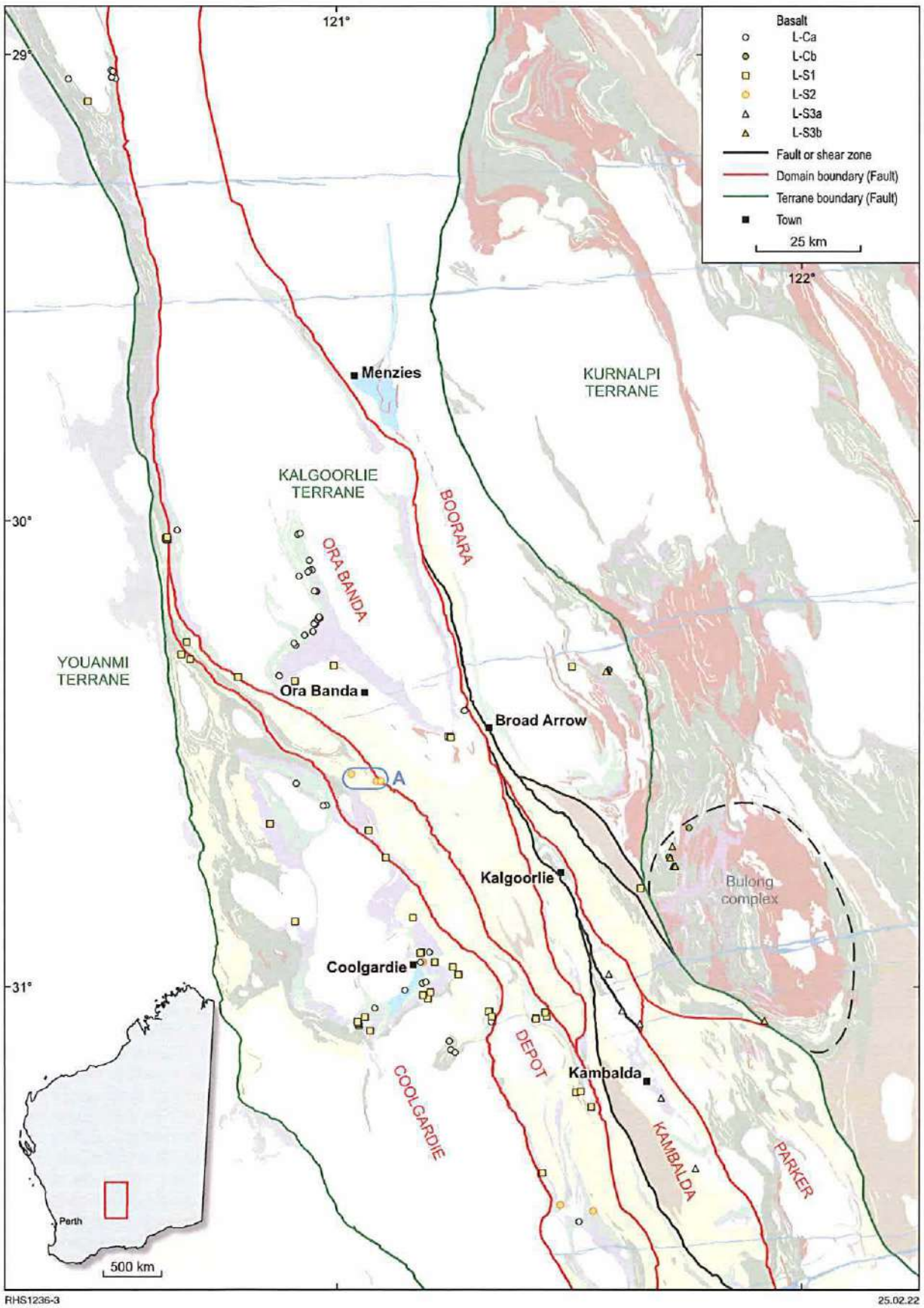


Figure 17. Geological map of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology and the distribution of L-C and L-S geochemical units (see Figure 3 for fault names)

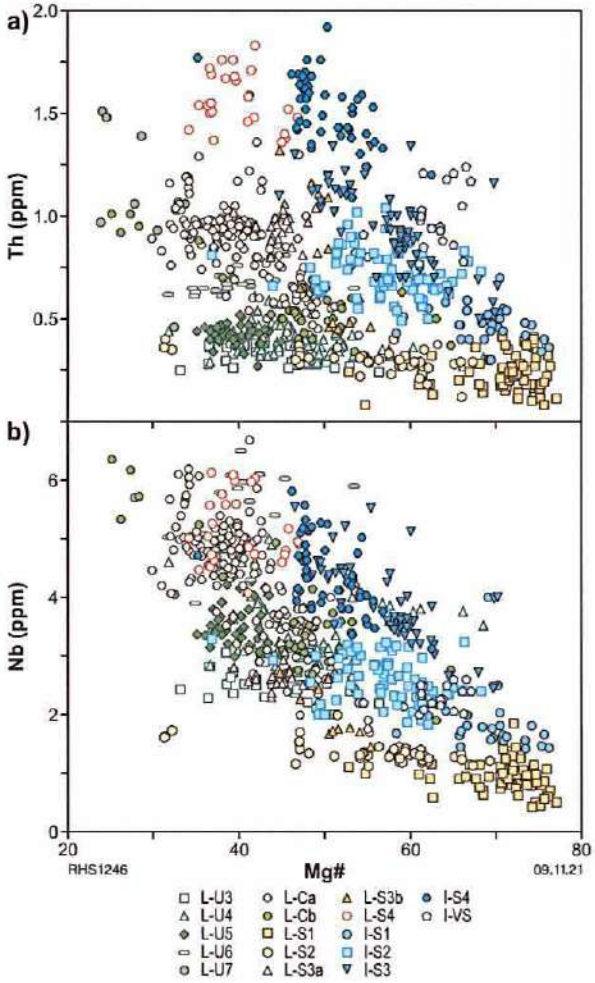


Figure 18. Variation in Th and Nb with Mg# for individual LTB and ITB geochemical groups and units

I-S4 falls at the enriched end of the I-S1–3 trend for some incompatible trace elements, but is a TiO_2-enriched (TiO_2 >1.13 wt%) unit readily distinguishable from other ITB and LTB by its combination of high Th concentration (>1 ppm) and moderately low Zr/Th ratio (median 0.017) (Figs 6a, 7). Like contaminated LTB units, such as L-S4, and uncontaminated units, such as L-U6 and L-U7, I-S4 shows many of the strongly fractionated tholeiitic compositional attributes typical of granophyre.

On many trace element variation diagrams, particularly those involving Th and La, I-VS shows trends towards low-Th outliers of HTSB (i.e. the H-LTh unit) and the komatiitic basalt unit KB3 (Fig. 21). Likewise, it has an MgO-rich composition (MgO 8.5–12.0 wt%; Mg# 67–61). The enrichment patterns that clearly distinguish I-VS reflect a scale of crustal contaminant similar to that required within HTSB. I-VS is best discriminated geochemically from H-LTh and KB3 by its unique position in plots of Zr or Y vs Th (slightly lower Th/Zr and Y/Th ratios). It is not known to be stratigraphically associated with these units or with any HTSB units.

Distribution of ITB units

The ITB are widely distributed throughout most of the study area (Fig. 22), although less commonly in the Boorara Domain and Bulong complex. Most commonly, they occur in the middle basaltic portions of the stratigraphy, overlying or in the upper portions of the komatiite-dominated stratigraphic levels (Fig. 9). I-S1 and I-S3 together dominate the basaltic component of the Hampton Hill Formation in the upper Coolgardie succession and in the stratigraphy of the Depot Domain. I-S2 is the dominant component of the Devon Consols Basalt in the Hannans Subgroup and in the upper part of the Big Dick Basalt in the Vetters Subgroup. I-S4 is the main component of the Bent Tree Basalt in the upper part of the Vetters Subgroup and is unique to that formation. Similarly, I-VS is unique to the Greenmont Basalt in the lower Coolgardie succession and the sole component of that formation.

High-Th siliceous basalts (HTSB)

The HTSB are perhaps the most geochemically distinctive mafic unit within the study region. Compared with LTB and ITB, the HTSB are characterized by higher Th/TiO_2 and La/TiO_2 ratios in particular, slightly higher Nb/TiO_2 and Zr/TiO_2 ratios (Fig. 6), and higher concentrations of Th and LREE at a given Mg# (but similar concentrations of Nb and Zr) (Fig. 23). They also show higher (and constant) ratios of Th and La against Nb and Zr (i.e. larger negative Nb anomalies) (Fig. 21), and higher SiO_2 contents at a given Mg# (Fig. 23). Such features reflect significant early contamination by particularly Th- and LREE-enriched crust. The other notable feature of HTSB is their generally very primitive composition with Mg# as high as ~77 (MgO ~21.7 wt%), likely reflecting derivation from an ultramafic parental magma. In both respects, these are similar to I-VS.

We have used an apparent gap in the Mg# range to subdivide HTSBs into:

1. A unit with Mg# between 63 and 78 (H-HMg#)
2. A unit with Mg# between 43 and 62 (H-LMg#)
3. A unit with low Mg# and consistently lower concentrations of Th (H-LTh).

Both H-HMg# and H-LMg# include examples of pillowed lavas. Therefore, the apparent break in the range of Mg# clearly does not simply reflect the distinction between lavas (lower Mg#) and cumulate intrusions (higher Mg#). However, the range to high Mg# (i.e. >70) and MgO >18 wt% in H-HMg# requires those lavas to include a significant cargo of olivine phenocrysts. It is also possible that many samples of H-HMg# that do not show clear indications of an extrusive origin are indeed cumulate or partial cumulate intrusions.

The petrogenesis of HTSB is less understood than that of LTB- and ITB-type basalts, probably reflecting the range of processes that can potentially achieve such enriched primitive compositions (e.g. Barnes et al., 2021), and also because the distinction between HTSB, KB, and primitive LTB and ITB can be very difficult. Barnes and Arndt (2018) and Barnes et al. (2021) use a plot of Al/Ti vs Mg# (Fig. 24) to distinguish komatiitic basalts derived through high-pressure fractionation (not involving plagioclase) of komatiitic magmas, from typical tholeiitic basalts. The tholeiites are believed to have been generated directly in the mantle as primary high-Mg basaltic partial melts that subsequently undergo low-pressure fractionation involving plagioclase. Most HTSB suites show high, mantle-like, Al_2O_3/TiO_2 ratios (~20), signifying little or no plagioclase fractionation) over much of their range in Mg#. Hence, their enriched compositions have been attributed to early contamination of komatiitic magmas in deep crustal chambers where pressures were too high to stabilize plagioclase. HTSB taken from the study area likewise show a remarkably constant Al_2O_3/TiO_2 ratio ~20.3 across a wide range of Mg# (from 77 to 54). Subsequent decreases in Al_2O_3/TiO_2 ratio to values as low as 17 at Mg# around 45 can be related to plagioclase fractionation and scattered values higher than 20 probably reflect plagioclase accumulation.

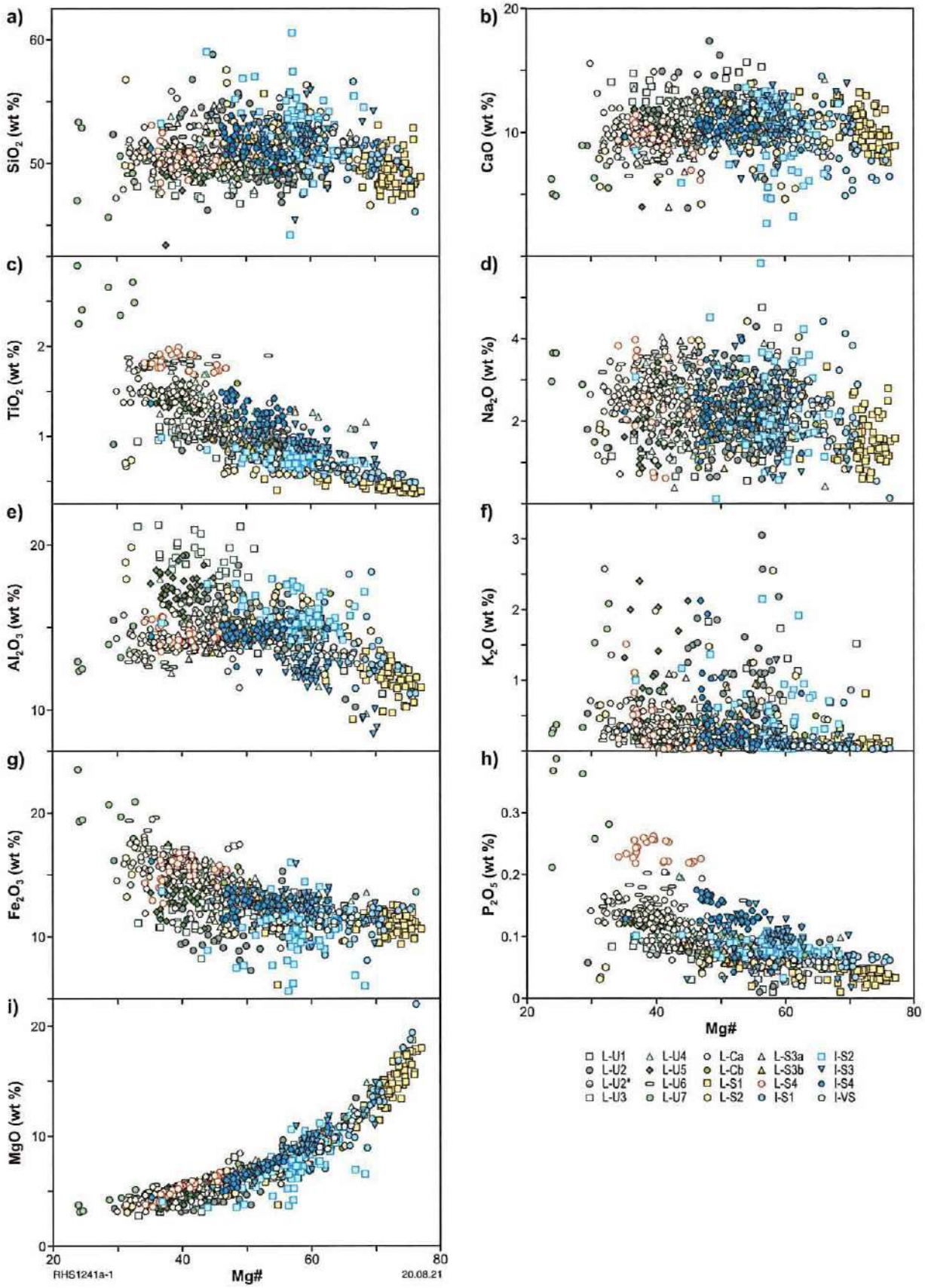


Figure 19. Variation in major and selected trace elements with Mg# for individual LTB and ITB geochemical groups and units

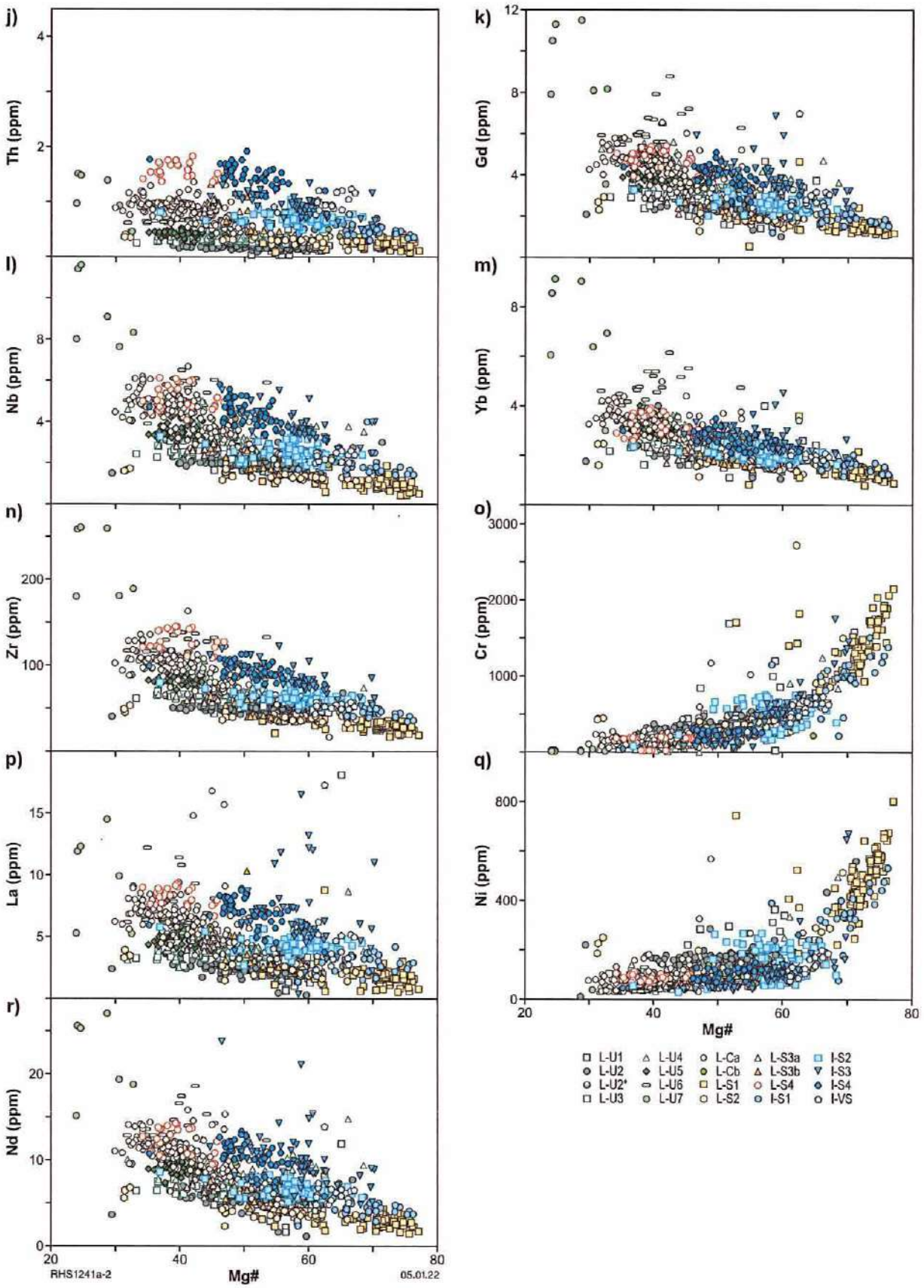


Figure 19. continued

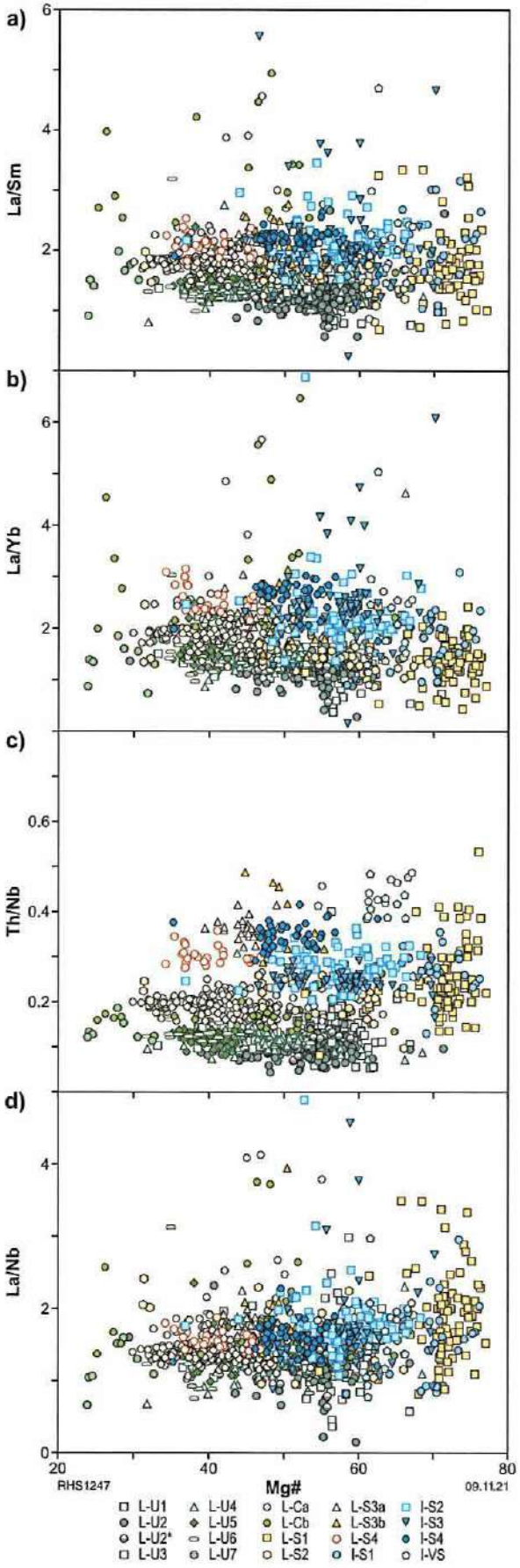


Figure 20. Variation in various trace element ratios with Mg# for individual LTB and ITB geochemical groups and units

Discrimination of H-HMg# and H-LMg#

These two groups of HTSB are discriminated from each solely based on Mg#. HTSB are completely discriminated from all LTB and ITB units based on their extremely high Th (e.g. Figs 4, 5, 23). Although >65% of all HTSB belong to the Paringa Basalt (Hannans Subgroup), there is no clear compositional distinction between these and samples found in the neighbouring domains.

Distribution of H-HMg# and H-LMg#

Most (>65%) of the HTSB samples presented here are from the Paringa Basalt, forming the upper part of the Hannans Subgroup (Fig. 9). This is a relatively unbiased reflection of the typically low abundance of such magmas within that part of the study area outside the Kambalda Domain. Nearly all the remaining HTSB samples lie to the north, in the Broad Arrow Subgroup or east, in the Parker Domain and Bulong complex (Fig. 25). HTSB do not appear to be common to the west of the Kambalda Domain, within the upper Coolgardie succession or the Vetters Subgroup. Outside of the Kambalda Domain, the stratigraphic position of the HTSB is difficult to constrain with the available data. It appears to relate to the upper parts of the stratigraphy at least within the Bulong complex. Within the Paringa Basalt, the relationship between H-HMg# and H-LMg# is clearly constrained within only one drillhole intersection examined (SE18), where H-LMg# broadly overlies H-HMg#.

Discriminating H-LTh

H-LTh has a very similar major and trace element concentration range to H-LMg#, including a similar range of Mg#, but has Th concentrations that lie between H-LMg# and ITB units (Figs 21, 23). One unit of komatiitic basalt (KB3) is notably more enriched in Th than the others (see below) and has major and trace element concentrations and ratios similar to those of H-LTh, but at higher Mg#.

Distribution of H-LTh

Although relatively rare, H-LTh shows a similar distribution to H-LMg#. Of the 16 H-LTh samples identified, five are found in the upper part of the Hannans Subgroup, and in cases where lithostratigraphic relationships can be inferred (e.g. in core from CD339W3), lie in the lower part of the Paringa Basalt, possibly mixed with or contaminated by the underlying Devon Consols Basalt.

Komatiitic basalts (KB) other than HTSB

As discussed above, in addition to the HTSB, which likely represent strongly contaminated komatiite, many ITB units, and LTB units such as L-S1 and L-S2, and possibly primitive L-U1, also combine high MgO contents, high Mg#, and Al_2O_3/TiO_2 ratios ~20 that possibly suggest a link to komatiitic parental magmas. There are several other groups (KB1, KB2, and KB3) that show similar characteristics but have probably remained strongly under sampled in our dataset because they typically occurred directly associated with komatiite-dominated horizons, which were generally sampled specifically for komatiites. Generally, these komatiitic basalts are primitive (Mg# >60, many >70), typically with MgO between 10 wt% and 18 wt% and Al_2O_3/TiO_2 close to 20 (17.5–22.5) (Figs 23, 24).

KB1 appears to bridge the compositional gap between the primitive LTB and L-Ca groups and komatiite. This does not necessarily imply a genetic relationship, as the trend simply reflects minimal contamination, although it clearly indicates a very similar mantle source composition. Similarly, KB2 potentially

bridges the compositional gap between komatiite and the IBT magmas forming the I-S1–3 trend. KB3 has incompatible trace element characteristics very similar to H-LTh and shares a similar geographical distribution, although unlike the latter, the concentrations of incompatible trace elements increase at constant, rather than decreasing, Al_2O_3/TiO_2.

The various identified komatiitic basalts are generally not difficult to distinguish geochemically from other mafic units (see below), and in particular from the HTSB. However, it is difficult to make a petrogenetic distinction between these units, other than the lack, in the case of the HTSB, of any clear and direct association with komatiite. KB and HTSB almost certainly are derived from ultramafic parental melts (in the case of KB, associated with those forming komatiites) and have undergone minor but variable (KB) or significant (HTSB) crustal contamination. It might be that differences between these units simply reflect aspects of the melt/crust-mixing process or the composition of the crustal component incorporated into the mantle-derived melts.

Distribution of KB units

One or more of the KB units occur throughout all the regional stratigraphies, although they are rarely sampled (Figs 9, 25). By definition, they are directly associated with komatiite or fine-grained peridotite and hence, are typically found around the middle komatiite-bearing parts of most stratigraphies.

Domain chemostratigraphic columns–Building the barcode

A significant finding of this study is that individual lithostratigraphic units, typically of formation level, seldom comprise a single geochemical unit (Fig. 9). Those that do, include the Greenmont Basalt (I-VS– lower Coolgardie succession), Gleesons Basalt (L-S1– upper Coolgardie succession), Paringa Basalt (HTSB– upper Hannans Subgroup), and Athena Basalt (L-S3a– upper Hannans Subgroup). Most stratigraphic formations comprise at least two distinct geochemical units, and in most cases, these units cannot be related back to a common parental magma composition. Thus, individual greenstone formations combine overlapping flow fields, each representing the products of a discrete eruptive event and/or eruptive centres supplied by genetically unrelated magma sources. This potentially makes some formation boundaries arbitrary reflections of changing source eruptions. Nevertheless, the subdivision of the regional (domain) stratigraphy into three levels (lower, middle, and upper basalts) reflects broader changes in magma compositions (see below) and provides useful markers of relative stratigraphic level.

Because textural or mineralogical features often cannot effectively discriminate between compositionally distinct basaltic units, mapping the discrete units at a stratigraphic 'Member' level might not be possible without a level of geochemical sampling density even greater than employed here.

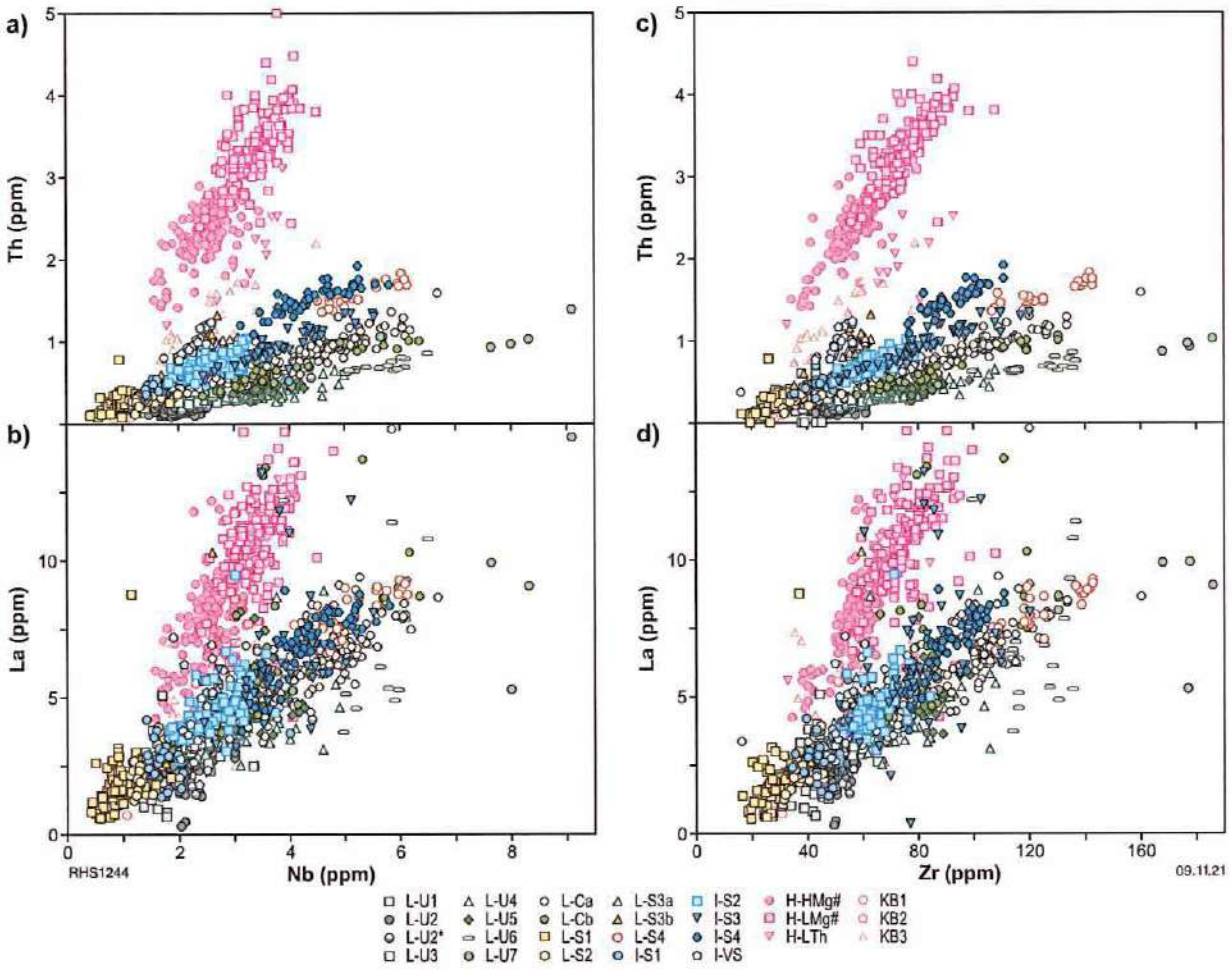


Figure 21. Variation in Th and La with Nb and Zr, for individual LTB, ITB, HTSB and KB geochemical groups and units

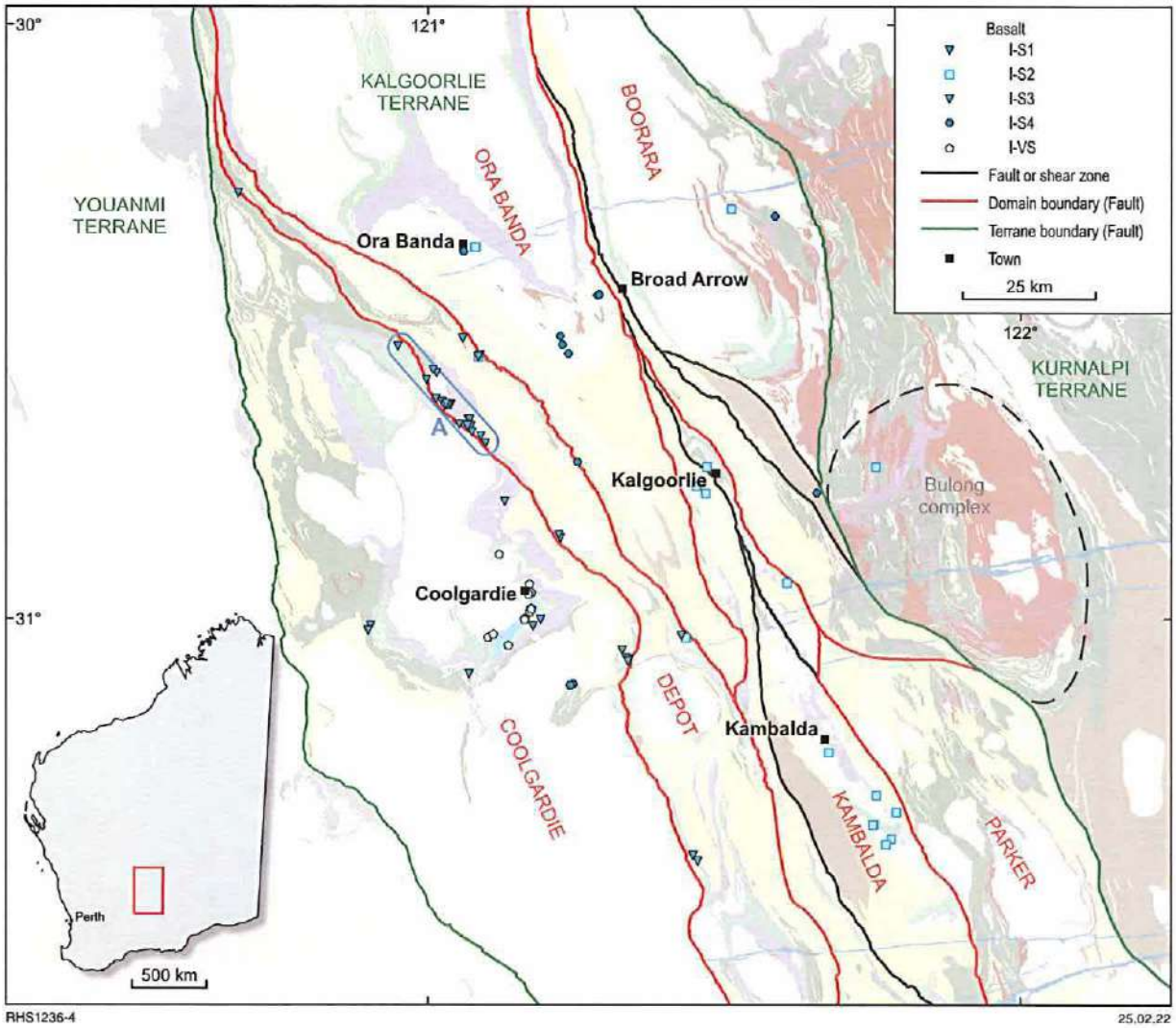


Figure 22. Geological map of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology and the distribution of I-S and I-VS geochemical units (see Figure 3 for fault names)

This would almost certainly not add any clarity to the potential ambiguities involved in identifying higher-order lithostratigraphic boundaries. This has implications for using geochemistry to infer stratigraphic position because, except in very rare cases, a single or low number of analyses will not be sufficient to uniquely identify a lithostratigraphic position. However, more comprehensive sampling, particularly if it incorporates the underlying and overlying stratigraphic units almost certainly will identify a unique lithostratigraphic position. Here, we describe the range of basaltic geochemical types forming various evolutionary stages of the regional stratigraphies.

Coolgardie Domain

Immediately apparent is that there are significant geochemical differences between the lower and upper stratigraphic successions of the Coolgardie Domain (Fig. 9). Here, the lithogeochemistry provides a clear indication that the upper and lower successions do not represent a structural repetition, although it remains plausible that they might be in thrust contact. The lower Coolgardie succession almost certainly should not be considered part of the Coolgardie Subgroup, as is presently the case.

Lower Coolgardie succession

The lower Coolgardie succession is presently limited to outcrops around the eastern and southern margins of the Calooli Monzogranite (Fig. 26). As currently mapped, the Burbanks Formation, forming the lowest exposed unit, comprises approximately:

- 56% L-U6
- 26% L-U4
- 18% primitive L-U2*.

The strongly TiO_2-enriched L-U6 unit is virtually unique to the Burbanks Formation. L-U6 is found at only one other locality within the study region, near Davyhurst in the Ora Banda Domain to the north-northwest, immediately east of the Zuleika Shear Zone (A in Figs 15, 27). There, it is also associated with interlayered L-U2* and L-U4, as well as another Ti-rich LTB unit, L-U7. It seems quite plausible that this exposure also belongs to a lower greenstone succession, possibly correlated with the Burbanks Formation.

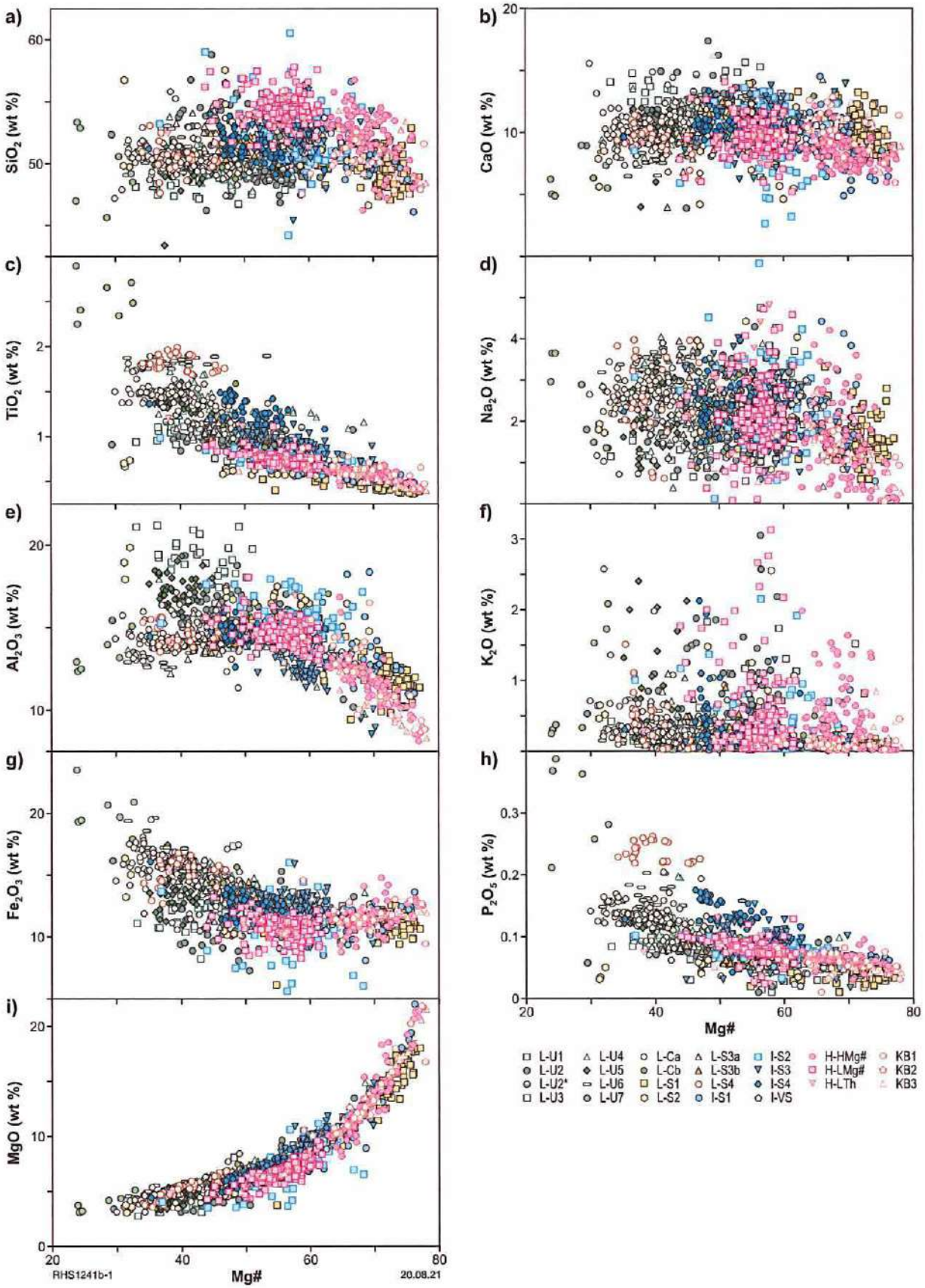


Figure 23. Variation in major and selected trace elements with Mg# for individual LTB, ITB, HTSB and KB geochemical groups and units

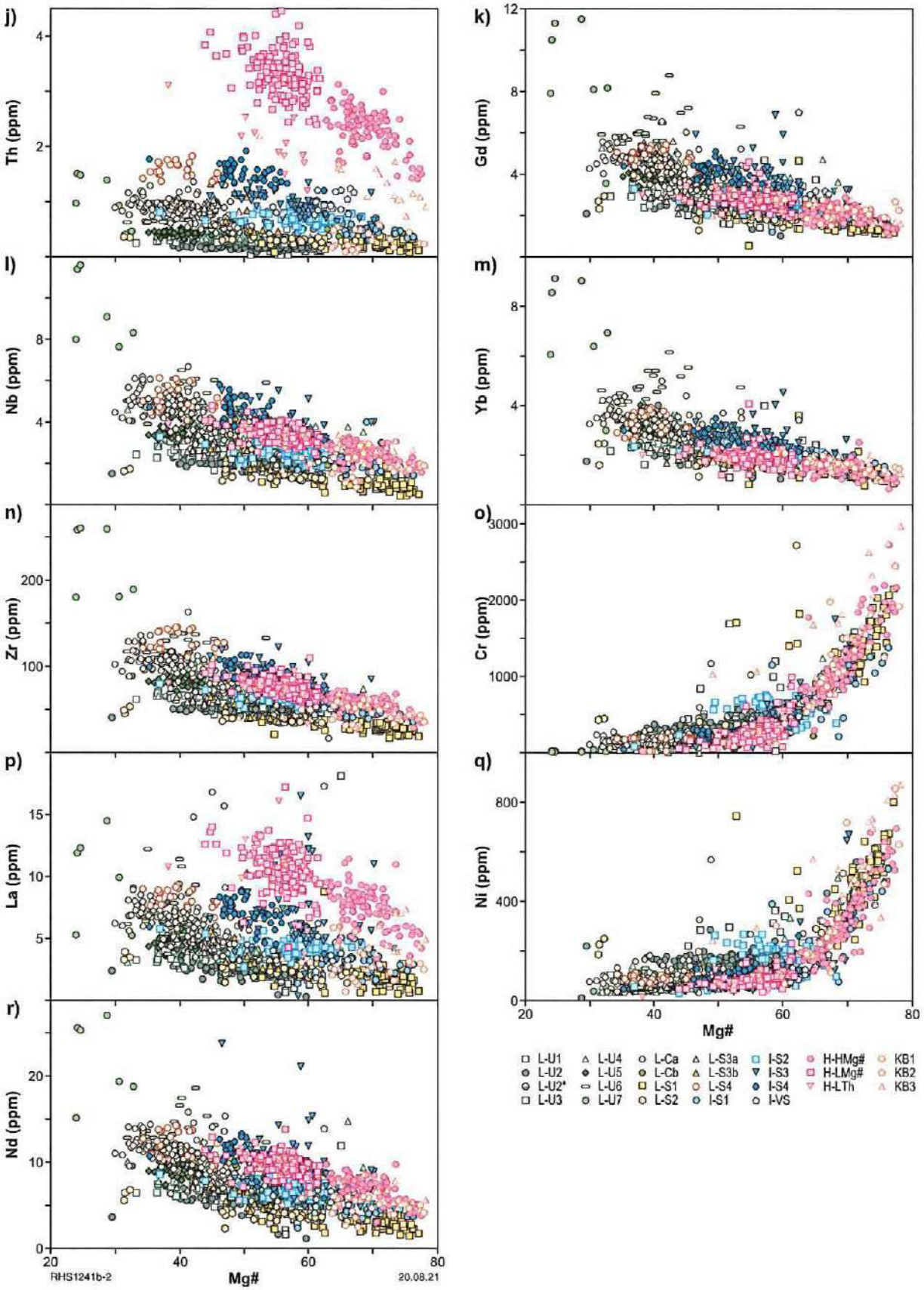


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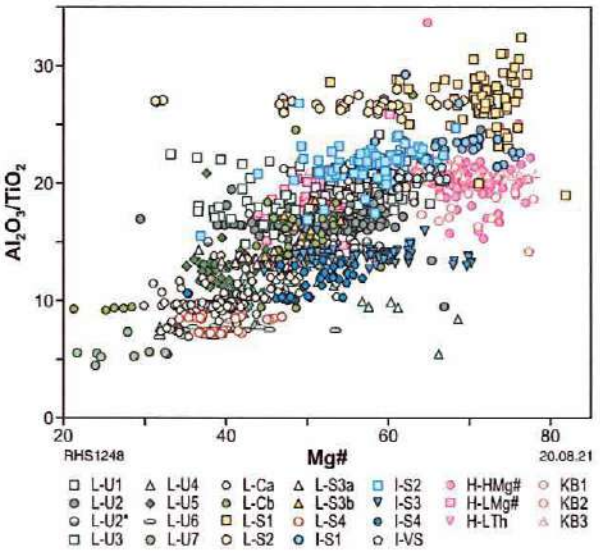


Figure 24. Variation in Al_2O_3/TiO_2 with $Mg^{\#}$ for individual LTB, ITB, HTSB and KB geochemical groups and units

The Little Blow Member of the Burbanks Formation, within the limits of the presently available data (3 samples only), consists of primitive L-U2* (2 samples) and L-U4 (one sample).

The Greenmont Basalt (possibly a sill) overlies the Burbanks Formation and is entirely composed of I-VS, which is unique to this stratigraphic unit.

Our dataset contains only three samples of the basaltic component of the typically ultramafic Brilliant Formation, including two samples of KB1 and a single sample of KB2.

Upper Coolgardie succession (Coolgardie Subgroup)

The Lindsays Basalt at the base of the upper Coolgardie succession comprises approximately:

- 42% L-Ca
- 36% primitive L-U1
- 13% primitive L-U2
- 9% primitive L-U2*.

Where stratigraphic relationships can be inferred within the Lindsays Basalt, it appears that primitive basalts L-U1, L-U2, including minor L-U2*, occur mainly in the lower stratigraphic regions, underlying a sequence dominated by L-Ca. Clear identification of Lindsays Basalt in the field, and in particular, distinguishing it from basaltic units of the overlying mafic–ultramafic Hampton Hill Formation, is hampered by poor outcrop and is potentially problematic. Current GSWA mapping shows vast areas in the Coolgardie region surrounding the Bali–Calooli Monzogranite (including regions of the Depot Domain) as comprising either 1) interleaved Lindsays Basalt and Hampton Hill Formation or 2) Hampton Hill Formation ranging from basalt to komatiite. Where Lindsays Basalt can be confidently identified, including where basalt directly overlies the lower Coolgardie sequence, it is volumetrically dominated by L-Ca. Since L-Ca has not been found in sequences clearly belonging to the Hampton Hill Formation, it can be regarded as a reliable geochemical signature of Lindsays Basalt when identified in the Coolgardie region. The Three Mile Dolerite, which intrudes Lindsays Basalt, is likewise almost entirely formed of L-Ca (Table 3). It is unclear, if minor L-U2* found in the lower Lindsays Basalt truly belongs to that formation or represents isolated (tectonically interleaved?) samples of the underlying Burbanks Formation.

The Hampton Hill Formation, as presently defined on GSWA mapping, is very complicated structurally and lithologically, and consequently geochemically. Overall, it is dominated by komatiite, but locally contains abundant basaltic geochemical components that also characterize the overlying Gleesons Basalt (L-S1), suggesting the stratigraphic boundaries are either tectonically or magmatically interleaved, or both. In addition, there are regions along the northeastern part of the Bali Monzogranite and the eastern side of the Calooli Monzogranite, where intercalated Lindsays Basalt and Hampton Hill Formation (undivided) (GSWA terminology) underlies the intercalated mafic and ultramafic components of the Hampton Hill Formation. This in turn, underlies the Gleesons Basalt, representing the upper part of the upper Coolgardie sequence. The basalt components of the intercalated mafic and ultramafic Hampton Hill Formation in these regions are very distinctive ITB units (I-S1 and I-S3, with interleaved KB1-3) (A in Figs 22, 26). It seems very likely that these represent a previously unrecognized ITB formation that at least locally separated the Hampton Hill Formation from Gleesons Basalt, and is now tectonically interleaved with the underlying Hampton Hill Formation.

Nevertheless, taking the Hampton Hill Formation as presently mapped, our data indicate a range of possible basaltic geochemical components including:

- 42% L-S1 – 25% of which, in the upper section is probably Gleesons
- 16% primitive L-U1
- 15% L-S2
- 11% I-S3 – all of which might constitute an upper ITB unit
- 11% KB3 – all of which might constitute an upper ITB unit
- 5% I-S1 – all of which might constitute an upper ITB unit.

Where inferred to lie in the upper part of the Hampton Hill Formation, L-S1 is possibly a tectonic slice of, or is magmatically transitional to, the overlying Gleesons Basalt. Where inferred to lie in the lower portions of the formation, L-S1 is often associated with L-S2 and more clearly forms true basaltic components of the Hampton Hill Formation.

L-S1 is also the sole geochemical unit within the stratigraphically overlying Gleesons Basalt, and elsewhere only occurs at the top of the Depot Domain stratigraphy.

Depot Domain (superseded)

Current GSWA mapping no longer recognizes the Depot Domain, which has been re-incorporated into the Coolgardie Domain. However, there remains significant confusion in this region as lithostratigraphic codes (GSWA mapping) appear to have been arbitrarily extrapolated from either the Coolgardie or Kambalda Domain or are generic Eastern Goldfields Superterrane codes. The former Depot Domain restricted the Coolgardie Domain to the area west of the Kunanalling Shear Zone. The limited outcrops in the Depot region do not allow the same degree of confidence in the broad stratigraphic context of the various geochemical components as it does in many of the other domains. However, from the previous discussion, it is quite clear that the upper Coolgardie succession and the superseded Depot Domain stratigraphy show very close lithogeochemical similarities (Fig. 9).

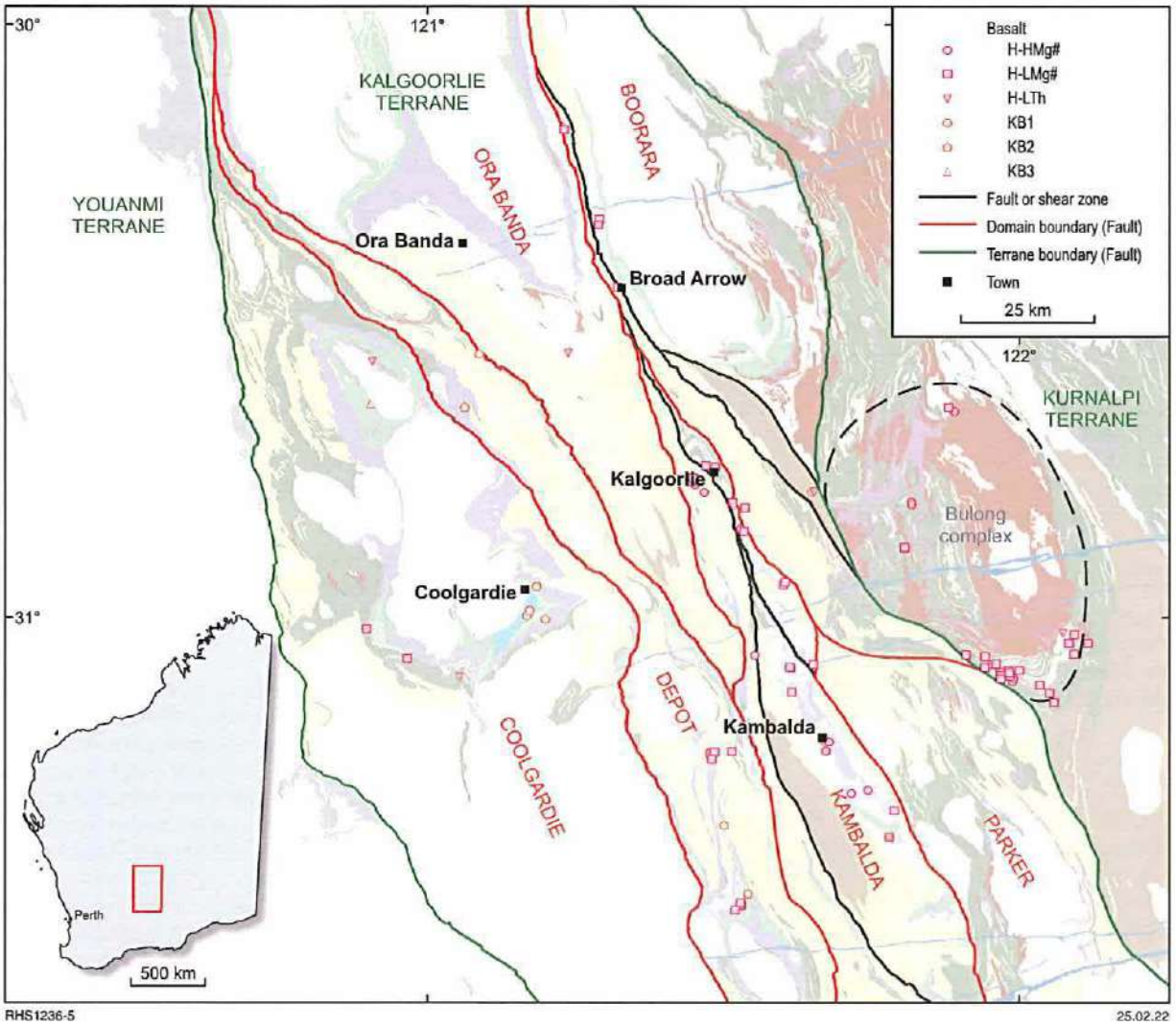


Figure 25. Geological map of the study area within the Kalgoorlie Terrane showing a simplified version of the current interpreted bedrock geology and the distribution of HTSB and KB geochemical units (see Figure 3 for fault names)

For example, the available data from the superseded Depot Domain indicate:

- A lower basaltic stratigraphy dominated by LTB with a high L-U1/L-U2 ratio (2.4) similar to that of Lindsays Basalt (1.6), locally containing L-Ca (12% of the lower basalt), and including L-S1 in some areas
- Apart from minor occurrences in the upper Coolgardie succession, L-S2 is unique to the eastern part of the former Depot Domain, with a concentration of 28 samples from three separate drillholes (BDDD0017, BDDD0002, and PXGD0089), in one hole apparently interleaved with komatiite (B in Fig. 26)
- A middle stratigraphy with a basaltic component almost exclusively of I-S3, either magmatically or tectonically interleaved with komatiite. This is overlain by a unit dominated, at least in one drillcore (BDDD0006: Fig. 26), by I-S1 (85%) and locally interleaved with I-S2 (15%), overlain by a discrete unit with L-S1 compositions identical to Gleesons Basalt of the upper Coolgardie succession.

These very close chemostratigraphic similarities with the upper Coolgardie succession strongly support re-incorporation of the Depot Domain into the Coolgardie Domain. However, there are a few anomalies. For example, I-S2 overlies I-S1 in drillcore (BDDD0017) immediately west of the Zuleika Shear Zone (approximately 10 km to the north of the Powder Gabbro) (C in Fig. 26), in what is inferred here to be the middle to upper part of the stratigraphy, and is otherwise unique within the Coolgardie Domain. I-S2 is a significant component of the stratigraphy of the Ora Banda and Hannans Subgroups and forms part of the I-S1–3 trend. We cannot determine whether its presence in this area reflects a stratigraphic transition or is simply part of a fractionation sequence from L-S1 to L-S2, although the latter process cannot be demonstrated in any other regional stratigraphy. In addition, in the Kundana region near Bullock Hole (D in Fig. 26), immediately west of the Zuleika Shear Zone that marks the boundary between the Coolgardie Domain and the Ora Banda Domain (in the north) or Kambalda Domain (in the south), stratigraphy identical to the upper part of the Vetters Subgroup is intersected in drillcores (PGDD15001, MEDD148, MEDD144, and FLRD04) and outcrop. Presumably, either the (GSWA) trace of the

Zuleika Shear Zone in the region is incorrectly located or the Ora Banda and Coolgardie stratigraphies impinge. Likewise, between the Connolly Gneiss and the Maloneys Dam Granite, approximately 13 km southeast of Londonderry Siding (E in Fig. 26), the greenstone stratigraphy is dominated by an I-S4–L-Ca association that is reminiscent of the upper part of the Vetters Subgroup and otherwise unique within the upper Coolgardie succession. However, this observation is based on limited data (7 analyses).

Ora Banda Domain

The Wongi Basalt is the basal formation of the Vetters Subgroup and comprises 61% L-Ca and 39% primitive L-U1. The relationships between the two geochemical components cannot be confidently assessed based on the current data (no mutual contacts are observed), although outcrop distribution suggests that L-Ca lies near the top of the sequence. The overlying Missouri Basalt is comprised entirely of L-Ca (Fig. 9) and it is plausible that the L-Ca component of the Wongi Basalt actually belongs to the Missouri Basalt or represents very fine-grained feeder dykes or sills related to the Missouri Basalt.

The Big Dick Basalt overlies the ultramafic component of the Vetters Subgroup (the Walter Williams Formation) that separates it from the Missouri Basalt and appears to comprise two geochemically distinct units:

- 55% I-S2
- 45% L-S1.

The I-S2 unit that forms the upper part of the Big Dick Basalt also forms a small component of the upper Coolgardie succession, potentially indicating close chemostratigraphic similarities between the lower and middle parts of the Vetters Subgroup and upper Coolgardie succession.

Overlying the Big Dick Basalt, the Bent Tree Basalt is overwhelmingly dominated (89%) by I-S4, which appears regionally unique to this formation. Six samples (11%) of fine-grained L-Ca from drillcore KND15001, collared within the Bent Tree Basalt, are tentatively assigned to this formation.

At the top of the Vetters Subgroup, the Victorious Basalt comprises three geochemically distinct units:

- 54% L-U5
- 39% L-U3
- 7% L-U4.

At least two of these geochemical units appear interlayered in several drillcores (e.g. L-U5 and L-U4 in the case of PMPD0080) (B in Fig. 27). L-U3 and L-U5, in particular, are strongly enriched in Al_2O_3 and this is locally expressed in a distinctive plagioclase megacrystic or even glomeroporphyritic texture in outcrop.

Kambalda Domain

The basal unit of what has traditionally been defined as the Hannans Subgroup is the Lunnon Basalt, which is almost entirely composed of primitive L-U1 (~43%) and L-U2 (~65%) (Fig. 9), with rare (<2%) samples of komatiitic basalts (KB1 and 2), particularly in the higher stratigraphic levels close to the onset of komatiitic magmatism. The Lunnon Basalt appears restricted to the southern half of the Kambalda Domain (Fig. 28), not having been found in outcrop or drillcore in the Kalgoorlie region. Where L-U1 and L-U2 are found in the same drillcore (e.g. in DDH KD1029), each typically dominates specific intervals (a flow or sequence of flows) but no consistent stratigraphic relationship was noted; hence, L-U1 and L-U2 magmatism is considered to be synchronous.

Apart from the notable absence of L-Ca from the Hannans Subgroup, a clear difference between the Lunnon Basalt and the primitive L-U rich portions of the Lindsays Basalt (and Wongi Basalt in the Vetters Subgroup) is the much lower L-U1/L-U2 ratio (0.49) in the Lunnon Basalt (cf. 1.6 to 2.4 for the Lindsays Basalt), although this ratio varies widely between locations within the Kambalda Domain (Fig. 28).

The basaltic components of the Kambalda Komatiite (not shown) appear to be limited to samples of komatiitic basalt (KB2>KB1) and this distinguishes the formation from the Hampton Hill Formation of the upper Coolgardie sequence, which appears to include locally voluminous ITB (I-S) and L-S LTB components. However, this might reflect a better understanding of the stratigraphy of the Kambalda Domain.

Overlying the Kambalda Komatiite, the Devon Consols Basalt is almost entirely composed of I-S2 (84%), with ~16% komatiitic basalt (KB2>KB3) mainly found stratigraphically close to the lower contacts with the Kambalda Komatiite.

The Paringa Basalt is the only clearly stratigraphically defined occurrence of HTSB within the study region, although H-LMg# forms a significant component of the upper Highway Formation of the Broad Arrow Subgroup and both H-HMg# and H-LMg# are significant components of the stratigraphy of the Parker Domain and Bulong complex. H-HMg# and H-LMg# are equally abundant components (49% each) of the Paringa Basalt. Relationships established in drillhole SE18 (Fig. 28) suggest that H-LMg# broadly overlies H-HMg# and similar observations were made by Morris (1993). Close to the contact with the Devon Consols Basalt, H-LTh also forms a component of the Paringa Basalt, possibly reflecting either tectonic or magmatic mixing with I-S2 of the underlying unit.

Strongly contaminated units overlying the Paringa Basalt – i.e. L-S4 and Athena Basalt (L-S3a) – are a relatively minor but widespread component of the Kambalda Domain (Fig. 28). Both units overlie (apparently conformably) the Paringa Basalt, separating it from the Black Flag Group (Fig. 9), with the Athena Basalt (L-S3a) locally present throughout the southern part of the Kambalda Domain and L-S4 restricted to its northern part. In a narrow region of overlap, drillhole intersections (MSC27310-1) indicate that L-S4 is overlain by the Athena Basalt (Fig. 28).

Although younger than the Hannans Subgroup, fine-grained mafic rocks occur within the overlying Black Flag Group. Locally, they show peperitic contacts with volcaniclastic rocks of the Black Flag Group and have been interpreted as lavas or synvolcanosedimentary sills (McMann, 2018; Cas et al., 2019). Six samples were taken during the current study and a further 10 samples were taken during a more detailed analysis of the Golden Mile Dolerite (McMann, 2018). These samples are dominantly of L-U2 with other major components comprising L-U4 and L-U5. These three units also overwhelmingly dominate the Golden Mile Dolerite itself (Table 3), which has been interpreted as a major synvolcanosedimentary sill (McMann, 2018; Cas et al., 2019). The fine-grained and peperitic rocks are interpreted as marginal facies associated with the Golden Mile Dolerite. Similar fine-grained quenched intrusions are associated with other high-level layered mafic–ultramafic sills emplaced within the upper basaltic units of many of the domain subgroups and into the overlying Black Flag Group or stratigraphically equivalent volcanosedimentary units.

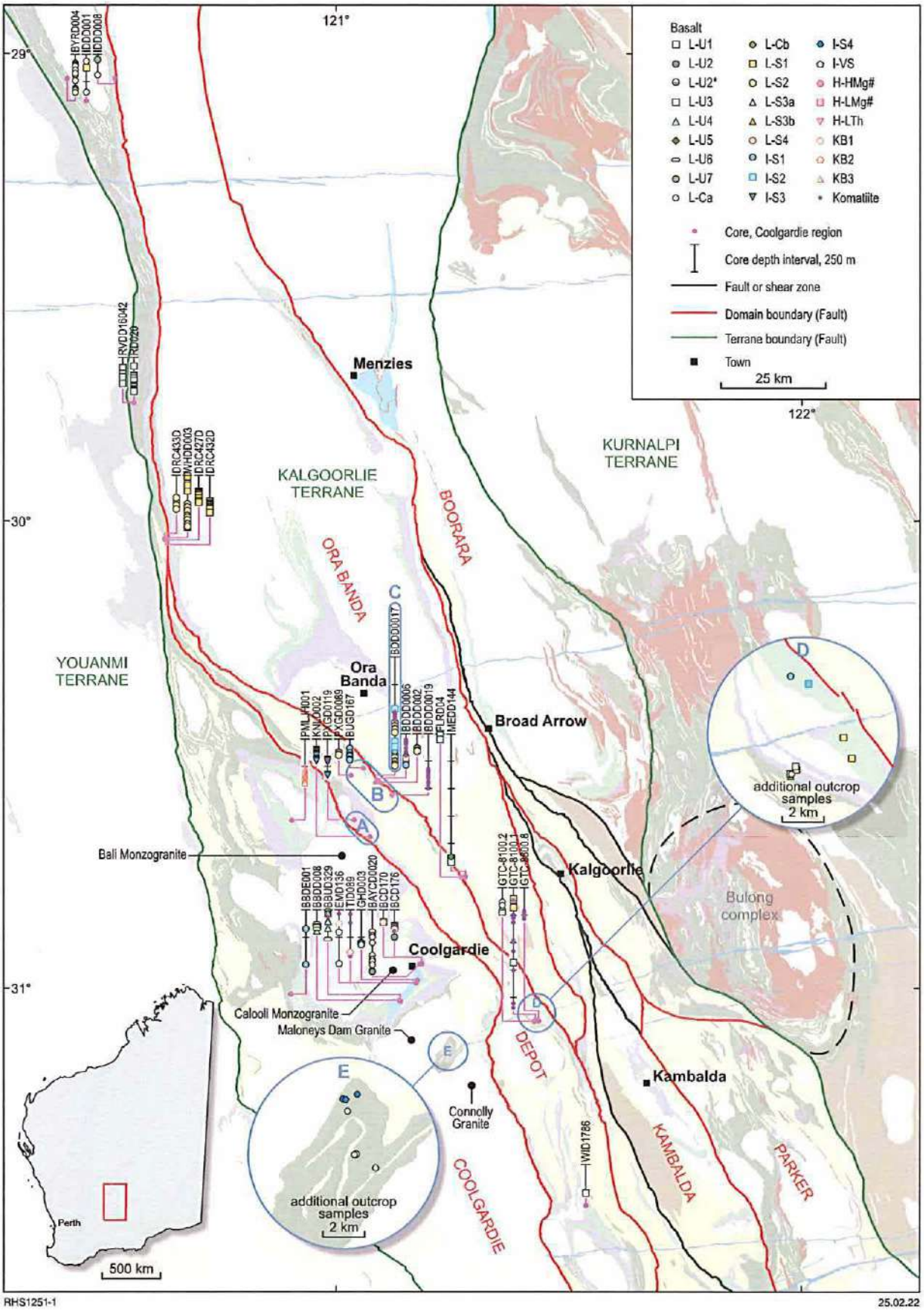


Figure 26. Geological map of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology and the distribution of diamond drillcore sample sites within the Coolgardie (and Depot) Domain (see Figure 3 for fault names)

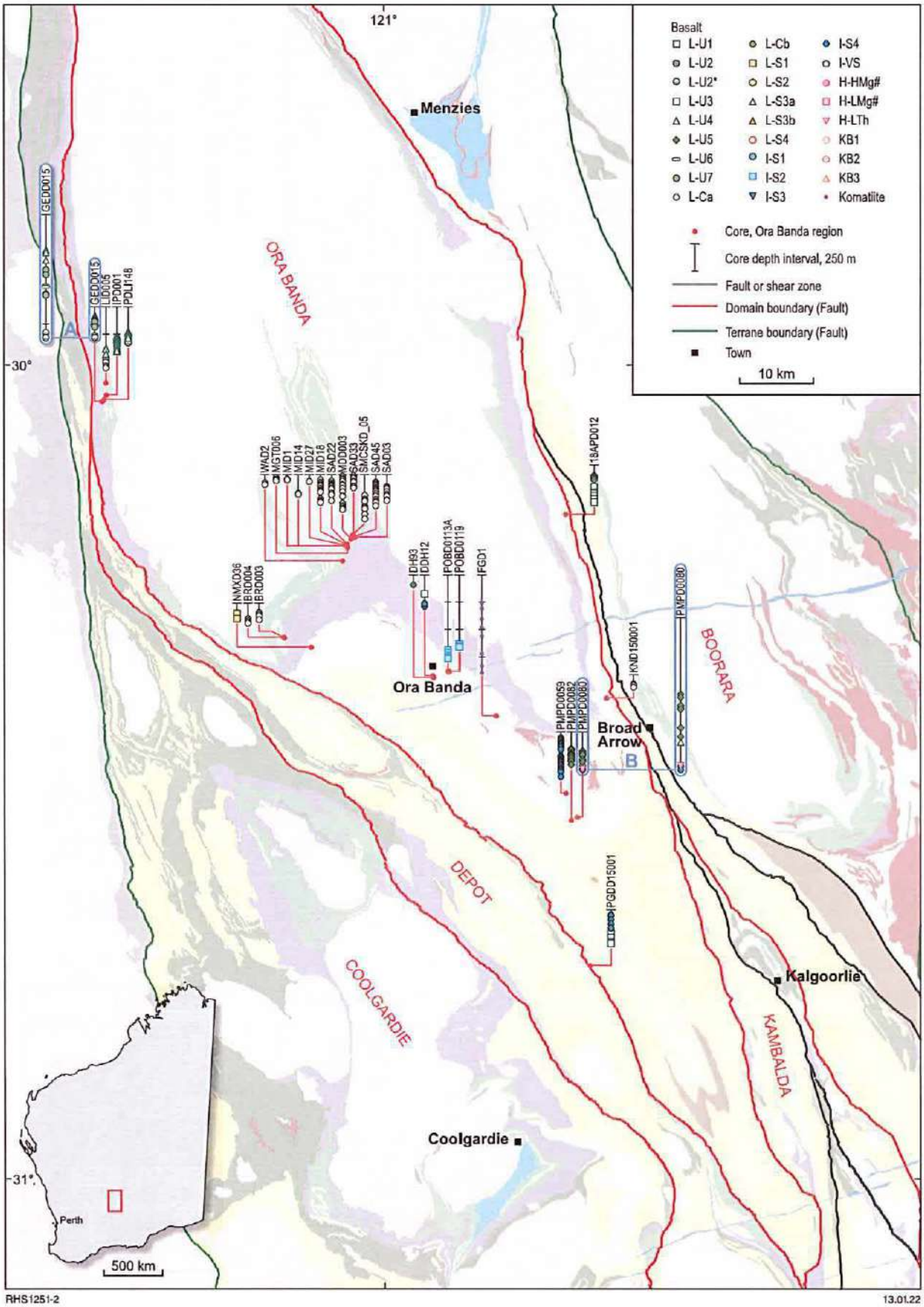


Figure 27. Geological map of part of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology and the distribution of diamond drillcore sample sites within the Ora Banda Domain (see Figure 3 for fault names)

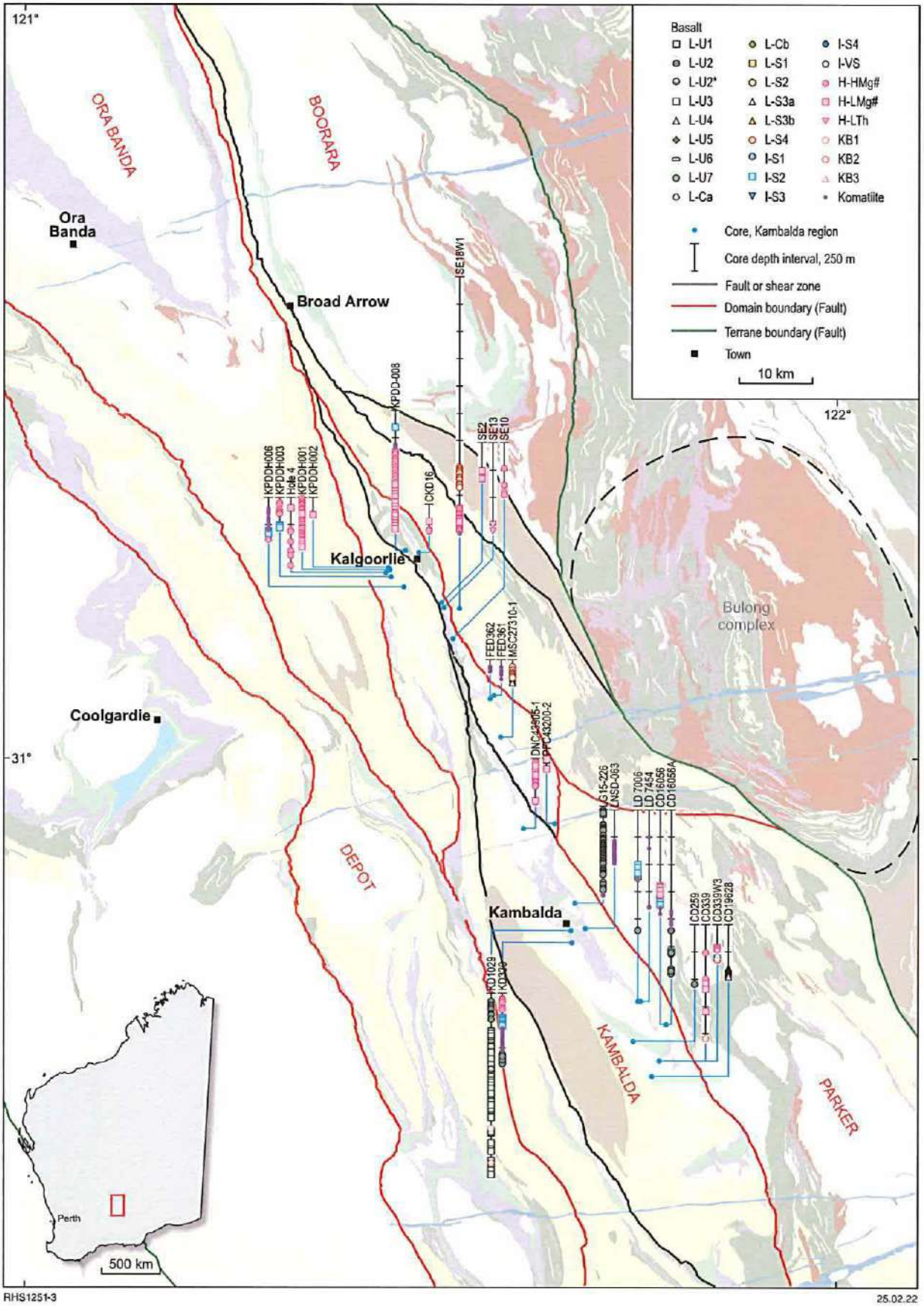


Figure 28. Geological map of part of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology and the distribution of diamond drillcore sample sites within the Kambalda Domain (see Figure 3 for fault names)

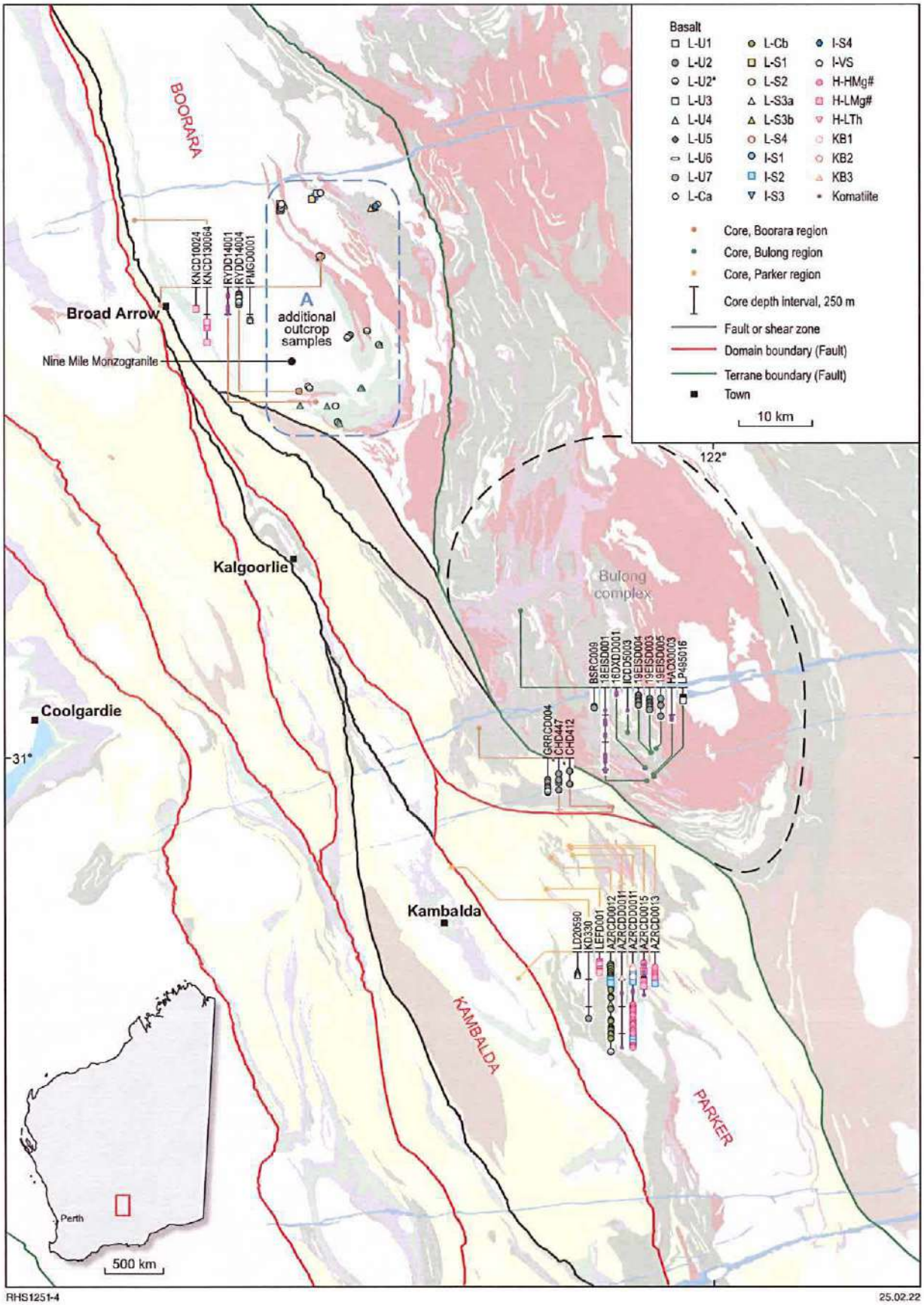


Figure 29. Geological map of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology and the distribution of diamond drillcore sample sites within the eastern domains (see Figure 3 for fault names)

Most of the sills for which we presently have significant data, comprise L-U2 >L-U1 parental compositions but are dominated by more differentiated compositions typical of the L-U3–7 trend. The presence of the sills and their dominant L-U3–7 fill is consistent with some lavas that form the upper levels of the regional mafic stratigraphies reflecting eruptions from evolved upper crustal chambers.

Domains to the east of the Kambalda and Ora Banda Domains (eastern Domains)

We currently have significantly fewer basalt data (~380) from the Boorara and Parker Domains and Bulong complex than for the regions to the west; nevertheless, these data provide valuable insight into the compositional variations in basaltic rocks within these regions. We have attempted to place these data into a stratigraphic context using available relationships in drillcores and current field mapping, and hence, produce a broad chemostratigraphy to compare with the regions to the west. Of all of these regions or stratigraphic levels, the basal portion of the Broad Arrow Subgroup, the Scotia Basalt (Fig. 9), is probably best characterized. However, the overlying Highway Formation comprising a basaltic stratigraphy with interleaved ultramafic rocks, including komatiite, remains poorly constrained.

A notable feature of all of these areas is an L-U dominated lower basalt sequence with L-U2 >>L-U1, similar to the Hannans Subgroup (Fig. 9). The Scotia Basalt contains some additional complexity with a lowest sequence, including L-U2* and L-U4, an association only seen elsewhere within the Burbanks Formation of the lower Coolgardie succession and suspected equivalents in the Ora Banda Domain near Davyhurst. This raises the possibility that a basaltic basement to the Broad Arrow Subgroup, possibly equivalent to the lower Coolgardie succession, is exposed on the southern flank of the Nine Mile Monzogranite, south of the Scotia–Kanowna Anticline (A in Fig. 29).

The Bulong complex stratigraphy apparently has none (or we have not yet sampled any) of the ITB that characterizes the komatiite-rich central portions of the domain stratigraphies (Figs 9, 28). However, I-S2 and I-S4 (one sample each) accompany L-S1 and L-S2 (4 samples) to form what might reflect a very poorly sampled equivalent of the middle stratigraphy in the Broad Arrow Subgroup, and I-S2 occupies possibly the same stratigraphic position in the Parker Domain. Also in common with the Hannans Subgroup, is an upper basalt stratigraphy dominated by HTSB and fractionated LTB, including L-S3b and L-Cb (Bulong complex) or L-U3, L-U7, L-S3a, and L-Cb (Parker Domain). Overall, this stratigraphic pattern appears to show considerably greater similarities with the Hannans Subgroup than with the upper Coolgardie succession or the Vetters Subgroup.

Discussion

Our geochemical analysis of local greenstone stratigraphy uncovers a surprisingly complex, but largely systematic, geochemical variation within and between established local and more regional lithostratigraphies. The chemostratigraphic variations, packages, and sequences presented here, particularly when accompanied by additional geological context, should provide a relatively reliable means of establishing the lithostratigraphic position of poorly constrained geochemical samples. The reliability of this approach will increase as more data are added. However, these data also provide potentially significant insights into the evolution of the greenstone belts within the region, including the evolution of lithospheric architecture. In addition to viewing the regional basaltic sequences lithostratigraphically, we can also examine the ways that factors such as source composition, degree of contamination, crustal level of fractionation, and emplacement mechanism (direct ascent or stalled ascent) changed throughout accumulation of the total magmatic greenstone sequences, i.e. the tectonomagmatic evolution.

Potential geochemical relationships between various units and trends

As previously discussed, it is common that during early fractionation of a mantle-derived magma and/or low to moderate degree partial melting of mantle sources, there are no mineralogical changes in the crystal assemblages that might significantly affect mineral/melt partitioning of strongly incompatible trace elements. In such situations, all magmas derived from a given mantle composition should plot on straight lines (in plots of incompatible elements, for example, Th vs Nb or Zr) that extrapolate to the origin (i.e. x, y both = 0 ppm). The slope of these lines gives the ratio of those elements in the source. The L-U units form a relatively consistent trend (well defined in the case of the L-U3–7 trend) that extrapolates to the origin and has the lowest Th/TiO_2, Th/Nb, and Th/Zr ratios of all magmas examined here (Figs 6, 7, 10). The source for these L-U units can be considered broadly similar in composition, although not identical as indicated by minor variations in, for example, Th/Nb ratios between L-U1 and L-U2 (Fig. 10). This mantle source is broadly representative of the ambient southeastern Eastern Goldfields Superterrane mantle at c. 2700 Ma and has compositions lying between those of primitive mantle and N-MORB (Fig. 10), like most Archean primitive basaltic rocks (Barnes et al 2021). The observation that komatiites plot close to the primitive part of the L-U trend, between L-U2 and the origin (Fig. 30), shows that komatiitic sources and non-komatiitic sources had broadly similar mantle compositions, although komatiitic sources range to slightly more depleted compositions (e.g. Barnes et al., 2021).

Although varying degrees of partial melting and fractional crystallization can result in similar straight-line trends in bivariate plots of strongly incompatible trace elements, we attribute the L-U3–7 trend primarily to fractional crystallization. The main reasons are that the trend is dominated by samples with low $Mg^{\#}$ (<50), significantly out of equilibrium with peridotite at realistic mantle pressures and temperatures. In addition, there is no clear difference in composition (including $Mg^{\#}$ and Nb/Yb ratios, which are both sensitive to changes in melt fraction) between primitive members of the stratigraphically lowest L-U2 units (e.g. Lunnon Basalt) and L-U2 magmas parental to sills that intrude the upper levels of the greenstone stratigraphy, although associated L-U samples within the sills (L-U3–7) vary widely in composition through fractionation.

We do, however, see evidence for considerable variation in the degree of mantle melting throughout the evolution of the greenstone sequences (i.e. Kalgoorlie Group). The occurrence of komatiitic magmatism and the dominance of ITB suggests that the middle basaltic sequences of the Kalgoorlie Group generally reflect a higher degree of partial melting than magmatism forming the lower basaltic sequences. ITB typically have slightly higher incompatible trace element concentrations and Th/Nb, Th/Zr, and Th/La ratios than LTB, despite ITB clearly reflecting the products of higher degree mantle melting based on typically higher MgO contents and $Mg^{\#}$ of primitive end members. This indicates that the parental magmas for ITB were also contaminated (enriched) at an earlier stage than the parental magmas for L-U compositions.

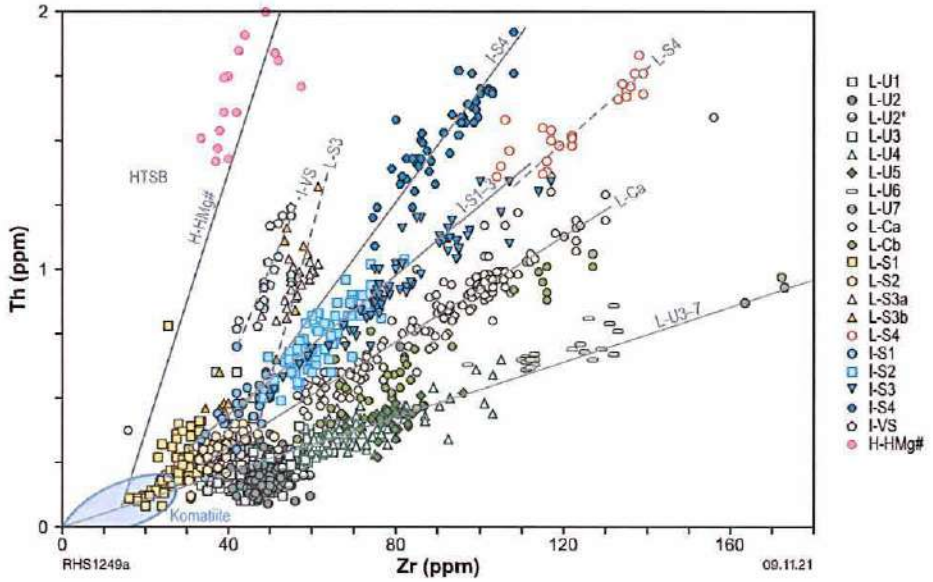


Figure 30. Variation in Th with Zr for individual LTB, ITB and HTSB geochemical groups and units. Also shown is the field for komatiites sampled during this project and the geochemical trends defined by many of the specific units

Most ITB are derived from ultramafic or komatiitic basaltic parental compositions with a higher thermal capacity to assimilate crustal material and at a stage when the base of the crust had already been considerably thermally weakened. The HTSB in the upper greenstone sequences represents an even more extreme case of this (e.g. Arndt and Jenner, 1986; Barnes and Arndt, 2018; Barnes et al, 2021).

Units of the L-U3–7 trend were not only derived from a compositionally similar and broadly primitive mantle source, but they all evolved via the same range of fractionation processes without significant additional contamination from the crust. Most other units or trends of LTB, ITB, and HTSB plot on lines that do not extrapolate to the origin, although many intersect the L-U or komatiite trend at various points (Fig. 30). A plausible interpretation of these relationships is that these mafic magmas reflect episodes when primitive L-U or komatiite magmas (if there was a significant difference) became contaminated by crustal material and subsequently evolved along different trace element trends, each reflecting the composition and proportion of the crust that was incorporated. Most of these subsequent trends are also linear, suggesting that once contaminated by crustal material, subsequent evolution (fractionation) is simple and involves no further crustal contamination. Hence, it is plausible that all ITB units and L-C units were derived through contamination of L-U or parental komatiite magmas more primitive than L-U1 and L-U2. Rare cases, including I-VS, I-S4, L-S3, and L-S4, show evidence of representing a second stage of either contamination, magma mixing or a major change in the mineralogy of the fractionating assemblage. We see no clear evidence with any unit or trend, for continuous contamination. With the exception of L-S1 and L-S2, we also see no clear evidence requiring melting of a mantle source already contaminated with crust, consistent with the findings of Smithies et al. (2018a). This means that contamination primarily occurred in or at the base of the crust through assimilation into mantle-derived melts. It remains plausible that prior crustal contamination of mantle sources did occur, but if this is the case, it was so minor and/or inhomogeneous that the compositional affects were swamped by subsequent crustal assimilation.

However, L-S1 and L-S2 potentially provide an exception in that their compositions might reflect a slightly more depleted bulk mantle source than that of most of the other magmas. Within the limits of the present data, it is difficult to assess the relationship between the L-S1 and L-S2 magmas and the other LTB units. Many incompatible trace elements trends are consistent with either L-S1 or L-U1 being a potential parental magma to L-Ca, although based simply on when these compositional variants appear in the regional stratigraphies, a relationship between L-U1 and L-Ca seems more likely and equally consistent with spatial relationships.

For HTSB and most ITB, trends away from the main L-U trend happen at a relatively primitive stage along that trend (Fig. 30). This indicates contamination of a high MgO, possibly komatiitic magma that was considerably less evolved than L-U1 or L-U2, probably at or just above the mantle-lower crust interface. In the case of more evolved HTSB and ITB magmas (e.g. I-S3 and I-S4), Al_2O_3/TiO_2 ratios below mantle values (~20) reflect subsequent plagioclase fractionation (without further contamination) at mid- to upper crustal levels.

We do see some deviation or scatter from straight-line trends at high TiO_2 contents in the L-U3–7 trend, with Th/TiO_2 ratios increasing in L-U6 and scattering in L-U7 (Fig. 6), as a result of magnetite saturation and fractionation, although this is unrelated to contamination of L-U composition magmas since trends for Zr/Th and Nb/Th (Fig. 7) remain relatively constant.

Individual units of the L-U3–7 and I-S1–3 trends have mantle-normalized trace element plots that broadly parallel each other (L-U3 parallels L-U4 and L-U5 etc.) at different element concentrations (Fig. 31); i.e. they show very limited ranges in incompatible trace element ratios and wide ranges in concentrations. For both trends, ~three-fold increases in Th concentrations indicate a ~70% reduction in magma volume, assuming a closed system. This reflects the role that fractionation of minerals (such as plagioclase, olivine and orthopyroxene), with exceptionally low (and similar) abilities to concentrate REE and HFSE, can have during cooling of mafic magmas.

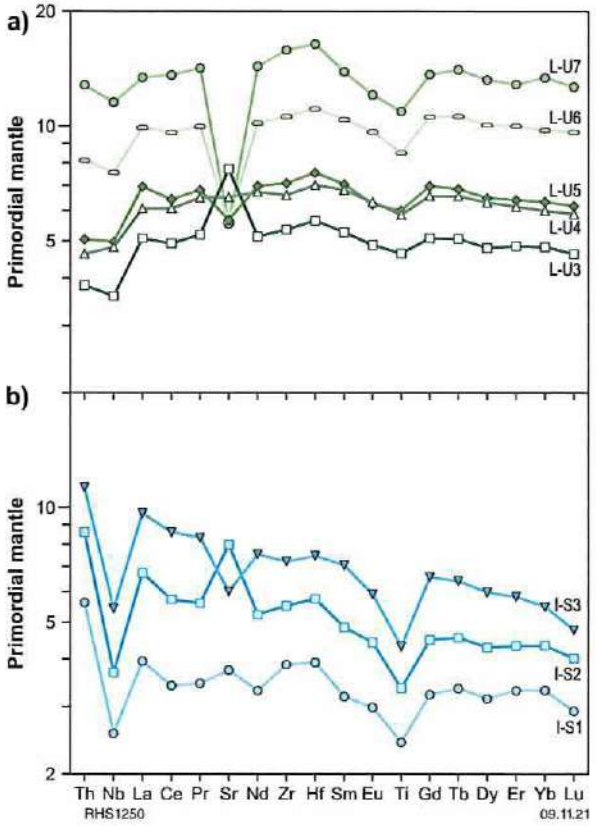


Figure 31. Average mantle-normalized trace element patterns for various LTB and ITB geochemical groups and units (normalizing values from Sun and McDonough, 1989)

Wide ranges in Al_2O_3/TiO_2 and in mantle-normalized Sr-anomalies (positive to negative) (Figs 24, 31) indicate the significant role of plagioclase fractionation (accumulation or removal); however, variations in Al_2O_3 and $Mg^{\#}$ vs Al_2O_3/TiO_2 (Figs 12, 24) are not systematic. Hence, although different units within each trend have similar incompatible trace element ratios, they have evolved through fractionation of different combinations of incompatible trace element-poor minerals. This emphasizes the dominant control that initial source composition and early contamination have on mafic magma composition. The evidence for extensive closed-system crystal fractionation trends suggests evolution in, and eruption from, standing crustal magma chambers at mid- to upper crustal levels.

Regional stratigraphic and geographic relationships and implications for greenstone evolution

The range of mantle-derived parental magmas feeding the mafic greenstone stratigraphies in the study area is limited mainly to L-U and komatiitic compositions (themselves not clearly distinguishable in source composition). We also infer that a range of conditions and processes operating at various crustal levels has resulted in significant compositional diversity in the erupted products. This includes changes in the regional deformation regime or in crustal thickness that favoured the rapid transfer of only weakly fractionated and uncontaminated magmas (L-U1 and L-U2) from the source to the deposition site. A crustal shortening regime and/or the presence of thicker crust would have favoured ponding in mid- to upper crustal chambers, prior to eruption of variably fractionated L-U3–7 magmas. In addition, we can infer a broad but non-systematic increase in the extent of magma contamination by crust, despite the occurrence of L-U3–7 magmas in the upper stratigraphies. Viewing these variations in stratigraphic and geographic contexts allows a generalized sequence of regional greenstone evolution to be inferred, and potentially identifies crustal architectural control on magma compositional distribution.

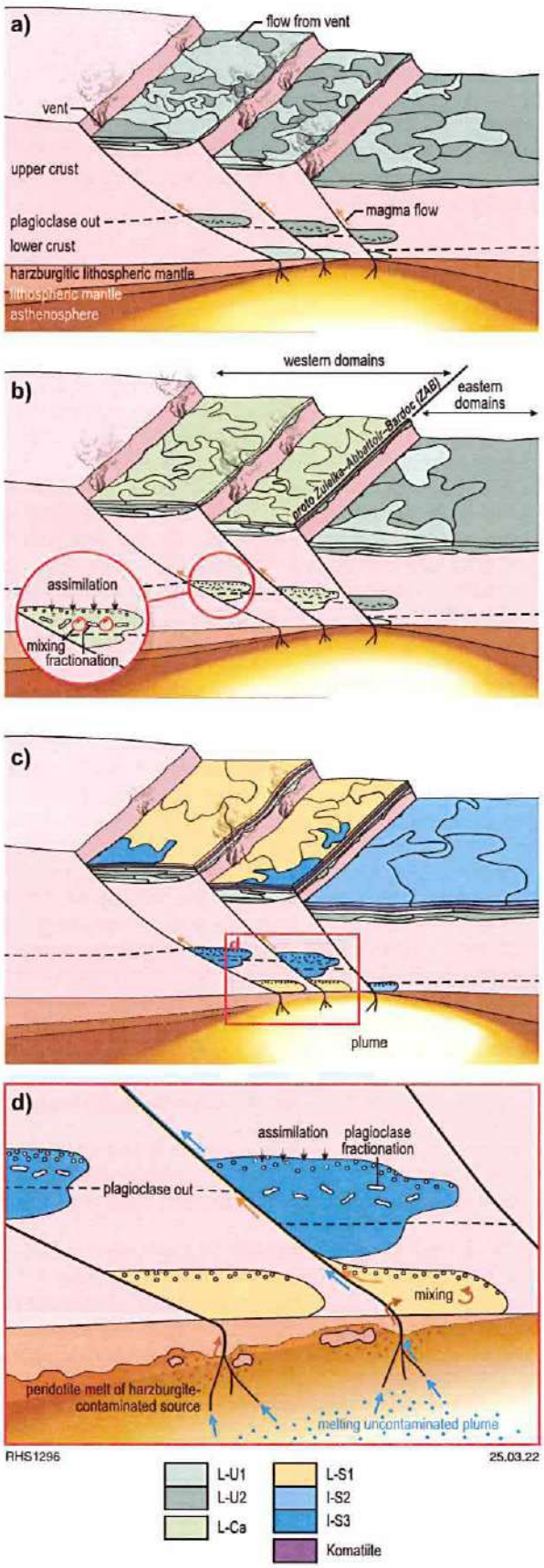
Lower basaltic stratigraphy

Primitive L-U magmas, L-U1 and L-U2 are the most sampled mafic units and form the basal component of all stratigraphies. As well as a stratigraphic pattern, there is a broad geographic pattern in their distribution. Both occur throughout the region but L-U1 greatly dominates the lower stratigraphy in the western domains (Coolgardie and Ora Banda), although it does not occur in the lower Coolgardie sequence. Conversely, L-U2 (main population) is more common and dominates the lower stratigraphy of the eastern domains (i.e. Kambalda, Boorara, and Parker Domains and the Bulong complex). With $Mg^{\#}$ typically <65, the parental magmas for L-U1 and L-U2 almost certainly fractionated to some extent before and during emplacement in the crust, but were not contaminated to any significant degree. In the case of L-U1, mantle-like Al_2O_3/TiO_2 ratios ~20 (Fig. 12) suggest that plagioclase was probably not stable during early fractionation, which probably occurred within the spinel stability field at or beneath lower crustal depths (broadly below ~35km). In the case of the main population of L-U2, a slightly lower Al_2O_3/TiO_2 (~16–17) ratio probably does reflect limited early plagioclase fractionation, perhaps as a result of thinner lithosphere. Slightly higher Th/Nb and Th/Zr ratios also suggest that L-U1 (Fig. 7) incorporated a slightly greater (although still very minimal) proportion of a crustal component. Thus, there is a broad but weak variation between eastern and western domains, in terms of the early history of the primitive L-U unit that dominates the lower basaltic stratigraphy. These small differences are consistent with L-U1 magmas ponding and fractionating at deeper levels than L-U2 magmas, in an environment that permitted slightly greater crustal contamination (i.e. within thicker crust) (Fig. 32). The relatively narrow compositional range for the primitive L-U1 and L-U2 magmas of the lower stratigraphy suggests a regional crustal deformation regime that permitted ascent through the full crustal column at rates rapid enough to avoid further significant compositional evolution.

However, the upper portion of the lower basalt succession in the western domains (the upper part of the Lindsays Basalt in the upper Coolgardie sequence and the upper part of the Wongi Basalt as well as the overlying Missouri Basalt in the Vetters Subgroup) is dominated by L-Ca. In contrast, with the primitive L-U units, L-Ca requires significant early contamination and homogenization of primitive magmas, prior to undergoing significant fractionation at low to mid-crustal levels, where plagioclase was stable (as indicated by positive correlations between $Mg^{\#}$ and Al_2O_3 with Al_2O_3/TiO_2 and by Al_2O_3/TiO_2 <20) (Figs 12, 24). This unit is unique to the western domains and clearly requires a change in the early ascent history of these magmas to conditions that have not affected the eastern domains at that time. Relaxing extension as the axis of rifting migrated eastwards is potentially one cause for ascending primitive magmas to stall and pond in the mid to lower crust (Fig. 32).

Middle basaltic stratigraphy

In all the regional stratigraphies, the lower basalt sequence is overlain by a sequence of komatiite and komatiitic basalt. Although not examined in this study, this c. 2705 Ma komatiite-dominated magmatism is traditionally attributed to melting in the tail of a



mantle plume (e.g. Campbell et al., 1989; Begg et al., 2010; Barnes et al., 2012; Mole et al., 2019) in an active magmatic rift setting. In the well-studied Hannans Subgroup of the Kambalda Domain, the stratigraphy is thought to be relatively simple, with the Kambalda Komatiite conformably overlying primitive L-U magmatism of the Lunnon Basalt (in places with an intervening shale unit) and in turn being conformably overlain by ITB of the Devon Consols Basalt. Tectonic interleaving locally complicates this stratigraphy, although in many other areas, structural complexity or magmatic interleaving typically causes significantly more complicated stratigraphy, particularly near the contacts with lower and upper basaltic sequences. It is clear, for example, that the komatiite-bearing parts of the upper Coolgardie sequence (Hampton Hill Formation), Vetters Subgroup (Walter Williams Formation and lower Big Dick Basalt), and Broad Arrow Subgroup (Highway Formation), include a much greater proportion of basalt than the Kambalda Komatiite. In the lower parts of the Hampton Hill Formation, it is not clear if L-U1 intercalated with komatiite is part of that formation or a component of the underlying Lindsays Basalt. In no other regional stratigraphy is extrusive L-U1 found above the lower basalt succession, although intrusive L-U1 forms a very minor component of major mafic sills emplaced at the end of mafic greenstone deposition.

Figure 32. Schematic block interpretation of the magmatic evolution of lower and middle parts of the mafic stratigraphy within the study region, showing the possible position of the ZAB shear zone (see text) and the contrasting magmatic evolution of the western and eastern domains. Earliest evolution of the mafic assemblages (a) involves L-U1 and L-U2 magmatism throughout the region in response to crustal extension and mantle upwelling (cause/response?) with the relative proportions of L-U1 and L-U2 decreasing eastwards as the proportion of primitive magma coming from lower crustal staging chambers, lying at pressures too high to crystallize plagioclase, decreases with crustal thinning. Eastwards migration of the point of maximum upwelling and possibly minor incorporation into L-U magmas of material from harzburgitic lithosphere lying only under the western region (b) resulted in less transient lower to mid-crustal chambers open to assimilation (i.e. L-Ca). L-U1 and L-U2 magmatism continues in the eastern domains. With arrival of an upwelling/plume head (c), regional komatiite magmatism blanketed the region. In the western domains, komatiite was preceded and followed by L-S lavas derived from ultramafic magmas formed as the outer plume margin assimilated harzburgitic lithosphere (d). Melting of slightly deeper, uncontaminated outer plume mantle, formed komatiitic to komatiitic basalt magmas that ponded in the now thermally weakened middle-crust where they fractionated plagioclase as well as assimilated surrounding country rock to produce predominately I-S3 magmas. To the east of the ZAB, beneath the eastern domains, the absence of a harzburgitic lithospheric layer meant that L-S magmas were not produced and the more extensional environment directly overlying the plume prevented crustal magma chambers forming at levels above the crust/mantle interface. Here, assimilation of thermally weakened low crust still occurred but was at pressures too high to be accompanied by plagioclase fractionation and resulted in I-S2 compositions. Dashed line approximates the position of the plagioclase-out phase boundary. Note the interpreted progressive easterly migration of the mantle-upwelling zone, which evolves into a distinct plume-like body. Harzburgitic lithosphere is interpreted to only lie west of the ZAB, based on the compositional features consistent with an influence of such a source component on specific magmas only in that region. The ZAB is, therefore, interpreted as a potential trace of a lithospheric domain boundary (see later text)

A middle basaltic succession distinctly dominated by ITB (and the I-S1–3 trend) is found in all regional stratigraphies (other than in the Bulong complex), either clearly overlying or intercalated within the upper parts of the komatiite-dominated stratigraphy. In the case of the western domains, the onset of ITB magmatism is immediately preceded by L-S magmatism, L-S1 in the case of the lower Big Dick Basalt and L-S1 and L-S2 in the case of the Hampton Hill Formation. Although constrained by only four samples, there is also a hint that minor L-S magmatism occurs at the same stratigraphic level in the Broad Arrow Subgroup.

As suggested earlier, L-S1 and L-S2 can probably be related through fractional crystallization. Like L-U1, they have high Al_2O_3/TiO_2 ratios that remain relatively constant over large decreases in $Mg^{\#}$ (>75 to <50) (Fig. 24), suggesting minimal fractionation at crustal pressures low enough to crystallize plagioclase. However, in the case of L-S1 and L-S2, Al_2O_3/TiO_2 ratios are unusually high (L-S1 ~27, L-S2 ~26), and where L-S magmas overlap in $Mg^{\#}$ with L-U magmas (other than L-U1), they typically range to slightly higher concentrations of SiO_2 and lower Ni (Fig. 33). This potentially points to a slightly more refractory harzburgitic mantle source component (possibly a modified lithospheric mantle component) than for other magmas studied here. Hence, a plausible interpretation of this magmatism involves extraction from a bulk mantle source that was mineralogically slightly discrete from that of all the other magmas (Fig. 32). L-S1 was contaminated either directly from that source (i.e. a metasomatized source) or immediately upon emplacement into the lowest crust but subsequent fractionation to L-S2 compositions was at pressures too high for plagioclase crystallization.

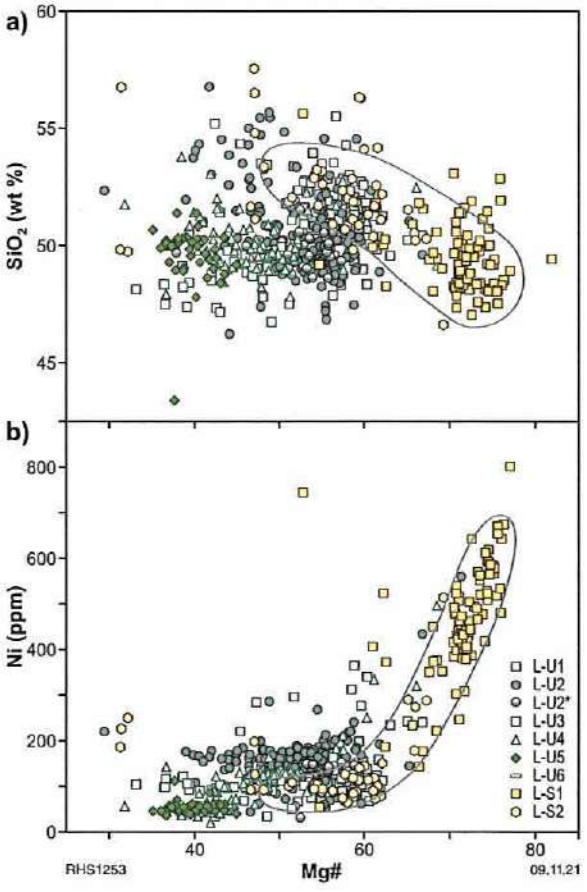


Figure 33. Variation of SiO_2 and Ni with $Mg^{\#}$ for individual LTB geochemical groups and units

The virtual absence of L-S1, L-S2, and L-Ca from the eastern domains also highlights a clear lithostratigraphic difference between the western and eastern domains. It is likely that L-U1 forms part of this association, showing some of the same geochemical attributes (e.g. higher SiO_2, lower Ni) and geographical and stratigraphic distribution as the L-S and L-Ca magmas. Because these differences between the western and eastern domains likely relate to variations in mantle source compositions and the depths at which various magmas are either contaminated or fractionated, they almost certainly reflect a variation in crustal architecture, possibly including lithospheric thickness. Hayman et al. (2015) suggested that a zone of enriched lower basalts (broadly equivalent to L-Ca) was restricted to the eastern domains and this is clearly corroborated here. Our eastern boundary to this zone is a north-northeast line comprising segments of (from south to north) the Zuleika, Abattoir and Bardoc shear zones (ZAB) (Fig. 34). Furthermore, the observation that the lower basalt (Never Can Tell Basalt) of the <c. 2720 Ma greenstone succession in the Agnew region, in the far north of the Ora Banda Domain, comprises the same enriched (L-Ca) component permits the intriguing possibility that the same lithospheric architectural controls on early basalt compositions existed in that region. It also suggests that such regional chemostratigraphic correlations may indeed be valid (Hayman et al., 2015).

Overlying komatiite in the Hannans Subgroup, or interleaved komatiites and L-S1 and L-S2 in the western domains, is a prominent succession comprising units of the I-S1–3 trend. Like the komatiite unit and the underlying L-U succession, the I-S1–3 magmas represent a reliable regional first-order stratigraphic marker, reflecting a certain stage in a process common to the greenstone stratigraphic evolution in all areas studied here. The relevance of this to greenstone evolution in other areas or of a significantly different age is unclear. The lower Coolgardie sequence, for example, contains no I-S1–3 magmas. It contains a strong crustally contaminated high-Mg ITB unit (I-VS) with compositional features transitional with HTSB and this underlies rather than overlies ultramafic units of the Brilliant Formation.

The compositional differences between the ITB in each of the stratigraphies examined here is relatively small. I-S2 represents the overwhelmingly dominant component of the Devon Consols Basalt. It is also the sole component in the upper part of the Big Dick Basalt and a major component of the lower part of the basaltic sequence overlying komatiites in the domains to the east of the Kambalda and Ora Banda Domains (although few data are available here). It forms only a very minor component of the ITB succession in the upper part of the Hampton Hill Formation, but is accompanied there by the more (I-S3) and less (I-S1) fractionated equivalents within the I-S1–3 trend. This diversity of ITB magmas is diagnostic of the upper-middle basaltic stratigraphy of the upper Coolgardie sequence. So too is an additional phase of L-S1 magmatism, forming the sole component of the overlying and regionally extensive Gleesons Basalt.

All ITB magmas of the I-S1–3 trend are derived from either L-U magmas or, more likely, komatiitic to komatiitic basalt parental magmas that were significantly more primitive (i.e. MgO rich and with a higher $Mg^{\#}$) than L-U1 or L-U2 when they underwent crustal contamination. Together, I-S1–3 magmas reflect a broad increase in the degree of crust interaction with mantle-derived magmas (e.g. Arndt and Jenner, 1986; Morris, 1993). This probably reflects their formation in the waning thermal wake of komatiite magmatism, where very primitive magmas reflecting relatively high-degree partial melts of the mantle were more readily contaminated at the base of the crust by material that was already close to solidus conditions.

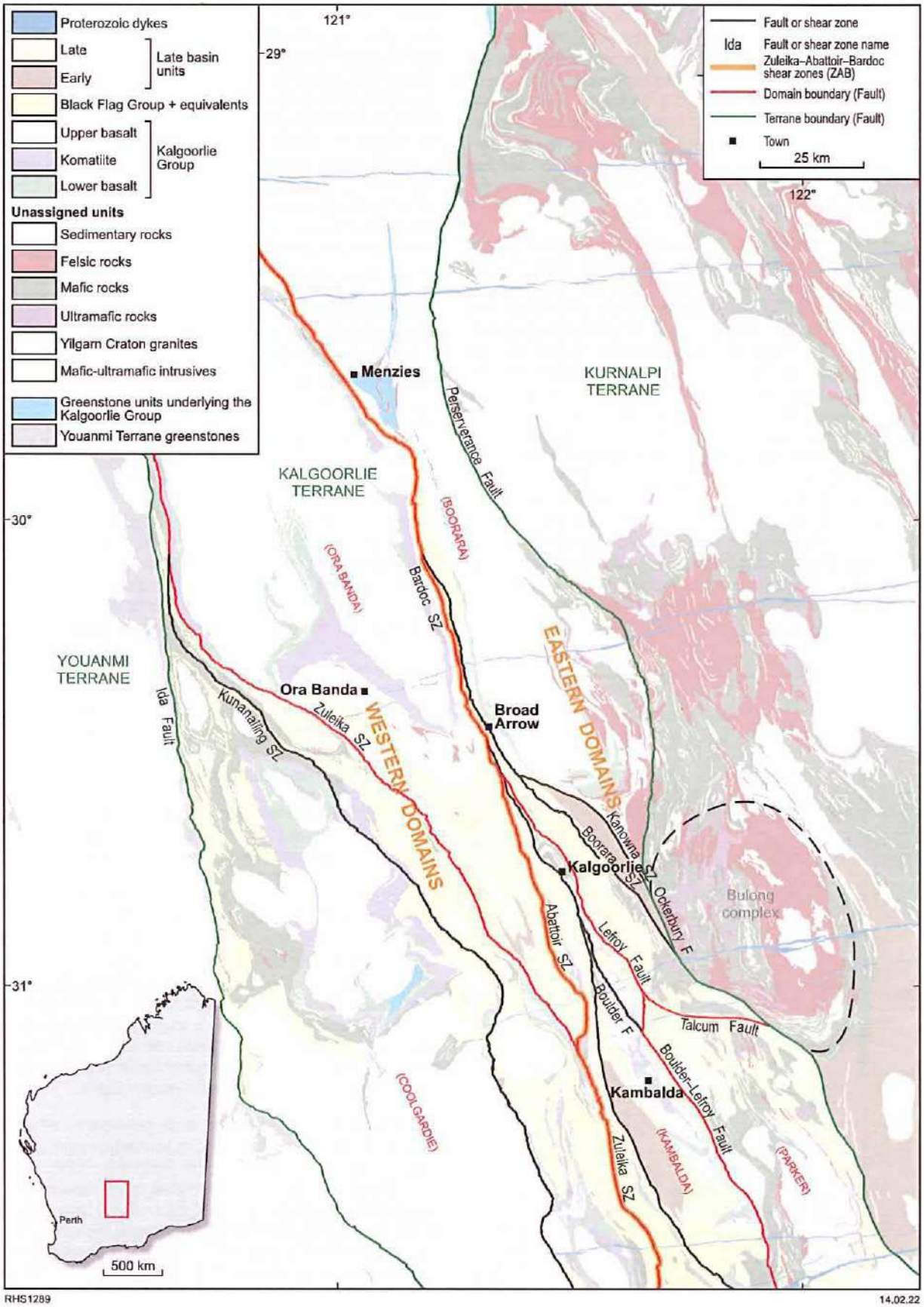


Figure 34. Geological map of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology. Also shown are the major faults and shear zones, including those forming proposed domain and/or terrane boundaries (from Cassidy et al., 2006). The trace of the combined ZAB shear zones forming a potential alternative boundary between areas of common chemostratigraphic evolution is highlighted

Upper basaltic stratigraphy

The upper basaltic stratigraphy is dominated either by HTSB or strongly fractionated LTB (LTB>>ITB) or both. A clear and important exception is the upper Coolgardie sequence, where the basaltic stratigraphy overlying the Gleesons Basalt is very poorly developed or absent. The scarcity or absence of this upper stratigraphy was noted by Swager et al. (1997) and appears to be the only significant chemostratigraphic difference between the upper Coolgardie sequence and the Vetters Subgroup.

Overall, magma diversity is perhaps greatest within the upper basaltic units of the regional stratigraphies and much of this diversity reflects processes that occurred within the upper crust. In contrast with the lower basaltic stratigraphies, where most of the extruded LTB magmas underwent relatively little compositional change since leaving the mantle, lavas in the upper basaltic stratigraphies erupted from mid- to upper crustal chambers large enough for parental compositions to fractionate to a range of evolved compositions.

Where HTSB magmatism is present, it typically forms the earliest component of the upper basaltic stratigraphy, with the Paringa Basalt of the Hannans Subgroup deposited at or after c. 2690 Ma (Carey, 1994). Although stratigraphic relationships are not as well established throughout the domains to the east of the Kambalda and Ora Banda Domains, HTSB clearly forms a dominant component within parts of these regions. HTSB is rare or absent (either does not occur or is in such low abundances that it was overlooked) in the Ora Banda and Coolgardie Domains, although rare intrusive equivalents, including notable (but younger) examples (such as Powder Gabbro) occur high in the stratigraphic sequence, close to the Zuleika Shear Zone, possibly suggesting the volcanic equivalents have been removed (eroded).

While most HTSB magmatism pre-dates deposition of the volcanosedimentary Black Flag Group (Powder Gabbro being an obvious exception), the fractionated LTB>>ITB magmatism overlying the HTSB arguably pre-dates and is syndepositional with respect to the Black Flag Group. We see no geochemical differences between the various compositions sampled from large mafic intrusions, such as the Golden Mile Dolerite and the Mount Pleasant Gabbro, emplaced within or immediately overlying the basaltic greenstone sequences, and the evolved LTB lavas forming the uppermost stratigraphic levels of the Vetters Subgroup and Highway Formation. Nor are we aware of any geochronological data that disallows such a potential genetic link. Tripp (2013, 2019) interprets the volcanosedimentary sequence of the Black Flag Group to conformably overlie the Victorious Basalt.

The LTB-dominated (LTB>>ITB) magmatism in the upper part of the upper basaltic stratigraphy forms two broad compositional groups. A group of significantly Th-enriched basalts directly overlies the Paringa Basalt in the Hannans Subgroup [L-S3a (Athena Basalt) and L-S4], overlies HTSB in the upper part of the Bulong complex (L-S3b), and forms the Bent Tree Basalt (I-S4) in the Vetters Subgroup. L-S3 (a and b) is a weakly to moderately fractionated LTB magma with major element similarities to compositions lying between L-U2 and L-U3. L-S4 is a strongly fractionated magma with compositions typical of granophyre-like magmas reflecting late-stage evolution in a high-level mafic magma chamber. Enrichments in, and ratios between, Zr, Nb, and La, in L-S3 (a and b) and L-S4 (e.g. Fig. 6) suggest evolution within an L-U3–7 chamber. However, if this were true then the distinctive and extreme enrichments in Th (>>La>Nb, Zr; Fig. 6) in these units can only be explained, if this chamber became contaminated or mixed, and homogenized prior to eruption with minor amounts of a component similar in composition to HTSB. Given their stratigraphic position in the upper part of the Hannans Subgroup (or upper Highway Formation in the case of L-S3b), this is a clear possibility. Similarly, I-S4 compositionally resembles L-S4 in many respects and can be broadly modelled through mixing and homogenizing a relatively small amount (~20%) of HTSB into an L-Ca chamber.

The second broad compositional group of LTB>>ITB magmas in the upper part of the upper basaltic stratigraphies belongs to the L-U3–7 trend. These form the lavas of the Victorious Basalt at the top of the Vetters Subgroup, and lavas contributing to the upper mafic stratigraphy of the Boorara and Parker Domains. They derive from upper crustal chambers large enough for parental L-U2 (>>L-U1) compositions to fractionate to a range of evolved compositions. Large mafic intrusions, such as the Golden Mile Dolerite and the Mount Pleasant Gabbro, emplaced within or immediately overlying the basaltic greenstone sequences between 2690 Ma and 2680 Ma are the youngest and uppermost crustal level expressions of these chambers. Their compositional range essentially mimics that of the L-U2 and L-U3–7 lavas (Table 3). These intrusions include fine-grained, quenched, L-U2 marginal facies material. In the case of the Golden Mile Dolerite, L-U2 forms peperitic contacts with the host sedimentary rocks of the Black Flag Group (McMann, 2018; Cas et al., 2019).

Thus, during deposition of the upper basaltic stratigraphic levels, conditions favourable for the establishment and maintenance of mid- to upper crustal magma chambers prevailed, perhaps for the first time, at least in the evolution of the greenstones of the eastern domains. However, evidence that mixing and homogenizing between HTSB and already-ponded L-U3–7 chambers produced a range of unusual hybrid compositions, also reflects a dynamic mid- to upper crustal environment that apparently did not exist in the Coolgardie region, west of the Zuleika Shear Zone.

Hence, in broad terms, the three commonly identified regional stratigraphic levels (lower, middle and upper, e.g. Swager et al., 1990) correspond to changes in tectonomagmatic processes. However, the distribution of the subgroups, processes, and products is distinctly asymmetric, producing regional-scale chemostratigraphic differences (e.g. Hayman et al., 2015) between the western and eastern domains that probably reflect a fundamental crustal architectural control and potentially provide some basis for reassessing present domain boundaries. The boundary between the two regions – the ZAB (Fig. 34) – separates a region to the west characterized by slightly thicker crust (no plagioclase fractionation in L-U1), earlier crustal contamination of primitive magmas, a broadly more refractory bulk mantle source, and a mafic magmatic history missing a significant upper stratigraphy, from the eastern domains (Kambalda, Boorara and Parker Domains and possibly also the Bulong complex). The Ora Banda Domain is clearly a part of the western domains throughout evolution of the lower and middle greenstone stratigraphic evolution, but also appears transitional to the eastern domains in terms of its upper stratigraphy characterized by the diversity of fractionated L-U3–7 magmatism formed in and erupted from layered sills.

Felsic rocks

Introduction

Felsic rock samples (andesite to rhyolite; i.e. intermediate and acid) taken during this study are mainly from plagioclase (±hornblende or quartz) porphyritic dykes or sills, representing subvolcanic intrusions and from the volcanic or volcaniclastic rocks, including probably poorly sorted and very immature proximal turbidite units. The majority of these samples are from the Black Flag Group, which is interpreted to have been deposited between 2690 and 2665 Ma across all of the domains of the Kalgoorlie Terrain within the study area. However, the presence of older felsic volcanic rocks is speculated within the lower Coolgardie sequence, and demonstrated at numerous localities within the study area. For example, felsic tuff and andesite-derived sandstone interleaved with ultramafic rocks have been dated at 2705 ± 4 Ma (Nelson, 1995a) and 2708 ± 5 Ma (Barley et. al., 2002) respectively, from the Bulong complex. Similarly, dacite interbedded with komatiite from the Highway Formation (Boorara Domain) has been dated at 2708 ± 7 Ma (Nelson, 1995b).

It is important to note there is no clear difference in the composition or compositional range of the intrusive, volcanic, and volcaniclastic rock populations; i.e. no evidence that a specific depositional process has significantly modified the primary magmatic compositions. Of the original set of 845 samples, filtering for alteration eliminated a total of 213 (~25%) samples. Approximately 50% of these were excluded based on a high ASI [molar $Al_2O_3/(CaO+Na_2O+K_2O)$] up to 2.2, reflecting a secondary alteration or significant sedimentary reworking or both. After filtering, the remaining samples were grouped based on distinct compositional trends reflecting igneous processes. At this stage, the samples initially removed from the dataset based on alteration screens were compared with the various geochemically classified groups. Considering only fluid immobile geochemical components, 113 of the altered samples could be confidently matched to a geochemically classified group. These altered samples were not considered for any petrogenetic interpretations but were used to examine the spatial distribution of the geochemically classified groups.

The majority of subvolcanic intrusions and associated volcanic or volcaniclastic deposits forming the Black Flag Group have been shown to have the composition of sanukitoids and comprise one of the first recognized examples of Archean sanukitoid volcanic rocks, and one of the most voluminous intrusive/extrusive Archean sanukitoid systems globally (Smithies et al 2018b). Sanukitoids are relatively rare hornblende-bearing rocks, typically in the monzonitic to dioritic and granodioritic (andesitic to dacitic but reaching rhyolitic) compositional range. They have been attributed to direct extraction as low-degree felsic partial melts from a hydrated mantle source (Shirey and Hanson, 1984; Smithies and Champion, 2000). However, more recently they have been interpreted to be (or include examples that are) fractionated from mafic to ultramafic parental magmas (including calc-alkaline lamprophyres), themselves partial melts of metasomatically hydrated and enriched mantle lithosphere (Smithies et al., 2018b, 2019). Sanukitoids are amongst the most hydrated and oxidized Archean magmas and their presence implicitly defines a translithospheric plumbing system that transferred fertile mantle material to upper crustal levels (Smithies, 2019; Smithies et al., 2019). In the Yilgarn Craton and other Archean regions, sanukitoids have a clear spatial and temporal association with gold mineralization (Champion and Cassidy, 1998; Witt et al., 2013, 2015; Fayol and Jébrak, 2017). In the Eastern Goldfields, many of the structures used as magma pathways by sanukitoids are faults that also played a primary role in the structural evolution of the 2664–2657 Ma sedimentary basins (Late Basins – Fig. 2) that overlie the Black Flag Group. Smithies et al. (2018b) and Smithies (2019) suggest Black Flag Group magmatism represents a long-lived, multiphased, crustal-scale magma system that along with the associated hydrothermal systems fueled by exsolved magmatic water and water entrained from intrusion-related country-rock fluid convection had the capacity to transfer large amounts of Au from the mantle source and crustal pathways, into the upper crust (Fig. 35). These systems are envisaged to resemble modern porphyry copper systems in many respects, including their distinctive high Sr/Y primary magmatic composition. As suggested in the case of many Archean gold deposits in the Abitibi Subprovince of the Canadian Superior Province (Fayol and Jébrak, 2017), the apparent orogenic characteristics of many gold deposits in the Eastern Goldfields, may reflect the upgrading of pre-existing magma-related concentrations during post-c. 2680 Ma orogenic-style deformation (Smithies et al., 2018b; Smithies, 2019). This is supported by sulfur isotope studies on Eastern Goldfields gold deposits that have shown early mineralizing fluids with mantle-like isotopic compositions overprinted by late-stage mineralizing fluids with crust-derived sulfur (e.g. Thébaud et al., 2018; Sugiono et al., 2021).

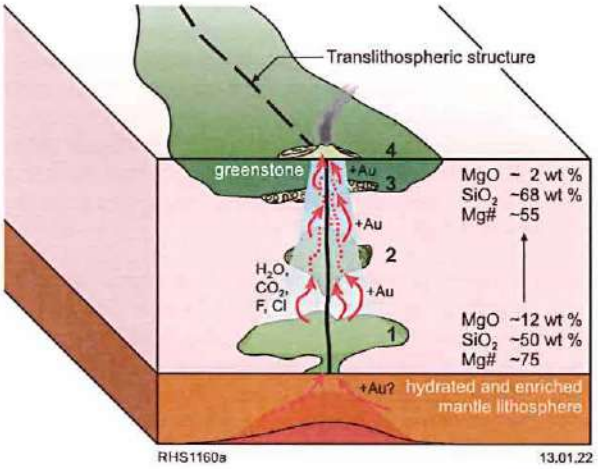


Figure 35. Schematic diagram showing the potential relationship between sanukitoid magmatism (e.g. Black Flag Group), translithospheric structures and gold mineralization (Smithies, 2019). We suggest that sanukitoids can form long-lived, multiphased, crustal-scale magma systems involving multiple staging chambers, where magmas undergo variable degrees of hornblende-dominated fractionation and frequent recharge events that typically incorporate a cargo of partially to largely crystallized material from earlier sanukitoid magma batches (i.e. 'cognate' xenoliths). Composite lower crustal chambers are filled with primitive, mantle-derived lamprophyric to high-Mg dioritic magmas (1 in Fig. 35), with subsequent higher-level chambers (2–3 in Fig. 35) having sequentially more evolved bulk compositions. In rare cases, such as the Black Flag Group, eruption of sanukitoid material occurs (4 in Fig. 35). Significant hornblende fractionation results in early and ongoing exsolution of strongly oxidized magmatic fluids. The associated hydrothermal systems (red arrows and blue screen) fuelled by this exsolved magmatic water, and by country-rock fluid convection, have the capacity to transfer large amounts of Au and other metals originating from magmatic (enriched mantle) and country-rock sources into the upper crust. In this respect, it might be irrelevant whether the primary lamprophyre/sanukitoid magmas and their mantle source regions were intrinsically Au rich, since the sanukitoid-related crustal-scale magmatic/hydrothermal system had the capacity to sequester metals over the entire crustal column. The volcanic environment exposed by the Black Flag Group also offers the opportunity for epithermal type processes to further enrich the country rock systems in Au, prior to any late upgrading caused by orogenic processes (e.g. Fayol and Jébrak, 2017)

Classification of felsic rocks

A workflow for classifying felsic rocks is provided in Appendix 5. Figure 36 attempts to graphically show aspects of this workflow. Although this classification is based primarily on geochemical data, the geographical and lithological association of the samples and mineralogy remain extremely important considerations. Sanukitoids typically have quite specific field appearances and mineralogy, particularly at the lower silica end of their compositional range, where they are invariably hornblende-rich and often contain cognate (hornblende-plagioclase±pyroxene) xenoliths. However, at silica contents higher than ~68 wt%, sanukitoids and High-Ca granites can be virtually impossible to distinguish from geochemistry alone; field association or relationships, and mineralogy become critical factors in classification. Similarly, High-Ca granites are typically plagioclase-rich and K-feldspar-poor, but can fractionate to K-feldspar-bearing examples that geochemically resemble Low-Ca granites.

The granite classification scheme of Champion and Sheraton (1997) and Cassidy et al. (2002) was initially applied to the felsic rocks within the study area. This classification divides Archean granites into five broad groups including:

- High-Ca granites, typically with SiO_2 >68 wt% and a compositional range broadly consistent with the Archean tonalite–trondhjemite–granodiorite (TTG) suites, reflecting deep crustal melting of dominantly mafic source regions
- Low-Ca granites, typically also with SiO_2 >68 wt% but generally more potassic in composition (broadly equivalent to Archean K-rich granite series) and more enriched in incompatible trace elements, reflecting higher temperature melting of dryer, more heterogeneous and slightly more refractory sources, including mafic crust and earlier-formed TTG crust
- HFSE granites, that are generally very silica rich (SiO_2 >70 wt%), Fe rich, and strongly enriched in Nb, Zr, and REE, and commonly showing characteristics consistent with the involvement of tholeiitic parental magmas or sources
- Syenites, typically very alkali rich, quartz-poor, and Fe-rich rocks, also enriched in Nb, Zr, and REE, and in large ion lithophile elements (LILE) such as Sr and Ba. They are thought to be derived through melting of a metasomatically enriched deep lithospheric source (Smithies and Champion, 1999)
- Mafic granites, typically hornblende-bearing rocks showing a wide range of silica concentrations upwards of ~57 wt% with a significant proportion of samples with SiO_2 <68 wt%. Sanukitoids form a large component (>50%) of this group. The remainder includes suites that have $Mg^{\#}$, and/or Cr, Ni, and/or Sr or Ba concentrations lower than expected for sanukitoids, but generally slightly higher than is typical of modern calc-alkaline magmas, possibly as a result of a hotter Archean mantle (e.g. Barnes and Van Kranendonk, 2014).

Since sampling was primarily directed at greenstone successions, the emphasis was on volcanic, volcaniclastic rocks or subvolcanic intrusions. For the most part, sampling of large plutonic bodies was avoided. Nevertheless, it is somewhat surprising that very few clear examples of felsic rocks with compositions similar to Low-Ca granites were identified despite these being the regionally dominant felsic magma type and mainly formed in the period immediately post-dating the evolution of the Black Flag Group.

The entire felsic dataset (Fig. 37) was divided between samples with compositions equivalent to:

- Mafic granites 87% – of which 66% are true sanukitoid and 34% are sanukitoid-like High-Ca granites (10%)
- HFSE granites (3%)
- Low-Ca granites (<<1%).

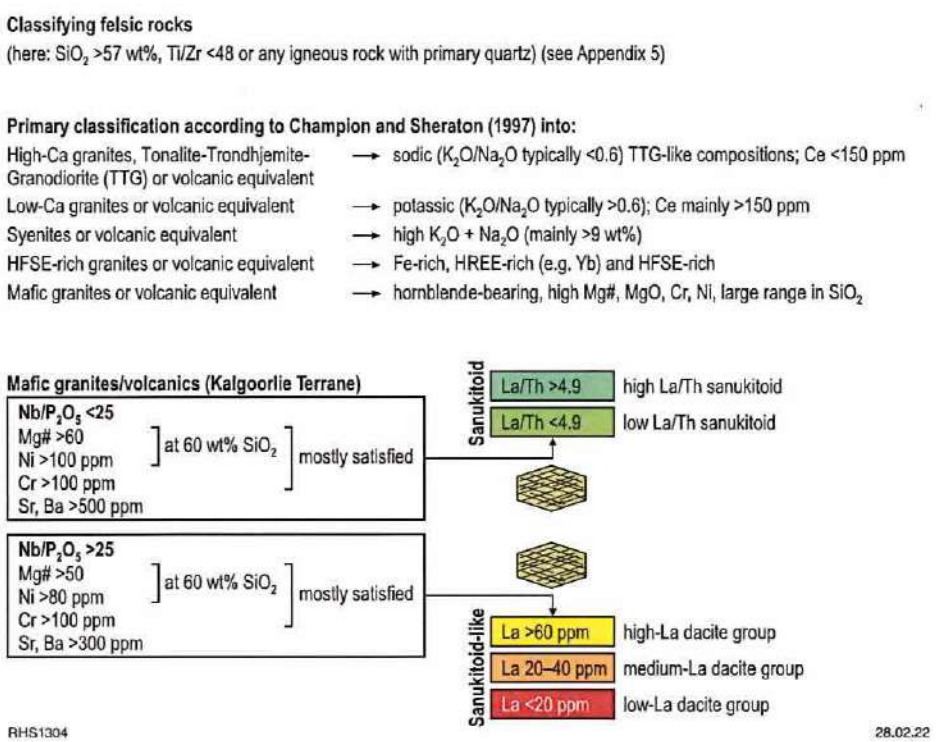


Figure 36. Flow sheet explaining the methodology used here to classify felsic units

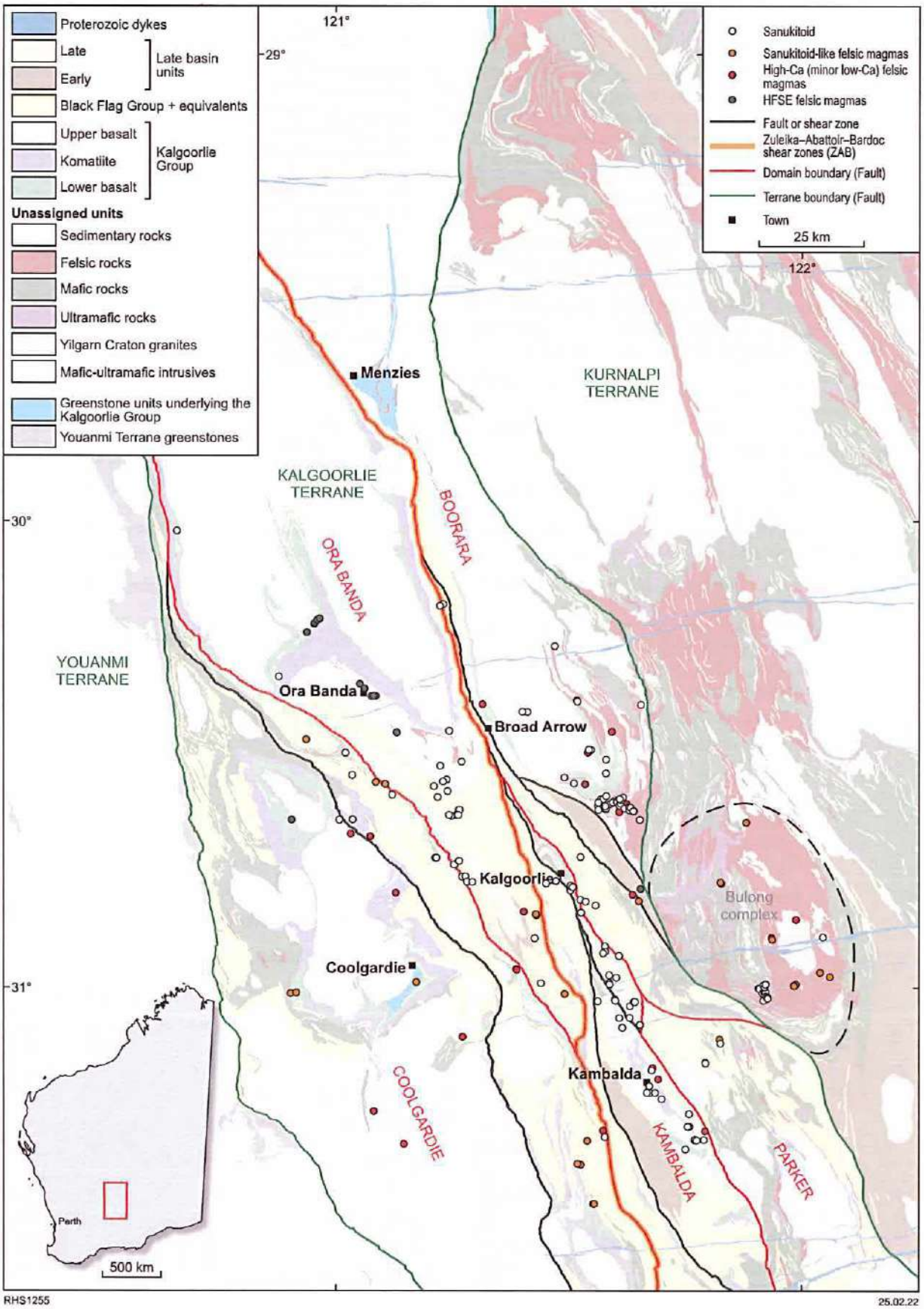


Figure 37. Geological map of the study area within the Kalgoorlie Terrane, showing the distribution of samples of various felsic volcanic and subvolcanic rock studied here (see Figure 34 for fault names)

Rocks compositionally equivalent to Mafic granites

Sanukitoids

Geochemical identification of sanukitoids is not always straightforward and is particularly problematic in populations that contain few samples with silica contents below ~65 wt%. Felsic rocks with low silica contents are relatively uncommon in Archean terrains. They are, however, unusually common in the Kalgoorlie–Kambalda region, particularly in the Black Flag Group. Strictly, a sanukitoid suite should be a geochemically coherent (genetically related) group of relatively sodic (K_2O/Na_2O <1.0) rocks, which at low silica values (at 60 wt% SiO_2 or reasonably extrapolated to that value), are dominated by members with $Mg^{\#}$ ≥60, Cr and Ni concentrations ≥100 ppm, Sr and Ba concentrations ≥500 ppm, and La concentrations ≥25 ppm. Concentrations of HFSE, including Nb and Zr, largely depend on the extent and nature of metasomatic enrichment of the mantle source, i.e. either fluid mediated (low HFSE) or melt mediated (higher HFSE) (Smithies and Champion 2000). At silica content >68 wt% sanukitoid is very difficult to geochemically distinguish from High-Ca granites or rocks of TTG suites. Smithies et al. (2019) suggest some High-Ca granites or TTGs, particularly those showing flat mantle-normalized HREE patterns that reflect hornblende fractionation, are probably fractionated sanukitoids. Nevertheless, at least at silica contents approaching 68 wt%, most sanukitoid suites (and Mafic granites, and volcanic equivalents) should still range to a higher $Mg^{\#}$ and higher Ni and Cr concentrations.

From a more practical point of view, the important aspects of a sanukitoid petrogenesis emphasize derivation from a highly MgO-rich hydrated source. Therefore, any magmatic series that includes low silica (<~65 wt%) members containing abundant primary hornblende, and characterized by anomalously high $Mg^{\#}$, and Cr and Ni concentrations, might also be considered sanukitoid-like (Fig. 36).

Most of the felsic magma series in the Kalgoorlie–Kambalda region that satisfies all or most aspects of the sanukitoid definition (Fig. 38) has a Nb/P_2O_5 ratio <25, whereas most of the other felsic magma types have higher ratios (Fig. 39). Whilst this discrimination appears to be useful for the Kalgoorlie–Kambalda region, even here it fails to identify all relatively hydrated sanukitoid-like magmas (see low-La dacites below). The concentrations of both Nb and P are likely to be strongly related to lithospheric mantle-enrichment processes. We do not recommend the application of this discrimination plot to other regions without further testing.

Nevertheless, considering only the sanukitoids from the Kalgoorlie–Kambalda region (Nb/P_2O_5 ratio <25 where P_2O_5 >0.1 wt%), two main groups – high La/Th sanukitoid (La/Th ≥ 4.9) and low La/Th sanukitoid (La/Th <4.9) – can be distinguished using a plot of La vs Th (Figs 40, 41). These groups are broadly equivalent to the high- and low-LILE subgroups of Mafic granites described by Champion and Cassidy (1998). Both high- and low-La/Th sanukitoid groups span the full silica range from <8 to >75 wt% (Fig. 42). A third, smaller, group has La/Th <4.9, but is restricted to high silica values between 70 wt% and 74 wt%. It has incompatible trace element patterns similar to those of high La/Th sanukitoid, but which are also transitional with silica-rich examples of low La/Th sanukitoids. We referred to this third group as evolved low La/Th sanukitoid (Fig. 40). It shows an extensive overlap with rocks that have compositions equivalent to High-Ca granites (Figs 42–44) (see below) but typically has higher Th and LREE concentrations and lower HREE concentrations. Hence, they have higher La/Yb and Th/Yb ratios as well as higher La/Nb and Sr/Y ratios (Figs 43, 44).

The low La/Th sanukitoid group can be further subdivided. A small subgroup of samples, the P-rich low La/Th sanukitoid subgroup, forms a distinct high-Th (U)-LREE trend. It is characterized by typically higher P_2O_5 concentrations (generally >0.24 wt%) (Fig. 42), although all other major element variations are within those of the main low La/Th sanukitoid group. This subgroup also has distinctly high La/Nb, Th/Nb, and Nd/Zr ratios reflected in mantle-normalized trace element patterns showing more steeply fractionated profiles with a distinct negative Zr (Hf) anomaly (Fig. 45).

The high La/Th sanukitoid group can be further subdivided based on distinctive trends in plots of Ni (and Cr) against SiO_2, which identify a low-Ni subgroup (low-Ni group referred to only where a distinction is required) (Fig. 43). The low-Ni sanukitoid has Ni values lower than those of typical sanukitoids but clearly meets all other requirements of the classification. There is a considerable overlap in Cr, Ni, MgO, and $Mg^{\#}$ between these two subgroups at high silica content, and other than for Cr, Ni, MgO, and $Mg^{\#}$, they are compositionally indistinguishable across the entire silica range (Figs 42, 43).

High La/Th sanukitoids have distinctly more radiogenic Nd-isotope compositions (Table 4), with initial $\varepsilon_{Nd\ (2690\ Ma)}$ values between 1.5 and 2.7 (a single outlier with a value of −3.28 almost certainly reflects an analytical problem and is not considered further), compared with a range for all types of low La/Th sanukitoids between 0 and 0.8 (Fig. 46).

Non-sanukitoid samples compositionally equivalent to Mafic granites

A large population of subvolcanic, volcanic, and volcaniclastic rocks with Mafic granite compositions has La/Th ratios overlapping those of the high and low La/Th sanukitoid groups, but has Nb/P_2O_5 ratios >25 (see 'Other local sanukitoid-like magmas' in Fig. 39). Three groups can be recognized based on distinctive mantle-normalized trace element patterns (Fig. 45d) and are reasonably well distinguished based on their La concentrations (Fig. 43). These will be referred to here as low-, medium- and high-La dacites, noting that the low-La dacite group also includes abundant andesite and the medium- and high-La dacite groups also include rhyolite.

Low-La dacites (La <20 ppm) have relatively flat mantle-normalized trace element patterns with La/Yb ratios <20 and clear positive normalized Zr (Hf) anomalies (Nd/Zr <0.15). Medium-La dacites (La = 20–60 ppm) have more steeply inclined mantle-normalized trace element patterns (La/Yb = 15–70) with only very subtle positive or negative mantle-normalized Zr (Hf) anomalies (Nd/Zr = 0.125 – 0.25). High-La dacites (La>70 ppm) have steeply inclined mantle-normalized trace element patterns (La/Yb >60) and clear negative mantle-normalized Zr (Hf) anomalies (Nd/Zr >0.275). All three types have a range of initial $\varepsilon_{Nd\ (2690\ Ma)}$ values between 0.25 and 2.1, overlapping the range for the high La/Th and low La/Th sanukitoids (Fig. 46).

Apart from generally lower LREE concentrations, the low-La dacites have compositions that closely approach those of sanukitoids, with relatively high $Mg^{\#}$ (mainly 50–55 at 60 wt% SiO_2), Ni (75–100 ppm at 60 wt% SiO_2), and Sr and Ba (both mainly ~300 ppm at 60 wt% SiO_2) concentrations. The concentrations of Cr (at 60 wt% SiO_2) are typical of sanukitoids and the low $Mg^{\#}$ can be attributed to high FeO rather than low MgO, with MgO concentrations within the range for true sanukitoids at a given silica value. We consider these rocks to be sanukitoid-like.

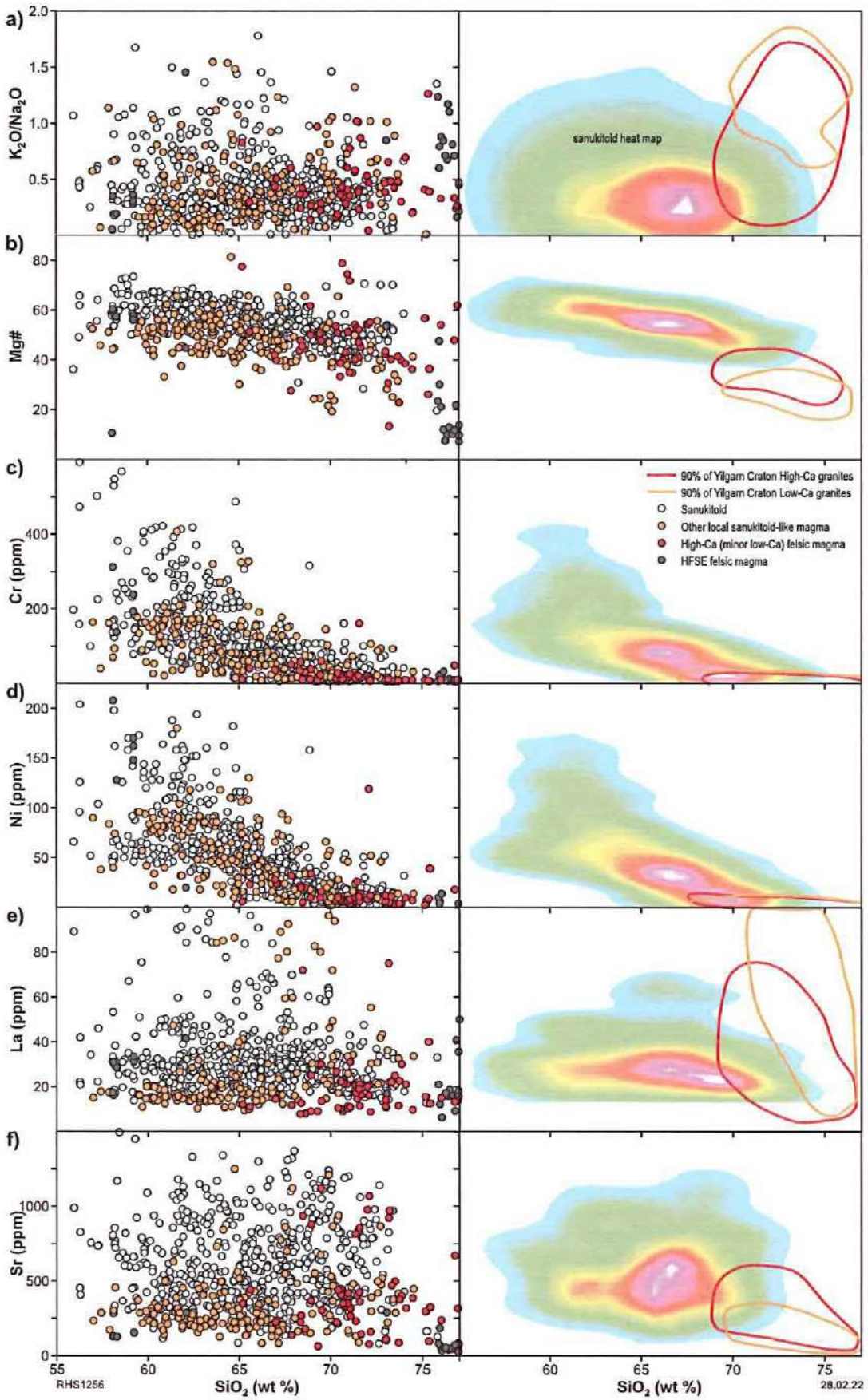


Figure 38. Variation with SiO_2 of various chemical parameters considered useful in the identification of sanukitoids, including K_2O/Na_2O, $Mg^{\#}$, Cr, Ni, La, and Sr. Right-hand column shows heat maps that outline the relative sample density for >95% of sanukitoid samples (hotter colours represent high density of samples) and compares them with the compositional range for High- and Low-Ca granites from the Yilgarn Craton

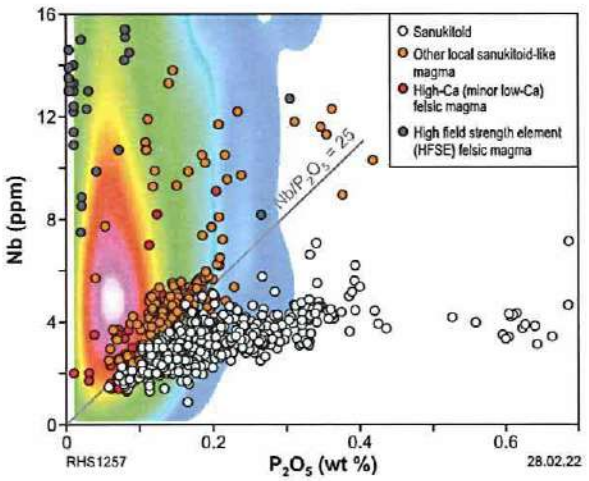


Figure 39. Variation of Nb with P_2O_5 highlighting the compositional contrast between sanukitoids and other felsic rock groups. The heat map shows a density distribution of >90% of granites (other than the Mafic granites) from the Yilgarn Craton

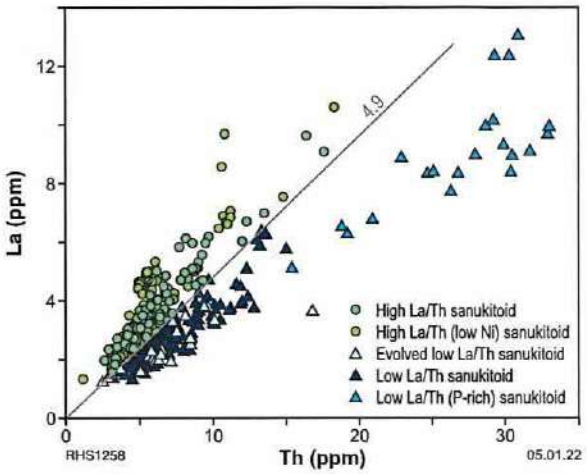


Figure 40. Variation of La with Th for rocks classified as sanukitoids

Nevertheless, the low-La dacites have slightly lower LILE concentrations than typical sanukitoids, and are enriched in HFSE (Ti, Nb, and Y) and HREE and depleted in LREE, with lower La/Nb ratios (Figs 43, 44). The possible explanations for these differences include derivation from a source that was more mafic (e.g. basaltic crust), rather than ultramafic, and less metasomatically enriched than the source for the sanukitoids. Alternatively, the differences might indicate mixing (diluting) of primitive sanukitoid magmas with contemporaneous basaltic magma. It is interesting to note that the high silica end of the compositional range for the low-La dacites merges with the range for local High-Ca granites and these two magma types share a similar geographical range. Hence, an additional hypothesis is that the low-La dacites reflect mixing of a mantle-derived mafic magma with high-Ca felsic crustal melts (or any average Archean felsic component). Barnes and Van Kranendonk (2014) constructed this model to explain the compositions of widespread low-La andesitic to dacitic rocks that formed within the main 2708–2690 Ma period of mafic magmatism in the Kurnalpi Terrane. However, in the case of the low-La dacites in the central Kalgoorlie Terrane, extrapolating the composition of available felsic components through the composition of the lowest-silica low-La dacites (at ~57 wt% SiO_2) to silica contents appropriate for a mafic end-member to any binary mix, identifies a putative mafic end-member with $Mg^{\#}$ >50, TiO_2 >1.0 wt%, P_2O_5 >0.2 wt % and a significant positive mantle-normalized Zr (Hf) anomaly. None of the mafic magmas studied here, except for mafic sanukitoid or the lamprophyres described by Smithies et al. (2018b) meet all these criteria.

Compared with the medium- and high-La dacites, the low-La dacites show a slightly narrower range in silica, but a significantly narrower range in incompatible trace element ratios (La/Sm, La/Yb, Th/Yb, Th/Zr, Dy/Yb) (Fig. 44) sensitive to fractionation of common igneous mafic minerals (e.g. hornblende and pyroxene). Visual descriptions of the low-La dacites indicate hornblende porphyritic intrusive and amygdaloidal (subvolcanic and volcanic) rocks with a very inhomogeneous distribution of mafic and felsic minerals resulting in mafic accumulations, and hornblende-rich layers, bands, and clots. Hence, it is possible that a significant proportion of the silica variation in the low-La dacites might reflect physical processes related to mixing and unmixing of mafic and felsic mineral components, perhaps during eruption from and/or recharge of high-level crystal-rich magma chambers. However, the relatively constant Nb and Th concentrations irrespective of silica content do not support this model. The prospect of an intermediate rather than mafic parental liquid permits the suggestion that these are lower crustal melts rather than mantle melts. However, most $Mg^{\#}$ values concentrate in the 50–55 range, 20% of the rocks have higher Mg#, with <10% of the population having Mg# >60. Such values appear inconsistent with experimental data that suggests that the $Mg^{\#}$ of intermediate melts of mafic crust are typically <40 (Lu et al., 2015). Hence, irrespective of the processes that have promoted compositional variation within the low-La dacites, it seems likely that their parental magmas included mafic melts of hydrated mantle. Our preferred petrogenetic model is that the low-La dacites represent hybrid magmas formed through early mixing of sanukitoid and basaltic magmas.

Medium-La and high-La dacites show many compositional similarities with low-La dacites but extend to more felsic compositions, have steeper (more fractionated) mantle-normalized trace element patterns with minimal or positive Zr anomalies, higher Na_2O+K_2O, and higher FeO and/or lower MgO (and correspondingly lower $Mg^{\#}$ <50) at a given silica content (Fig. 42). The high-La dacites also consistently have high concentrations of other incompatible trace elements including Th, Nb, MREE, and Zr, in some cases similar to (e.g. Th) or greater than (e.g. Nb) the enrichments seen in the sanukitoids (Fig. 43). The medium-La dacites show more scattered enrichments in these trace elements, although typically at higher concentrations than seen in the low-La dacites.

Champion and Cassidy (1998, 2002a,b) show that the Mafic granites of the Eastern Goldfields region can be divided into a series of geochemical populations based on minor variations in major and trace element compositions, probably reflecting similar ranges in the bulk source compositions. These authors recognized two broad end-members – low- and high-LILE – which they subdivided into a number of subgroups (clans). The medium-La dacites have compositions very similar to what Champion and Cassidy (2002a) called the Kanowna Belle clan. The enrichments in incompatible trace elements seen in the medium-La dacites make a petrogenesis through mixing basaltic and felsic (TTG) magmas very unlikely, as with the low-La dacites.

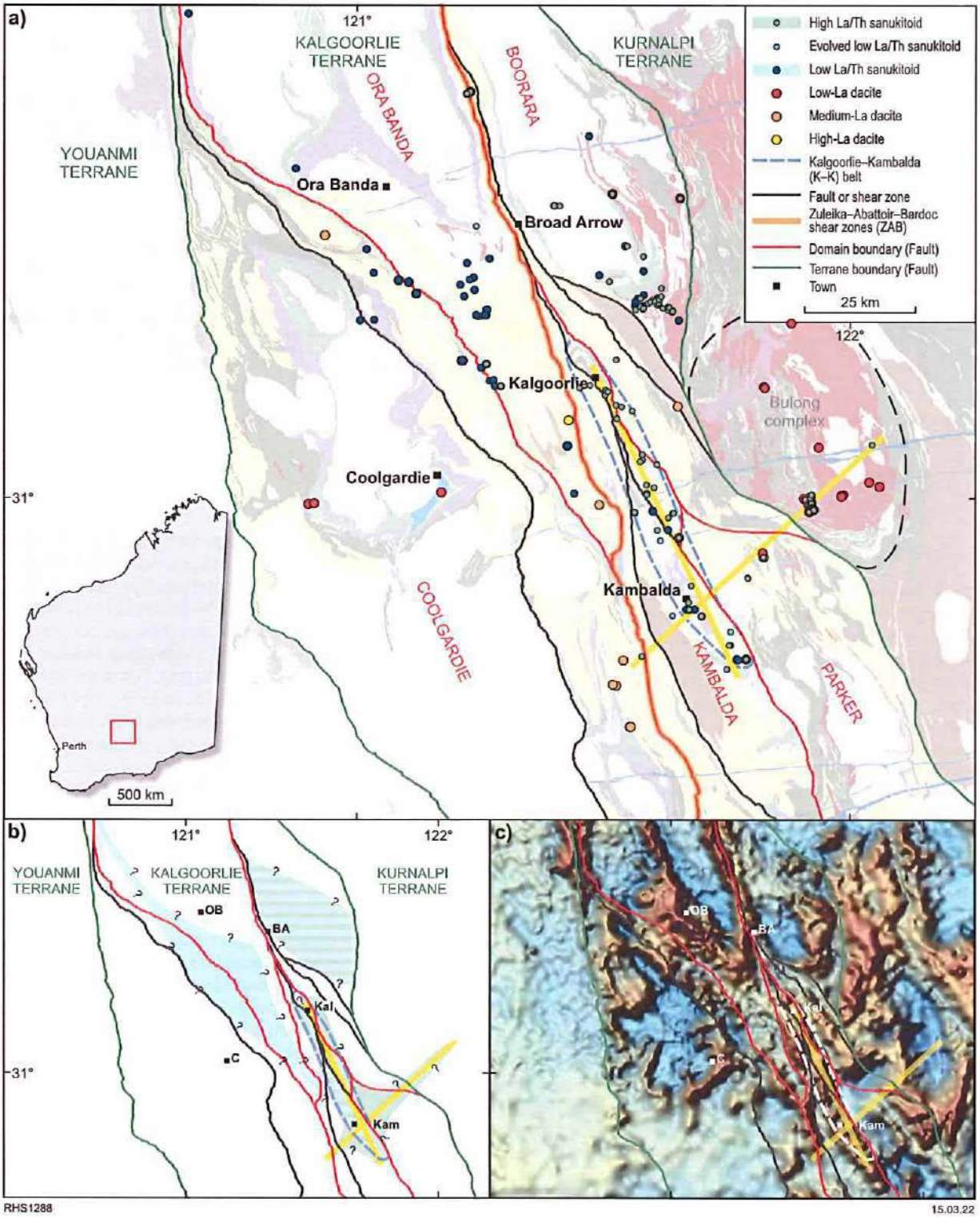


Figure 41. Geological map of the study area within the Kalgoorlie Terrane showing a) the distribution of samples of the various sanukitoid types as well as high-, medium-, and low-La dacites. Also shown is the Kalgoorlie–Kambalda belt (K-K belt: dashed line) and other apparent trends outlined by the sanukitoids (yellow). A summary of these trends and the broad outline of sanukitoid distribution is given in b) and c) compares this distribution with potential trends identified in regional geophysical datasets (in this case 0.5VD Bouguer gravity) (see Figure 34 for fault names)

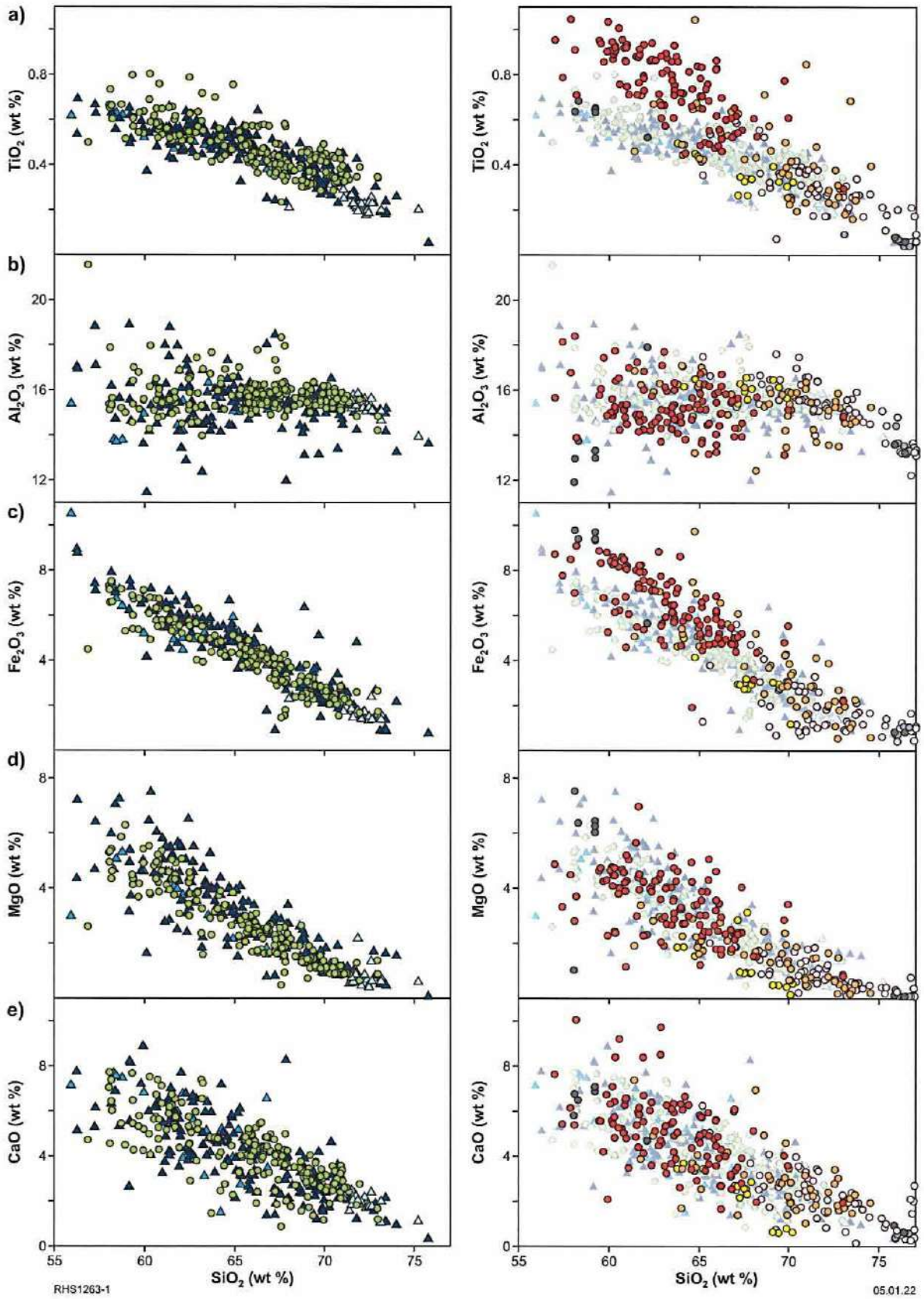


Figure 42. Variation in major elements and $Mg^{\#}$ with SiO_2 for the various felsic rock types. The left-hand column shows sanukitoids. The right-hand column shows the sanukitoid-like rocks (low-, medium-, and high-La dacites) and other felsic units (bold symbols) and compares them with the sanukitoids (faded symbols)

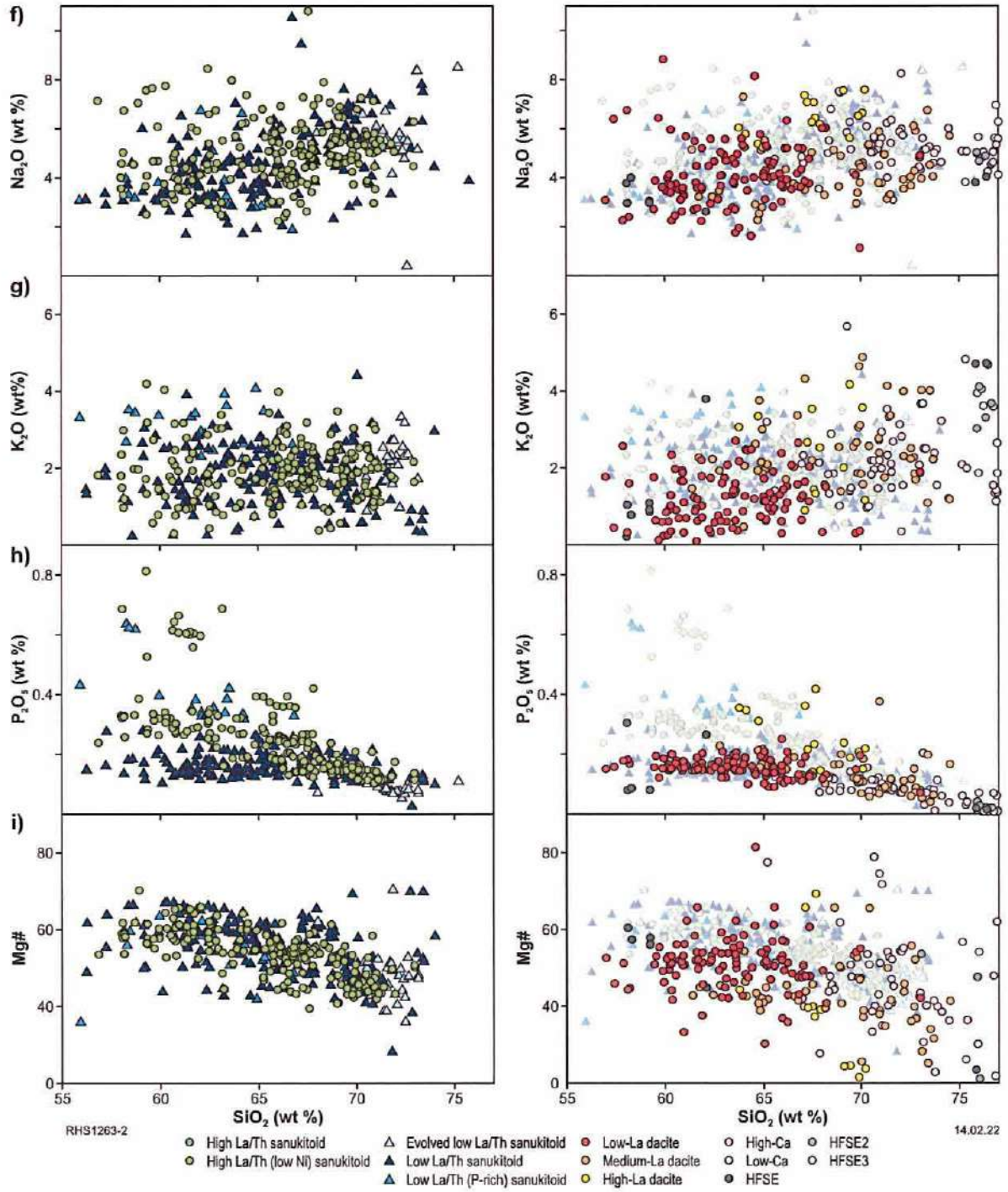


Figure 42. continued

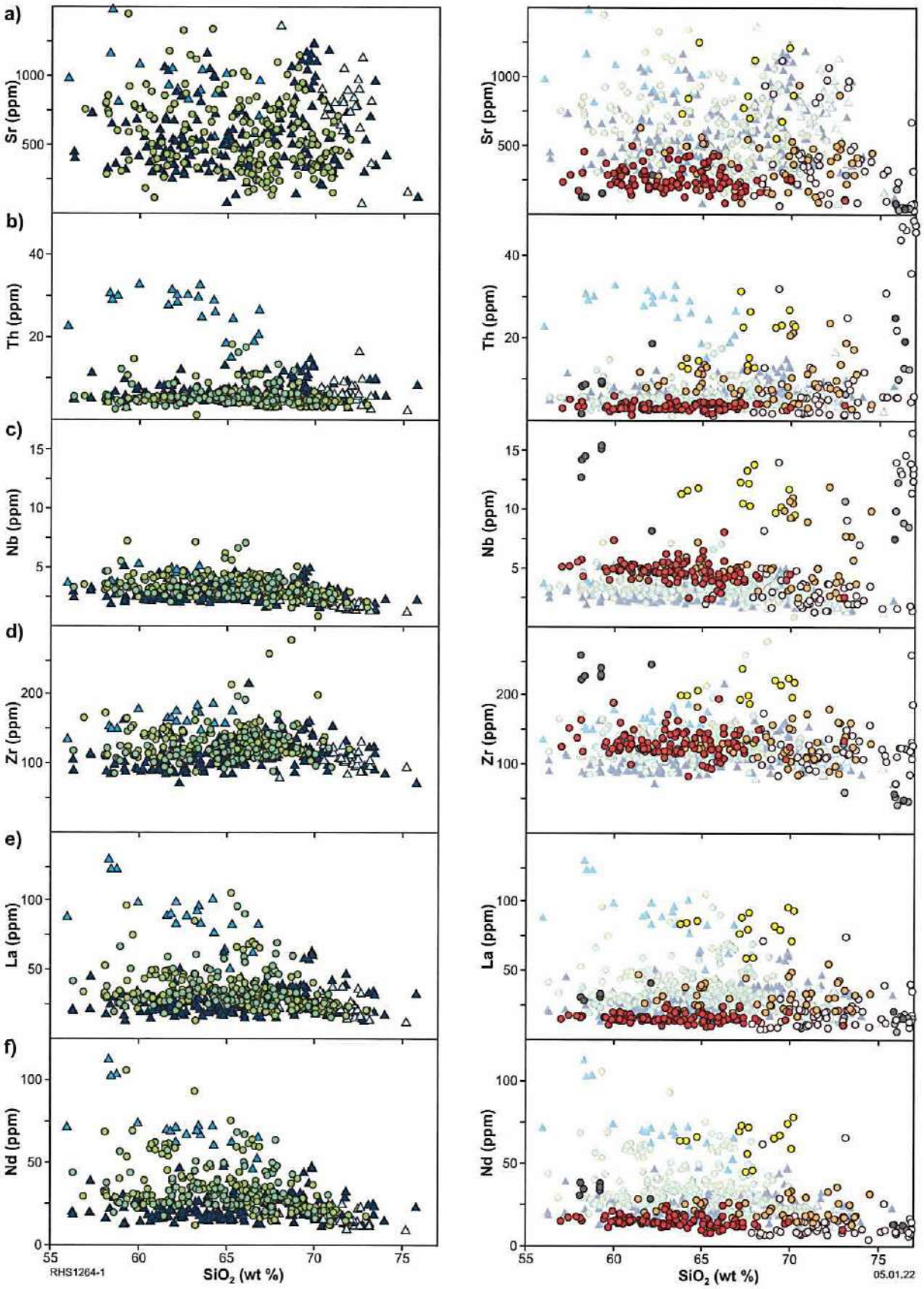


Figure 43. Variation in trace elements with SiO_2 for the various felsic rock types. The left-hand column shows sanukitoids. The right-hand column shows the sanukitoid-like rocks (low-, medium-, and high-La dacites) and other felsic units (bold symbols) and compares them with the sanukitoids (faded symbols)

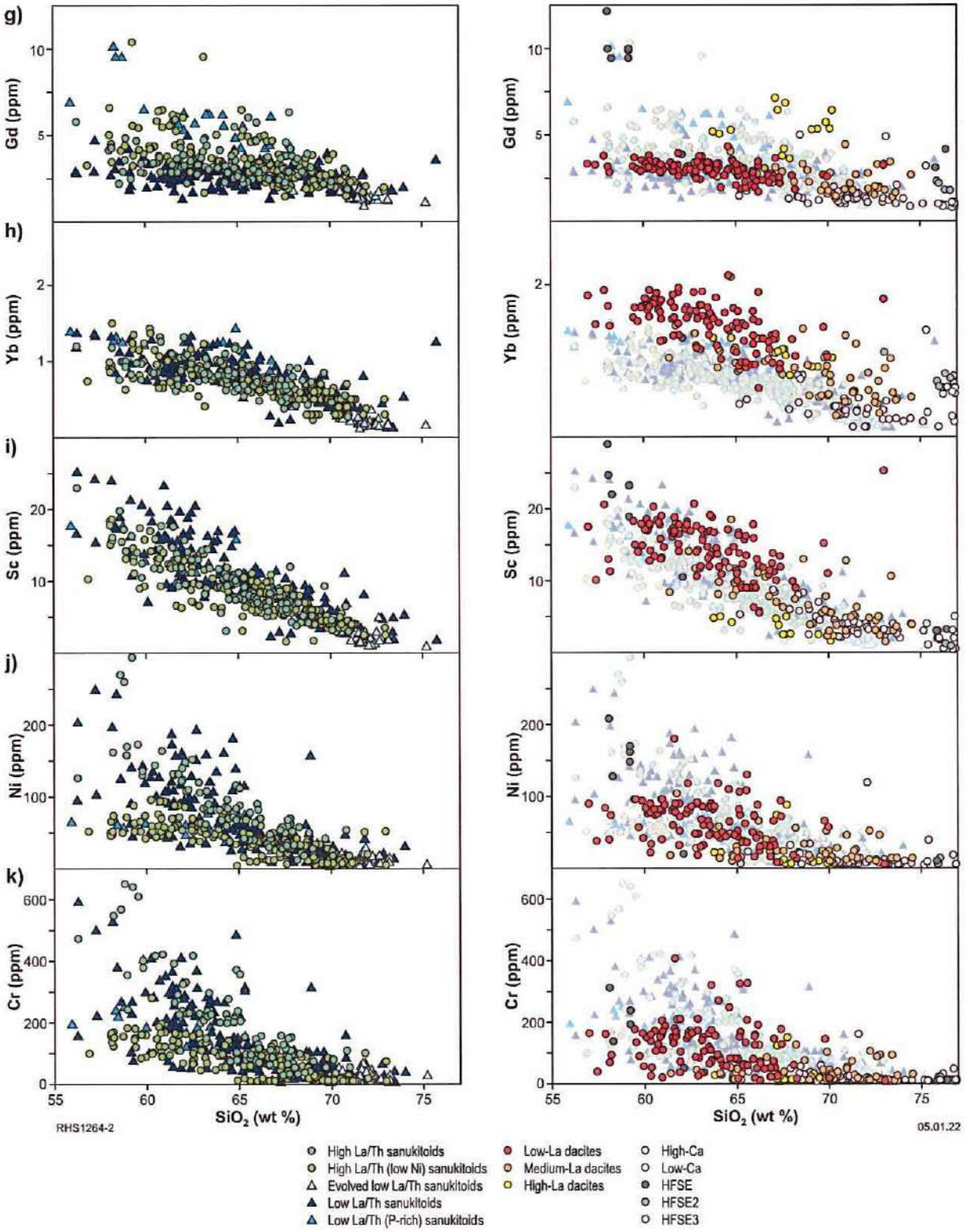


Figure 43. continued

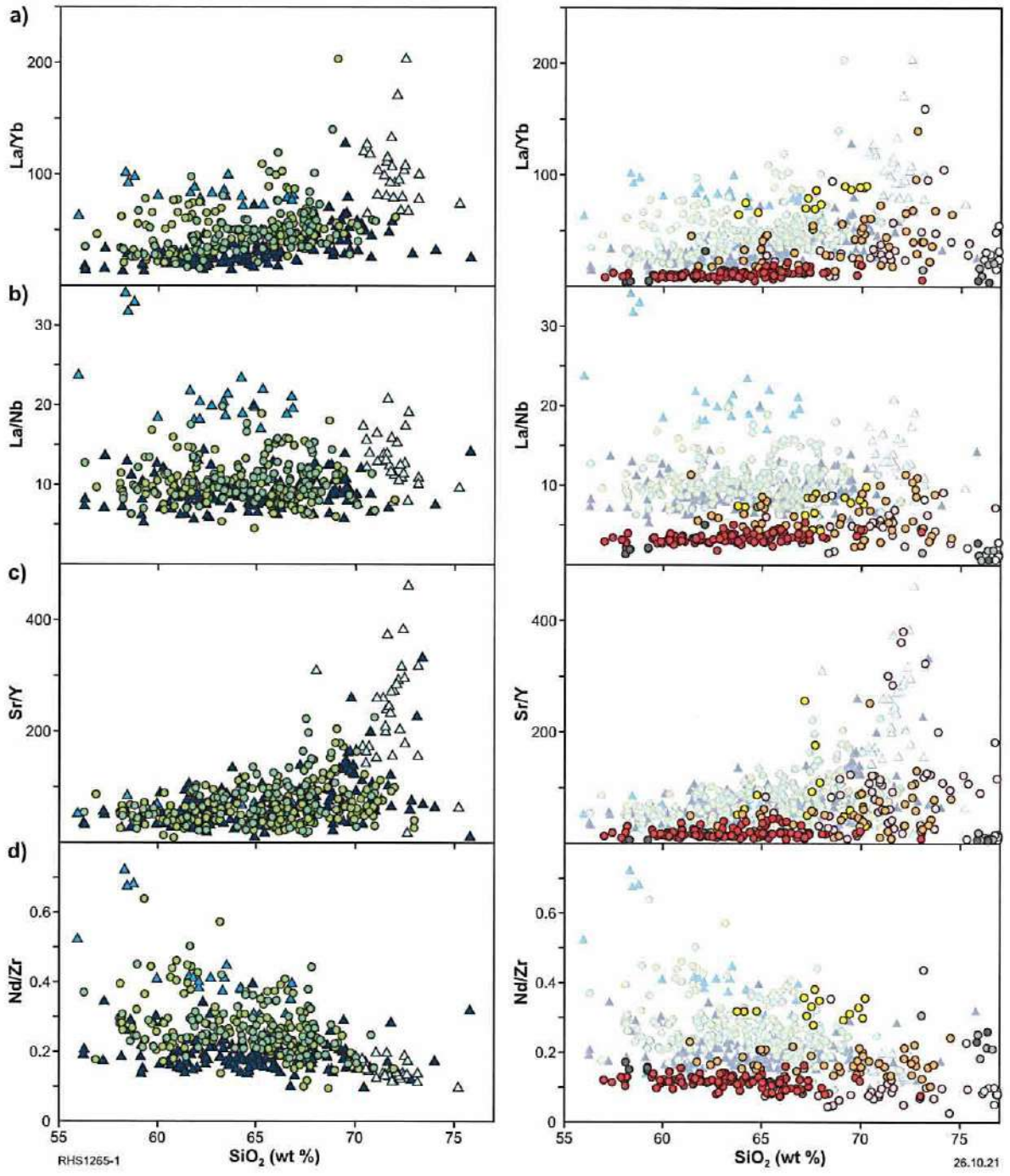


Figure 44. Variation in trace element ratios with SiO_2 for the various felsic rock types. The left-hand column shows sanukitoids. The right-hand column shows the sanukitoid-like rocks (low-, medium-, and high-La dacites) and other felsic units (bold symbols) and compares them with the sanukitoids (faded symbols)

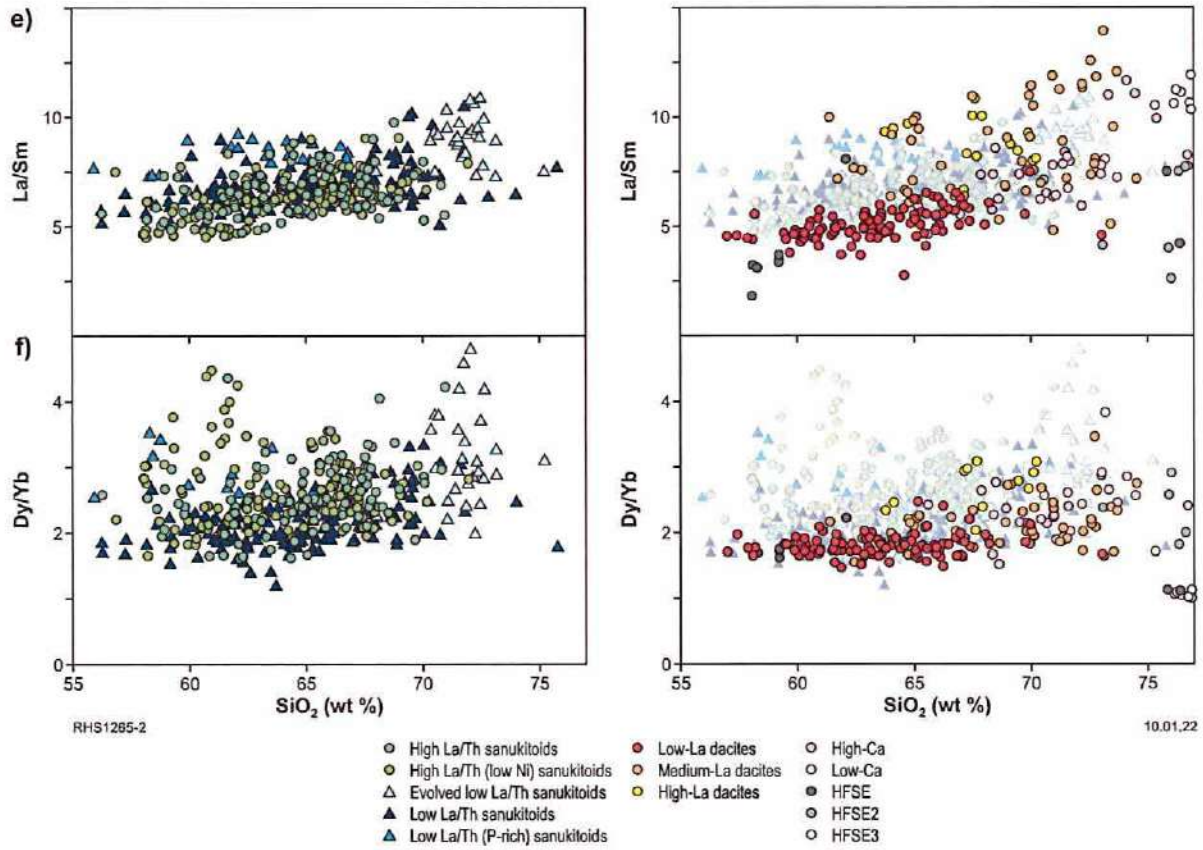


Figure 44. continued

Table 4. Nd isotope data for felsic rocks of the Black Flag Group (sample locations are provided in Appendix 1)

Sample	Classification	Source	Age (Ma)	Age source	Sm (ppm)	Nd (ppm)	$^{147}Sm/^{144}Nd$	$^{143}Nd/^{144}Nd$	1σ	ε_{Nd} (at T)	1σ	T_{DM}^2 (Ga)
230261	Evolved Low La/Th sanukitoid	GSWA	2680	assumed	3.67	25.00	0.0878	0.510742	0.000012	0.49	0.24	2.97
233755	Evolved Low La/Th sanukitoid	GSWA	2680	assumed	2.25	13.70	0.0929	0.510837	0.000014	0.60	0.27	2.96
223074	High La/Th sanukitoid	GSWA	2690	assumed	3.97	21.30	0.0959	0.510953	0.000005	1.94	0.10	2.87
223138	High La/Th sanukitoid	GSWA	2700	assumed	5.27	29.90	0.0961	0.510979	0.000005	2.66	0.10	2.83
223141	High La/Th sanukitoid	GSWA	2700	assumed	10.40	57.30	0.0935	0.510918	0.000005	2.41	0.10	2.85
223153	High La/Th sanukitoid	GSWA	2700	assumed	5.91	33.70	0.0941	0.510957	0.000005	2.98	0.10	2.81
227865	High La/Th sanukitoid	GSWA	2680	assumed	4.06	24.30	0.1042	0.511083	0.000012	1.50	0.24	2.89
227954	High La/Th sanukitoid	GSWA	2690	assumed	11.10	67.10	0.1004	0.510766	0.000005	−3.28	0.10	3.26
229904	High La/Th sanukitoid	GSWA	2690	assumed	8.12	50.00	0.0939	0.510897	0.000005	1.56	0.10	2.90
229913	High La/Th sanukitoid	GSWA	2690	assumed	6.22	36.40	0.0956	0.510923	0.000005	1.47	0.10	2.91
2000969007	High La/Th sanukitoid	Barley et al. (2003)	2662	Champion (2013)	7.50	44.00	0.1034	0.511103	0.000020	1.80	0.39	2.85
211529	High La/Th (low Ni) sanukitoid	GSWA	2690	assumed	7.68	40.50	0.1022	0.511101	0.000005	2.67	0.10	2.81
211547	High La/Th (low Ni) sanukitoid	GSWA	2690	assumed	5.40	34.00	0.0921	0.510908	0.000005	2.41	0.10	2.83
227886	High La/Th (low Ni) sanukitoid	GSWA	2680	assumed	4.77	26.30	0.1067	0.511158	0.000011	2.10	0.21	2.85
227894	High La/Th (low Ni) sanukitoid	GSWA	2690	assumed	6.52	33.00	0.1083	0.511189	0.000005	2.28	0.10	2.84
227909	High La/Th (low Ni) sanukitoid	GSWA	2690	assumed	7.55	45.40	0.0912	0.510881	0.000005	2.17	0.10	2.85
230907	High La/Th (low Ni) sanukitoid	GSWA	2680	assumed	9.80	57.80	0.0986	0.511019	0.000012	2.18	0.24	2.84
204681	Low La/Th sanukitoid	GSWA	2690	assumed	4.96	30.10	0.0994	0.510969	0.000005	1.05	0.10	2.94
224513	Low La/Th sanukitoid	GSWA	2690	assumed	3.54	17.00	0.1022	0.510973	0.000005	0.17	0.10	3.00
233603	Low La/Th sanukitoid	GSWA	2680	assumed	4.22	25.90	0.0970	0.510923	0.000011	0.85	0.22	2.94
233640	Low La/Th sanukitoid	GSWA	2680	assumed	3.56	18.50	0.1078	0.511073	0.000011	0.03	0.22	3.00
230730	Low La/Th (P-rich) sanukitoid	GSWA	2680	assumed	6.39	40.50	0.0963	0.510878	0.000011	0.20	0.22	2.99
230799	Low La/Th (P-rich) sanukitoid	GSWA	2680	assumed	11.1	74.50	0.0894	0.510782	0.000009	0.72	0.17	2.95
224528	Low La/Th (P-rich) sanukitoid (altered)	GSWA	2690	assumed	9.00	47.50	0.0932	0.510842	0.000005	0.70	0.10	2.96
114144	UF Low-La dacite	GSWA	2690	assumed	2.34	12.10	0.1099	0.511208	0.000005	2.07	0.10	2.86
204687	UF Low-La dacite	GSWA	2680	assumed	2.28	11.80	0.1231	0.511354	0.000011	0.25	0.21	2.99
223086	UF Low-La dacite	GSWA	2690	assumed	2.32	10.10	0.1159	0.511283	0.000005	1.46	0.10	2.91
230883	UF Low-La dacite	GSWA	2680	assumed	3.14	15.50	0.1306	0.511528	0.000019	1.09	0.37	2.93
204146	UF Medium La dacite	GSWA	2690	assumed	4.69	33.60	0.0834	0.510715	0.000005	1.64	0.10	2.89
230040	UF Medium La dacite	GSWA	2680	assumed	4.27	26.60	0.1012	0.510719	0.000012	−4.60	0.24	3.35
2001969035	UF Medium La dacite	Barley et al. (2003)	2660	Champion (2013)	2.20	13.00	0.0926	0.510901	0.000020	1.53	0.39	2.86
240405	High-La dacite	GSWA	2680	assumed	7.41	56.10	0.0788	0.510629	0.000009	1.39	0.17	2.90
230996	High-Ca	GSWA	2680	assumed	2.62	13.20	0.1148	0.511245	0.000013	0.99	0.25	2.93
236483	High-Ca	GSWA	2680	assumed	1.53	8.24	0.1050	0.511136	0.000011	2.24	0.21	2.84
2001967017A	High-Ca	Barley et al. (2003)	2667	Champion (2013)	2.00	11.50	0.0917	0.510830	0.000020	0.54	0.39	2.94
2001967019A	High-Ca	Barley et al. (2003)	2675	Champion (2013)	1.80	10.50	0.0955	0.510884	0.000020	0.39	0.39	2.96
223068	High Ca (altered)	GSWA	2690	assumed	2.53	8.90	0.1153	0.511273	0.000005	1.46	0.10	2.91
2001969001	Low-Ca	Barley et al. (2003)	2632	Champion (2013)	3.80	26.00	0.0969	0.510926	0.000020	0.17	0.39	2.94
223194	HFSE	GSWA	2680	assumed	9.06	35.80	0.1478	0.511798	0.000012	0.38	0.24	2.98
2001969039	HFSE	Barley et al (2003)	2650	Champion (2013)	2.80	13.40	0.1224	0.511386	0.000020	0.69	0.39	2.92

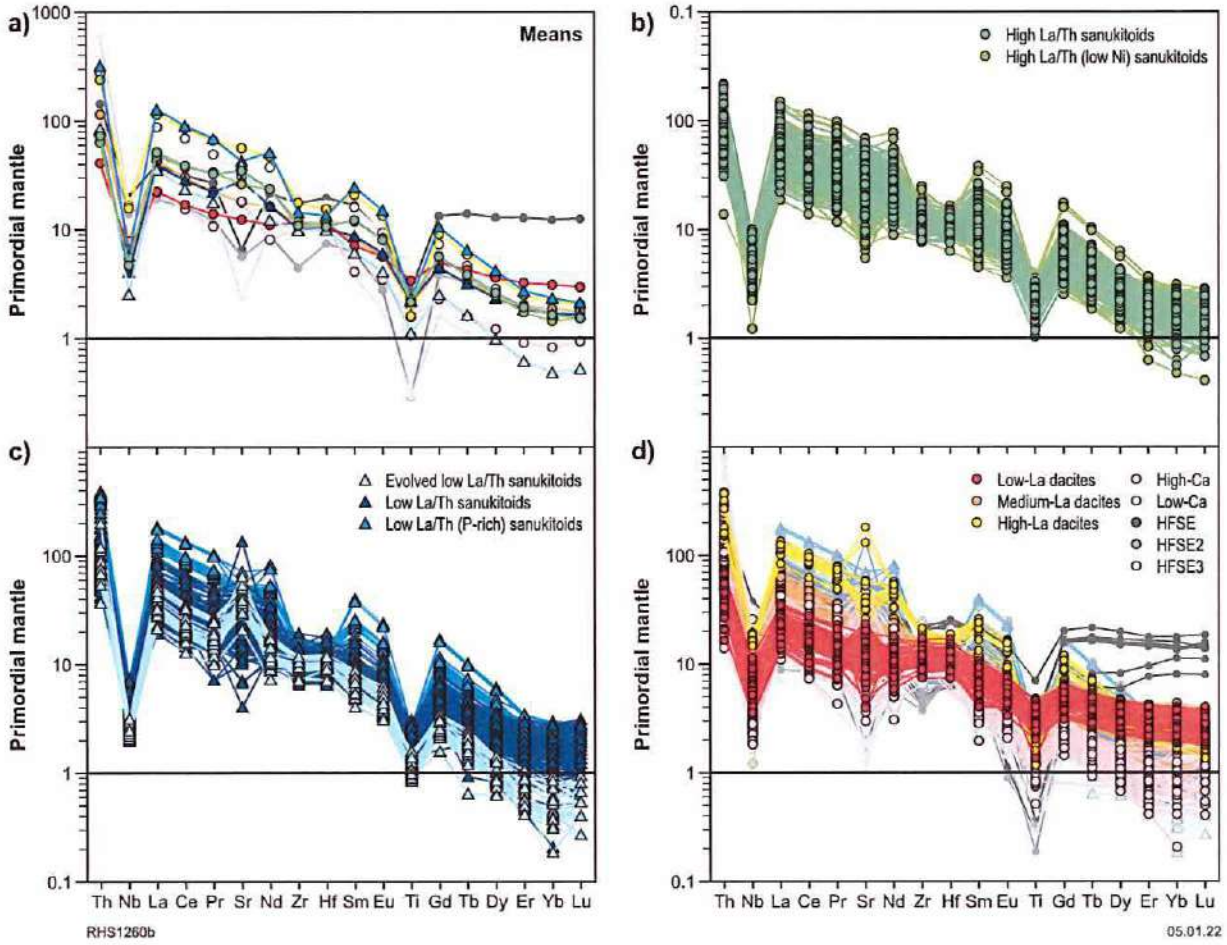


Figure 45. Mantle-normalized trace element patterns for various felsic geochemical groups and units (Normalizing values from Sun and McDonough, 1989)

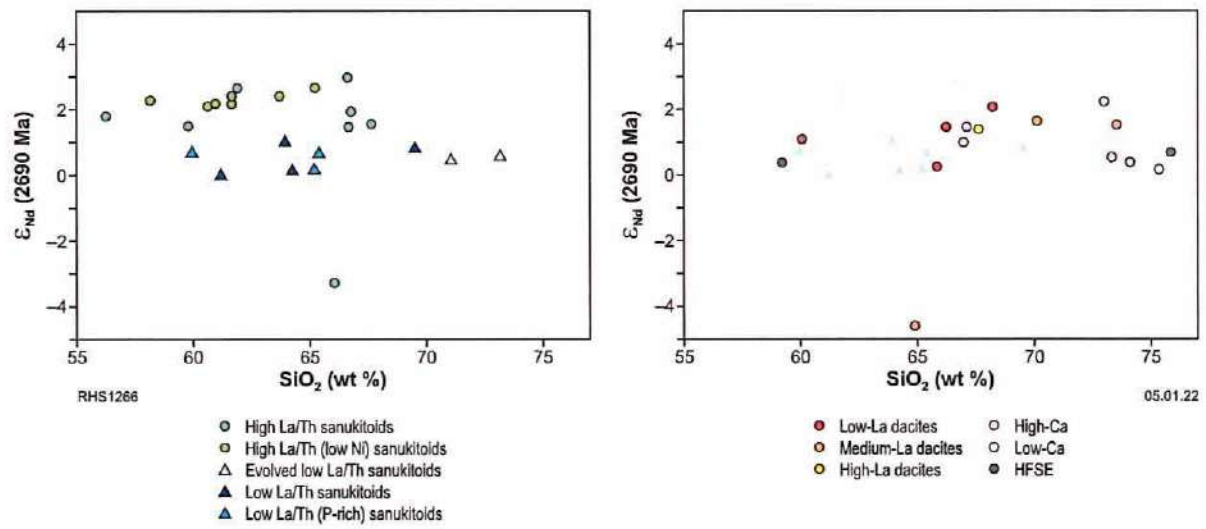


Figure 46. Variation in initial $\varepsilon_{Nd\ (2690\ Ma)}$ values with SiO_2 for the various felsic rock types. The left-hand column shows sanukitoids. The right-hand column shows the sanukitoid-like rocks (low-, medium-, and high-La dacites) and other felsic units (bold symbols) and compares them with the sanukitoids (faded symbols)

The $Mg^{\#}$ for the medium-La dacites is still as high as ~45 at minimum silica contents of ~63 wt% (hence, $Mg^{\#}$ should be even higher, if extrapolated to primitive compositions at ~60 wt% silica), which as with the low-La dacites, also appears to question simple crustal melting as the origin of the parental magmas. Many of the high-La dacites lie at the low silica end of the range for High-Ca granites but are otherwise compositionally very similar to High-Ca granites, particularly at higher silica contents. An exception is for P_2O_5 contents, which are higher in the high-La dacites than in High-Ca granites. For these reasons and the generally hydrated (hornblende-rich) nature of the medium-La dacites (and perhaps the high-La dacites), we suggest that these are also sanukitoid-like. Mixing between sanukitoid magmas and basaltic magmas at or near the source of sanukitoid formation remains our preferred model for producing low-La dacites and possibly most of the medium-La dacites. However, few regionally sampled basaltic melts, when hybridized with melts of sanukitoid composition, have the capacity to evolve to compositions as Nb-enriched as the high-La dacites and some members of the medium-La dacites. Two exceptions include the high-Mg andesitic volcanic rocks that are grouped as geochemical equivalents of HFSE granites (referred to below as HFSE1), and evolved tholeiitic magmas (e.g. L-U7) that otherwise form granophyres within fractionating mafic sills. Rocks with HFSE1 compositions are very rare, whereas tholeiitic sills fractionating to L-U7 compositions are known to intrude the Black Flag Group.

Rocks compositionally equivalent to High-Ca granites

These rocks typically have a restricted range of high silica values (SiO_2 mainly >67 wt%) and are amongst the most Al_2O_3-rich and K_2O-, HREE-, and Th-poor rocks when compared with most sanukitoids and sanukitoid-like rocks (i.e. rocks equivalent with Mafic granites). However, they are indistinguishable from evolved low La/Th sanukitoids in major element composition. Like the latter sanukitoids, the high-Ca rocks have steeply inclined (strongly fractionated) mantle-normalized trace element patterns, but in the case of samples within the study region, these can be distinguished from the evolved sanukitoids, which have slightly less prominent positive Zr (Hf) anomalies, higher Th and LREE concentrations, and lower HREE concentrations. Hence, they have slightly higher La/Yb and Th/Yb ratios as well as higher La/Nb and Sr/Y ratios (Figs 43, 44). The high-Ca rocks examined have La concentrations of ~10 ppm, which is within the lower range of La concentrations expected for High-Ca granites (or TTG), and is consistent with ~20% partial melting of mafic crust with compositions similar to the LTB that forms the main mafic magma composition sampled from the regional greenstones (assuming LTB La concentrations of ~2 ppm).

Rocks compositionally equivalent to high field strength element (HFSE) granites

Three minor groups have been tentatively equated with HFSE granite-like compositions (Figs 42–44). Although many of these samples are of high-level intrusive rocks, volcanic examples also occur (e.g. DDH PMLJH001). These are variably characterized by high Na_2O and/or K_2O concentrations and high total Fe, but they are specifically identified based on prominent enrichments in various incompatible trace elements. Strong enrichments in Nb, Zr and particularly, Yb, at silica contents <60 wt% are diagnostic of HFSE1. Enrichments in U, Y, and Yb and depletions in Zr at silica contents of 72–77 wt% characterize HFSE2 and enrichments in Th and Nb at silica contents of 76–77 wt% characterize HFSE3. The HFSE1 group, in particular, is extremely enriched and has the major element compositions of a high MgO andesite. HFSE1 is characterized at SiO_2 ~59 wt%, by MgO contents typically from 6 to 7.5 wt%, $Mg^{\#}$ >55 (but significantly more FeO rich than the sanukitoids at the same SiO_2 content), and flat mantle-normalized trace element patterns at unusually high concentrations (e.g. Nb >12 ppm, Zr >220 ppm, Yb >6 ppm). The extremely enriched tholeiitic magmas reflect potential parents to A-type granites.

Rocks compositionally equivalent to Low-Ca granites

Three samples have compositions equivalent to Low-Ca granites, these are silica-rich (SiO_2 >68 wt%) rocks, which compared to High-Ca granites are typically potassic (K_2O >3 and mostly >4 wt%), low in CaO and show enrichments in incompatible trace elements, with strong enrichments in LREE being particularly diagnostic. Not considering the magma series that extend to low silica values (i.e. rocks compositionally equivalent to the Mafic granites), a plot of Ce against K_2O proves particularly useful at distinguishing between the high- and low-Ca series (Champion and Cassidy, 2002a).

Spatial and temporal trends

Sanukitoids

Of the diamond drillcores examined, approximately 20 provided information (e.g. stratigraphic succession or intrusive contacts) that might be used to determine relative age differences between felsic rocks of different geochemical groups within segments of the Black Flag Group stratigraphy (Fig. 47). Within the study region, there are 31 samples that have both a published U–Pb zircon radiometric age determination, and have readily available whole-rock geochemical data; these provide absolute age constrains on magmatism (Table 5).

The most striking feature of the geographical distribution of the various felsic geochemical groups relates to the sanukitoids (Fig. 41). Of the two main compositions of sanukitoids, the high La/Th group is concentrated in the southern and eastern parts of the region, while the low La/Th group (excluding the evolved low La/Th subgroup) is mainly concentrated in the north. Sanukitoids in the northwestern part (Kundana region) are exclusively low La/Th types, although high La/Th and low La/Th sanukitoids occur to the northeast, and locally show mutually intrusive (subvolcanic) or interlayered (volcanic) relationships (Fig. 47). There appears to be a very strong structural control on the occurrence of the high La/Th sanukitoids, in a broad north-northwest trending band connecting Kalgoorlie and Kambalda (referred to as the 'K-K belt') within or proximal to the Boulder–Lefroy Fault. To some extent, the association with the fault (and hence the strong linearity of the trend) reflects a sample bias to drillcores sited on mineral exploration targets. Nevertheless, the distribution of the high La/Th sanukitoids in the K-K belt is clearly constrained to within specific fault-bound blocks of the Black Flag Group. In the northern (Kalgoorlie) portion of the K-K belt, the sanukitoids are exclusively of the high La/Th type but in the southern portion high and low La/Th types occur, showing locally mutually intrusive (subvolcanic) or interlayered (volcanic) relationships.

Table 5. U–Pb zircon geochronology of felsic rocks with readily available geochemical data

Sample No.	Classification	Age	Sample material	Latitude	Longitude	Description/notes	Reference
203736	Evolved low La/Th sanukitoid	2680 ± 8	Drillcore (CD20297-W1)	-31.3331	121.7804	Quartz-feldspar porphyry, Victory Flames, Saint Ives, Kambalda region, age of igneous crystallization	Wingate et al., 2022a
230328	Evolved low La/Th sanukitoid	2682 ± 14	Drillcore (LD81635)	-31.2391	121.6997	Feldspar porphyritic intrusion, Kambalda region	Lu et al., 2020a
2004967310	Evolved low La/Th sanukitoid	2693 ± 18	Drillcore	-30.5987	121.5991	Volcanic metasandstone, Kanowna Belle, Ballarat Grit, tentative age from 6 analyses- significant range of common-Pb and discordance	Sircombe et al., 2007
2004967316	Evolved low La/Th sanukitoid	2695 ± 6	Drillcore	-30.6062	121.5896	Volcanic metasandstone, Kanowna Belle, Ballarat Grit, poor quality zircon	Sircombe et al., 2007
2004967377	Evolved low La/Th sanukitoid	2655 ± 15	Outcrop	-30.4496	121.2432	Biotite monzogranite, Mt Ellis, Owen Monzogranite, age of igneous crystallization, low zircon yield	Sircombe et al., 2007
2001969039	HFSE granite	2650 ± 3	Outcrop	-29.8266	121.7281	Seriate to quartz-feldspar porphyritic granite, Donkey Rocks, age of igneous crystallization	Dunphy et al., 2003
2001967017A	High-Ca granite	2657 ± 3	Outcrop	-30.6703	121.0315	K-feldspar megacrystic biotite monzogranite, Camel Dam, Bali Monzogranite, age of igneous crystallization	Dunphy et al., 2003
2001967019A	High-Ca granite	2675 ± 3	Outcrop	-31.2650	121.0806	Biotite quartz-feldspar gneiss, Quairnie Rock, age of igneous crystallization	Dunphy et al., 2003
E7S	High-Ca granite	2698 ± 6	Drillcore	-30.8010	121.6373	Dacite lava, Golden Ridge	Krapež et al., 2000
N8M011	High-Ca granite	2702 ± 4	Drillcore	-30.7883	121.6543	Dacite, Mimbus Mine site, age of igneous crystallization	Hollis et al, 2017
222662	High La/Th sanukitoid	2653 ± 9	Outcrop	-30.5753	121.6164	Porphyritic dacite, Four Mile Hill, 4 km north-northwest of Kanowna	Wingate et al., 2021a
229913	High La/Th sanukitoid	2655 ± 7	Drillcore (LD 7006)	-31.2978	121.7557	Diorite intrusion, Lake Lefroy, southeast of Kambalda	Lu et al., 2020b
245105	High La/Th sanukitoid	2654 ± 4	Drillcore (GDD534)	-30.6191	121.5699	Coarse-grained volcaniclastic or epiclastic (?) rock, Panglo Sandstone, Golden Valley Prospect	Wingate et al., 2021b
245115	High La/Th sanukitoid	2688 ± 6	Drillcore (RHDD18007)	-30.5956	121.6095	Sericitized plagioclase-porphyritic dacitic intrusion, Red Hill porphyry, Kanowna Consols	Wingate et al., 2021c
9796703BG	High La/Th sanukitoid	2656 ± 3	Drillcore (WCVXD-73)	-30.4089	121.4111	Biotite-hornblende granodiorite (Golden Cities granodiorite), collar location Woodcutters prospect, age of igneous crystallization	Fletcher and McNaughton 2002
20009670060	High La/Th sanukitoid	2661 ± 3	Drillcore (MAD2)	-30.7201	121.5253	Aphanitic feldspar-amphibole porphyry clast in polymictic conglomerate, collar ~2.5 km southeast of Parkeston, age of igneous crystallization and maximum depositional age	Fletcher et al., 2001
2000969007	High La/Th sanukitoid	2662 ± 3	Drillcore (SHD13)	-30.9717	121.5783	Massive equigranular to feldspar porphyritic hornblende quartz diorite/tonalite phase of Ellington granite, intrudes Black Flag Group. Collar location ~4 km southeast of Hannan South Mine, age of igneous crystallization	Fletcher et al., 2001
2004967308	High La/Th sanukitoid	2658 ± 7	Drillcore (GVD97)	-30.6105	121.5628	Volcanic metasandstone, Kanowna Belle, Golden Valley Conglomerate, age of igneous crystallization and maximum depositional age	Sircombe et al., 2007
20009670110	Low-Ca granite	2667 ± 3	Drillcore (C029)	-30.8372	121.4030	Intensely (sericite-carbonate-albite) altered fine-grained strongly feldspar-phyric porphyry, intrudes the Black Flag Group. Collar location in area now occupied by the Centurion Pit, age of igneous crystallization	Fletcher et al., 2001
2001969001	Low-Ca granite	2632 ± 4	Outcrop	-30.8542	121.9876	Medium-grained, equigranular biotite monzogranite (Juglah Monzogranite),~4.8 km from Boundary Dam, age of igneous crystallization	Dunphy et al., 2003
E179	Low-Ca granite	2676 ± 5	Outcrop	-30.8447	121.4302	Dacitic conglomerate clasts: Gibson Honman Rock, maximum depositional age	Krapež et al., 2000
211051	Low La/Th sanukitoid	2690 ± 7	Unknown	-30.6319	121.2619	Fragmental hornblende-phyric andesitic rock, west end of White Flag Lake, age of igneous crystallization/maximum depositional age.	Lu et al., 2017
240481	Low La/Th sanukitoid	2694 ± 8	Drillcore (18APD012)	-30.1816	121.2238	Felsic volcanic to volcaniclastic rock probably of dacitic composition, Aphrodite prospect, age of igneous crystallization/maximum depositional age	Wingate et al., 2022b
245127	Low La/Th sanukitoid	2682 ± 4	Drillcore (NGDD19021)	-30.9909	121.4409	Plagioclase porphyritic, volcaniclastic sandstone, Ellen Dam, age of igneous crystallization/maximum depositional age	Wingate et al., 2021d
2004967367	Low La/Th sanukitoid	2696 ± 2	Drillcore	-30.5788	121.2397	Volcanic metasandstone, Three-in-Hand prospect, age of igneous crystallization/maximum depositional age	Sircombe et al., 2007
2004967369	Low La/Th sanukitoid	2689 ± 2	Drillcore	-30.5787	121.2397	Porphyritic micro–diorite sill, Three-in-Hand prospect, age of igneous crystallization	Sircombe et al., 2007
236002	Low-La dacite	2702 ± 6	Drillcore (18EISD001)	-31.0278	121.9182	Metadacite-felsic volcaniclithic breccia, Daisy Milano, age of igneous crystallization	Wingate et al., 2022c
E353	Low-La dacite	2708 ± 5	Outcrop	-30.7758	121.8276	Silicified feldspathic volcanogenic sandstone, Bulong, age of igneous crystallization/maximum depositional age	Kositcin et al., 2008
E357	Low-La dacite	2706 ± 5	Drillcore	-30.3934	121.6552	Volcanogenic conglomerate, Black Swan, age of igneous crystallization/maximum depositional age	Kositcin et al., 2008
E359	Low-La dacite	2708 ± 5	Outcrop	-30.9956	121.9835	Feldspathic volcanogenic sandstone; Mt Monger; age of igneous crystallization/maximum depositional age	Kositcin et al., 2008
245122	Medium-La dacite	2701 ± 5	Drillcore (PDDD18003)	-30.4689	120.9358	Plagioclase porphyritic volcaniclastic sandstone, Paradigm sandstone, Carbine Prospect, Mount Burges Station, age of igneous crystallization/ maximum depositional age	Wingate et al., 2021e

NOTE: Ages are quoted with 95% uncertainties

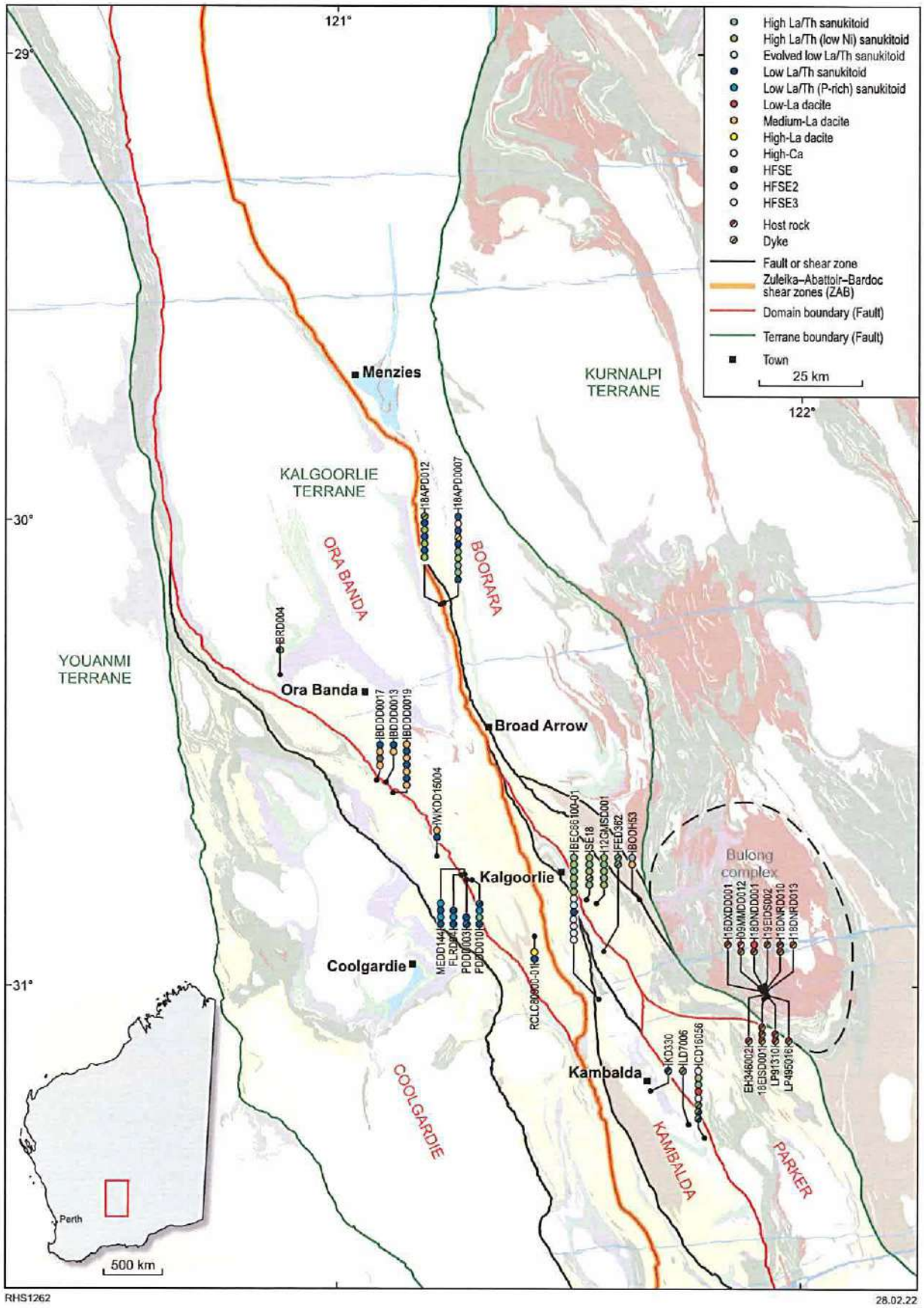


Figure 47. Geological map of the study area within the Kalgoorlie Terrane, showing the locations of sampled diamond drillcores where information on the relative stratigraphic positions of the volcanic or volcaniclastic rocks or intrusive relationships could be obtained

A smaller number of exclusively high La/Th sanukitoids form a prominent northeasterly trend, oblique to the main structural and stratigraphic trends of the greenstones, and intersect the K-K belt in the Kambalda – St Ives region. There is no clear offset in the surface geological features related to this trend, although it is broadly coincident with the southern margin of a regional geophysical anomaly (gravity low) and has the same orientation as cryptic geophysical, isotopic, and geochemical trends previously interpreted to reflect the boundaries of deep basement domains (Smithies et al., 2018c; Gessner et al., 2017). However, the interpretation of the high La/Th sanukitoid trend requires caution, as these samples are primarily from mineral exploration drill targets, themselves possibly investigating the geophysical anomaly itself. Much of the unsampled basement in the region on both sides of the trend is mapped as Black Flag Group volcanic and volcaniclastic rock (possibly high La/Th sanukitoid).

Magmatism relating to the Black Flag Group extends broadly from c. 2690 to c. 2660 Ma (e.g. Squire et al., 2010; Tripp, 2013) (Fig. 48; Table 5). However, sanukitoid magmatism in the form of high-level intrusions appears to extend to younger ages. Examples include the Kanowna Porphyry (a high La/Th sanukitoid according to our data) with ages as young as 2655 ± 6 Ma and 2649 ± 4 Ma (Ross et al., 2004) and other high La/Th sanukitoids within the Kanowna Belle area [e.g. 2653 ± 9 Ma (222662) and 2656 ± 3 Ma (97967038)], although many of these ages remain within uncertainty of 2660 Ma.

Eight U–Pb zircon dates are available for rocks that can be geochemically verified as high La/Th sanukitoids and indicate magmatism between at least 2688 ± 6 Ma and 2653 ± 9 Ma (Fig. 48). Six U–Pb zircon dates are available for rocks that can be geochemically verified as low La/Th sanukitoids (not including evolved low La/Th sanukitoids) and indicate magmatism between at least 2696 ± 2 Ma and 2682 ± 4 Ma. In many cases, the low quality and low yields of zircon and complex age populations are recognized as reasons for treating these data with caution (e.g. Sircombe et al., 2007). Relative age relationships between the evolved low La/Th sanukitoids and other magmatic phases can be determined from several drillhole intersections (Fig. 47). The evolved low La/Th and high La/Th sanukitoids are interlayered in DDH BEC66100-01, but in other intersections, evolved low La/Th sanukitoid volcanic layers typically form the oldest phase. Four out of five U–Pb zircon dates on geochemically verified evolved low La/Th sanukitoids give ages between 2695 ± 6 Ma and 2680 ± 8 Ma. The remaining sample gave a significantly younger age of 2655 ± 15 Ma. Ignoring the younger age, the group of old ages suggests that the evolved low La/Th sanukitoid volcanic rocks form a part of an early phase of sanukitoid magmatism, but only the contact relationships in drillcore provide any indication that these evolved rocks formed before the other low La/Th sanukitoids (Fig. 47).

Contact relationships (Fig. 47) clearly require that high and low La/Th magmatism was comagmatic for at least a portion of the depositional history of the Black Flag Group. However, evidence from the geochemically matched geochronological data (Fig. 48) suggests that this period of overlap may have been quite limited, with much of the high La/Th magmatism (seven of the eight dated samples) occurring after much of the low La/Th magmatism. In some regions, the igneous component of the entire preserved Black Flag Group stratigraphy reflects accumulation from only one of the main compositional types of sanukitoid magmatism. According to the stratigraphic interpretation of Tripp (2013, 2019) (Fig. 49), the northwestern Kundana – Ora Banda region exposes mainly the lower (White Flag Formation) and middle (Gibson–Honman Formation) parts of the Black Flag Group in a south-southeast plunging anticline.

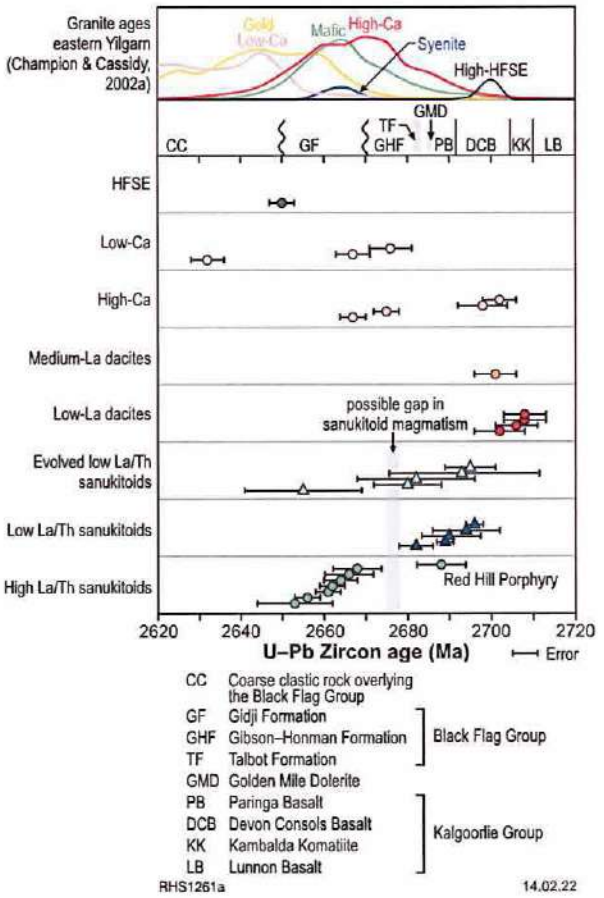


Figure 48. Summary of geochronological data and relationships for felsic units within the study region. See Table 5 for individual U–Pb zircon geochronology results and references. Black Flag Group stratigraphy was modified from Tripp (2013). Histograms showing the age distribution of granite types are from Czarnota et al. (2010), using data from Champion and Cassidy (2002a). Ages are shown with 95% uncertainties

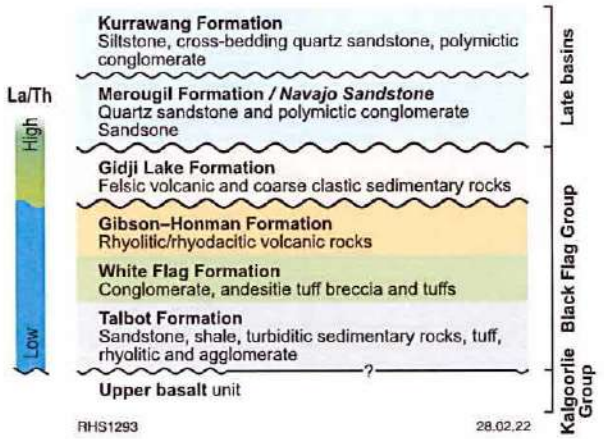


Figure 49. Stratigraphy of the Black Flag Group (modified from Tripp; 2013, 2019)

Two dated samples geochemically confirmed to be low La/Th sanukitoid lie in the region assigned to the White Flag Formation and have crystallization ages of 2696 ± 2 Ma (2004967367) and 2689 ± 2 Ma (2004967369), which is consistent with deposition in the lower stratigraphic levels of the Black Flag Group units (Tripp, 2013, 2019). To the south-southeast, a single dated sample from the Gibson–Honman Formation with available geochemistry (E179; from Gibson–Honman Rocks) has an age of 2678 ± 5 Ma, interpreted to represent the maximum depositional age of a dacite clast in a volcanolithic conglomerate (Krapež et al., 2000). This age is used to characterize the Gibson–Honman Formation (Tripp, 2013, 2019). Although this sample has a composition equivalent to Low-Ca granite, the age is clearly relevant to the low La/Th sanukitoid volcanic rocks that dominate the Black Flag Group of this area.

Although numerous geochronological data exist from the northern part of the K-K belt, geochemical data required to link the compositions of the samples are only readily available for two samples of high La/Th sanukitoids. These give ages of magmatic crystallization of 2661 ± 3 Ma (2000967006) and 2662 ± 3 Ma (2000969007). Our geochemical data indicate that all sanukitoids (intrusive or extrusive) in this region are of the high La/Th variety. These ages are characteristic of the Gidji Lake Formation, which unconformably overlies the Gibson–Honman Formation (Tripp, 2013, 2019) and forms the upper unit of the Black Flag Group.

In the southern part of the K-K belt where high-, low-, and evolved low La/Th sanukitoids occur, dates of geochemically verified samples are only available for high-La/Th sanukitoids and evolved low-La/Th sanukitoids and are from subvolcanic intrusions within greenstone. The single sample of high La/Th sanukitoid gave an age of 2667 ± 7 Ma (229913).

Hence, in broad terms, it seems that the northwestern Kundana – Ora Banda region records or preserves mainly the lower part (Gibson–Honman Formation and below) of the Black Flag Group, overwhelmingly dominated by low La/Th sanukitoids and the northern part of the K-K belt records only the uppermost part (Gidji Lake Formation), overwhelmingly dominated by high La/Th sanukitoids. In this regards, the La/Th ratio of Black Flag Group rocks might provide a first-order estimate of stratigraphic position. The geochemically matched geochronological data (Fig. 48) potentially identifies a small amagmatic gap from 2675–2678 Ma (taking analytical uncertainty into account; although with one notable exception, the high La/Th Red Hill Porphyry sample 245115 dated at 2688 ± 6 Ma). This gap potentially dates the unconformity between the Gidji Lake Formation and the Gibson–Honman Formation described by Tripp (2013, 2019).

Geochronological constrains in the other area of mixed high and low La/Th sanukitoids to the north-northeast of Kalgoorlie are mainly from samples from or near Kanowna Belle. Five of these can be geochemically classified into a specific magma type. Available geochemical data, mainly from intrusive phases, indicate that felsic magmatism in the Kanowna Belle region (i.e. within 5 km of Kanowna Belle mine itself) is dominated by high La/Th sanukitoids (71% high La/Th sanukitoid, 12% low La/Th sanukitoid, 11% high-Ca magmatism, 4% evolved low La/Th sanukitoid volcaniclastic rocks, 1% HFSE intrusion), although the ratio of high- to low-La/Th sanukitoid samples throughout the wider northeastern region is close to unity. Geochemically confirmed high La/Th sanukitoid from the Kanowna Belle region has been dated at 2653 ± 9 Ma (222662) and 2656 ± 3 Ma (97967038), which is consistent with forming subvolcanic equivalents of the Gidji Lake Formation. Geochemically confirmed high La/Th sanukitoid forms a clast within the Golden Valley Conglomerate, with an igneous crystallization age of 2668 ± 7 Ma (2004967308), within uncertainty of the age of both, the Gidji Lake and Gibson–Honman Formation. However, a recent date of the Red Hill Porphyry (245115: Table 5), also a geochemically confirmed high La/Th sanukitoid, gave an igneous crystallization age of 2688 ± 6 Ma, significantly older than the other high La/Th sanukitoids. Several other dated intrusive rocks, unaccompanied by readily available geochemistry, give older ages (e.g. 2674 ± 6 Ma; 2676 ± 4 Ma; 2677 ± 5 Ma; 2682 ± 5 Ma: Kositcin et al., 2008, Krapež et al., 2000, Ross et al., 2004) consistent with forming subvolcanic equivalents of the Gibson–Honman Formation, although not necessarily inferring they are of low La/Th sanukitoid. Two samples from a unit locally known as the Ballarat Grit and geochemically similar to be evolved low La/Th sanukitoid (assuming a relatively juvenile volcaniclastic origin and hence a relatively primary igneous composition) give igneous crystallization ages of 2693 ± 18 Ma (2004967310) and 2695 ± 6 Ma (2004967316) (Sircombe et al., 2007), consistent with forming part of the White Flag Formation.

It should be stressed that Tripp's geochronological compilations (2013, 2019) suggest that volcanism related to the Black Flag Group occurred broadly from <2690 Ma to >2665 Ma, with intrusive magmatism from <2691 Ma to <2650 Ma. We have mainly addressed compositional aspects of Black Flag Group magmatism here, but our data are broadly consistent with that of Tripp (2013, 2019) in that high La/Th compositions have a higher proportion of intrusive samples than the generally older low La/Th sanukitoids.

Geographical trends in Nd-isotope data

One of the most significant compositional distinctions between the high La/Th and low La/Th sanukitoids is the consistently more radiogenic Nd-isotope values of the former (Fig. 46). Hence, although most of the high La/Th sanukitoids appear to be equally or slightly more enriched in incompatible trace elements than the low La/Th sanukitoids at any given $Mg^{\#}$ or silica content (Fig. 43), they are isotopically more primitive (mantle-like). The Nd-isotope data also suggest that the bulk source for the low La/Th sanukitoids included a component that was significantly older (>2.9 Ga) (Fig. 50) than the model ages for the high La/Th sanukitoids (T_{DM}^{2} ~2.85 Ga). Incorporating a larger proportion of a younger mantle component into the source for the high La/Th sanukitoids might satisfy the Nd-isotope constraints but not the observations that the bulk source for the high La/Th sanukitoid may have been slightly more enriched. A more plausible explanation is that the enriched component of the bulk source for high La/Th sanukitoids was compositionally different (relatively enriched in incompatible trace elements) and younger than that in the bulk source for low La/Th sanukitoids. From the spatial viewpoint, the same inferred crustal boundary (ZAB shear zone boundary) separating the western and eastern domains based on asymmetric regional-scale chemostratigraphy differences, also appears to approximate an isotopic boundary. To the west of this boundary, strongly radiogenic high La/Th sanukitoids are absent.

Other magma types

The sanukitoid-like low-La dacite occurs mainly in the southern part of the study region, on either side of the K-K belt (Fig. 41), but appears to be more common in the Mount Monger Station region in the western part of the Bulong complex. Unlike the sanukitoids, the low-La dacites show no relationship between La/Th ratio and geographical distribution. However, the only low-La sanukitoid to the west of the ZAB boundary with existing Nd-isotope data does have a significantly less radiogenic composition (initial $\varepsilon_{Nd\ (2690\ Ma)}$ = 0.25) than the three analysed samples to the east of that boundary (1.1 – 2.1), consistent with the relationships shown by the sanukitoids.

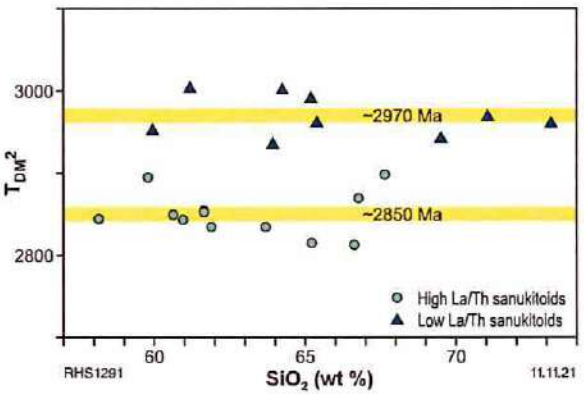


Figure 50. Variation in depleted mantle model ages (T_{DM}^2) with SiO_2 for the high- and low- La/Th sanukitoids

Four samples geochemically verifiable as low-La dacites have been dated, all to the east of the K-K belt, with three of these in the Mount Monger Station region. All dates fall within a narrow age range from 2708 to 2702 Ma. In all 10 drillcores to the east of the K-K belt that show contact relationships between low-La dacite and other units, the dacite was consistently intruded by subvolcanic dykes of high La/Th sanukitoid and was the oldest observed felsic component (Fig. 47). Of the samples collected to the west of the K-K belt (and ZAB), one is a felsic intrusion into rocks of the Brilliant Formation in the lower Coolgardie sequence and the other two are felsic volcanic rocks stratigraphically within or beneath the Lindsays Basalt, and subsequently at least as old as the lowest part of the upper Coolgardie sequence. The first sample possibly pre-dates the Black Flag Group, but the second two samples clearly pre-date the Black Flag Group. The two low-La dacites sampled within the K-K belt (DDH PFC43200-2 and CD16056) do not show clear stratigraphic relationships; neither sample GSWA 233792 from PFC43200-2 nor the units above and below it show clear evidence of a volcanic origin. Hence, based on available data, it is quite likely that the low-La dacite represents a c. 2706 Ma felsic volcanic event clearly older than the Black Flag Group. Thus, the low-La dacite compositions potentially provide a stratigraphic time marker. Along with the La/Th ratio of sanukitoids, these potential compositional age references might prove useful in elucidating field relationships, although they should be considered with caution.

Medium-La dacite was not found within the K-K belt but was sampled west and east of that belt and along the full north–south extent of the study region. It was also sampled in several drillcore intersections from the Kundana region (e.g. BDDD0019; Fig. 47) to the west-northwest of Kalgoorlie, where it is interlayered with low La/Th sanukitoid. These two compositional types are difficult to distinguish visually and include juvenile volcanolithic deposits, where the clasts and matrix are compositionally indistinguishable from each other. A single age determination from a geochemically verifiable medium-La dacite (245122, Table 5) from the Mount Burges Station (Carbine Prospect) area yielded an interpreted magmatic crystallization age of 2701 ± 5 Ma. This age is possibly slightly younger than low-La dacite magmatism and slightly older than low La/Th sanukitoid magmatism, but within uncertainty of both, although some depositional overlap with low La/Th sanukitoid is required, based on crosscutting relationships observed in drillcore BDDD0019.

Like the sanukitoids, the medium-La dacites can be divided into low- and high-La/Th subgroups, each with a distinctive spatial distribution. Medium-La dacites with high La/Th ratios occur mainly to the east of the K-K belt, like the high La/Th sanukitoid. Medium-La dacites with low La/Th occur along a distinct north-northwesterly trend that extends the full north–south extent of the study region.

High-La dacite is less common than the low- and medium-La dacite but forms part of the Black Flag Group to the west of the K-K belt. No radiometric ages exist for geochemically verifiable high-La dacite, although it is interlayered with low La/Th sanukitoid and must share at least part of the magmatic age range for those rocks.

Volcanic and subvolcanic rocks with compositions similar to High-Ca granites are widespread throughout the study region (Fig. 37). They also occur within the K-K belt, where they form volcanic interlayers with high La/Th sanukitoid and are likewise interlayered with low La/Th sanukitoid in the Kundana region (Fig. 47). Two samples of geochemically verifiable high-Ca volcanic rocks, give U–Pb ages of 2702 ± 4 Ma and 2698 ± 6 Ma (Table 5). However, composite granite plutons within the region, including the Bali Monzogranite, also have High-Ca granite components dated between 2675 Ma and 2667 Ma (Table 5).

Volcanic and subvolcanic rocks with compositions similar to Low-Ca granites are rare and seem restricted to the central part of the region (east and west of Kalgoorlie). Two closely spaced geochemically verifiable samples form components of the Black Flag Group, one with a U–Pb age of 2667 ± 3 Ma (Fletcher et al., 2001) and the other, from Gibson–Honman Rock (Krapež et al., 2000), with an age of 2676 ± 5 Ma. Low-Ca granite forming domal composite granite bodies within the region have crystallization ages as young as 2632 ± 4 Ma (Dunphy et al., 2003).

Volcanic and subvolcanic rocks with compositions similar to HFSE granites are mainly restricted to the northern part of the study region. They form volcanic interlayers with medium-La dacite within the Kundana (DDH BDDD0017) and Nimbus (DDH BODH53) regions, suggesting immediately pre-Black Flag Group ages around c. 2700Ma. However, a single U–Pb age determination from an intrusive HFSE magma (informally know as Donkey Rocks Monzogranite) gave an age of 2650 ± 3 Ma (Dunphy et al., 2003).

Regional tectonomagmatic evolution and implications

One outcome of constructing the greenstone geochemical barcode, apart from the chemostratigraphic functionality, is that it also provides data that permit the effective mapping of magmatic and lithospheric architectural evolution. Both these appear to ignore many of the boundaries previously used to construct domains and potentially even terranes. The new data also maps specific events of potential importance in magmatic evolution in the lead up to Au mineralization.

As discussed earlier, changes in compositional proxies for crustal contamination (e.g. Th/Ti, La/Nb, Th/Nb) and fractional crystallization (e.g. Al_2O_3/TiO_2, $Mg^{\#}$), particularly above lower crustal levels (e.g. low Al_2O_3/TiO_2), reveal systematic changes in magma ascent history throughout the evolution of the mafic sequences. Minimal evidence for either crustal contamination or fractional crystallization during emplacement of the oldest mafic units (L-U1, L-U2) is consistent with rapid magma transfer through dyke

intrusion, which is also the likely case for the ascent of the slightly later komatiitic magmas. This, along with high inferred magma volumes supports a general rifting environment for the early history of the Kalgoorlie Group. Clear evidence for contamination and fractional crystallization begins before komatiite magmatism (e.g. L-Ca) in the western domains (i.e. the Coolgardie and Ora Banda Domains), possibly as extension decreases in the thicker crust to the west (Fig. 32). Such evidence only occurs after komatiite magmatism in the eastern domains (i.e. Kambalda, Boorara, Parker Domains and Bulong complex). The subsequent increase in the degree of contamination (LTB→ITB→HTSB) with increasing stratigraphic height is likely a combined effect of the production of higher degree partial melts of the mantle, and of the progressive thermal weakening of the lower crust through ongoing mantle magma input (including komatiite magmatism at c. 2706 Ma), making assimilation progressively easier. The higher degree melts have significantly lower incompatible trace element concentrations and hence, even small additions of crustal material produce significant changes in trace element geochemistry (and particularly in ratios), often with little change in major element composition.

The formation of the upper basaltic unit, and the transition into and the early stages of Black Flag Group deposition, perhaps reflect the most magmatically dynamic period of the Eastern Goldfields greenstone evolution. Many units forming the upper basaltic stratigraphy clearly erupted from mid- to upper crustal chambers (many initially filled by L-U2 magmas) undergoing extensive fractional crystallization (e.g. L-U3–7). This change to a more magmatically dynamic upper crustal regime probably accompanies an overall decrease in extension, at least in the upper crust, and probably some magmatic thickening of the crust. The result was a tendency for magmas to pond, forming chambers large enough to undergo significant fractionation prior to eruption triggers. The HTSB lavas are also abundant in the upper mafic stratigraphy of the eastern domains. Their scarcity in the western domains presumably either reflects an eastward migration in the axis of magma flux or topographical controls (i.e. fault movement) after (i.e. erosion), before or during magmatism. Recharge by and mixing of HTSB into some of the mid- to upper crustal (L-U3–7) chambers locally produced extremely enriched L-S units (e.g. L-S3a, 3b, L-S4) and likely provided eruption triggers.

A dynamic lower crustal/upper mantle environment is also suggested by the formation of HTSB itself. These have ultramafic parental compositions (Barnes et al., 2021) like earlier komatiitic magmatism, represent a sudden change in magma composition with little, if any, compositional transition, and in the case of HTSB magmatism, include an unprecedented influx of crustal material. There is very little difference in major element compositions between the more primitive ITB (I-S1) and the more primitive HTSB (H-HMg#) magmas (Fig. 21). Both approximate the transition between komatiite and komatiitic basalt compositions in terms of MgO vs Mg#. The primary distinction is in the evidence in the HTSB, either for significantly more crustal contamination or contamination by a bulk crustal component with a higher Th/Nb ratio, or both. For both magma types, contamination of the very primitive parental compositions occurred at a very early stage and at very deep crustal levels, and no significant contamination accompanied further compositional evolution. Barnes et al., (2021) suggest that the high and constant Th/Nb ratio in HTSB probably reflects ultramafic parental magmas with very low initial incompatible trace element concentrations that have assimilated enough crust for the initial magmatic Th/Nb ratios to be overprinted by values reflecting the contaminant. Nevertheless, within the range of magmas with similar MgO, Mg#, and Al_2O_3/TiO_2 ratios as the HTSB, none have Th/Nb ratios as high as HTSB and only the rare KB3 and H-LTh magmas have Th/Nb ratios that partially bridge the compositional gap between the HTSB and all other magmas (Fig. 51). The H-LTh can be adequately explained in terms of mixing or contamination between komatiite and ITB at the emplacement scale. Similarly, KB3 is typically stratigraphically associated with komatiite and probably simply represents significantly, but only locally, contaminated komatiites.

Compositional similarities between L-U3–7 lavas (e.g. Victorious Basalt) and samples from layered sills (e.g. Golden Mile Dolerite) emplaced within and above the Kalgoorlie Group, as well as emplacement of HTSB into the Black Flag Group (e.g. Powder Gabbro), indicate that the latest stages of upper basalt deposition and the early stages of Black Flag Group deposition reflect a period of maximum magmatic diversity, which also included low La/Th sanukitoid magmatism. This magmatic stew was potentially initiated by a catastrophic lower crustal event that also initiated HTSB magmatism.

Sanukitoid-like low-La dacites (Fig. 36) formed as early as c. 2708 Ma to 2702 Ma, more or less synchronously with komatiite magmatism (Fig. 48). Evidence presented here suggests that the hydrated, hornblende-bearing, low-La dacites are unlikely to represent any form of hybridization between felsic crustal melts and mantle-derived mafic or ultramafic magmas.

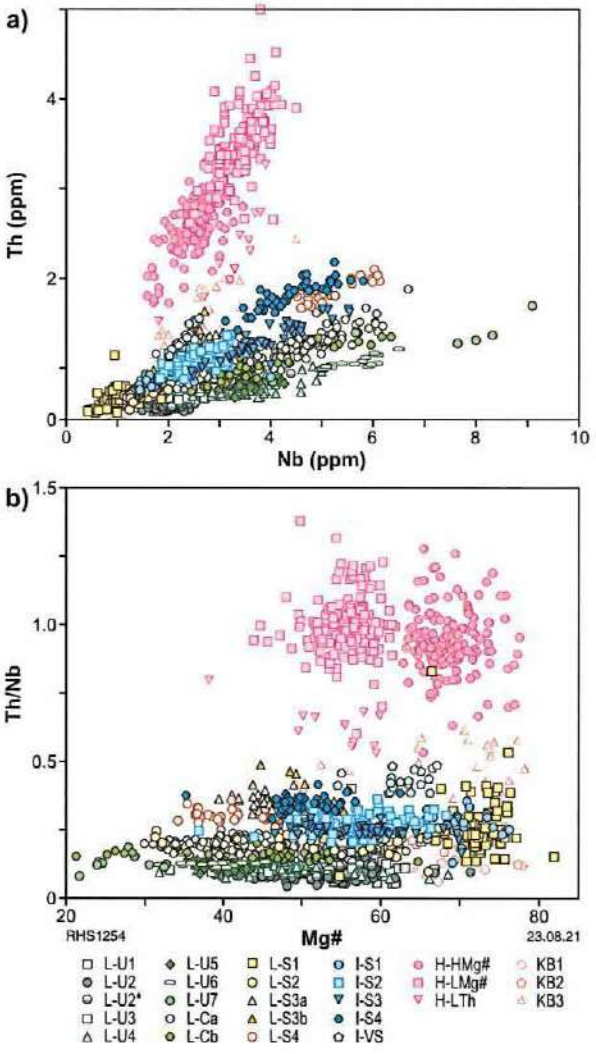


Figure 51. Variation in Th and Th/Nb with Mg# for individual LTB, ITB, HTSB and KB geochemical groups and units

In addition, in the absence of clear evidence that subduction processes directly influenced the evolution of the Kalgoorlie Terrane at this stage, the low-La dacites are unlikely to reflect melting of a recently established metasomatized mantle wedge. Nevertheless, they do reflect melting of hydrated ultramafic lower crust or mantle lithosphere. The hydrated and hornblende-bearing, medium-La dacites appear to have formed at a slightly later stage (c. 2701 Ma and younger) that overlaps with low La/Th sanukitoid magmatism at a stage when HTSB and L-U3–7 (and hence also remnant L-U2) magmatism was still active (Fig. 48). Thus, despite the dominance of voluminous and nominally dry, mafic, and ultramafic magmatism, hydrated lithospheric mantle source regions were still either preserved or being created and then partially melted throughout the period from c. 2708 Ma to c. 2660 Ma.

Recent work by Murphy et al., (2021) has provided clear evidence that varioles in basalt (i.e. in variolitic basalts), like the HTSB Paringa Basalt (but also the slightly older and immediately post-komatiite I-S2 Devon Consuls Basalt), can reflect flux melting of a hydrated mantle source. Processes that might hydrate a mantle source include subduction-like processes or lower crustal delamination or dripping events (e.g. Murphy et al., 2021). An issue posed by the abundance of HTSB magmatism, 5–10 Myr after komatiite magmatism, and by a proposed hydrated mantle source for voluminous variolitic ITB and HTSM magmas, is whether this represents an intractable anomaly within the plume paradigms commonly employed to explain komatiite. Irrespective of this issue, it is inviting to suggest that a significant post-komatiite (post-plume?) mantle disturbance event reinvigorated high-degree mantle melting more or less at the stage where evidence for primarily extension-related magma transfer was waning. Whatever the process, these ultramafic parental HTSB magmas underwent significantly greater interaction with or contamination by, lower crustal material than any preceding or subsequent mafic or felsic magma, including the sanukitoids and sanukitoid-like magmas, many of which are isotopically more primitive than the HTSB. One possibility is that the collapse of the mantle upwelling feeding komatiite magmatism resulted in a rapid and temporary transition to compression (e.g. Ueda et al., 2008; Gerya et al., 2015) that destabilized thermally weakened lower crust and remnant lithospheric mantle. The suggestion that variolitic basalt formation can reflect transfer of significant amounts of volatile components to the mantle (Murphy et al., 2021), also provides a mechanism of hydrating the lithospheric mantle sources for sanukitoids. Hence, HTSB-forming events might be crucial steps in sanukitoid petrogenesis and in providing a source of fluids that eventually might feed mineral systems.

The greenstone geochemical barcode presents a very asymmetric picture of the distribution of various characteristic mafic and felsic lithologies across the study area. The observed lithogeochemical variations, on their own, provide no basis for separating the Coolgardie Domain (upper sequence) from the Ora Banda Domain or the Kambalda Domain from either the Boorara and Parker Domains or the Bulong complex. These variations do, however, suggest that the bulk lithospheric source component contributing throughout mafic magmatic evolution in the western domains was either compositionally distinct or architecturally distinct, and most likely both, from that contributing to the magmatic evolution of the eastern domains. The surface expression of the boundary between these two bulk mantle source regions is now separated along a combined ZAB shear zone, as previously hinted by Hayman et al. (2015), based on the distribution of their enriched lower basalt unit.

Hence, prior to komatiite magmatism, early L-U1 and L-U2 magmatism in the western domains was replaced by L-Ca magmas, with compositions not seen in the east. At higher stratigraphic levels, L-S1 and L-S2 magmas followed komatiite magmatism in the west but not the east. In L-S1 and L-S2 magmas, the combination of unusually high Al_2O_3/TiO_2 (>>20), slightly higher SiO_2, and lower Ni concentrations suggests an already refractory (melt depleted), and probably more harzburgitic lithospheric mantle source contributed to magmatism in this region. The absence of this magmatism to the east of the ZAB suggests either that the refractory harzburgitic source was already removed (? delaminated, thermally eroded) as the focus of rifting migrated eastwards or that the ZAB is the upper crustal expression of the boundary between two already distinct lithospheric domains.

Compositional differences in lithospheric source components across the ZAB are reflected in the geochemical and Nd-isotope data for the sanukitoids and also reflected in the Nd isotopes data from the low-La dacite. Virtually all high La/Th sanukitoid magmatism is restricted to the east and is characterized by radiogenic $\varepsilon_{Nd\ (2690\ Ma)}$ values >1.5. The slightly older low La/Th sanukitoid occurs east and west of the ZAB. Nd-isotope data are available for two samples within the K-K belt and seven samples to the west of the ZAB and all give $\varepsilon_{Nd\ (2690\ Ma)}$ values <1.05 (Fig. 45). The model ages (T_{DM}^2) for the two groups are clearly distinct (Fig. 50), with average values for low La/Th sanukitoids of c. 2970 Ma (n=9) and c. 2850 Ma (n=12) for high La/Th sanukitoids. Thus, the ZAB is not only the surface expression of the boundary separating the bulk source regions for mafic magmatism, but also of the zone where fundamental changes occurred in the sources of sanukitoid magmatism.

The available data provide no clear indication whether the differences in $\varepsilon_{Nd\ (2690\ Ma)}$ or T_{DM}^2 values between high- and low-La/Th sanukitoids reflect distinct source compositions or different mixing proportions between low ε_{Nd} lithospheric mantle (and crust) and high ε_{Nd} juvenile asthenospheric mantle. For example, although the high La/Th sanukitoids appear to be slightly more enriched in incompatible trace elements than the low La/Th sanukitoids, there is no clear correlation between La/Th ratios and either $\varepsilon_{Nd\ (2690\ Ma)}$ or (T_{DM}^2) values within or between the sanukitoid groups (Fig. 52), which would support a separate source. Conversely, both La/Sm and Th/Nb ratios appear to broadly decrease with increasing $\varepsilon_{Nd\ (2690\ Ma)}$ or decreasing T_{DM}^2, suggesting lithosphere–asthenosphere mixing. Irrespective, these data indicate that crustal material at least as old as c. 2970 Ma forms a major component of the bulk source for the low La/Th sanukitoids and material at least as old as c. 2850 Ma forms a component of the bulk source for the high La/Th sanukitoids.

Pawley et al., (2012), Mole et al., (2015, 2019), and Smithies et al., (2018a) have suggested that the Eastern Goldfields Superterrane developed as a rift over a basement of Youanmi and/or Burtville Terrane crust. There is no clear requirement that this suggested rift ever evolved to the extent that new oceanic crust and micro-continental ribbons were created, and subsequently reassembled (e.g. Smithies et al., 2018a), although models involving subduction and arc accretion during the c. 2730–2660 Ma period (e.g. Morris and Kirkland, 2013; Witt et al., 2018) cannot be discounted. Nevertheless, the enriched signatures interpreted to reflect subduction-like processes, seen in some mafic and felsic magmas associated with formation of the Superterrane, are more easily accounted for in terms of melting lithospheric mantle that was metasomatized during subduction events related to the evolution of the Youanmi Terrain, rather than the Eastern Goldfields Superterrane (Smithies et al., 2018a).

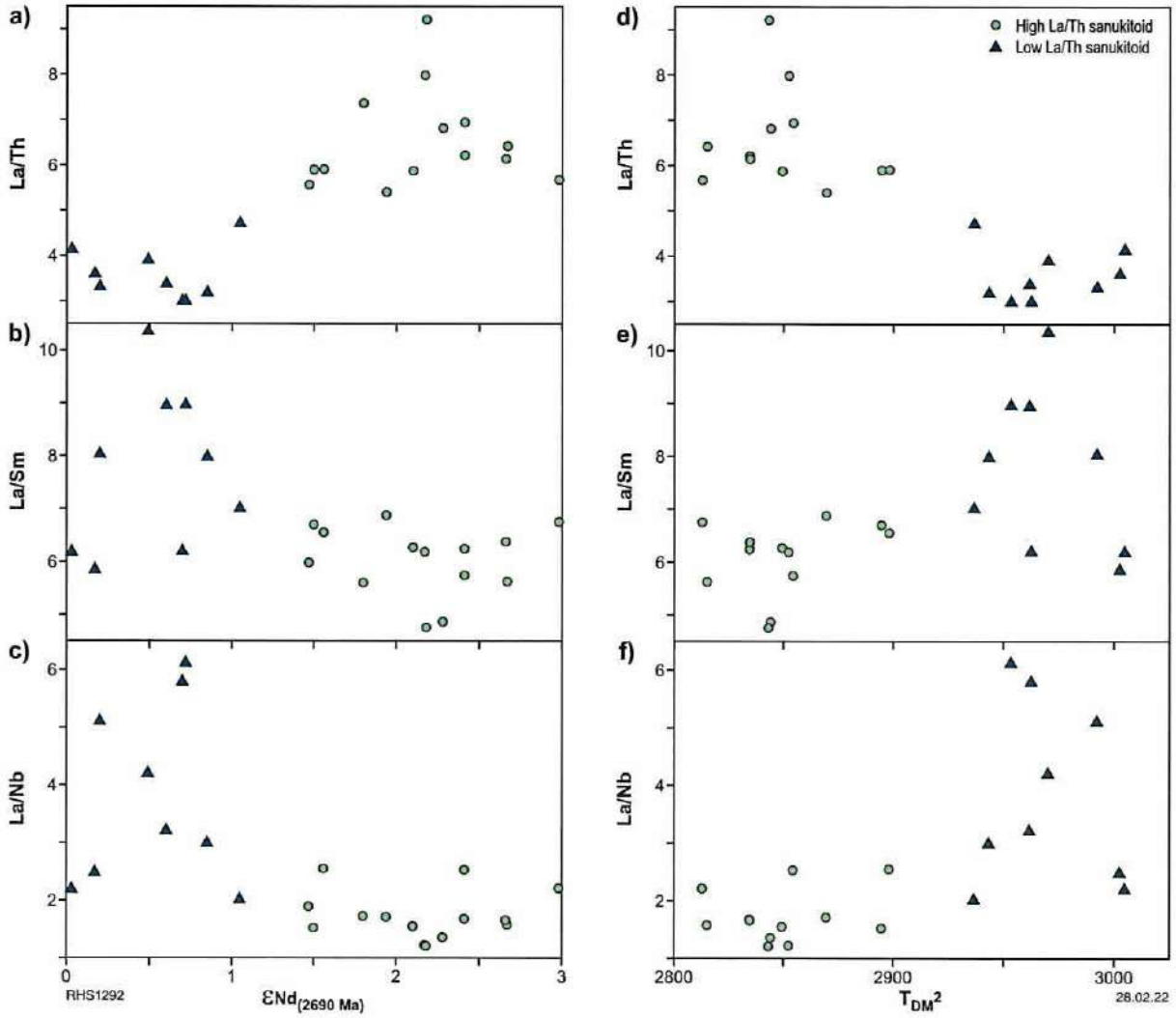


Figure 52. Variation in La/Th, La/Sm, and La/Nb ratios with initial $\varepsilon_{Nd\ (2690\ Ma)}$ values and depleted mantle model ages (T_{DM}^2) for the high- and low- La/Th sanukitoids

Such events correspond with the c. 2970 and c. 2850 Ma T_{DM}^2 ages for the low- and high- La/Th sanukitoids, respectively (e.g. Lowrey et al., 2020). The simplest explanation is that hydrated and enriched c. 2970 Ma and c. 2850 Ma Youanmi lithospheric mantle, perhaps additionally hydrated as a result of events leading to HTSB formation, was the main source for these sanukitoids.

The broadly north–south ZAB segment defines the local contours in the Nd-isotope compilation map of the Yilgarn Craton (Lu et al., 2021) (Fig. 53), separating felsic magma sources with T_{DM}^2 = 2920–3000 Ma in the west from those with T_{DM}^2 = 2800–2920 Ma to the east. From the isotopic data perspective, this contour indicates some form of lithospheric structure or gradient considerably more clearly than any data or region of data corresponding with the Ida Fault. If the distinct $\varepsilon_{Nd\ (2690\ Ma)}$ values shown by the available data for the high and low La/Th sanukitoids reflect two compositionally distinct lithospheric mantle sources, then the ZAB is the surface expression of a lower lithospheric boundary. Alternatively, and probably less likely, if the distinct $\varepsilon_{Nd\ (2690\ Ma)}$ values shown for the high and low La/Th sanukitoids reflect a significant change in the mix of asthenosphere mantle melt with melts derived from a single, homogeneous, lithospheric mantle source, then T_{DM}^2 gradients in this region might define a significant change in the extent of lower crustal extension (rifting).

Irrespective of these alternatives, it seems clear that the highly Au-endowed region examined here, directly overlies a region where geochemical and isotopic gradients define a significant feature in the regional lithosphere architecture. The arguments presented by Hayman et al. (2015) further suggest that the same or similar feature is observed in the Agnew area in the far north of the Ora Banda Domain, and it is possible that its trace will be effectively defined northwards and southwards by detailed mapping of the Nd-isotopic gradients and more integrated geochemical-stratigraphic work.

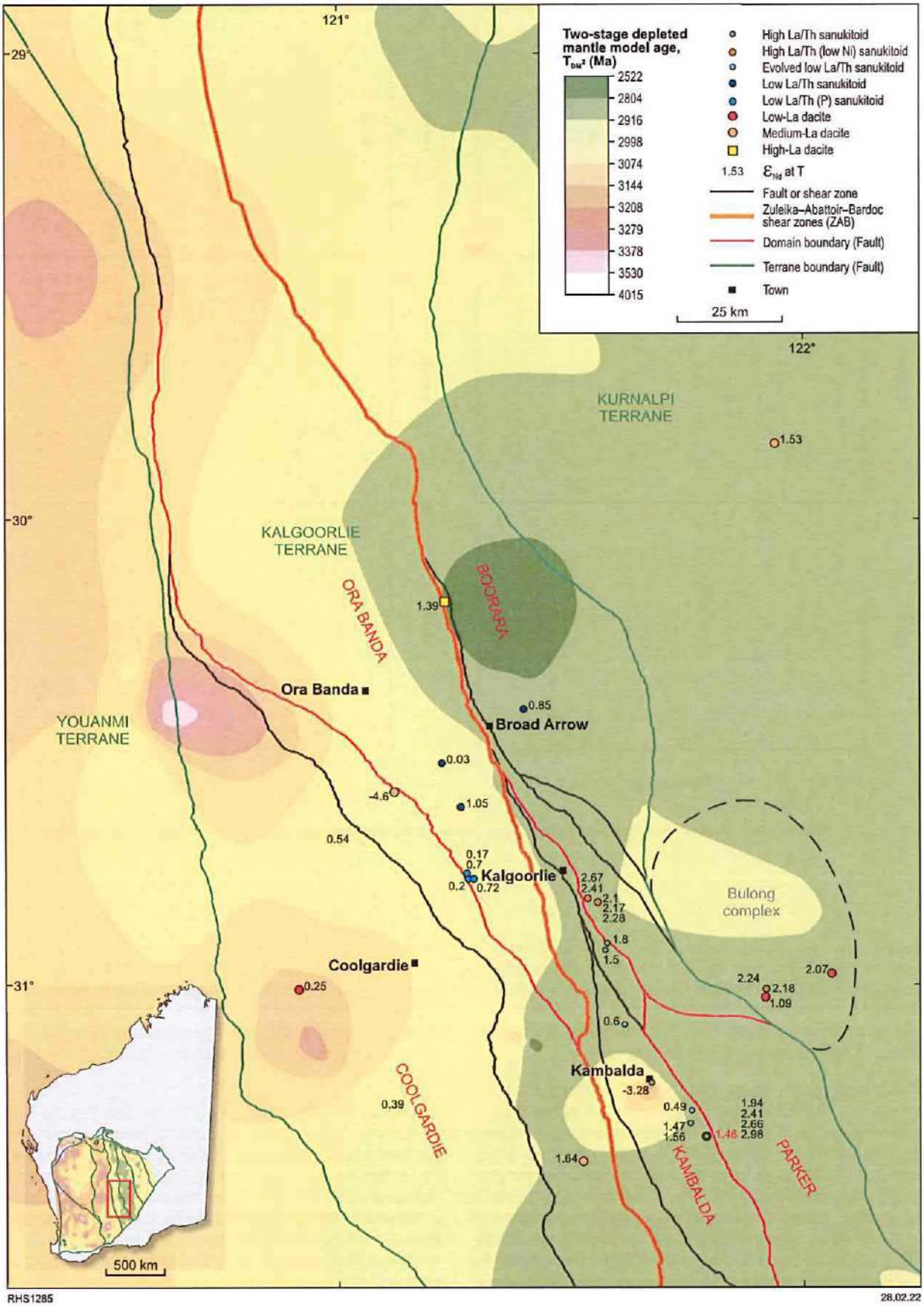


Figure 53. Part of the Yilgarn Craton Nd-isotope map (Lu et al., 2021) covering the study area and contoured for two-stage depleted mantle model ages (T_{DM}^2). The location and initial $\varepsilon_{Nd\ (2690\ Ma)}$ values of samples taken for the present study (Table 4) are also indicated and show the close correspondence of the ZAB shear zones with maximum T_{DM}^2 contour. This correspondence breaks down to the north and the relationships in that area are constrained by very few Nd-isotope data

Conclusions

The interpretations of data presented here have been significantly influenced by a wealth of field and laboratory experience and expertise from government, industry and university investigators. The interpretations, nevertheless, remain heavily based on geochemical trends and their validity or applicability needs to be carefully considered on a case-by-case basis in light of local geological context. However, the data do show clear, persistent and systematic composition variations that should be highly informative on a number of levels.

This new dataset of ~2800 high-quality whole-rock geochemical analyses of subvolcanic, volcanic, and volcaniclastic rocks from the greenstone sections of the Kalgoorlie Group are used to construct chemostratigraphic barcodes within the broader Ora Banda – Kambalda region, encompassing most of the previously defined domains of the Kalgoorlie Terrane as well as the Bulong complex area of the Kurnalpi Terrane. Probably the most functional aspect of this geochemical dataset is in the augmentation of current lithostratigraphy with clear geochemical constraints that uniquely identify a large number (26) of distinct geochemical groupings in commonly generic basalt. These distinctions highlight the controls that various fundamental igneous processes and source compositional variations exerted on evolving magma compositions.

These geochemical barcodes are based on the currently better-known and sampled stratigraphic sections and provide templates for lithogeochemical data sourced from other parts of a specific greenstone belt to be evaluated for possible lithostratigraphic or chemostratigraphic equivalence. We emphasize that in many cases, a single or low number of analyses will not be sufficient to uniquely identify a lithostratigraphic position, but more comprehensive sampling, particularly if it incorporates the underlying and overlying stratigraphic units, almost certainly will. It is hoped that the geochemical barcodes provide an accurate and useful stratigraphic tool from the mine site to the regional scale.

The applicability of the barcodes to greenstone stratigraphy outside of the study area has not been tested yet, although it is clearly expected to decrease. This is because legitimate geochemical units not considered within this study will be expected and the lithospheric sources causing contamination in various common mantle-derived parental magmas (e.g. primitive L-U units) will vary regionally.

According to the new barcode data, individual established lithostratigraphic units (typically at formation level) seldom comprise a single geochemical unit. Most formations comprise at least two distinct geochemical units, most of these distinct units occur at multiple stratigraphic levels and in most other regional stratigraphic columns. Thus, individual greenstone formations combine overlapping flow fields, each representing the products of a discrete eruptive event and/or eruptive centres supplied by genetically unrelated magma sources. This potentially makes some formation boundaries arbitrary reflections of changing source eruptions. Although complex, this diversity ensures that, in most cases, the position of stratigraphically unknown geochemical samples can be uniquely established as long as samples are taken over a continuous stratigraphic interval that incorporates the overlying and underlying units.

The application of chemostratigraphic barcoding is more reliable or appropriate for the Kalgoorlie Group than for the overlying Black Flag Group because of the greater continuity of the low-viscosity flows that characterize its basalt-dominated stratigraphy. Geochemical characteristics and relationships within the felsic-dominated Black Flag Group, nevertheless provide broad stratigraphic information and significant insight into crustal architecture and regional geological evolution. The subvolcanic, volcanic, and volcaniclastic rocks of the Black Flag Group have the composition of sanukitoids, likely derived from a metasomatically enriched lithospheric mantle source. Within the magmatic history of the Black Flag Group, a change from relatively low to high La/Th ratio sanukitoids correlates quite strongly with the spatial and temporal distribution of sanukitoid magmas. The low La/Th sanukitoid is typically older (>c. 2678 Ma) and more common in the region's west, while the high La/Th sanukitoid is typically younger (<c. 2675 Ma) and virtually restricted to the east. We also identify a compositionally specific, sanukitoid-like, low-La dacite group possibly forming a regionally relevant time marker at c. 2706 Ma.

The subdivision of the regional mafic stratigraphy into three levels (lower, middle and upper basalts; e.g. Swager et al., 1990) broadly reflects changes in magma compositions and provides useful markers of relative level within a particular stratigraphy. These changes reflect a range of conditions and processes operating at various crustal levels and at various stages and continue to influence a systematic compositional change in the felsic rocks of the Black Flag Group, overlying the basalt-dominated stratigraphy. They include a change from a crustal deformation regime that allowed rapid transfer from source to deposition site, through dykes, of weakly fractionated and uncontaminated magmas (L-U1 and L-U2) – to an environment favouring ponding in mid- to upper crustal chambers prior to eruption of variable fractionated L-U3–7 magmas. We can also see a broad, and non-systematic increase in the extent that mafic magmas became contaminated by crust, despite the occurrence in the upper stratigraphy of L-U3–7 magmas. An additional consequence of these changes is an increase in the compositional diversity of magmatism with decreasing age, reaching a climax broadly at the transition between the Kalgoorlie and Black Flag Groups, where evidence suggests comagmatic intrusion and/or extrusion of the L-U magmas, L-S magmas, HTSB magmas, low- and medium-La dacite group magmas, and sanukitoid magmas. At this time, the resurgence of locally voluminous ultramafic magmas parental to the HTSB might reflect a catastrophic event at the crust–mantle boundary, at a stage when the hydrated mantle source of sanukitoids was either created, tapped or both.

The regional distribution of the mafic chemostratigraphic associations is distinctly asymmetric, in many cases, ignoring previously established domain boundaries. However, a broadly north-trending structure incorporating segments of (from south to north) the ZAB does appear to separate the western domains (Coolgardie and Ora Banda) from the eastern domains (Kambalda, Boorara, Parker, and the Bulong complex). These domains have distinctive chemostratigraphic variations that probably reflect a fundamental crustal architectural control and potentially provide some basis for reassessing the present domain boundaries. In particular, the mafic units in the western domains probably reflect melting of a more depleted source beneath thicker lithosphere. The Ora Banda Domain is clearly part of the western domains throughout evolution of the lower and middle greenstone stratigraphic evolution, but also appears transitional to the eastern domains, in terms of its diversity of fractionated, layered sill-derived magmatism in its upper stratigraphy. The chemostratigraphic differences between the western and eastern domains extend to the felsic units of the Black Flag Group. The subvolcanic, volcanic, and volcaniclastic rocks of the Black Flag Group show a wide range in La/Th ratios, which broadly correlates with the crystallization age and the geochemical and isotopic compositions of their source. In particular, the low La/Th sanukitoids, which are more common west of the ZAB have less radiogenic Nd (i.e. is more 'crustal'), whereas the high La/Th sanukitoids, virtually restricted to east of the ZAB, have

more mantle-like isotopic compositions. The simplest explanation for the regional compositional variations in the mafic and felsic rocks is that the ZAB is the exposed trace of a structure separating compositionally contrasting lithospheric domains. Based on the geochemical and isotopic data, the northern and southern extensions of that trace should be entirely traceable.

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Appendices

Appendix 1

Whole-rock major and trace element data for samples used in the Report

Available with the PDF online as an accompanying digital resource

Appendix 2

Diamond drillcores used for geochemical sampling

Available with the PDF online as an accompanying digital resource

Appendix 3

Layered PDF versions of all geochemical variation diagrams

Available with the PDF online as an accompanying digital resource

Appendix 4

Summary statistics relating to the geochemistry of mafic units

Available with the PDF online as an accompanying digital resource

Appendix 5

Workflow for classifying geochemical samples

The following workflow provides a method of visually or graphically reproducing the specific compositional features that uniquely identify the individual units or groups described here. It does not reproduce the process originally used to identify the units or groups. Users should be aware that this workflow will not identify, or will erroneously assign, data from any legitimate new geochemical groups not encountered within our dataset and it will not allow for significant legitimate expansion of specific compositional fields. For these reasons, it may not be appropriate to apply this workflow to areas outside the study region. Preliminary applications to data from the Agnew region, for example, clearly identify significant migration in specific fields, likely relating to slight changes in magma bulk compositions (i.e. including melt source and any subsequent contaminant) as expected. Nevertheless, the procedure employs sufficient critical compositional variation diagrams that any completely new units or significant expansions to the fields of old units, should be apparent to the user.

The importance of spatial, stratigraphic, and more broadly geological context cannot be over emphasized. In a core intersection or in a specific geographical region, if all samples are L-U2, except for a single (or a few) sample that is L-U1 on some but not most plots, common sense would suggest that it too is probably L-U2 and that either analytical errors or weak alteration (for example) might account for any discrepancies.

It is also important to note that fields isolated using the following workflow, typically encompass 90–95% of the data (i.e. up to 10% outliers excluded). Using the following method, we have reclassified our data for mafic rocks, with a 95% success rate. Most of the 5% of samples given alternative classifications relate to examples where we originally (and arguably correctly) exercised geological discretion in classification.

Workflow

If major element oxide data does not exist, recalculate:

Al (ppm) to Al_2O_3 wt% (Al/10000*1.8895)

Ti (ppm) to TiO_2 wt% (Ti/10000*1.6681)

Mg (ppm) to MgO wt% (Mg /10000*1.6579)

Fe (ppm) to FeO wt% (Fe/10000*1.2865)

Fe (ppm) to Fe_2O_3 wt% (Fe/10000*1.4297)

Ca (ppm) to CaO wt% (Ca/10000*1.3992)

Na (ppm) to Na_2O wt% (Na/10000*1.3480)

K (ppm) to K_2O wt% (K/10000*1.2046)

P (ppm) to P_2O_5 wt% (P/10000*2.2914)

Calculate:

Total Fe as Fe_2O_3 (Fe_2O_3T) [i.e. Fe2O3 + (FeO/1.1113)]

Total Fe as FeO (FeOT) [i.e. FeO + (Fe_2O_3*1.1113)]

$Mg^{\#}$ [=100*(MgO/40.3)/((MgO/40.3)+(FeOT/71.85))]

ASI (if Na data present) =[(Al_2O_3/101.96)/((CaO/56.1)+(Na_2O/61.98)+(K_2O/94.2))]

Quarantine ultramafic data (not used further in barcoding):

Set aside all data with MgO >21 wt%

Set aside all data with Ti/Zr >70 AND MgO >18 wt%

Group the remaining data as:

Mafic (basic), if Ti/Zr >48

Felsic (intermediate and acid), if Ti/Zr <48

For mafic samples

Unless clear evidence for anomalism in the major or trace elements was detected, mafic samples have not been filtered for alteration. Most of the plots we used for discrimination and interpretation are based on elements typically regarded as moderately or strongly fluid immobile. No measures of compositional evolution or crustal contamination (e.g. La/Sm, La/Yb, Th/Nb etc., or even U/Th) show a clear correlation with concentrations of CaO, Na_2O, or K_2O and these element oxides did not vary systematically with loss on ignition (LOI). Throughout the entire basalt dataset, we could identify 81 analyses that show some form of anomalism on a mantle-normalized trace element diagram, but removal of these analyses had no perceptible effect on compositional trends or fields; all analyses were retained.

Isolate all samples with Th/TiO_2 >2 (plot of Th vs TiO_2) as HTSB and KB3:

- Classify all samples with Th/TiO_2 >3.33 (plot of Th vs TiO_2) as either H-$HMg^{\#}$ ($Mg^{\#}$ >62) or H-$LMg^{\#}$ ($Mg^{\#}$ <62) (set aside)
- Classify all remaining HTSB+KB3 samples with Ni <150 ppm AND Th/Nb >0.5 as H-LTh (set aside)
- Classify remaining samples as KB3, if a relationship to komatiite can be reasonably inferred (set aside).

Classify samples with Al_2O_3/TiO_2 between 25 and 28.5 AND with TiO_2 >0.54 wt% as L-S2 (set aside).

Classify samples with Al_2O_3/TiO_2 between 24.5 and 32.5 AND with Al_2O_3 <15.5 wt% as L-S1.

Also classify remaining samples with Zr <30 ppm AND TiO_2 <0.54 wt% as L-S1 (set aside).

Classify samples with Th >0.2 ppm AND TiO_2 <0.6 wt% as KB1 (set aside).

Classify samples with TiO_2 >2.0 wt% as L-U7 (set aside).

Isolate all samples with Th/Zr ratio >0.0143 (this group includes I-S4, I-VS, L-S3a, and L-S3b):

- Classify samples with Al_2O_3/TiO_2 >19 as I-VS (set aside)
- Classify samples with Zr >70 ppm as I-S4 (set aside)
- Group all remaining samples as L-S3
- Classify all L-S3 samples within the western domains as L-S3a (set aside)
- Classify all remaining samples as L-S3b (set aside).

Isolate, using a plot of Th (ppm) (y-axis) against TiO_2 wt% (x-axis), all samples above a line, y = 1.366x−0.55 (this groups the I-S trend):

- Classify samples with TiO_2 <0.6 wt% as I-S1 (set aside)
- Of the remainder, classify samples with Al_2O_3/TiO_2 <15 as I-S3 (set aside)
- Classify all remaining samples as I-S2 (set aside).

Return to a plot of Th vs Zr and isolate samples with a Th/Zr ratio >0.01 AND Th <0.55 ppm and add to L-S3 (L-S3b) (set aside).

Classify, using a plot of Th (ppm) against TiO_2 wt%, samples with TiO_2 >1.75 wt% AND Th <0.9 ppm as L-U6 (set aside).

Classify, using a plot of Zr (ppm) against TiO_2 wt%, samples with Zr <48 ppm AND TiO_2 <0.8 wt% as L-U1 (set aside).

Using a plot of Th (ppm) against TiO_2 wt%:

- Isolate all samples with a Th/TiO_2 ratio <0.41 [this includes L-U2 and most of the L-U3–7 trend (L-U6 and L-U7 already set aside)]. Of these samples (Th/TiO_2 ratio <0.41):
  - Classify all samples with Al_2O_3 >17 AND TiO_2 <1.17 as L-U3 (set aside)
  - Classify all samples with Al_2O_3/TiO_2 >15.3 as L-U2 (set aside)
  - Classify all samples with Nb <2.7 ppm as L-U2* (set aside)
  - Classify all samples with Al_2O_3 >15.6 wt% as L-U5 (set aside)
  - View remaining samples on plots of Th, Nb, Zr vs TiO_2, isolate significant trend outliers and classify remaining samples as L-U4 (set aside)
  - If trend outliers represent a rational geochemical trend, classify them as a new group.

  Samples of L-U2 (L-U2*) and L-U3 that fall in the area where these units almost merge, should be further considered from a geographical perspective, and using any additional geological context, allocated to the most likely group.

  The samples of L-U3, LU-4, and L-U5 that fall in the area where these units almost merge, should be further considered from a geographical perspective, and using any additional geological context, allocated to the most likely group.

- Isolate all samples with a Th/TiO_2 ratio >0.41 (this includes L-Ca and L-Cb as well as L-S4). Of these samples (Th/TiO_2 ratio >0.41):
  - Classify all samples with Th/Nb >0.27 as L-S4 (set aside)
  - Classify remaining samples as L-C. If L-Ca and L-Cb require discriminating, this can be achieved effectively using Na_2O concentration (L-Cb >3.5 wt% > L-Ca) or a plot of Fe_2O_3 (wt%) (y-axis) against Sc (ppm) (x-axis), where most L-Ca and L-Cb lie above and below, a line y = −0.1818x+20, respectively.

  Samples of L-C lying in regions merging with L-S3, L-U3, and L-U4 should be further considered from a geographical perspective, and using any additional geological context, allocated to the most likely group.

For felsic rocks

Unless there is reason to indicate the following conditions are not related to alteration, remove all data with:

Ti/Zr <48 AND CaO >8 wt%

Ti/Zr <48 AND Na_2O <0.5 wt%

Ti/Zr <48 AND K_2O >4.5 wt%

Ti/Zr <48 AND Na_2O+K_2O <3.5 wt%

Ti/Zr <48 AND ASI >1.2

Although it appears that felsic volcanic and subvolcanic magmatism in greenstone belts is seldom compositionally equivalent with the felsic magmas forming the main granitic regions separating greenstones, particularly in the case of the High-Ca and Low-Ca granites, this is not always the case. Therefore, the potential equivalents of High-Ca and Low-Ca granites (for example) need to be identified. This is typically not easy because of the extent to which established geochemical populations of felsic magmas overlap. Consideration of mineralogy and field descriptions is very important. Most syenites (feldspar rich, quartz poor) and sanukitoids (hornblende rich) are mineralogically distinctive and many High-Ca granites can be distinguished from many Low-Ca granites based on feldspar mineralogy and proportions.

Identifying the compositional affiliation of a single sample can be impossible. Identifying trends in geologically plausible suites of samples is more productive. One of the biggest challenges is the identification of Mafic granites or sanukitoids from High-Ca granites. Without the benefit of accompanying genetically related mafic (low silica) samples, isolated samples of sanukitoids with SiO_2 >66 wt% are very difficult to distinguish from High-Ca granites (and almost impossible at SiO_2 >72 wt%), although $Mg^{\#}$ >45 and Cr >35 ppm should distinguish most sanukitoids at these SiO_2 levels.

Based on a craton-wide compilation of granite compositions, Champion and Sheraton (1997) and Champion and Cassidy (2002a) present numerous geochemical variation diagrams that can be used to identify or discriminate analyses of High-Ca, Low-Ca, HFSE-rich, and Mafic granites (mainly sanukitoids), and syenites. No single diagram is without considerable overlap, particularly between the fields for High-Ca and Low-Ca granites, but when most or all diagrams are considered collectively, effective discrimination is possible in most cases. We have found the following plots to be most useful:

- Ce vs K_2O: discriminating High-Ca and Low-Ca granites (Fig. 54)
- K_2O vs Ce/K_2O: discriminating High-Ca and Low-Ca granites (Fig. 55)
- Al_2O_3 vs Ce/K_2O: discriminating High-Ca and Low-Ca granites (Fig. 56).

- SiO_2 vs Ce/K_2O: discriminating Mafic granites and syenites from High-Ca and Low-Ca granites (Fig. 57)
- Fe_2O_3(total) vs SiO_2: discriminating HFSE-rich granites from High-Ca and Low-Ca granites (Fig. 58)
- Yb vs SiO_2: discriminating HFSE-rich granites (Fig. 59)
- Fe_2O_3(total) vs SiO_2: discriminating syenites, Mafic granites, and HFSE-rich granites from each other (Fig. 60)
- K_2O+Na_2O vs SiO_2: discriminating syenites (Fig. 61)
- $Mg^{\#}$ vs SiO_2: discriminating Mafic granites from High-Ca and Low-Ca granites (Fig. 62)
- Ni vs SiO_2: discriminating Mafic granites from High-Ca granites (Fig. 63).

After identifying samples equivalent to High-Ca, Low-Ca, HFSE-rich granites, and syenites, the procedure for subdividing and classifying the remaining Mafic granites (including sanukitoids and sanukitoid-like rocks) is described below (Fig. 36). As discussed above, a degree of overlap in all compositional fields is expected. This means that, for example, a few samples from an otherwise compositionally coherent suite of sanukitoids might fail the first test used here to identify sanukitoids (i.e. Nb/P_2O_5 <25) and be classified as 'others', and then plot apparently anonymously as sanukitoids within all subsequent classification plots. Common sense suggests these are sanukitoids and in an iterative approach to classification, such discrepancies can be rectified.

Classifying Mafic granites:

- After applying the filters (above), view samples with Ti/Zr <48 on a plot of Nb vs P_2O_5.
  - Group all samples with Nb/P_2O_5 <25 as sanukitoids
  - Group all samples with Nb/P_2O_5 >25 as potential sanukitoid-like rocks.
- For sanukitoids: verify that all ***geologically related*** suites (e.g. hornblende-bearing rocks of broadly andesitic to dacitic composition from one specific area or stratigraphic position) form trends or fields where at 60 wt% SiO_2, most (not necessarily all) samples have $Mg^{\#}$ >60, MgO >5 wt%, Ba and Sr each >500 ppm, La >20 ppm, and Ni and Cr each >100 ppm.
  - For samples within the region covered in this Report, using a plot of La vs Th, classify all samples with La/Th ≥4.9 as high La/Th sanukitoid and all samples with La/Th <4.9 as low La/Th sanukitoid.
- At this stage, particularly if filtering samples taken from outside of the area considered in this Report (where Nb/P_2O_5 might not be a suitable discriminator of sanukitoid), it would be advisable to also examine potential sanukitoid-like rocks on plots of $Mg^{\#}$, MgO, Ba, Sr, La, Ni, and Cr. The suites that otherwise meet all requirements of sanukitoids should be classified as such.
- For remaining potential sanukitoid-like rocks sampled within the region covered in this Report:
  - Classify any sample or suite with anomalously high concentrations of La (>60 ppm) AND other LREE, AND Zr (>175 ppm), AND Nb (>9 ppm), AND Sr (>500 ppm), and compared with other potential sanukitoid-like rocks, relatively high Na_2O, P_2O_5, and low total Fe (as either Fe_2O_3T or FeOT), MgO and $Mg^{\#}$, and Sc concentration (extrapolated to 60 wt% SiO_2) <8 ppm as high-La dacite (includes dacites and rhyolites)
  - Classify as low-La dacite (includes andesites and dacites), any sample or suite that (at 60 wt% SiO_2) satisfies most of the following: $Mg^{\#}$ ~50, Fe_2O_3T >8 wt%, MgO >4 wt%, Ba and Sr each <500 ppm, La <20 ppm, and Ni and Cr each >70 ppm, Sc >15 ppm, and Th <5 ppm
  - Most or all remaining samples should be medium-La dacites (includes dacites and rhyolites). When trends are extrapolated to 60 wt% SiO_2, most (not necessarily all) samples should have $Mg^{\#}$ ~45, Fe_2O_3 (total Fe) ~8 wt%, MgO ~3.5 wt%, Ba between 500 ppm and 1000 ppm and Sr ~500 ppm, La between 20 ppm and 40 ppm, and Ni and Cr each <50 ppm, Sc ~15 ppm, and Th ~10 ppm.

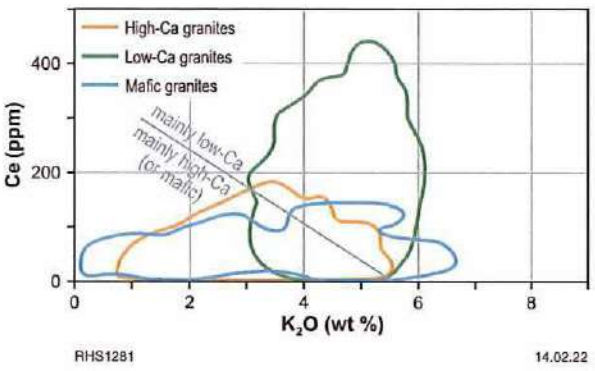


Figure 54. Variation of Ce with K_2O for granites of the Yilgarn Craton, distinguishing High-Ca, Low-Ca, and Mafic granites (data associated with Champion and Sheraton, 1997 and Champion and Cassidy, 2002a)

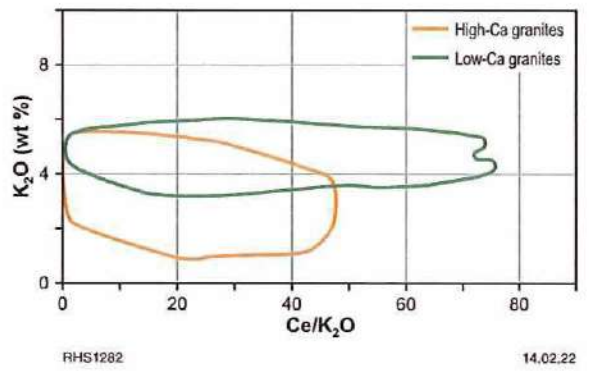


Figure 55. Variation of K_2O with Ce/K_2O for granites of the Yilgarn Craton, distinguishing High-Ca and Low-Ca granites (data associated with Champion and Sheraton, 1997 and Champion and Cassidy, 2002a)

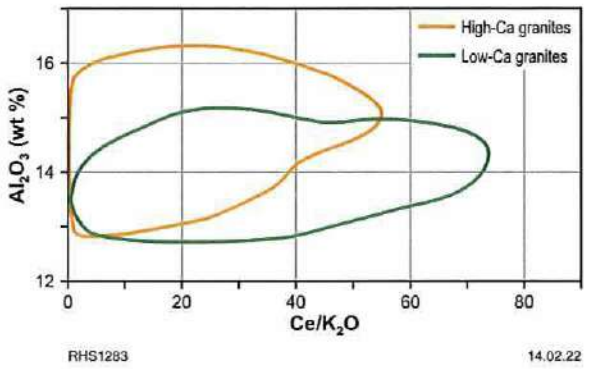


Figure 56. Variation of Al_2O_3 with Ce/K_2O for granites of the Yilgarn Craton, distinguishing High-Ca and Low-Ca granites (data associated with Champion and Sheraton, 1997 and Champion and Cassidy, 2002a)

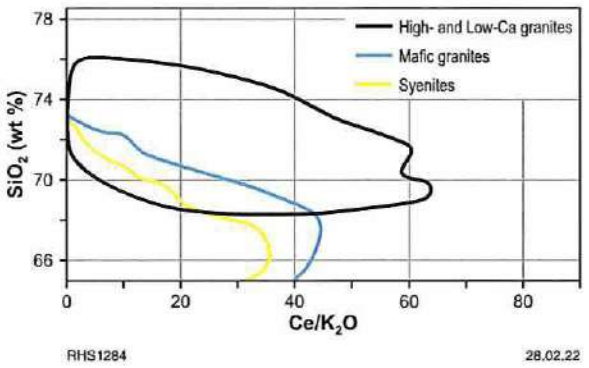


Figure 57. Variation of SiO_2 with Ce/K_2O for granites of the Yilgarn Craton, distinguishing High-Ca and Low-Ca granites, and syenites (data associated with Champion and Sheraton, 1997 and Champion and Cassidy, 2002a)

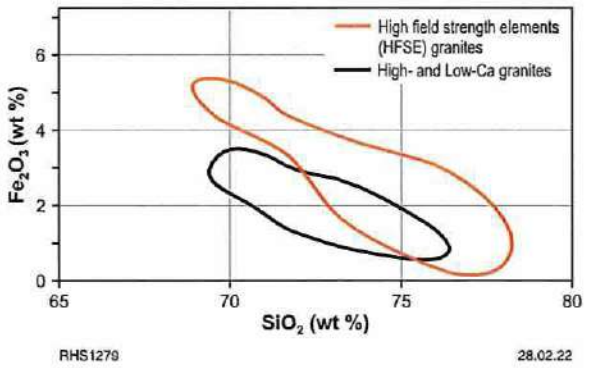


Figure 58. Variation of Fe_2O_3 with SiO_2 for granites of the Yilgarn Craton, distinguishing High-Ca, Low-Ca, and HFSE-rich granites (data associated with Champion and Sheraton, 1997 and Champion and Cassidy, 2002a)

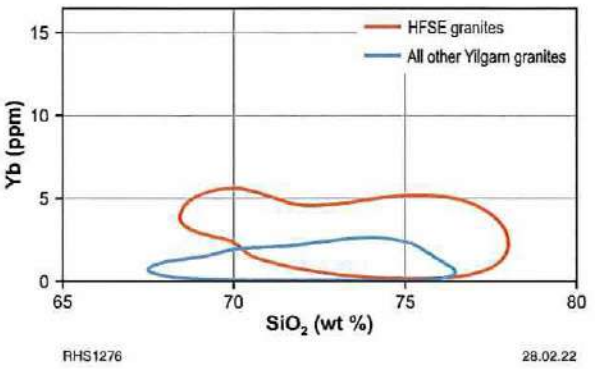


Figure 59. Variation of Yb with SiO_2 for granites of the Yilgarn Craton, distinguishing HFSE-rich granites from other granites (data associated with Champion and Sheraton, 1997 and Champion and Cassidy, 2002a)

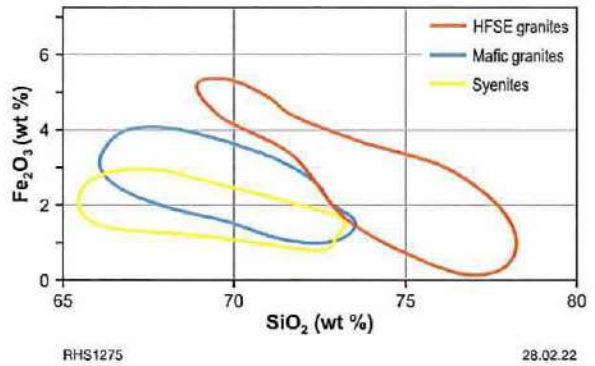


Figure 60. Variation of Fe_2O_3 with SiO_2 for granites of the Yilgarn Craton, distinguishing HFSE-rich granites, syenites, and Mafic granites (data associated with Champion and Sheraton, 1997 and Champion and Cassidy, 2002a)

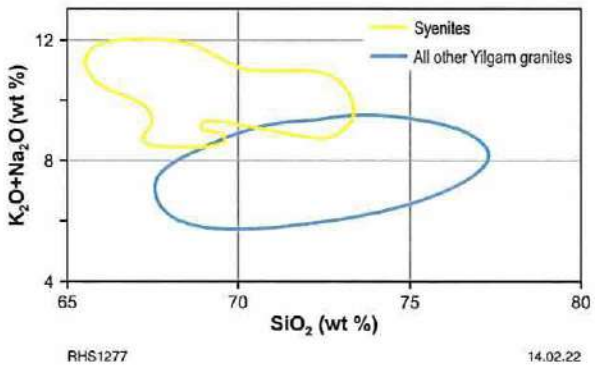


Figure 61. Variation of K_2O+Na_2O with SiO_2 for granites of the Yilgarn Craton, distinguishing syenites from other granites (data associated with Champion and Sheraton, 1997 and Champion and Cassidy, 2002a)

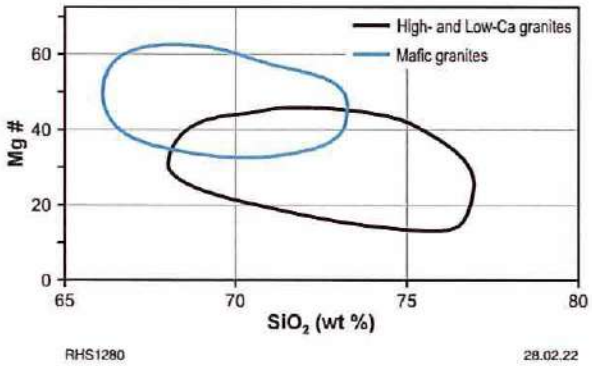


Figure 62. Variation of $Mg^{\#}$ with SiO_2 for granites of the Yilgarn Craton, distinguishing High-Ca, Low-Ca, and Mafic granites (data associated with Champion and Sheraton, 1997 and Champion and Cassidy, 2002a)

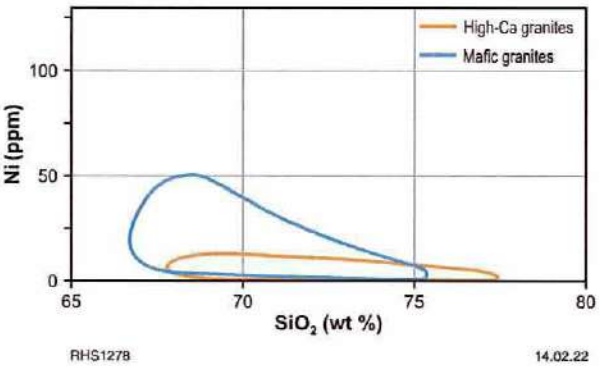


Figure 63. Variation of Ni with SiO_2 for granites of the Yilgarn Craton, distinguishing High-Ca and Mafic granites (data associated with Champion and Sheraton, 1997 and Champion and Cassidy, 2002a)

Appendix 6

Summary of the chemostratigraphy of the Ora Banda – Kambalda region and its practical field applications

Also available with the PDF online as an accompanying digital resource

This Appendix is designed as an independent practical guide to the potential field applications of the barcoding chemostratigraphy for mafic volcanic rocks of the Kalgoorlie Group and the felsic volcanic, volcaniclastic, and subvolcanic intrusive rocks of the Black Flag Group. The data are directly relevant to the region broadly between Ora Banda and Kambalda (Fig. 64). For more detailed geochemical, petrological, and geodynamic descriptions, interpretations and implications see the main text of Report 226.

Using a database of ~2800 high-quality whole-rock geochemical data and available stratigraphic information, we have attempted to identify the various geochemical components comprising the basaltic and felsic stratigraphy for parts of the domains lying within the region, broadly between Ora Banda and Kambalda. These domains include the Coolgardie, Kambalda, Ora Banda, Boorara, and Parker Domains of the Kalgoorlie Terrane and the western part of the Bulong Domain (Bulong complex) in the Kurnalpi Terrane. The mafic stratigraphy is shown in Figure 65. The mafic stratigraphy in each area is divided into three levels (lower, middle, and upper) according to previous lithostratigraphic interpretations and is based on the classic Kalgoorlie–Kambalda stratigraphy (e.g. Woodall, 1965; Gresham and Loftus-Hills, 1981; Roberts, 1988, Swager et al., 1990; Morris, 1993). Komatiite provides a convenient marker at the base of the middle level. We provide no information on the relative stratigraphic thicknesses of the levels within each column. The dashed line shown in the upper part of the Coolgardie and Ora Banda columns (Fig. 65) indicates that either an upper stratigraphic level does not appear to have been developed or preserved (Coolgardie, e.g. Swager et al., 1990) or that the boundary between the middle and upper levels is unclear (Ora Banda). Relatively low sample numbers mean that the chemostratigraphy for some parts of the Boorara and Parker Domains and for the Bulong complex is less robust than for the regions to the west. For units where this might be of concern, the number of samples collected is indicated in Figure 65. Otherwise, for each stratigraphic formation, the volume of each specific geochemical unit is proportional to the number of samples analysed. Whilst the relative number of samples of each specific geochemical unit might be broadly representative of true relative abundances in most cases, it is clearly open to sample bias. The geochemical units are organized vertically where the relative stratigraphic position of geochemical units within a formation is known or reasonably inferred. Otherwise, the geochemical units within formations are organized horizontally.

All samples collected as part of the greenstone geochemical barcoding project represent the visually least-altered examples of lithologically, texturally, and mineralogically homogeneous core intervals. We largely restricted our analysis to data from volcanic rocks covering the mafic and felsic (intermediate and acid) compositional range, i.e. SiO_2 from ~45 to 77 wt%, and MgO up to 18 wt% (except in the case of some partial cumulate layers). Komatiites and samples from mafic dykes and sills are not discussed in detail. All major element concentrations are quoted on a volatile-free basis (i.e. anhydrous).

For the classification and screening of rocks with broadly mafic compositions, we slightly modified the approach used by Barnes et al., (2012). In particular, the maximum SiO_2 content used to identify rocks of broadly basaltic composition was raised from 56 wt% to 57 wt%. The approach used by Barnes et al. (2012) uses variations in ratios of incompatible trace elements (e.g. Ti, Th, La, Nb, Zr) that are relatively immobile during hydrothermal alteration and low- to medium-grade metamorphism and insensitive to small variations in major elements related to common igneous processes.

Unless clear evidence for anomalism in the major or trace elements was detected, mafic samples have not been filtered for alteration. No measures of compositional evolution or crustal contamination (e.g. La/Sm, La/Yb, Th/Nb etc. or even U/Th) show a clear correlation with concentrations of CaO, Na_2O, or K_2O. Nor do these element oxides vary systematically with measures of alteration such as LOI, which in our dataset has values below 5 wt% for >80% of samples. Throughout the entire basalt dataset, we could identify 81 analyses that show some form of anomalism on a mantle-normalized trace element diagram but removal of these analyses had no perceptible effect on compositional trends or fields; most of these mafic analyses were retained.

The following screens and caveats were applied to the data:

- Intrusive rocks with SiO_2 <57 wt% and MgO <18 wt% were classified as dolerite or gabbro, depending upon grain size. Those samples with MgO >18 wt% were classified as peridotite
- Non-intrusive rocks with SiO_2 <57 wt% were classified as basaltic, if MgO <18 wt% and >3 wt% or komatiite, where MgO >18 wt%
- Many samples with MgO between 16 wt% and 18 wt% (and a few with MgO <16 wt%) have Al_2O_3/TiO_2 ratios ~20 (Al/Ti of ~1.0), a criteria Barnes et al. (2012) used to identify komatiitic basalts (KB). In nearly every case, these were directly associated with true komatiites (olivine spinifex-textured rocks or rocks intimately associated with olivine spinifex-textured rocks) and have been classified as KB. In some cases, these formed a compositionally distinct series that extended to much lower MgO contents
- Intrusive rocks with SiO_2 >57 wt% were almost invariably plagioclase-porphyritic and were classified as felsic dykes [andesitic (SiO_2 up to 63 wt%), dacitic (SiO_2 63–68 wt%) or rhyolitic (SiO_2 68–77 wt%)]
- Rocks with SiO_2 >57 wt% and evidence for a volcanic and volcaniclastic origin were classified as felsic volcanic or volcaniclastic (andesitic, dacitic or rhyolitic)
- Rocks with SiO_2 >57 wt% but with no clear evidence for a volcanic and volcaniclastic origin were classified as felsic (andesitic, dacitic or rhyolitic).

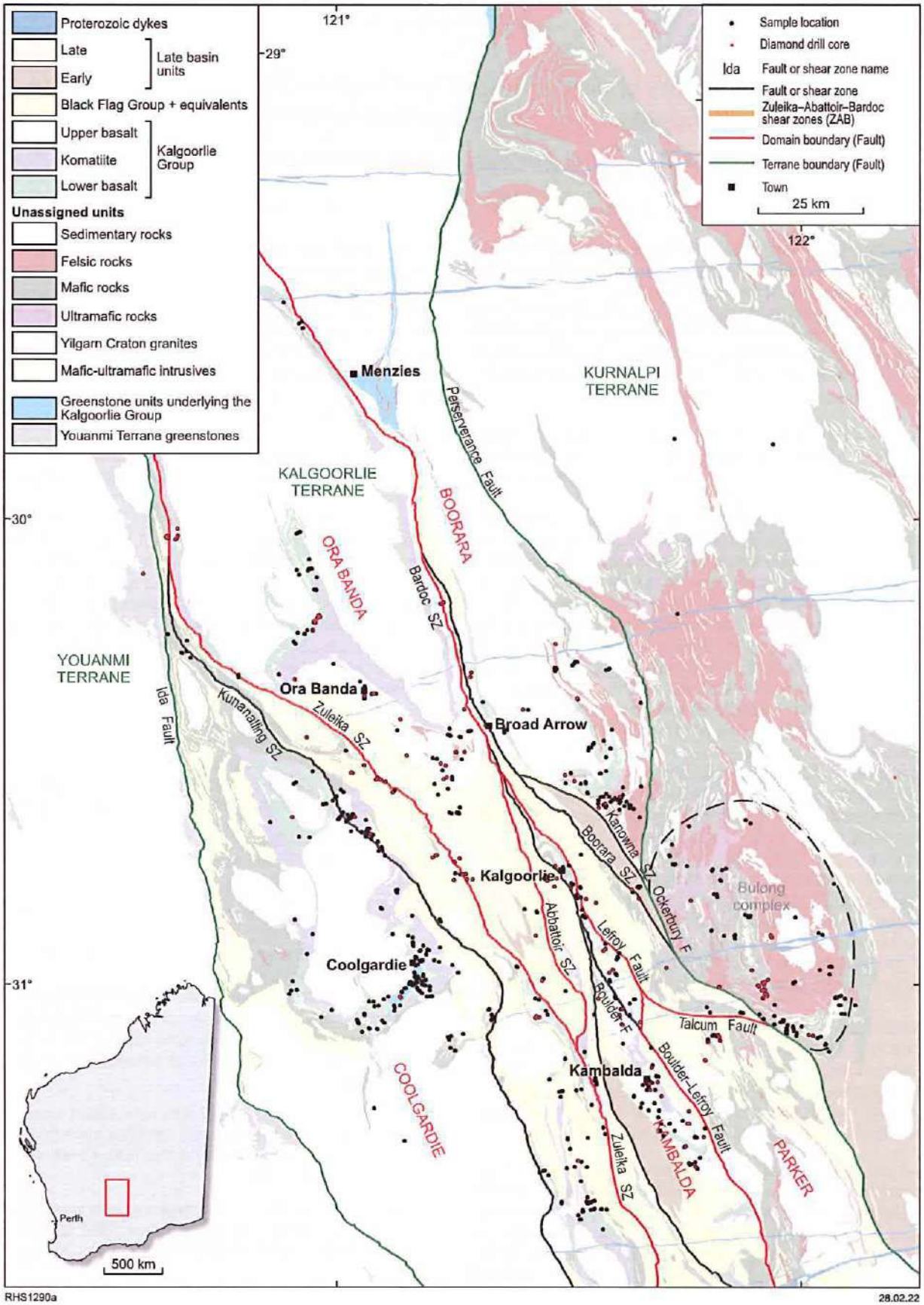


Figure 64. Geological map of the study area within the Kalgoorlie Terrane, showing a simplified version of the current interpreted bedrock geology, including the major faults and shear zones. Also shown is the distribution of sites of greenstone lithologies sampled for whole-rock geochemical data, including the locations where diamond drillcore has been sampled. Diamond drillcore locations are typically the sites of multiple (up to 55) individual samples

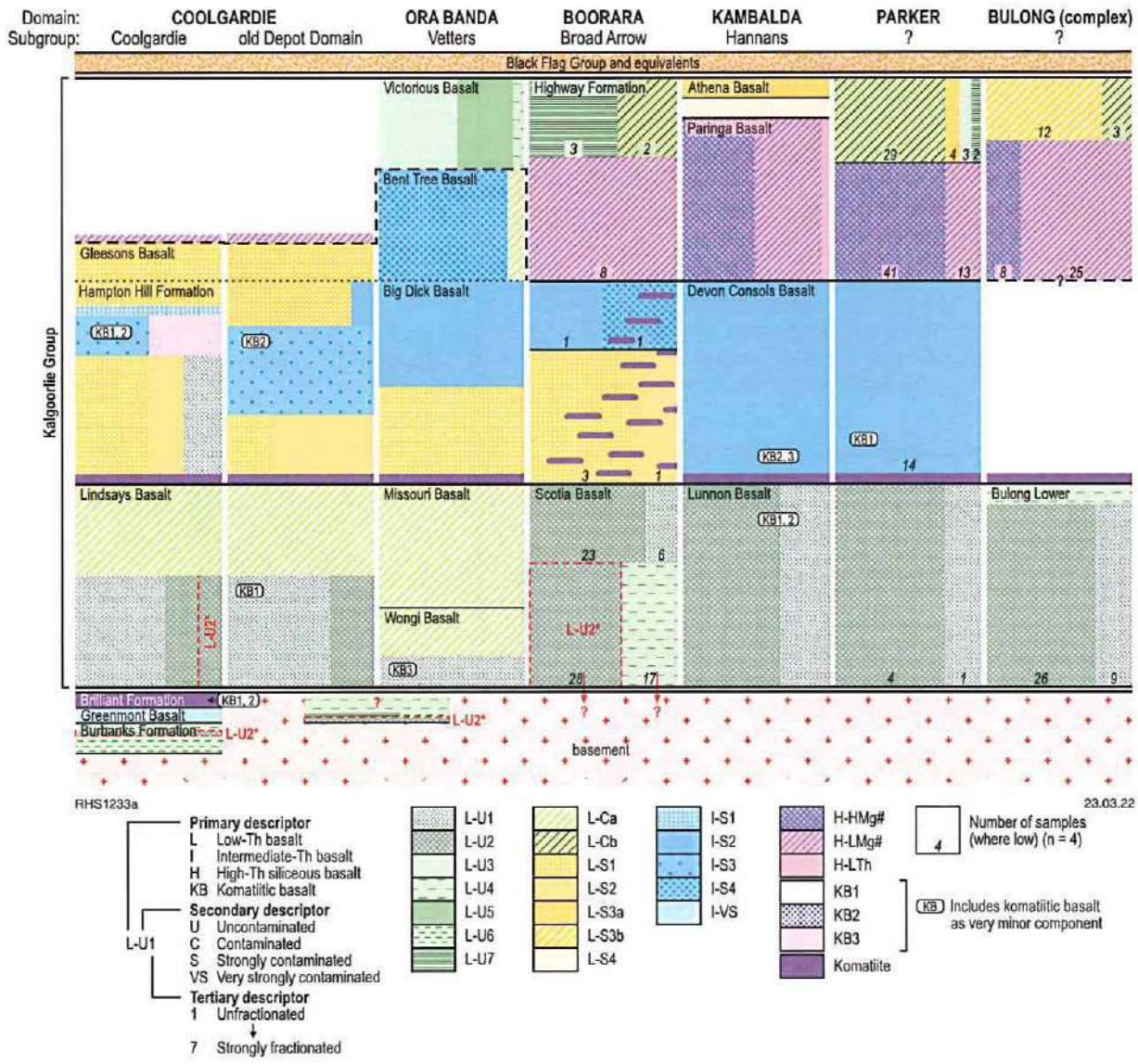


Figure 65. Chemostratigraphic columns representing each of the domain regions within the study area. Also included is a brief graphical explanation of the coding system used here (see text for further details and explanation)

- Rocks with SiO_2 >57 wt% were initially excluded from the dataset if:
  - The aluminium saturation index [ASI = molar Al_2O_3/($CaO+Na_2O+K_2O$)] exceeded 1.1 (i.e. the sample was peraluminous, in many cases indicative of feldspar-destructive alteration)
  - SiO_2 exceeded 77.5 wt% (i.e. silica concentrations higher than values achievable through normal igneous processes, i.e. silicified)
  - Na_2O was <1 wt% or Na_2O+K_2O <3.5 wt% (values suggestive of alkali loss).

Subdividing mafic rocks

Having applied the screens described above, mafic samples were further divided into primary lithochemical groups, including low-Th basalt (LTB, a proxy for any LREE-HFSE-Th-poor basalts), intermediate-Th basalt (ITB), and high-Th siliceous basalt (HTSB) groups as defined by Barnes et al. (2012) (Fig. 66). The geochemical range, in terms of Th (Nb, Zr, La) vs TiO_2 for each of these geochemical groups is tightly constrained and within each group, several discrete subgroups can be identified. These typically have a range of unique compositional features.

LTB, ITB, and HTSB groups, by definition, are commonly but not invariably distinguished on a plot of Th vs TiO_2 alone (see note below on precision limitations on low concentration Th data). Although Th concentrations are generally more sensitive to contamination and fractionation, the fluid immobile and only slightly less incompatible elements, Zr, Nb, and La should ideally also be used to discriminate broadly similar low, intermediate, and high groups. Instances where variation in one or more of these elements is slightly decoupled from the others, provide potential means of further subdivision. Hence, some LTB samples identified because of their low Th, Nb, and La concentrations might have slightly high Zr concentrations that result in Zr/TiO_2 ratios within the range typical of ITB, rather than LTB. In this case, a Zr-enriched LTB sample might be identified.

Similarly, if anomalism is identified with Th, the result might be a Th-enriched LTB. If this is a persistent characteristic within a specific geographical region and/or at a specific stratigraphic level, then we define an LTB group with high-Th (or Zr). Similar arguments apply for the identification of ITB. HTSB should be identified based only on Th and La vs TiO_2. We have used all four trace elements (each plotted against TiO_2) to identify our primary mafic groups. In cases where a sample plots in one field based on two elements (e.g. Th and Zr vs TiO_2) and in another based on the other two elements (La, Nb vs TiO_2), geological judgement is required to classify the sample. For example, you may need to consider whether this is an unusual or a persistent feature within a spatially or stratigraphically related series of samples. Although it is clearly feasible that a group of related samples might truly have perfectly transitional characteristics between LTB and ITB or ITB and HTSB, we have not yet identified any such cases.

Some basalts with MgO generally between 10 wt% and 18 wt% and $Mg^{\#}$ [molar ratio of Mg/(Mg+Fe), where Fe is total iron calculated as Fe^{2+}] from 60–78 (higher than typical LTB), have Al_2O_3/TiO_2 ratios ~20 but concentrations of Th, Zr, Nb, and La considerably lower than that of HTSB (and mostly lower than LTB). These are typically directly associated with komatiites and are classified as KB.

Primary mafic geochemical groups were further manually subdivided into smaller geologically plausible geochemical groups. This was an iterative process undertaken by simultaneously viewing the position of data points with respect to natural data clusters on plots of:

- TiO_2 vs Th, Nb, Zr, and La
- Th vs Nb and Zr
- Nb/Yb vs Th/Yb
- Al_2O_3/TiO_2 vs Al_2O_3 or Th/Nb
- Mantle-normalized incompatible trace element diagrams
- $Mg^{\#}$ vs all major elements, Cr, Ni, and Al_2O_3/TiO_2
- Longitude vs latitude vs position in drillcore (i.e. a map).

All these elements are strongly incompatible in mafic systems (until a TiFe oxide is stabilized in the case of Ti) and should increase in concentration with increased fractionation or decreased partial melting. Ratios between these elements will mainly reflect variations in bulk source compositions or mineralogy and provide an opportunity to subdivide magmas.

The Al_2O_3/TiO_2 ratio of a mantle-derived magma should not change significantly (from ~20) until plagioclase crystallizes and fractionates at low-pressure.

Compositional variations in the plots listed above, highlight the controls that various fundamental igneous processes and source compositional variations might have exerted on an evolving magma, and at least in the case of TiO_2, P_2O_5 and most of the selected trace elements are not strongly influenced by postmagmatic alteration.

A total of 26 minor units were identified and most of these were either LTB or ITB. These groups or units commonly show a moderate degree of compositional overlap with other groups or units in one or many of the compositional variation diagrams. However, each has either a compositional feature or a geographical distribution (or both) that allows it to be uniquely distinguished. Plots of TiO_2 vs Th and Zr and Al_2O_3/TiO_2 vs Al_2O_3 appear to be particularly useful in distinguishing these geochemical populations. These geochemical units cannot be given stratigraphic names because in most cases they occur at several stratigraphic positions, even within a single subgroup stratigraphy, and commonly form one of several distinct compositional units that together form an established stratigraphic unit. For example, the Big Dick Basalt in the Vetters Subgroup comprises at least two distinct geochemical units and the Lindsays Basalt in the upper Coolgardie succession comprises at least three units.

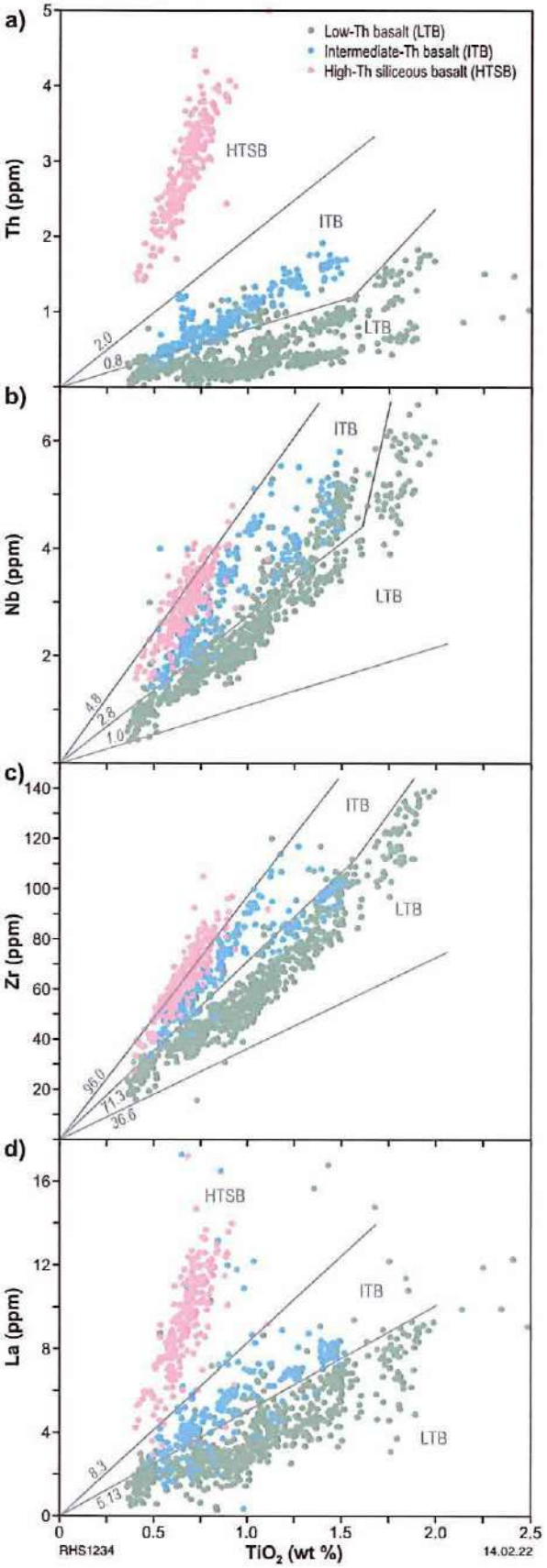


Figure 66. Variation in Th, Nb, Zr, and La with TiO_2 used for primary classification of basalts into LTB, ITB, and HTSB (after Barnes et al., 2012). Numbers indicate the x/TiO_2 ratio of the lines chosen to separate the various groups

Note on precision limitations on low concentration Th data

A degree of caution needs to be exercised regarding discrimination procedures such as the one outlined above, which utilizes trace elements typically present at concentrations approaching analytical detection limits. This is particularly the case for Th. Under such circumstances, it can be difficult to be certain that some of the typically more incompatible trace element-depleted groups are not simply an artefact of variation within the limits of analytical precision. In Figure 67, we show the estimated precision of analyses for Th, Nb, Zr, and La at very low concentrations, based on historical analytical results from BV Minerals (Champion et al., 2021). These uncertainties are generally small, but clearly, at least in the case of Th, potentially provide a plausible explanation for some of the compositional groups. However, we are confident that these groups are geologically meaningful for several reasons. Firstly, the precision estimates are indicative values based on analysis of a wide range of rock types and standards, most with significantly higher concentrations of Th and Nb (for example), and almost certainly would be improved, if calculated based only on samples and standards with very low concentrations. Secondly, mantle-normalized incompatible trace element patterns are typically smooth between Th, with a relatively high analytical uncertainty, and La and other LREE with a significantly lower analytical uncertainty, strongly suggesting the analytical precision for Th is much better than the indicated values. Thirdly, the same groups are often well-defined and are consistently identified on a wide range of compositional variation diagrams, including those using major element data at concentrations not subject to the same limitation in precision. Fourthly, most of the groups make geological sense in terms of their geographic and/or stratigraphic distribution. In addition, these compositional groups typically comprise samples analysed within more than one analytical batch.

Most of the basalt units were grouped and named primarily using plots of Th against TiO_2 and Th against Zr. The naming convention was constructed to emphasize the magmatic or petrogenetic history of the unit without any necessary implications in terms of direct genetic relationships to other units. It is clear from the systematic data spread in bivariate plots utilizing incompatible trace elements that varying degrees of crustal contamination reflects the greatest control over compositional differences between the units identified here. The primary classification itself is based on increasing amounts of crustal contamination from LTB to ITB to HTSB. However, we have added a secondary code that reflects the relative degrees of contamination of a specific unit within a given primary classification (Fig. 68). An additional code is added to indicate the relative extent of fractionation of a specific unit. Thus:

- For the primary classification descriptor,
  - LTB is denoted as 'L'
  - ITB is denoted as 'I'
  - HTSB is denoted as 'H'.
- The secondary measure of the amount of crustal material incorporated by a mafic parental magma was based broadly on the position of a unit in plots of Th against TiO_2 and Th against Zr (with Th/TiO_2 and Th/Zr ratios increasing with increasing contamination), relative to the other units of the same primary classification, such that:
  - U = uncontaminated or nearly so
  - C = contaminated
  - S = strongly contaminated
  - VS = very strongly contaminated.
- The degree of fractionation was a qualitative measure within each primary classification group for each specific secondary measure of contamination (e.g. L-S for strongly contaminated LTB). It was based on the concentration range of Th and Nb in each unit such that (for example) L-S1 had lower Th concentrations (was less fractionated) than L-S2, which in turn was less fractionated than L-S3, and so on.

Hence, L-U5 would be an uncontaminated but strongly fractionated LTB, whereas I-S2 would be a strongly contaminated but only weakly fractionated ITB.

Mafic units identified so far (GSWA Report 226) include:

- LTB
  - L-U1 and L-U2: two units of uncontaminated (or nearly so) and relatively unfractionated primitive LTB, including one local variant (L-U2*)
  - L-U3–7: five units of uncontaminated (or nearly so) but moderately to strongly fractionated LTB
  - L-Ca: a coherent unit of contaminated LTB showing a continuous fractionation trend, that [except for a possibly Zr-enriched variant (L-Cb)] cannot plausibly be divided into discrete units
  - L-S1 and L-S2: two units of strongly contaminated but relatively unfractionated LTB
  - L-S3a, L-S3b, and L-S4: three groups of strongly contaminated and strongly fractionated LTB.
- ITB
  - I-S1–3: three units of strongly contaminated and weakly to moderately fractionated ITB
  - I-S4: a strongly contaminated and strongly fractionated ITB
  - I-VS: a very strongly contaminated and relatively weakly fractionated ITB.
- HTSB: all HTSB by definition, are strongly contaminated with crustal material. The three groups recognized are:
  - H-HMg#: HTSB with high Mg#
  - H-LMg#: HTSB with lower Mg#
  - H-LTh: HTSB with lower Th concentration compared with other HTSB.
- KB: Three units (KB1 to 3).

Appendix 5 **(Workflow for classifying geochemical samples)** provides a method for visually or graphically reproducing specific compositional features that uniquely identify the individual units or groups described here. It does not reproduce the process used to identify the units or groups in the first place. Users need to be aware that it will either not identify, or erroneously assign, data from any legitimate new geochemical groups not encountered within our dataset and it will not allow for significant legitimate expansion of specific compositional fields. For these reasons, it may not be appropriate to apply this workflow to areas outside the study region. Nevertheless, the procedure employs sufficient critical compositional variation diagrams that any completely new units should become apparent.

It should be recognized that industry exploration assay datasets, typically collected over much more specific regions and in many cases, containing an order of magnitude more data than the dataset we use here, inevitably show a considerably greater scatter of data that hides much of the detail shown in our data (Fig. 67). Although these larger datasets might include discrete basalt geochemical populations not sampled in this study, it is highly likely that most of the scatter can be attributed to factors including:

- analytical techniques less accurate and precise than those used here
- less homogeneous sample material (i.e. mixed lithologies particularly in the case of rock chip samples or samples taken at or over pre-specified intervals)

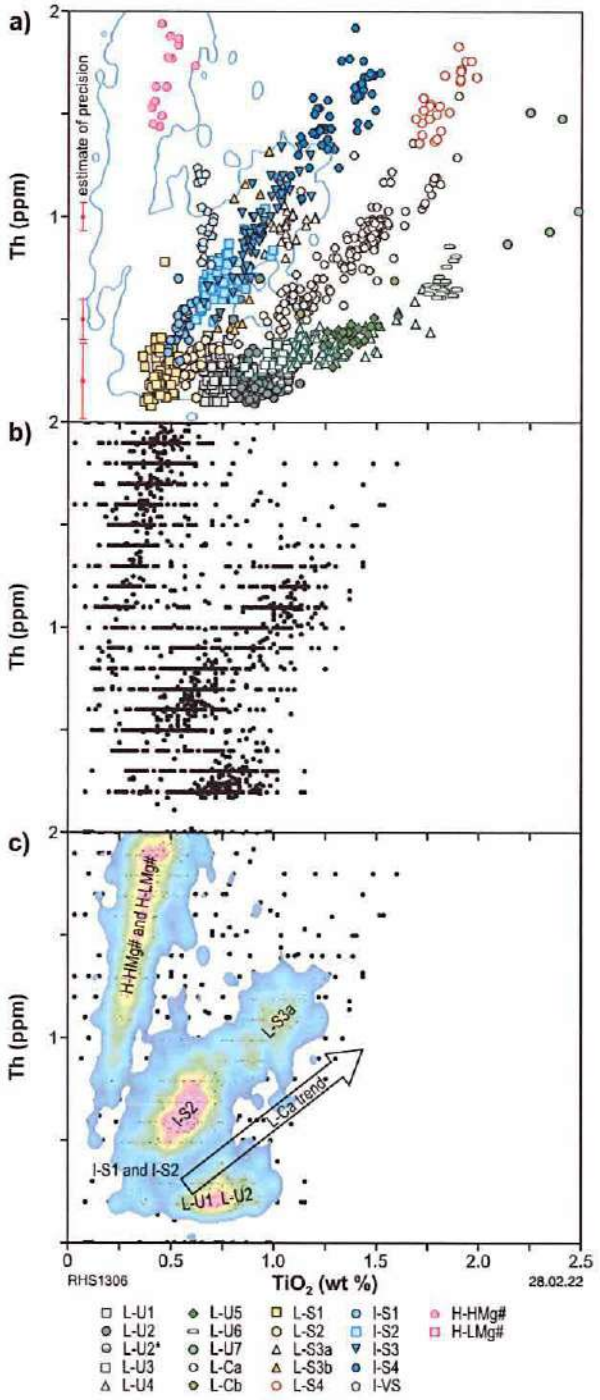


Figure 67. Variation in Th with TiO_2 showing: a) all basaltic data (this study) from within the study region (purple line outlines the extent of the data heat map shown in c below). The estimates of precision (error bars) for Th are based on historical analytical results from BV Minerals (from Champion et al., 2021) and relate only to data presented in part a; b) a subset of ~10 000 data points from a mineral exploration dataset collected from a specific area within the study region; c) a heat map of the data presented in b) showing specific data hotspots reflecting the range of basalt compositional groups expected for the Hannans Subgroup in the Kambalda Domain

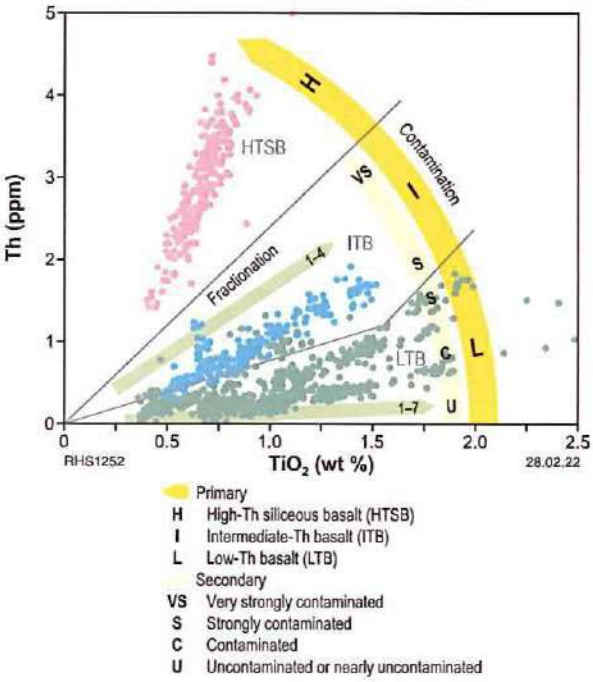


Figure 68. Variation in Th with TiO_2 used for the classification of basalts (after Barnes et al., 2012), showing the rationale behind the system used to give unique codes to each specific mafic geochemical unit (see text for details)

- sampling protocols that do not specifically target the least-altered material or specifically do target altered and/or mineralized material.

When using large assay datasets, employing heat maps can help reduce the impact of scatter related to these factors. Figure 67 provides a simple example of this. When we apply a simple heat map to the industry data, discrete compositional populations become apparent. In this case, it becomes clear that the overall compositional diversity strongly suggests the data are probably from the Kambalda Domain (see below) with compositional hotspots indicating the dominance of L-U1, L-U2, I-S3a, and H-HMg#– H-LMg# basalts. Although limited scatter occurs into the regions of L-S1, L-S2, and L-Ca basalts, the heat map clearly indicates that these compositional populations are at best, very insignificant, and certainly require verification, for example in this case, by looking at a corresponding heat map on a plot of Al_2O_3/TiO_2 vs Al_2O_3.

Using selected major and trace elements, we employ a selection of the compositional variation diagrams that together offer effective identification of most or all of the specific compositional groups. These include plots of TiO_2 vs Th and Zr and Al_2O_3/TiO_2 vs Al_2O_3. On each of these plots and for each of the domains or regions discussed in GSWA Report 226, we show where each compositional unit of the chemostratigraphic column lies within the wider population, comprising all analyses of mafic volcanic rocks within the study area (Figs 69–74). The digital versions of these figures (Appendix 3 of GSWA Report 226) are in layered PDF format so that users can toggle between the specific compositional fields of interest.

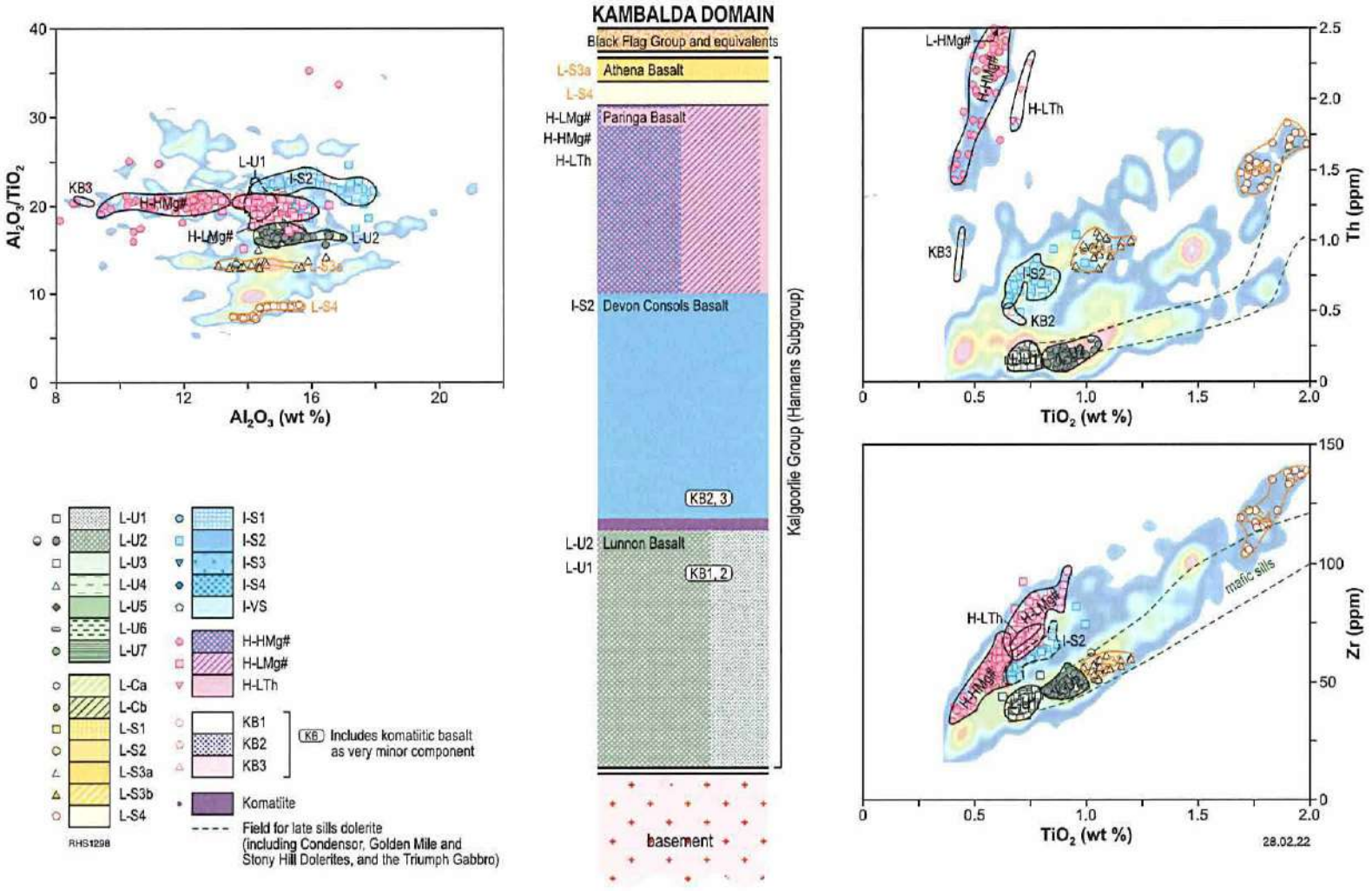


Figure 69. Stratigraphic column and selected geochemical variation diagrams identifying mafic units of the Kambalda Domain

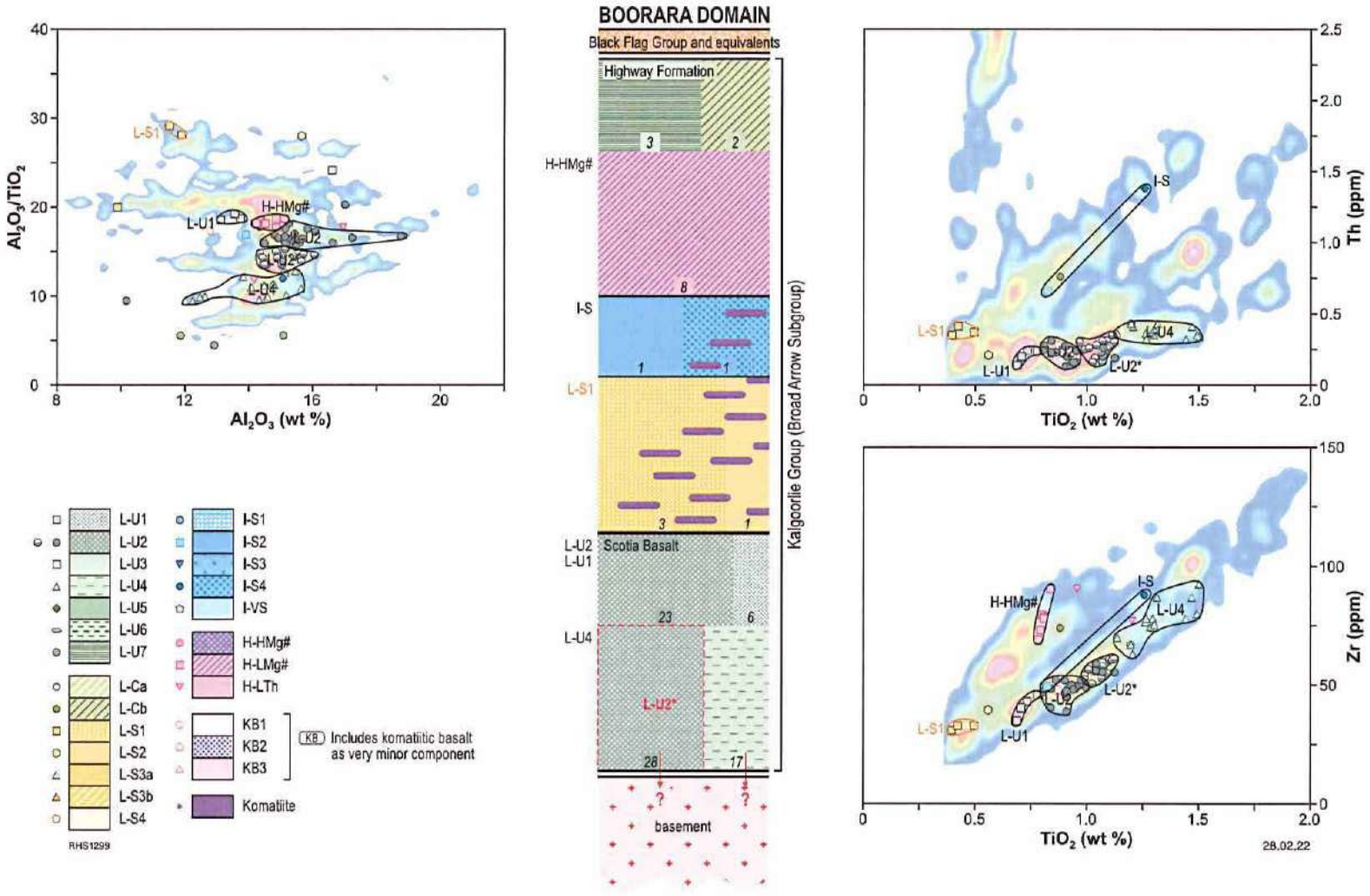


Figure 70. Stratigraphic column and selected geochemical variation diagrams identifying mafic units of the Boorara Domain

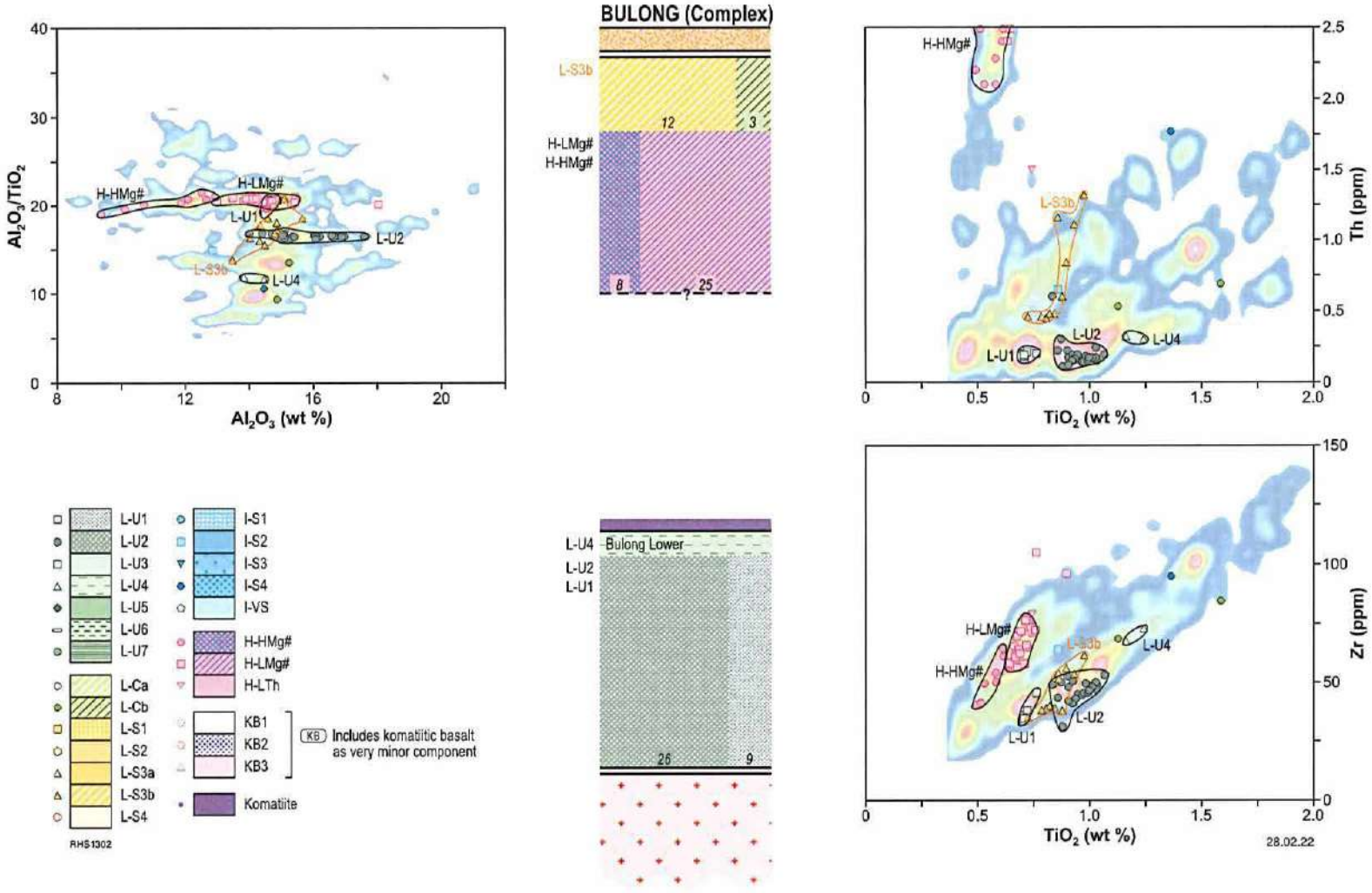


Figure 71. Stratigraphic column and selected geochemical variation diagrams identifying mafic units of the Bulong complex

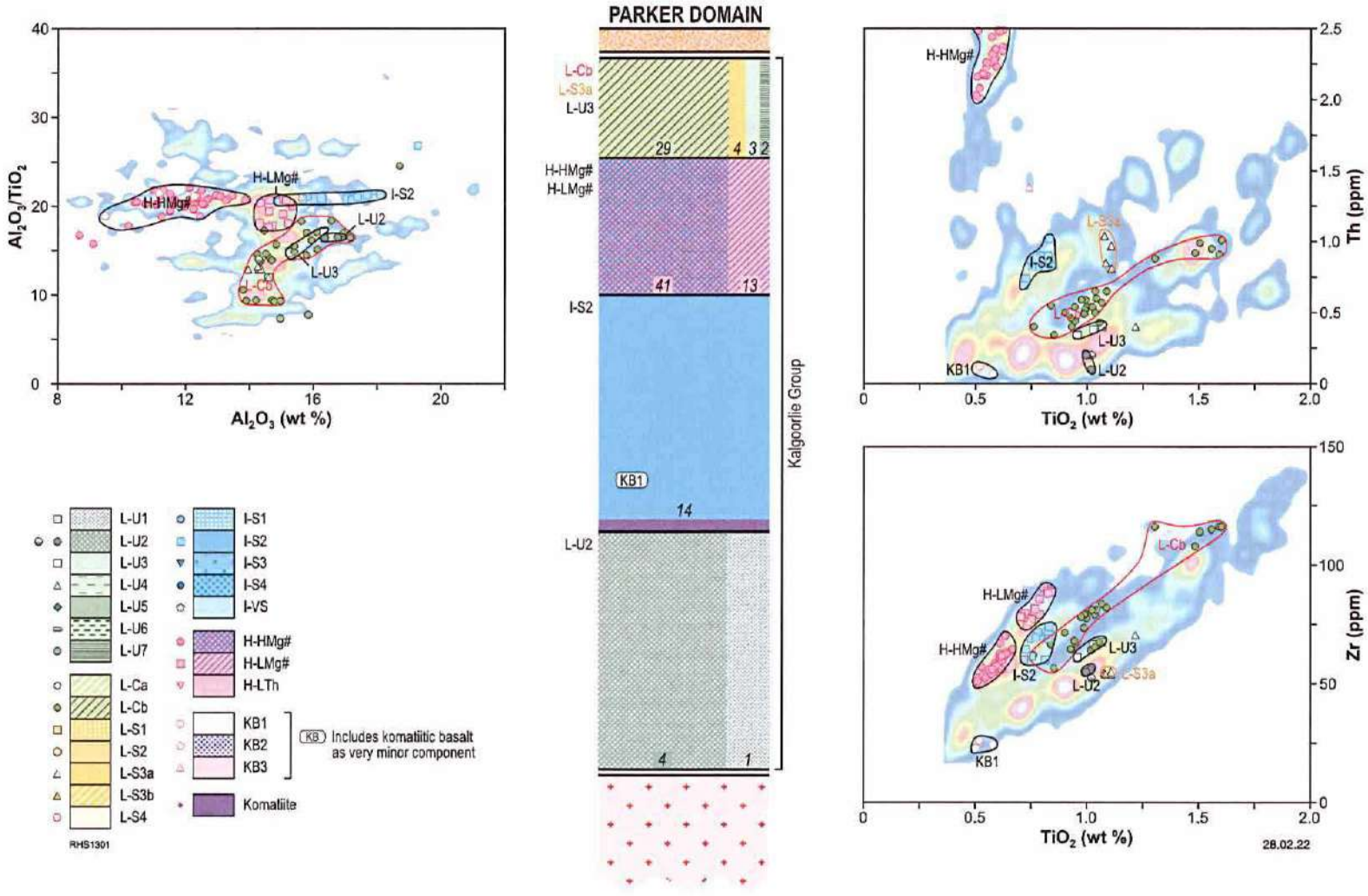


Figure 72. Stratigraphic column and selected geochemical variation diagrams identifying mafic units of the Parker Domain

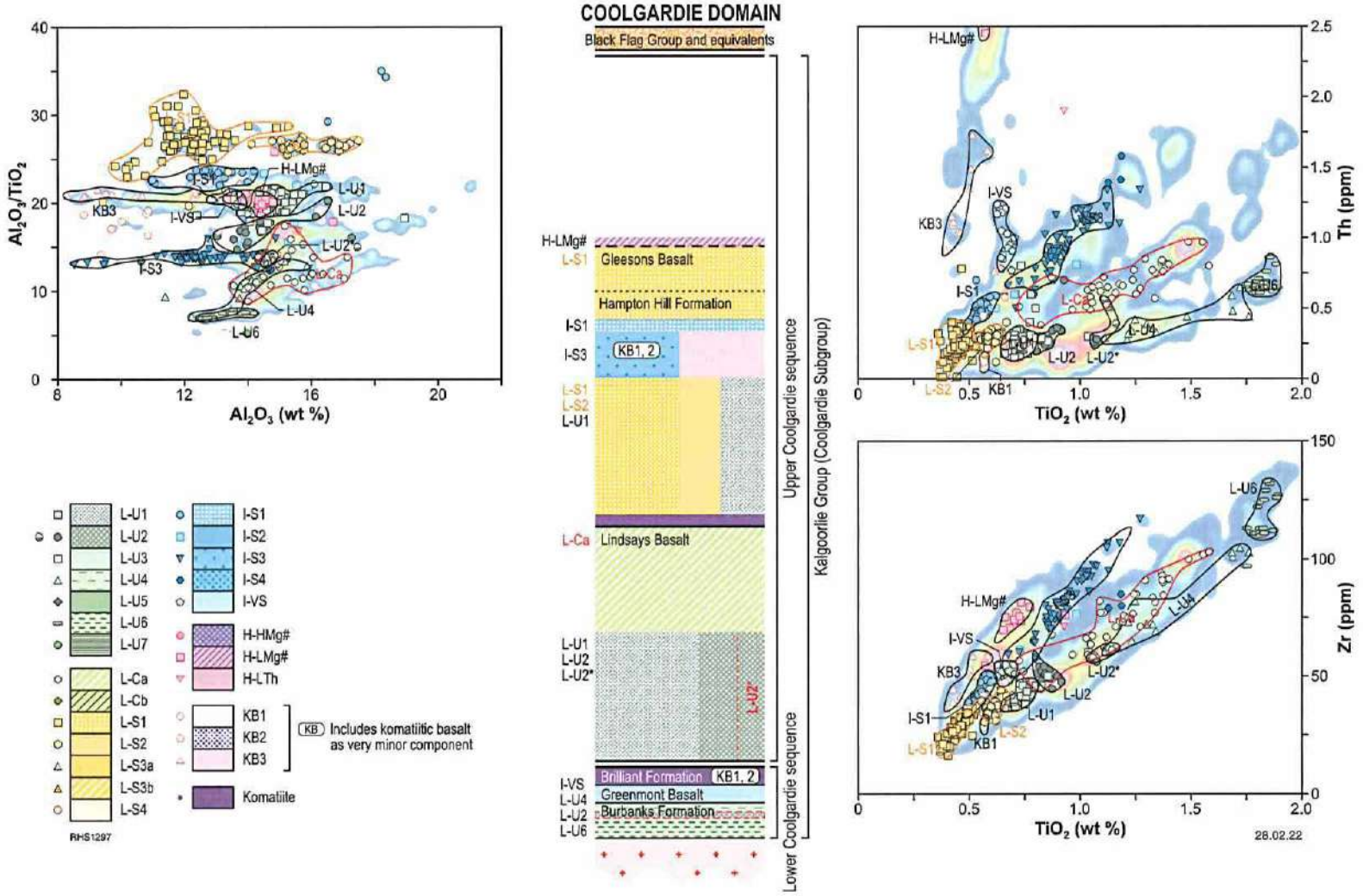


Figure 73. Stratigraphic column and selected geochemical variation diagrams identifying mafic units of the Coolgardie Domain

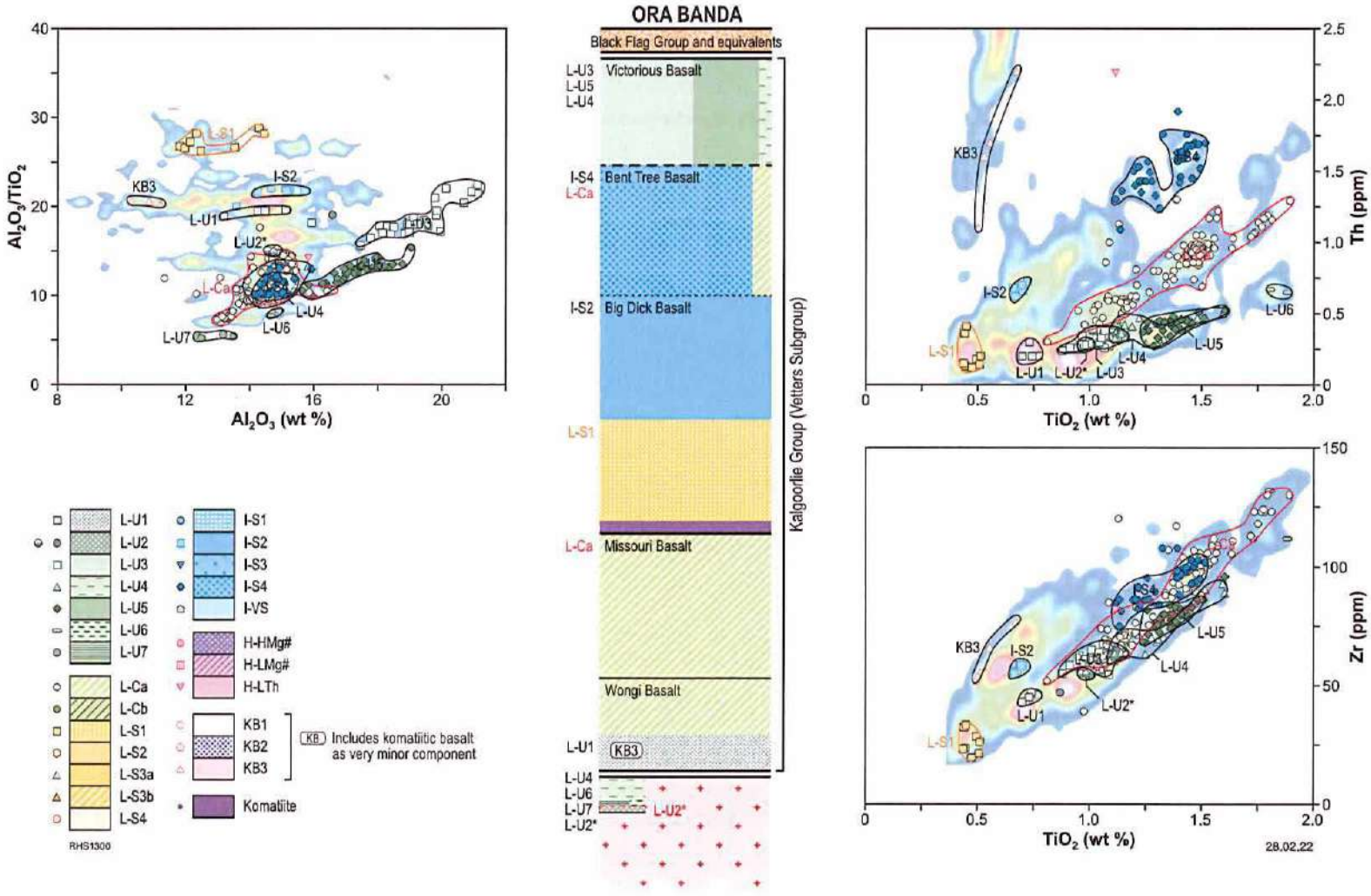


Figure 74. Stratigraphic column and selected geochemical variation diagrams identifying mafic units of the Ora Banda Domain

Subdividing felsic rocks

The geochemical barcoding approach is particularly applicable to basalt-dominated stratigraphy, since low-viscosity mafic (and ultramafic) lavas can travel long distances and blanket the underlying surface. However, the approach is less effective in the case of felsic volcanic and volcaniclastic accumulations, although broad but significant compositional variations can still be observed and utilized.

Felsic (andesite to rhyolite, or intermediate and acid) rock samples taken during this study are mainly from plagioclase (±hornblende or quartz) porphyritic dykes or sills representing subvolcanic intrusions and from volcanic or volcaniclastic deposits, including very immature proximal turbidite units (i.e. typically quartz-poor, poorly sorted, not conspicuously graded deposits). The majority of these samples are from the Black Flag Group, which is interpreted to have been deposited between 2690–2665 Ma across all of the domains of the Kalgoorlie Terrain within the study area. However, the presence of older felsic volcanic rocks is speculated within the lower Coolgardie sequence and demonstrated at numerous localities within the study area, with >2700 Ma felsic volcanic units occurring in (for example) the Bulong complex (e.g. Nelson, 1995a; Barley et al., 2002).

From the original set of 845 felsic samples, filtering for alteration removed a total of 213 (~25%) samples, 50% of which were excluded based on high ASI (up to 2.2), reflecting secondary alteration or significant sedimentary reworking or both. It is important to note that, once filtered, there is no clear difference in the composition or compositional range of the intrusive, volcanic or volcaniclastic rock populations; i.e. no clear evidence that a specific depositional process has significantly modified the primary magmatic compositions.

The majority of subvolcanic intrusions and associated volcanic or volcaniclastic deposits forming the Black Flag Group have the composition of sanukitoids and comprise one of the first recognized examples of Archean sanukitoid volcanic rocks, and one of the most voluminous intrusive/extrusive sanukitoid systems anywhere in global Archean terrains (Smithies et al., 2018). Sanukitoids are relatively rare hornblende-bearing rocks, typically in the monzonitic to dioritic and granodioritic (andesitic to dacitic but reaching rhyolitic) compositional range. They have been attributed to direct extraction as low-degree felsic partial melts from a hydrated mantle source (Shirey and Hanson, 1984; Smithies and Champion, 2000). However, more recently they were interpreted to be (or include examples that are) fractionated from mafic to ultramafic parental magmas (including calc-alkaline lamprophyres), themselves partial melts of metasomatically hydrated and enriched mantle lithosphere (Smithies et al., 2018, 2019). Sanukitoids are amongst the most hydrated and oxidized magmas known from the Archean and their presence implicitly defines a translithospheric plumbing system that transferred fertile mantle material to upper crustal levels (Smithies, 2019; Smithies et al., 2019). In the Yilgarn Craton, and in other Archean regions, sanukitoids have a clear spatial and temporal association with gold mineralization (Champion and Cassidy, 1998; Witt et al., 2013, 2015; Fayol and Jébrak 2017). In the Eastern Goldfields, many of the structures used as magma pathways by sanukitoids are faults that also played a primary role in the structural evolution of the 2664–2657 Ma sedimentary basins (Late Basins) that overlie the Black Flag Group.

Classification of the felsic rocks

A workflow for classifying felsic rocks is provided in Appendix 5. Figure 75 attempts to graphically show aspects of this workflow. Although this classification is based primarily on geochemical

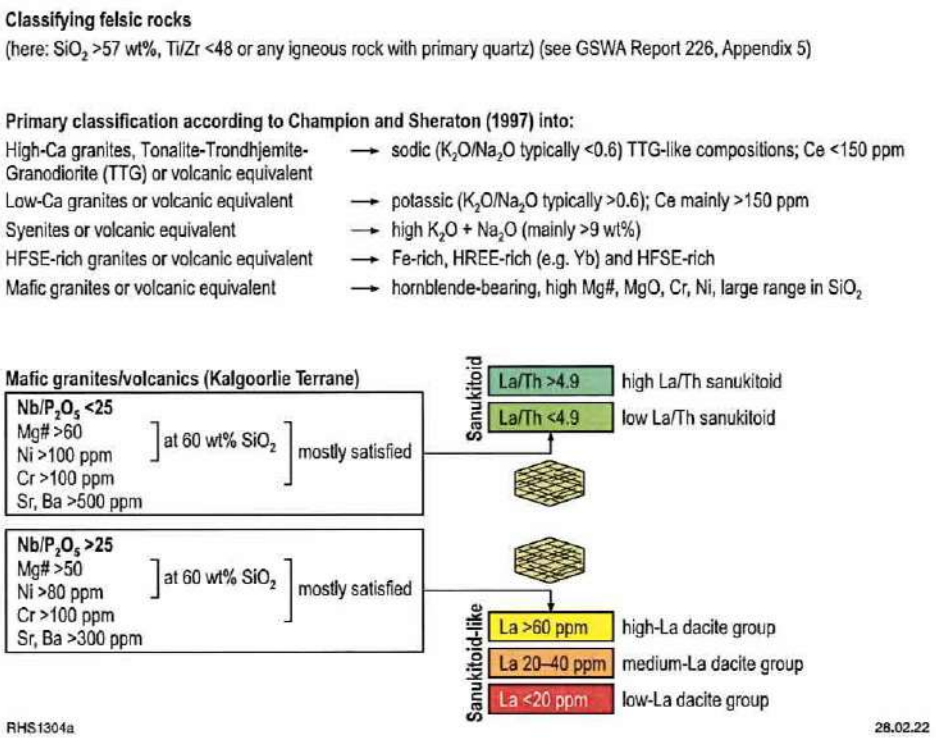


Figure 75. Flow sheet explaining the methodology used here to classify felsic units

data, the geographical and lithological association of samples and mineralogy remain extremely important considerations. For example, sanukitoids typically have quite specific field appearances and mineralogy, particularly at the lower silica end of their compositional range, where they are invariably hornblende-rich and often contain cognate (hornblende-plagioclase±pyroxene) xenoliths. However, whereas lower-silica sanukitoid is also typically distinctive geochemically, at the higher silica end of their range, sanukitoid can be virtually indistinguishable from High-Ca granite. Similarly, High-Ca granites are typically plagioclase rich and K-feldspar poor, but can fractionate to K-feldspar-bearing examples that geochemically resemble Low-Ca granites. In both cases, the spatial and temporal associations may help in classification.

The granite classification scheme of Champion and Sheraton (1997) and Champion and Cassidy (2002) was initially applied to the felsic rocks within the study area. This classification divides Archean granites into five broad groups including:

- High-Ca granites, typically with SiO_2 >68 wt% and a compositional range broadly consistent with that of Archean tonalite–trondhjemite–granodiorite (TTG) suites, reflecting deep crustal melting of dominantly mafic source regions
- Low-Ca granites, typically with SiO_2 >68 wt% but generally more potassic compositions (broadly equivalent to Archean K-rich granite series), and generally more enriched in incompatible trace elements, reflecting higher temperature melting of dryer, more heterogeneous and slightly more refractory sources, including mafic crust and earlier formed TTG crust
- HFSE granites, generally highly silica rich (SiO_2 >70 wt%), Fe rich and strongly enriched in (e.g.) Nb, Zr, and REE, usually with characteristics consistent with the involvement of tholeiitic parental magmas or sources
- Syenites, very alkali rich and typically quartz poor, but also Fe rich and enriched in (e.g.) Nb, Zr, and REE, and additionally in large ion lithophile elements (LILE; e.g. Sr and Ba), thought to be derived through melting of a metasomatically enriched deep lithospheric source (Smithies and Champion, 1999)
- Mafic granites, typically hornblende-bearing rocks showing a wide range of silica concentrations upwards of ~57 wt% with a significant proportion of samples with SiO_2 <68 wt%. Sanukitoids form a large component (>50%) of this group. The remainder includes suites that have $Mg^{\#}$, and/or Cr, Ni, and/or Sr or Ba concentrations lower than expected for sanukitoids, but generally slightly higher than is typical of modern calc-alkaline magmas (e.g. Barnes and Van Kranendonk, 2014), possibly as a result of a hotter Archean mantle.

The entire felsic dataset was divided between samples with compositions equivalent to:

- Mafic granites 87%, of which 66% are true sanukitoid and 34% are sanukitoid-like (see below)
- High-Ca granites (10%)
- HFSE granites (3%)
- Low-Ca granites (<<1%).

Rocks compositionally equivalent to Mafic granites

Sanukitoids

Geochemical identification of sanukitoids is not always straightforward and is particularly problematic in populations that contain few samples with silica contents below ~65 wt%. Felsic rocks with such low silica contents are relatively uncommon in Archean terrains. However, they are unusually common in the Kalgoorlie–Kambalda region, particularly in the Black Flag Group. A sanukitoid suite should be a geochemically coherent (genetically related) group of relatively sodic (K_2O/Na_2O <1.0) rocks, which at low silica values (at 60 wt% SiO_2 or reasonably extrapolated to that value) are dominated by members with $Mg^{\#}$ ≥60, Cr and Ni concentrations ≥100 ppm, Sr and Ba concentrations ≥500 ppm, and La concentrations ≥25 ppm. At silica contents >65 wt% sanukitoid is difficult to geochemically distinguish from High-Ca granites or rocks of TTG suites. Smithies et al. (2019) suggest that some High-Ca granites or TTGs, particularly those showing flat mantle-normalized HREE patterns that reflect hornblende fractionation might be fractionated sanukitoids. Nevertheless, at silica contents >65 wt%, most sanukitoid suites (and Mafic granites, and volcanic equivalents, in general) should range to higher Mg# and higher Ni and Cr concentrations and might show flat or reversely fractionated (spoon-shaped) mantle-normalized HREE patterns.

From a practical point of view, the important aspects of a sanukitoid petrogenesis emphasize derivation from a highly MgO-rich hydrated source. Therefore, any magmatic series that ranges to low silica (<~65 wt%) members containing abundant primary hornblende and having high $Mg^{\#}$ and Cr and Ni concentrations, might also be considered sanukitoid-like (see below).

Most felsic magma series in the Kalgoorlie–Kambalda region that satisfy all or most aspects of the sanukitoid definition (Fig. 76) have a Nb/P_2O_5 ratio <25, whereas most other felsic magma types have higher ratios (Fig. 77). Whilst this discrimination appears to be useful for the Kalgoorlie–Kambalda region, it fails to identify all significantly hydrated sanukitoid-like magmas (see low-La dacites below). The concentrations of both Nb and P are likely to be strongly related to lithospheric mantle-enrichment processes – the concentration of Nb in sanukitoids, for example, has been linked to variations in the relative proportions of fluid- and melt-mediated metasomatism (Smithies and Champion, 2000). We do not recommend the application of this discrimination plot to other regions without further testing.

Two main groups, high La/Th sanukitoids (La/Th = 4.9 and greater) and low La/Th sanukitoids (La/Th <4.9) can be distinguished using a plot of La vs Th (Fig. 78). These groups are broadly equivalent to the high- and low-LILE subgroups of Mafic granites described by Champion and Cassidy (1998). Both groups span the full silica range from <58 wt% to >75 wt%. A third, smaller group has La/Th <4.9, but is restricted to high silica values between 70 wt% and 74 wt%. It has incompatible trace element patterns similar to high La/Th sanukitoids, but is also transitional to silica-rich examples of low La/Th sanukitoids. We referred to this third group as evolved low La/Th sanukitoids. It shows an extensive overlap with rocks that have compositions equivalent to High-Ca granites, but typically has higher Th and LREE concentrations and lower HREE concentrations.

The low La/Th sanukitoid group includes a small P-rich low La/Th sanukitoid subgroup with a distinctly higher P, Th, LREE trend. The high La/Th sanukitoid group includes a small population with distinctly lower Ni concentrations.

High La/Th sanukitoids also have distinctly more radiogenic Nd-isotope compositions with initial $\varepsilon_{Nd\ (2690\ Ma)}$ values lying between 1.5 and 2.7 compared with a range for all types of low La/Th sanukitoids between 0 and 0.8.

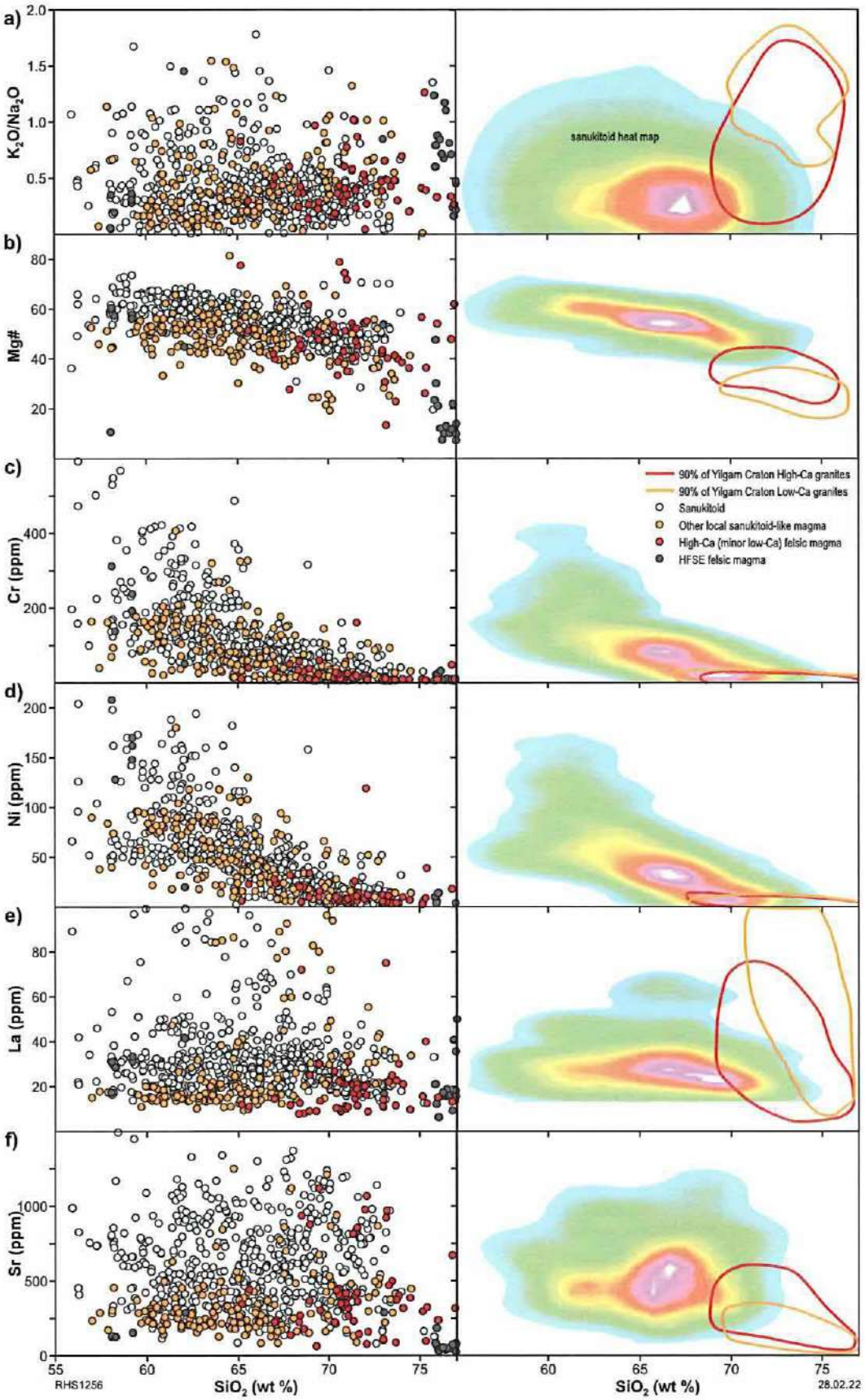


Figure 76. Variation with SiO_2 of various chemical parameters considered useful in the identification of sanukitoids, including K_2O/Na_2O, Mg#, Cr, Ni, La, and Sr. Right-hand column shows heat maps that outline the relative sample density for >95% of sanukitoid samples (hotter colours represent high density of samples) and compares this with the compositional range for High- and Low-Ca granites from the Yilgarn Craton

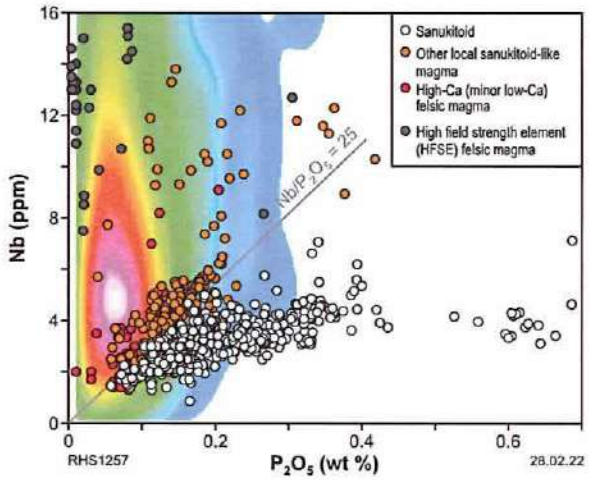


Figure 77. Variation of Nb with P_2O_5 highlighting the compositional contrast between sanukitoids and other felsic rock groups. The heat map shows the density distribution of >90% of granites from the Yilgarn Craton other than the Mafic granites of Champion and Sheraton, 1997

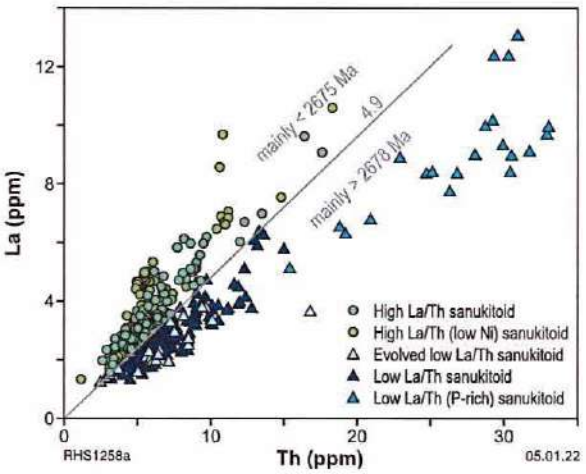


Figure 78. Variation of La with Th for rocks classified as sanukitoids. Available U–Pb (in zircon) dating on Black Flag Group sanukitoids that can be geochemically confirmed to be high La/Th or low La/Th suggest that the age of igneous crystallization of low-La/Th sanukitoids was older than c. 2678 Ma, whereas that of the high-La/Th sanukitoids was younger than c. 2675 Ma

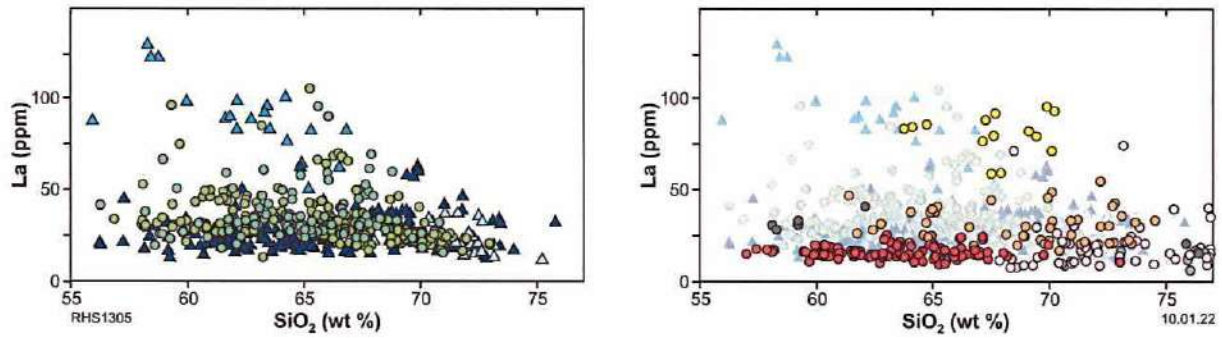


Figure 79. Variation in La with SiO_2 for the various felsic rock types. The left-hand column shows sanukitoids. The right-hand column shows the sanukitoid-like rocks (low-, medium-, and high-La dacites) and other felsic units (bold symbols) and compares these with the sanukitoids (faded symbols)

Non-sanukitoid samples compositionally equivalent to Mafic granites

A large population of subvolcanic, volcanic, and volcaniclastic rocks with Mafic granite compositions has La/Th ratios overlapping those of the high- and low-La/Th sanukitoid groups, but has Nb/P_2O_5 ratios >25. Three groups can be recognized based on distinctive mantle-normalized trace element patterns (see GSWA Report 226) and are reasonably well distinguished based on La concentrations (Fig. 79). These will simply be referred to as low-La (La <20 ppm), medium-La (La 20–60 ppm), and high-La (La >70 ppm) dacites, noting that the low-La dacite group also includes abundant andesite and the medium- and high-La dacite groups include rhyolite.

Apart from generally lower LREE concentrations, the low-La dacites have compositions that closely approach those of sanukitoids, with relatively high $Mg^{\#}$ (mainly 50–55 at 60 wt% SiO_2), Ni (mainly 75–100 ppm at 60 wt% SiO_2), Sr and Ba (both mainly ~300 ppm at 60 wt% SiO_2) concentrations. Indeed, concentrations of Cr (at 60 wt% SiO_2) are typical of sanukitoids. We consider these rocks to be sanukitoid-like.

Medium-La and high-La dacites show many compositional similarities with the low-La dacites but extend to more felsic compositions, have steeper (more fractionated) mantle-normalized trace element patterns with minimal or positive Zr anomalies, higher Na_2O+K_2O, higher FeO and/or lower MgO (and correspondingly lower $Mg^{\#}$, mostly below 50) at a given silica content (see main text).

Distribution and age

Sanukitoids

Of the two main compositional types of sanukitoids, the high La/Th group is concentrated in the southern and eastern parts of the region, while the low La/Th group (excluding the evolved low La/Th subgroup) is mainly concentrated in the north and west (Fig. 80). Sanukitoids in the northwestern part (Kundana region) are exclusively low La/Th types, but to the northeast, high La/Th and low La/Th sanukitoids occur, and locally show mutually intrusive (subvolcanic) or interlayered (volcanic) relationships.

Geochronological data (see GSWA Report 226) for high La/Th sanukitoids indicate magmatism between at least c. 2688 and c. 2653 Ma (Fig. 81), and between c. 2696 and c. 2682 Ma for low La/Th sanukitoids (not including evolved low La/Th sanukitoid).

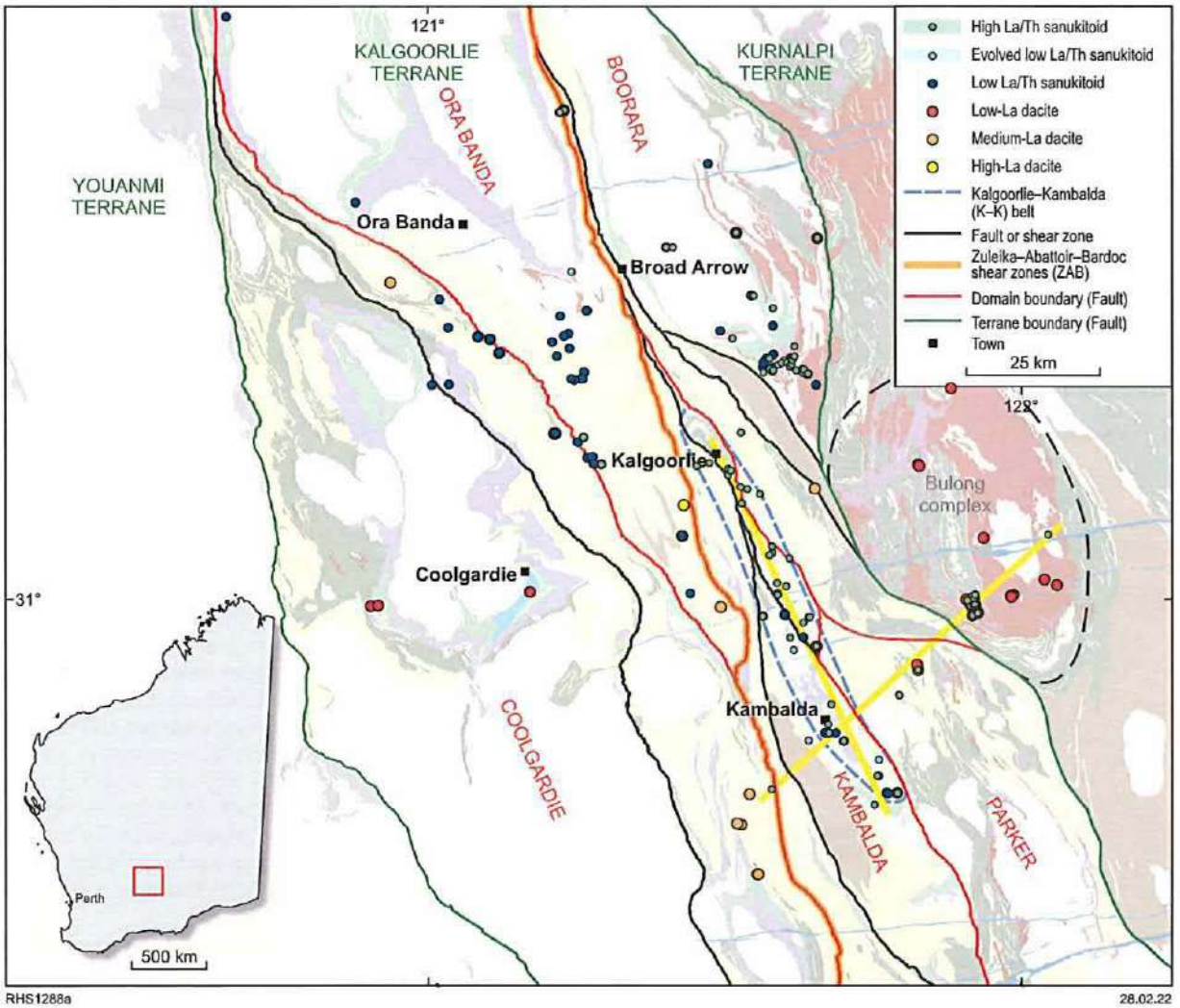


Figure 80. Geological map of the study area within the Kalgoorlie Terrane, showing the distribution of samples of the various sanukitoid types as well as high-, medium-, and low-La dacites. Also shown is the Kalgoorlie–Kambalda belt (dashed line) and other apparent trends outlined by the sanukitoids (yellow)

Evolved low La/Th sanukitoids have ages between c. 2695 and c. 2680 Ma.

Contact relationships clearly require that high- and low-La/Th magmatism was comagmatic although available dating suggests that this period of overlap may have been quite limited, with much of the high La/Th magmatism occurring after much of the low La/Th magmatism. In some regions, the igneous component of the entire preserved Black Flag Group stratigraphy reflects accumulation only from one of the main compositional types of sanukitoid magmatism. The spatial and temporal distribution of the sanukitoid groups is broadly consistent with Tripp's (2013, 2019) stratigraphic interpretations. Thus, at least in broad terms, it seems that the northwestern Kundana – Ora Banda region records or preserves mainly the lower part (Gibson–Honman Formation and below) of the Black Flag Group, overwhelmingly dominated by low La/Th sanukitoids, whereas the northern part of the Kambalda Domain records only the uppermost part (Gidji Lake Formation) overwhelmingly dominated by high La/Th sanukitoids. In this regard, the La/Th ratio of Black Flag Group rocks might provide a first-order estimate of age and stratigraphic position (Fig. 78).

It should be stressed that geochronological compilations by Tripp (2013, 2019) suggest that volcanism related to the Black Flag Group occurred broadly from <2690 Ma to >2665 Ma, with intrusive magmatism from <2691 Ma to <2650 Ma. We have addressed mainly the compositional aspects of Black Flag Group magmatism but our data are broadly consistent with that of Tripp (2013, 2019), in that high La/Th compositions have a higher proportion of intrusive samples than the older low La/Th sanukitoids.

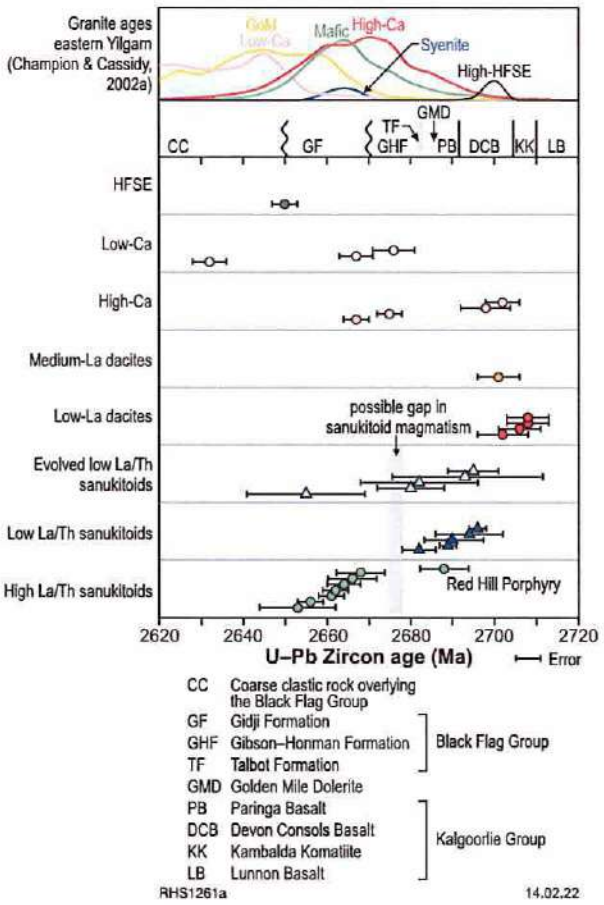


Figure 81. Summary of geochronological data and relationships for felsic units within the study region. See GSWA Report 226, Table 5 for individual U-Pb (zircon) geochronology samples and references. Black Flag Group stratigraphy modified from Tripp (2013). Histograms showing age distribution of granite types are from Czarnota et al. (2010), using data from Champion and Cassidy (2002a)

Other magma types

The sanukitoid-like low-La dacite occurs mainly in the southern part of the study throughout the Kambalda Domain, but appears to be more common in the Mount Monger Station region in the western part of the Bulong complex. Four samples have been dated (see GSWA Report 226) and all dates fall within a narrow age range from 2708–2702 Ma. In all drillcores that show contact relationships between low-La dacite and other units, the dacite was consistently intruded by subvolcanic dykes of high La/Th sanukitoids and was the oldest felsic component observed. Thus, low-La dacite compositions potentially provide a stratigraphic time marker. Along with the La/Th ratio of sanukitoids, these potential compositional age references might prove useful in elucidating field relationships, although they need to be considered with caution.

Some points from this Report with potentially significant implications in local to regional geological interpretation

- There are at least 12 major geochemically discrete units within the study area that have LTB-like compositions. This represents a significant problem or source of serious misinterpretation when LTB compositions are erroneously regarded as synonymous with Lunnon Basalt (i.e. the stratigraphic unit) or are even more broadly referred to as Lunnon-like. Similar problems arise regarding the Devon Consols Basalt and the ITB groups. Of the 12 LTB units, seven are L-U units and only two of these (L-U1 and L-U2) occur within the Lunnon Basalt.

  Hence, **LTB is not synonymous with Lunnon Basalt**. Only a basalt pile within the region between Kalgoorlie and Kambalda, comprising combinations of primitive L-U1 <L-U2 alone, overlain by komatiite and then by I-S2 (i.e. Devon Consols Basalt) is Lunnon Basalt. Likewise, ITB is not synonymous with Devon Consols Basalt.

- Individual lithostratigraphic units within the area, typically of formation level, seldom comprise a single geochemical unit. Those that do, include the Greenmont Basalt (I-VS, lower Coolgardie succession), the Gleesons Basalt (L-S1, upper Coolgardie succession), the Paringa Basalt (HTSB, upper Hannans Subgroup), and the Athena Basalt (L-S3a, upper Hannans Subgroup). Most formations comprise at least two or more distinct geochemical units, and in most cases these units cannot be related back to a common parental magma composition. Thus, individual greenstone formations combine overlapping deposits or flows from fissures or vents supplied by several genetically unrelated magma sources. This potentially makes some stratigraphic formation boundaries arbitrary reflections of changing source eruptions. Nevertheless, the subdivision of the regional (domain) stratigraphies into three levels (lower, middle, and upper basalts), does in most cases reflect broader changes in magma compositions and provides useful markers of relative level within a particular stratigraphy.

- In many cases, the mafic chemostratigraphic columns appear to ignore previously established domain boundaries. However, a broadly north-trending structure incorporating segments of (from south to north) the Zuleika Shear Zone, Abattoir Fault and the Bardoc Shear Zone (ZAB) does appear to separate the western domains (Coolgardie and Ora Banda) from the eastern domains (Kambalda, Boorara, Parker and the Bulong complex), having distinctive chemostratigraphic variations, which in the western domains, probably reflects melting beneath thicker lithosphere and possibly a slightly more melt-depleted source. The chemostratigraphic differences between the western and eastern domains extends to the felsic units of the Black Flag Group.
- Because textural or mineralogical features often cannot effectively discriminate between compositionally distinct basaltic units, mapping the discrete units at a stratigraphic member level might not be possible without a level of geochemical sampling density even greater than employed here. In any case, this would almost certainly not add any clarity to the potential ambiguities involved in identifying higher order lithostratigraphic boundaries. This also has implications for using geochemistry to infer stratigraphic position because, except in very rare cases, a single or low number of analyses will not be sufficient to uniquely identify a lithostratigraphic position, but more comprehensive sampling, particularly if it incorporates the underlying and overlying stratigraphic units almost certainly will.
- There are significant geochemical differences between the lower and upper stratigraphic successions of the Coolgardie Domain. Here, the lithogeochemistry provides a clear indication that the upper (Lindsays Basalt and above) and lower (below Lindsays Basalt) successions do not represent a structural repetition (although it remains plausible that they might be in thrust contact). The lower Coolgardie succession almost certainly should not be considered as part of the Coolgardie Subgroup, as is presently the case.
- A lack of significant chemostratigraphic difference strongly supports the re-incorporation of the Depot Domain into the Coolgardie Domain. However, there are a few anomalies. For example (see main text):
  - I-S2 overlies I-S1 in drillcore (BDDD0017), immediately west of the Zuleika Shear Zone (approximately 10 km to the north of the Powder Gabbro)
  - In the Kundana region near Bullock Hole, immediately west of the Zuleika Shear Zone that marks the boundary between the Coolgardie Domain and the Ora Banda Domain (in the north) or the Kambalda Domain (in the south), stratigraphy identical to the upper part of the Vetters Subgroup is intersected in drillcore (PGDD15001, MEDD148, MEDD144, and FLRD04) and in outcrop
  - Between the Connolly Gneiss and the Maloneys Dam Granite, approximately 13 km southeast of Londonderry Siding, the greenstone stratigraphy is dominated by a I-S4–L-Ca association very reminiscent of the upper part of the Vetters Subgroup and otherwise unique within the upper Coolgardie succession. However, this observation is based on limited data (7 analyses).
- The Black Flag Group is overwhelmingly dominated by volcanic, volcaniclastic, and subvolcanic rocks with compositions typical of sanukitoids. Slightly older felsic volcanic and subvolcanic rocks are also sanukitoid-like.
- Sanukitoid-like, low-La dacite group rocks possibly form a regional magmatic time marker at c. 2706 Ma.
- Sanukitoids of the Black Flag Group can be subdivided into:
  - an older (mainly pre-c. 2680 Ma) low La/Th group, which mainly occurs in the northwestern part of the region
  - a younger (mainly post c. 2670 Ma) high La/Th group, which dominates the eastern part of the area.
- Based on Nd-isotopic variations, the translithospheric ZAB structure, identified based on variations in the geochemistry of the underlying greenstone basalts, also separates the more radiogenic (mantle-like) source of the high La/Th sanukitoids to the east from the less radiogenic source of the low La/Th sanukitoids to the west.

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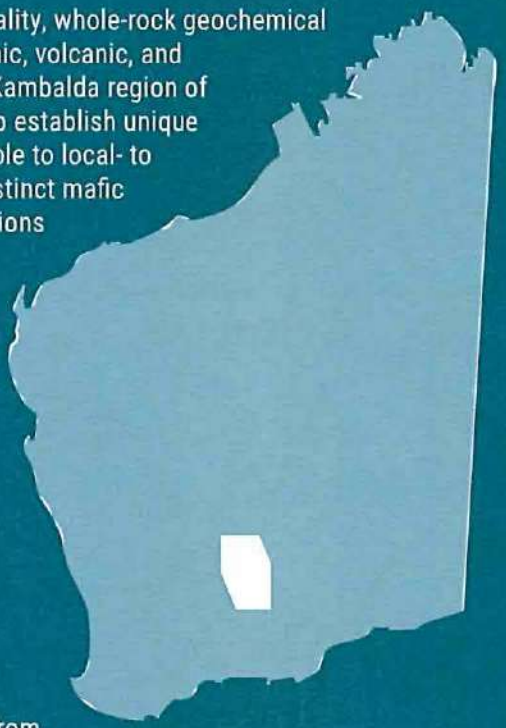
REPORT
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GEOCHEMICAL CHARACTERIZATION OF THE MAGMATIC STRATIGRAPHY OF THE KALGOORLIE AND BLACK FLAG GROUPS – ORA BANDA TO KAMBALDA REGION

RH Smithies, JR Lowrey, J Sapkota, MC De Paoli, P Hayman, SJ Barnes, DC Champion, Q Masurel, N Thébaud, LL Grech, M Drummond and R Maas

This Report discusses the use of ~2800 new, high-quality, whole-rock geochemical data from drillcore and outcrop samples of subvolcanic, volcanic, and volcaniclastic rocks collected from the Ora Banda – Kambalda region of the Eastern Goldfields Superterrane. The purpose is to establish unique lithogeochemical characteristics (bar codes) applicable to local- to regional-scale greenstone stratigraphy. Twenty-six distinct mafic compositional units are identified based on combinations of major and trace element variations and are used to erect chemostratigraphic columns that highlight trends in compositional changes in evolving magma compositions throughout the region. Although complex, this chemostratigraphic diversity ensures that, in most cases, the position of stratigraphically unknown geochemical samples can be uniquely established as long as enough samples are taken over a continuous stratigraphic interval that also incorporates the overlying and underlying units. The regional distribution of the mafic chemostratigraphic associations is distinctly asymmetric, in many cases ignoring previously established domain boundaries. However, a broadly north-trending structure incorporating segments of (from south to north) the Zuleika Shear Zone, Abattoir Fault, and the Bardoc Shear Zone does appear to separate western domains from eastern domains. These domain groups have distinctive chemostratigraphic variations that probably reflect a fundamental crustal architectural control and potentially provide some basis for reassessing the present domain boundaries. The chemostratigraphic differences between the western and eastern domains extend to the felsic units of the Black Flag Group.



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Deacon nickel sulfide deposit, Tramways Dome, Kambalda

J D Hicks[1] and R G Parkinson[2]

INTRODUCTION

The Lanfranchi nickel mine consists of a series of Kambalda-style komatiite-hosted nickel sulfide deposits developed about the southern and eastern flanks of the Tramways Dome, which is 40 km south of the Kambalda township in Western Australia, at -31.50°, 121.85° (Figure 1). Panoramic Resources Limited (Panoramic) acquired a 75% interest in Lanfranchi in 2004 from WMC Resources Ltd (WMC) and moved to 100% ownership in 2009.

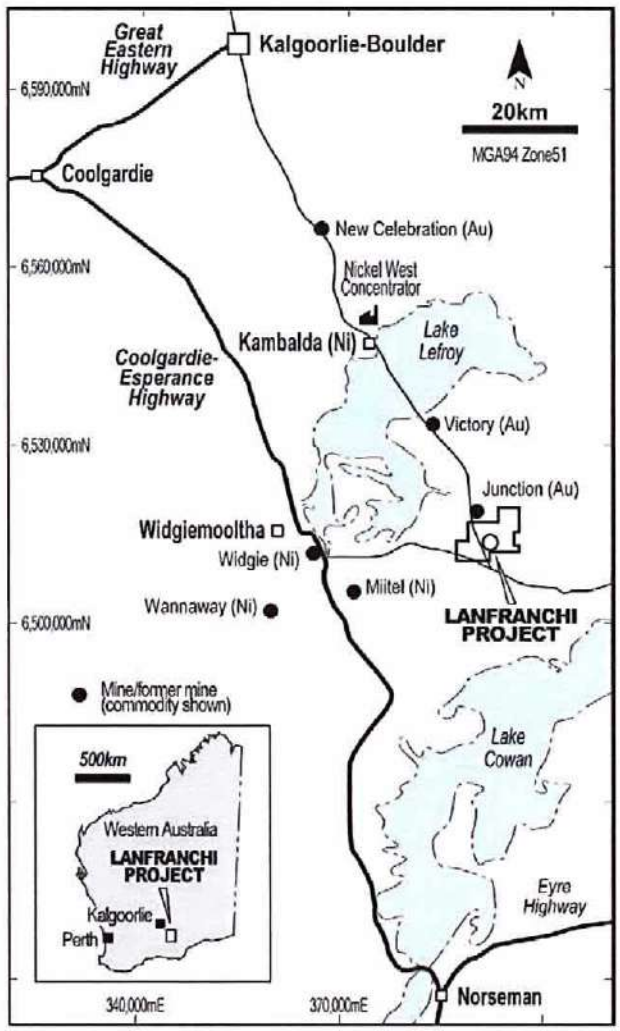


FIG 1 – Plan of the Tramways Dome showing depth contours for the Lunnon Basalt – Kambalda Komatiite contact and distribution of nickel sulfide resources.

Since Panoramic acquired Lanfranchi there has been a significant increase in the resource base of several existing deposits through the application of geological targeting, drilling and downhole electromagnetic surveying as exploration techniques. This included the discovery of several new deposits, including Deacon, the largest nickel sulfide deposit on the Tramways Dome to date. Deacon has a premining resource of 73 000 t contained Ni; other discoveries by Panoramic include Lower Schmitz (6700 t Ni), Metcalfe (6400 t Ni) and Helmut South Extended (7100 t Ni) deposits. The combined premined resource of the new discoveries totals 95 000 t contained Ni.

Exploration about the Tramways Dome has led to a revised understanding of its architecture. Previously, a major truncating thrust structure was interpreted to exist along the northern margin. Drilling conducted over the last decade demonstrates that the Tramways Dome is an overturned fold structure, which has important implications for the discovery of nickel sulfide mineralisation on its northern side.

EXPLORATION AND MINING HISTORY

Nickel sulfide mineralisation was discovered at Tramways in 1969. Exploration over the Dome identified ten nickel sulfide deposits with defined ore reserves. Mining commenced at the Edwin deposit in December 1975 and total Tramways nickel production between 1976 and 2002 – when mining operations were suspended – was approximately 3.09 Mt at 3.19% Ni for 98 500 t contained Ni. Most of this production occurred between 1987 and 2001 when the larger deposits of Lanfranchi, Schmitz and Skinner were mined. Remnant and undeveloped resources at the time of closure in 2002 were estimated at 1.11 Mt, averaging 3.34% Ni for 37 150 t contained Ni.

After acquiring Lanfranchi in 2004, Panoramic resumed production in 2005 and produced 3.86 Mt at 2.45% Ni for 94 725 t contained Ni up to September 2015, when the mine was placed on care and maintenance. Production during this period was from the existing Helmut South, Lanfranchi, Schmitz and Winner deposits and Deacon, Helmut South Extended and Metcalfe. Remnant and undeveloped resources at Lanfranchi at 31 December 2016 are estimated to be 5.65 Mt at 1.69% Ni for 95 500 t contained Ni (Table 1).

1. MAusIMM, Panoramic Resources Ltd, Perth WA, jhicks@panres.com; 2. Panoramic Resources Ltd, Perth WA.

TABLE 1 – Resources and Reserves estimate at 30 June 2016.

Mineral Resources[a]	Tonnage (t)	Ni grade (%)
Inferred Resources	2 956 000	1.52
Indicated Resources	2 517 000	1.73
Measured Resources	180 000	3.75
Total Resources	5 655 000	1.69
Ore Reserves	**Tonnage (t)**	**Ni grade (%)**
Probable Reserves	208 000	2.03
Proved Reserves	-	-
Total Reserves	208 000	2.03

a. Mineral Resources are reported inclusive of Ore Reserves. Source: Panoramic Resources Limited, Australian Securities Exchange announcement, 30 September 2016.

PREVIOUSLY PUBLISHED WORK ON THE DEPOSIT

The geological framework of the komatiite-hosted nickel deposits of the Kambalda district is well documented; however, no succinct work exists for the Tramways deposits. Descriptions of the geology and some of the associated nickel deposits are recorded in Marston (1984), Cowden and Roberts (1990), Stone and Masterman (1998), Stone and Archibald (2004) and Stone, Heydari and Seat (2004).

REGIONAL GEOLOGY

The Kambalda nickel district, including the Tramways Dome area, is located in the south-central part of the Norseman–Wiluna greenstone belt of the Yilgarn Craton (Gee *et al*, 1981; Cowden and Roberts, 1990). All deposits in the district are broadly recognised to be associated with the Kambalda Komatiite. The komatiite is part of the Kambalda domain, a volcanic and sedimentary sequence accumulated in the 40 Ma interval from 2710 to 2670 Ma. The sequence was subsequently intruded by granite, metamorphosed to amphibolite facies, and complexly faulted and folded during four episodes of deformation (Gresham and Loftus-Hills, 1981). During metamorphism and deformation, the komatiitic rocks hydrated to antigorite chlorite-magnetite assemblages or carbonated to talc-magnesite or dolomite-chlorite assemblages.

Studies of the Kambalda Komatiite have shown that the komatiites can be divided into channel flow facies and sheet flow facies (Cowden, 1988; Lesher, 1989). The channel flow facies is dominated by up to 100 m of magnesian (to 48% volatile-free MgO) komatiitic flows with thick olivine cumulate zones and relatively thin spinifex-textured zones. The channel facies typically occupy trough (or channel) structures in the underlying Lunnon Basalt. Individual flows have been mapped for up to 15 km in length and 500 m in width. Mineralisation is preferentially developed within the channel facies. The sheet flow facies consist of thin, less magnesian (≤36% volatile-free MgO) komatiite flows with thin cumulate zones and relatively thick spinifex-textured zones. They are often separated by thin iron sulfide ± graphite -rich sedimentary units and lack any significant nickel sulfide mineralisation.

Nickel sulfide mineralisation is typically restricted to the base of the lowermost channel facies flow and can occur as tabular-to ribbon-like bodies up to 3 km long and 300 m wide, with a thickness generally less than 5 m. Individual deposits typically range between <0.5 and 10 Mt in size (Marston and Kay, 1980). Evans, Cowden and Barratt (1989) described them as pod-like bodies hundreds of metres long and distributed along a linear trend. The distribution, form and tenor of nickel sulfide mineralisation within the channel flow facies varies between deposits. A typical mineralisation profile consists of a contact zone 2 m thick of massive sulfide at the base, overlain by a similar thickness of matrix sulfide followed by disseminated sulfides.

Tramways Dome geology

Similar to the Kambalda Dome, the Lunnon Basalt forms the core of the Tramways Dome, covering an area of 15 km^2 (Figure 2). Stratigraphically overlying the Lunnon Basalt at Tramways is the Kambalda Komatiite. The southern side of the dome, which hosts the majority of nickel sulfide mineralisation discovered to date at Tramways, contains the greatest thickness of komatiite. Here the komatiite sequence is up to 450–500 m thick and dips consistently between 30 and 35° towards the south. Talc-magnesite alteration of the Kambalda Komatiite at Tramways is dominant and widespread, replacing an earlier antigorite alteration event. In contrast, dolomite ± chlorite is observed at Cruickshank on the eastern margin of the Dome replacing the talc-magnesite alteration.

Poor outcrop and sparse drilling across the northern margin of the dome led previous workers to interpret a major truncating thrust structure (Tramways fault) along this margin. This interpretation is no longer supported by drilling and an overturned fold model is preferred. Exploration drilling completed over the last decade confirms that the Kambalda Komatiite on the dome's northern margin is overturned and dips back towards the south at approximately 50–60° beneath the outcropping Lunnon Basalt (Figure 3). The overturned model is supported by a 2D seismic survey completed across the Dome in 2006, and at Cruickshank, where the deposit and host komatiite sequence have been observed in drill holes to fold around the nose of the overturned fold structure.

ORE DEPOSIT FEATURES

The Tramways Dome nickel sulfide deposits are typical Kambalda-style komatiite-hosted deposits. The deposits are developed either at or within a few metres above the contact between the Lunnon Basalt and overlying Kambalda

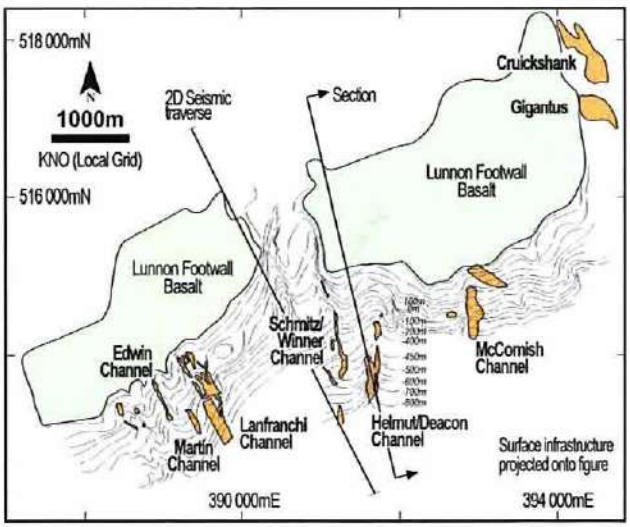


FIG 2 – Plan of the Tramways Dome showing depth contours for the Lunnon Basalt – Kambalda Komatiite contact and distribution of nickel sulfide resources (Note: UTM True North and KNO grid north are essentially the same).

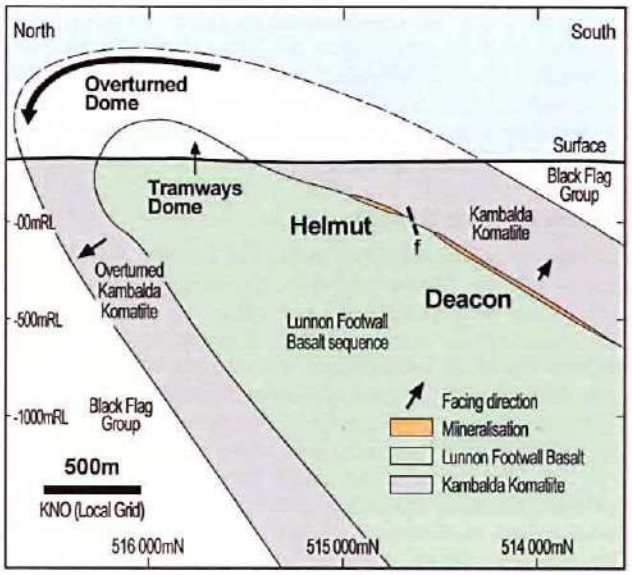


FIG 3 – Simplified geological section (Figure 1) through Tramways Dome.

Komatiite. They typically form tabular to ribbon-like bodies that are several hundred metres long, 25 m to 200 m wide and generally less than 5 m thick. Mineralisation can take the form of either massive, matrix or disseminated sulfides or a combination of all three styles, and is dominated by pyrrhotite + pentlandite ± pyrite ± chalcopyrite.

Thirteen nickel sulfide deposits have been delineated to date along the most intensely explored southern and eastern sides of the Tramways Dome. All but two deposits discovered to date occur on the south side of the dome within trough structures containing channel flow facies komatiite lithologies. The low-grade Cruickshank and Gigantus deposits located on the eastern margin consist dominantly of thin, tabular bodies of disseminated mineralisation. Neither deposit has a clear association with significant channel flow facies rocks or confining trough structures. No deposits have been identified at interflow positions higher up within the komatiite sequence. Several drill hole intersections of massive nickel sulfide mineralisation have been made on the northern overturned Tramways Dome, but to date no significant new deposits have been discovered in the area.

Deacon

The Deacon deposit is located at the base of a pronounced trough structure at the contact between the Lunnon Basalt and Kambalda Komatiite (Figure 4). Deacon was discovered within the Helmut – Helmut South trough structure in September 2006 and is the largest Tramways nickel deposit discovered to date, having a premining resource of 73 000 t contained Ni. The discovery also made the Helmut – Helmut South – Deacon trough structure the most Ni-endowed trough at Tramways with a combined premining resource of ~115 000 t contained Ni.

Deacon was a geological discovery based on the realisation that Helmut South was on the eastern wall of a major trough structure, and that the trough's core lay untested further to the west. The Deacon discovery was made when Helmut South grade control holes were extended into the trough area, and drill hole HS212 returned an intersection of 37.4 m grading 2.07% Ni.

The Deacon deposit extends down plunge for 650 m and is up to 100 m wide and 25 m thick. A typical mineralisation profile of contact massive sulfides overlain by matrix and

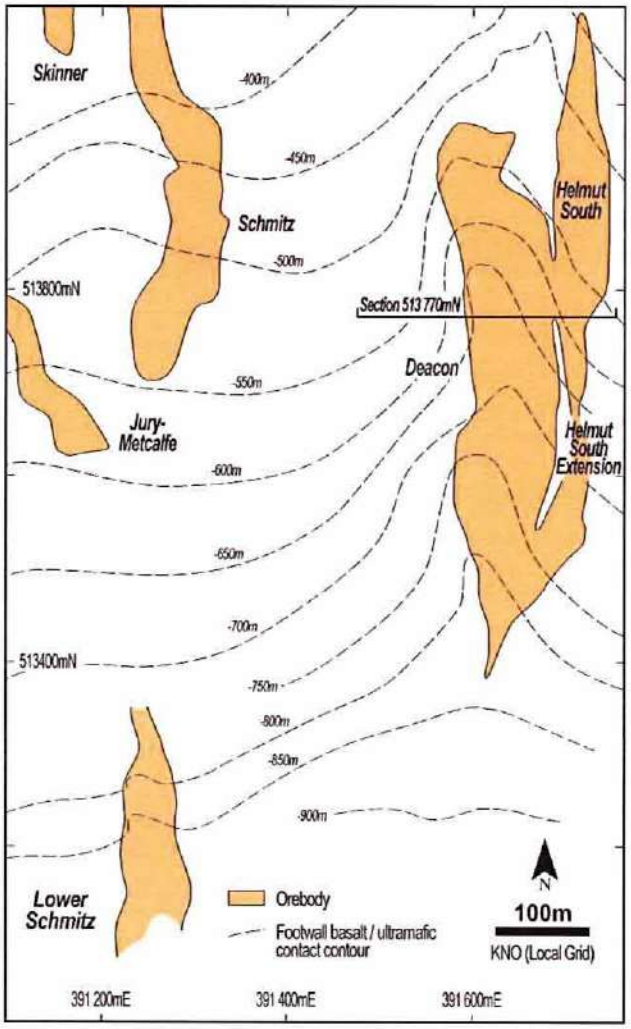


FIG 4 – Deacon Schmitz area plan showing Lunnon Basalt – Kambalda Komatiite contact contours (approximately true North).

disseminated sulfides is developed over a 300 m long central core of the deposit. Where the contact massive sulfide mineralisation is not present, matrix mineralisation directly overlies the footwall basalt contact. Matrix mineralisation averages 2.7% Ni and is up to 20–25 m thick in places. Where the Deacon and Helmut South deposits are developed adjacent to each other, a narrow unmineralised zone of Lunnon Basalt contact typically separates the deposits; however, in certain places a thin mineralised link is evident between the two deposits.

Deacon differs from the other Tramways deposits by the thickness of the matrix mineralisation and the depth and tightness of the host trough structure (Figure 5). Over the length of the deposit the trough structure that hosts Deacon is very pronounced, with steep basalt walls on either side that rise up well in excess of 100 m in places. To the north (beyond the Helmut deposit) and south of Deacon, the host trough structure becomes far less pronounced, exhibiting a much broader, open profile. No significant mineralisation has been discovered in these more open parts of the trough. All evidence points to the Deacon trough structure being a primary depression in the basalt substrate. Talc-magnesite altered olivine ortho to mesocumulate komatiite fills the trough to a height of 100 m before the first spinifex-textured flow top and thin sulfidic interflow sedimentary unit is recognised.

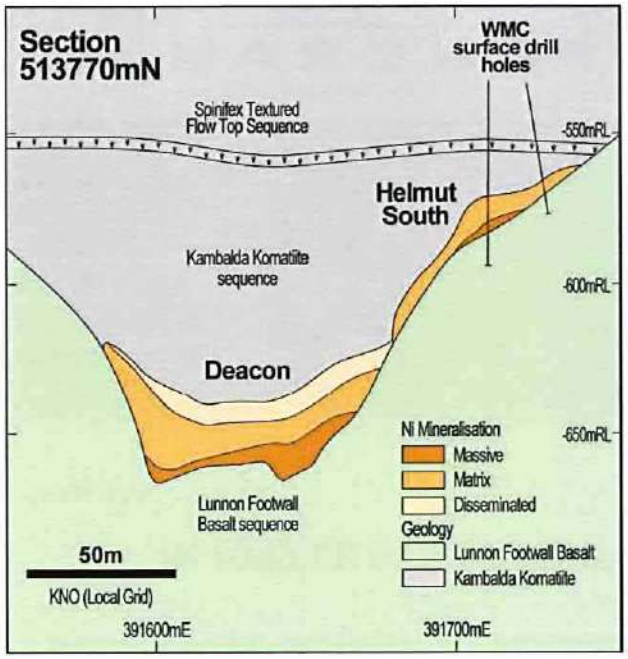


FIG 5 – Deacon geological section 513 770 mN.

ACKNOWLEDGEMENTS

The authors thank Panoramic Resources Limited for permission to publish this paper. The authors also acknowledge the many geoscientists and researchers who have worked in the Kambalda nickel district since the discovery of nickel sulfide mineralisation 50 years ago in 1966. The current depth of knowledge and understanding of the formation of komatiite-hosted nickel sulfide systems is due to their endeavours.

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News Release PJD-47

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28 April 2011 ASX: PAN

Cruickshank Resource Upgraded 26% to 33,560t Ni

Panoramic Resources Limited ("**Panoramic**") is pleased to announce an upgrade to the Cruickshank Mineral Resource at the Lanfranchi Project.

Significant Points

- Cruickshank Mineral Resource is between **70m and 500m below surface**
- The total Mineral Resource is estimated to contain **2.63Mt at 1.28% Ni for 33,560t Ni contained**
- Total contained nickel tonnes have **increased by 26%** from the 2007 estimate
- Indicated Resource total **2.02Mt at 1.42% for 28,720t Ni contained**
- Mine planning studies are underway to convert the Indicated Resource to a maiden Reserve

Fifty-four holes were drilled to upgrade the majority of the Resource from Inferred to Indicated (*Figure 1*). The resource estimate is constrained within a 3D mineralisation model constructed to a 0.5% nickel cut-off grade. At a resource block grade cut-off of 1.0% Ni inside the 3D model the Cruickshank Mineral Resource is estimated to contain 1.84mt at 1.51% Ni for 27,720t Ni contained (*Table 1*). The 42 mineralised drill holes within the Cruickshank 3D mineralisation model are listed in Table 2.

Table 1: Cruickshank – April 2011 Mineral Resource Estimate

Ni% cut-off	Class	Res-Zone	Tonnes	Ni %	Ni Tonnes	Cu %	Cu Tonnes
Inside 0.5%	Indicated	Main lens	2,018,300	1.42	28,720	0.12	2,465
model	Inferred	Main lens	610,850	0.79	4,840	0.07	405
Total			**2,629,150**	**1.28**	**33,560**	**0.11**	**2,870**
0.75%	Indicated	Main lens	1,907,400	1.47	28,060	0.13	2,390
	Inferred	Main lens	491,300	0.92	4,520	0.07	340
Total			**2,398,700**	**1.36**	**32,580**	**0.11**	**2,730**
1.0%	Indicated	Main lens	1,725,630	1.53	26,440	0.13	2,245
	Inferred	Main lens	114,770	1.12	1,280	0.08	95
Total			**1,840,400**	**1.51**	**27,720**	**0.13**	**2,340**

Notes:
The April 2011 Cruickshank Mineral Resource estimate has been prepared by BM Geological Services Pty Ltd and Panoramic geological personnel. The estimate is JORC compliant and is based on a 3D Surpac block model utilising a block size of 20m NS x 20m EW x 5m vertical with 5.0m x 5.0m x 1.25m sub-cells. All drill holes have been accurately located by an independent licensed surveyor and by down-hole Eastman and Reflex survey instruments. Grade interpolation is by Ordinary Kriging techniques using an oriented search ellipse based on the geometry of the mineralisation.

The tonnes and grade of the Mineral Resource estimate is the material constrained within the interpreted mineralisation model which was constructed on sectional wireframe interpretations to a 0.50% Ni cut-off grade. The sectional wireframes were joined to form a 3D model (solid) of the mineralisation. The Mineral Resource is largely classified as Indicated due to the good continuity of the mineralisation, the adequate drill hole spacing and the confidence gained from QA/QC checks and data validation. Small areas of mineralisation at the extremities of the resource have been classified as Inferred due to the lack of sample support, lower continuity of grade and lithological controls.

News Release



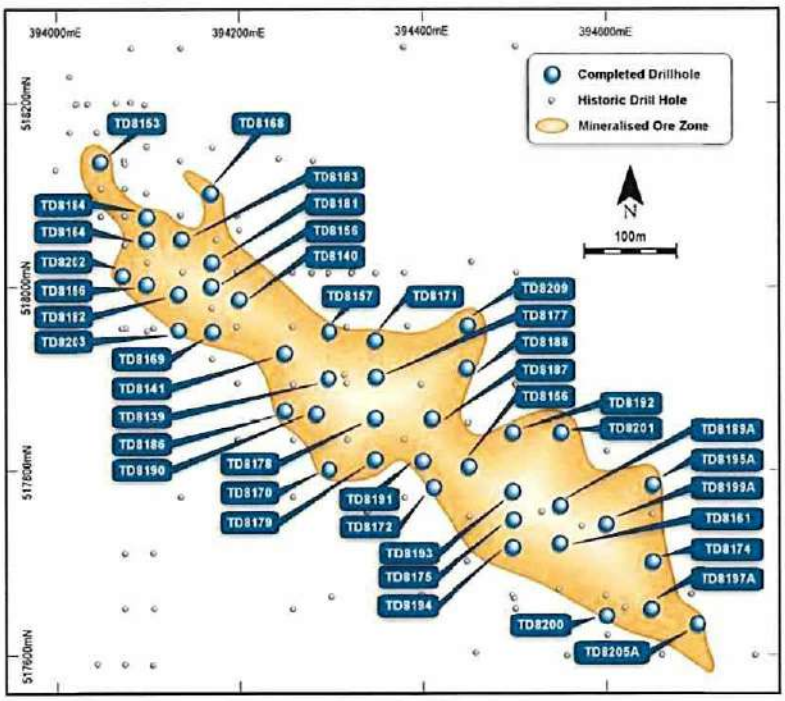
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Table 2: Cruickshank drill results

Drill hole	Section (mE)	Interval (m)	Ni %
TD8153	394050	9.00	1.19
TD8202	394075	1.00	1.05
TD8165	394100	2.30	1.85
TD8164	394100	3.40	0.95
and		3.50	1.54
TD8184	394100	2.00	2.12
and		2.00	1.24
TD8203	394135	1.00	1.45
TD8182	394135	5.00	1.11
TD8183	394138	5.00	1.34
TD8169	394170	4.60	2.03
TD8156	394170	4.55	1.45
and		2.70	3.06
TD8181	394170	4.00	1.37
TD8168	394170	1.40	0.87
TD8140	394200	7.00	1.72
TD8141	394250	6.98	1.79
TD8186	394250	8.40	1.84
TD8190	394285	3.85	1.41
TD8170	394300	3.15	1.62
TD8139	394300	5.45	1.42
TD8157	394300	1.00	0.83
TD8179	394350	2.45	1.87
and		1.45	0.98
TD8178	394350	0.20	1.75
TD8177	394350	3.80	1.68
TD8171	394350	1.50	1.22

Drill hole	Section (mE)	Interval (m)	Ni %
TD8172	394410	2.20	1.57
TD8191	394400	3.75	1.42
TD8187	394410	9.50	1.77
incl		5.50	2.37
TD8158	394450	6.30	2.08
TD8188	394450	5.00	1.05
and		1.05	2.91
TD8209	394450	1.25	1.30
and		4.10	1.09
TD8194	394500	5.90	2.21
TD8175	394500	13.25	2.01
TD8193	394500	13.00	2.16
and		7.50	1.32
TD8192	394500	1.00	0.88
TD8161	394550	16.50	2.10
TD8189A	394550	18.70	2.17
TD8201	394550	1.40	1.06
TD8200	394600	2.00	1.58
and		1.15	1.63
TD8199A	394600	7.80	1.56
TD8197A	394650	4.60	1.24
TD8174	394650	4.20	1.50
and		5.10	2.01
TD8195A	394650	9.00	1.16
TD8205A	394700	3.80	1.40

Figure 1: Plan Cruickshank Resource - showing mineralised infill resource drill holes





Background

Cruickshank is a large low to moderate grade disseminated/matrix style nickel sulphide deposit located on the eastern flank of the Tramways Dome, approximately 6.5kms north-east of the Lanfranchi mine. The deposit dips eastward at 35 degrees and plunges towards the south-east at approximately 25 degrees. It outcrops at surface and has been delineated to a depth of 500 metres below surface, where-upon it remains open. Previous resource estimates indicated the deposit contained between 12,700t Ni contained (WMC, 2003) to 26,600t Ni contained (Panoramic, 2007).

In June and July 2010, Panoramic drilled four HQ diamond core holes through the deposit to provide samples for a metallurgical test program. The results of the metallurgical tests were very positive with nickel recoveries of approximately 90% to a 9-10% Ni concentrate grade achieved on all four samples using the BHP Nickel West Kambalda Concentrator standard operational grind and reagent regime.

In November 2010, following on from the metallurgical testwork, Panoramic commenced an infill drill program at Cruickshank. The aim of the program was to infill the Cruickshank deposit in order to produce a JORC compliant Indicated Resource estimate for the majority of the deposit. The infill drill program comprising 54 drill holes for a total of 13,508 drill metres was completed in April 2011.

About the Company

Panoramic Resources Limited (ASX Code **PAN**, ABN 47 095 792 288) is an established Western Australian mining company operating two 100% owned underground nickel sulphide mines, the Savannah Project in the Kimberley, and the Lanfranchi Project 42km south of Kambalda. On a Group basis, Panoramic produced 17,458t nickel contained in FY2010 and is forecasting to produce around similar levels in FY2011. In February 2011, the Company acquired the Gidgee Gold Project, located 640kms north-east of Perth. Exploration and evaluation studies have commenced at Gidgee, with the aim of expanding the known 310,000oz gold resource. The Panoramic Group has strong cash reserves, no bank debt and is continually looking to grow its existing business through internal exploration success, outside acquisitions and/or joint ventures.

For further information contact:
Peter Harold
Managing Director
+61 8 9225 0999

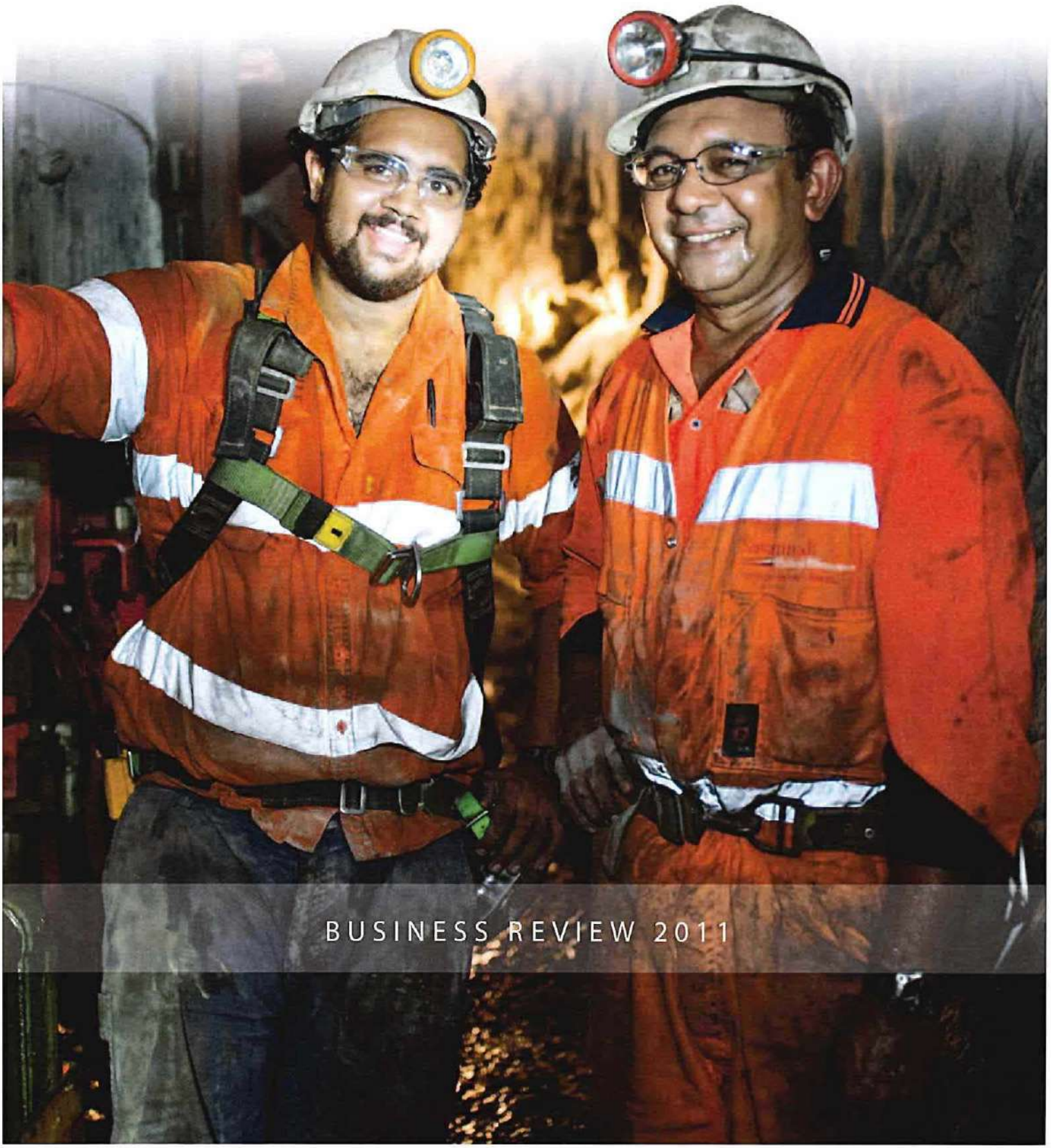
The information in this release that relates to Exploration Results is based on information reviewed by John Hicks. Mr Hicks is a member of the Australasian Institute of Mining and Metallurgy (AusIMM) and is a full-time employee of Panoramic Resources Limited. Mr Hicks has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which each person is undertaking to qualify as a Competent Person as defined in the 2004 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Mr Hicks consents to the inclusion in the release of the matters based on the information in the form and context in which it appears.

Information in this report relating to Mineral Resources has been completed by Mr Andrew Bewsher of BM Geological Services Pty Ltd. Mr Bewsher is a member of The Australasian Institute of Mining and Metallurgy (AusIMM) and has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2004 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Mr Bewsher consents to the inclusion in the release of the matters based on his information in the form and context in which it appears.

PJD-48

PANORAMIC
RESOURCES LTD

VISION | COMMITMENT | RESULTS



BUSINESS REVIEW 2011



SAVANNAH TEAM



LANFRANCHI TEAM

ABOUT US

Panoramic Resources Limited is a well established Perth based mining company. It owns and operates the Savannah (East Kimberley) and Lanfranchi (Kambalda) underground nickel mines and has recently acquired the Gidgee Gold Project (Murchison). **Panoramic** is an S&P/ASX Top 200 company with a growing nickel and gold resource base and a workforce of more than 500 people.

Panoramic is also engaged in exploration activities in Australia and Scandinavia, primarily focusing on nickel, copper, PGM's and gold, and in Canada and South America via equity investments in junior explorers. The Company is actively looking to add to this exploration portfolio with new projects.

The growth path is being led by an experienced exploration-to-production team that has already enabled **Panoramic** to develop a substantial and profitable revenue stream, providing it with the capital base to achieve its vision through advanced exploration programs, acquisitions and/or joint ventures.

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PANORAMIC
RESOURCES LTD

HIGHLIGHTS

- Net profit of $22.3 million
- Total dividends of 6 cents per share (fully franked)
- Cash and receivables of $126.4 million
- Group Production of 17,027t Ni
- Acquisition of the Gidgee Gold Project
- 26% increase of the Cruickshank Resource to 33,560t Ni
- Initial Helmut South Extension Resource of 7,080t Ni
- New nickel mineralisation down plunge of Deacon
- Three new JV's in Norway with Drake Resources

THREE PILLARS OF GROWTH

- Organic production growth at existing mines
- Exploration success (near mine, regional and entry into new mineralised provinces)
- Disciplined acquisitions or mergers with a focus on operating mines, development projects and prospective projects (adding copper, gold, PGM's and other essential industrial minerals to our portfolio)



OUR ACHIEVEMENTS

- Successfully grown nickel production from 6ktpa (2005) to 17ktpa (2011)
- Successfully grown nickel Resources from 63kt Ni to ~220kt Ni and produced ~93kt Ni
- Reserves have grown significantly from nil to ~112kt Ni not including 93kt Ni depletion from production
- We have controlled costs (C1 Cash Cost ~ US$5-6/lb)
- Both of our operating mines are in good shape in terms of mineral inventory and upside

COMMITTED TO GROWTH

Nickel	Targeting +25,000tpa Ni for 10+ years
Gold	Targeting 100,000ozpa within two to three years
Resources	Focused on exploration to grow our existing resource base
Acquisitions	Committed to making strategic acquisitions that increase shareholder value

10 YEAR PLAN

- Improve our safety culture
- Optimise metal production
- Grow the existing resource and reserve base
- Maintain dividend payments
- Acquire additional assets

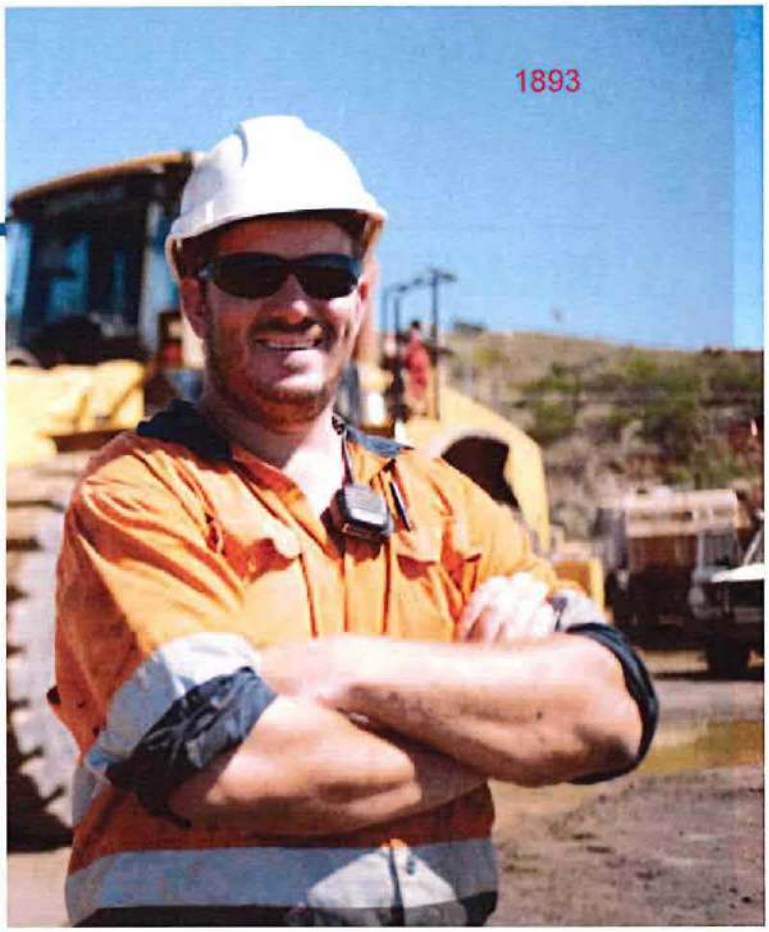


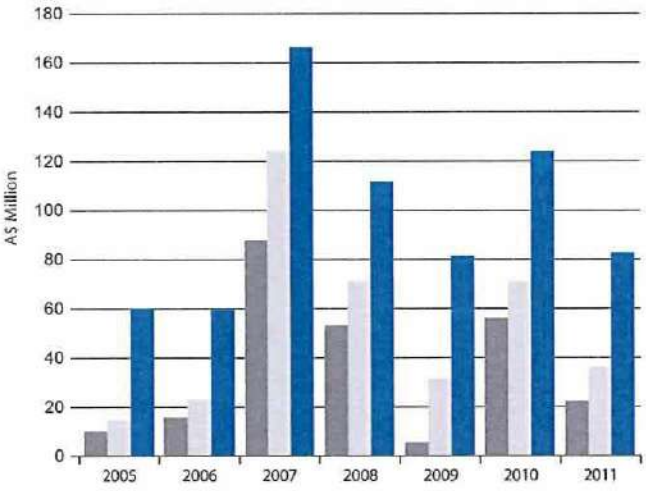
THROUGH ORGANIC GROWTH, EXPLORATION SUCCESS AND SELECTIVE ACQUISITIONS WE WILL ACHIEVE OUR OBJECTIVE TO BECOME A SIGNIFICANT DIVERSIFIED MINING COMPANY.

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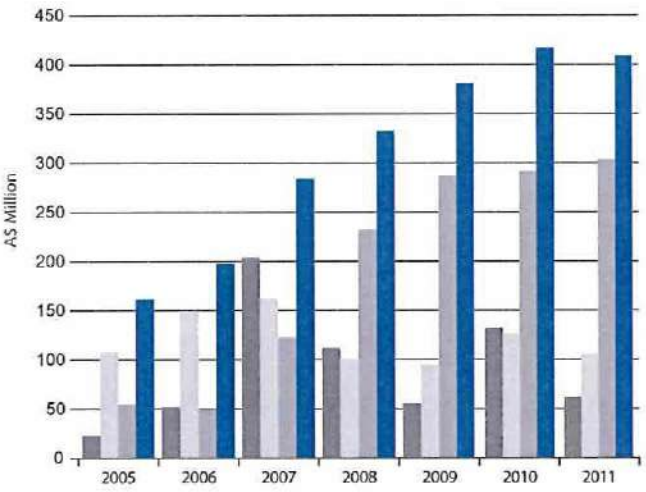
FINANCIAL HIGHLIGHTS

- Seventh straight year of profits
- Revenue of $249.6 million
- Cash and receivables of $126.4 million
- Net profit after tax of $22.3 million
- Total dividends of 6.0 cents per share (fully franked)
- A 56% pay-out ratio of after tax earnings
- Net assets of $303.1 million

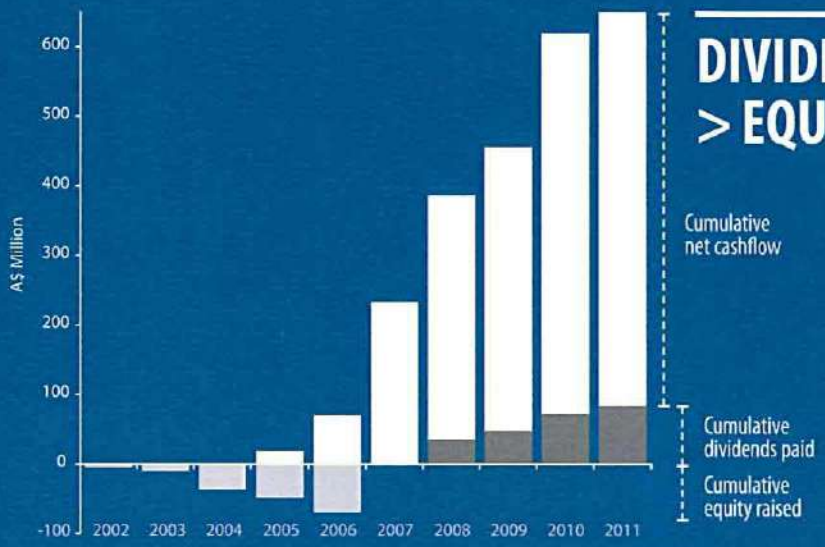




SEVEN STRAIGHT YEARS OF PROFIT 2005-2011



SHAREHOLDER EQUITY & TOTAL ASSETS CONTINUE TO GROW



DIVIDENDS PAID > EQUITY RAISED

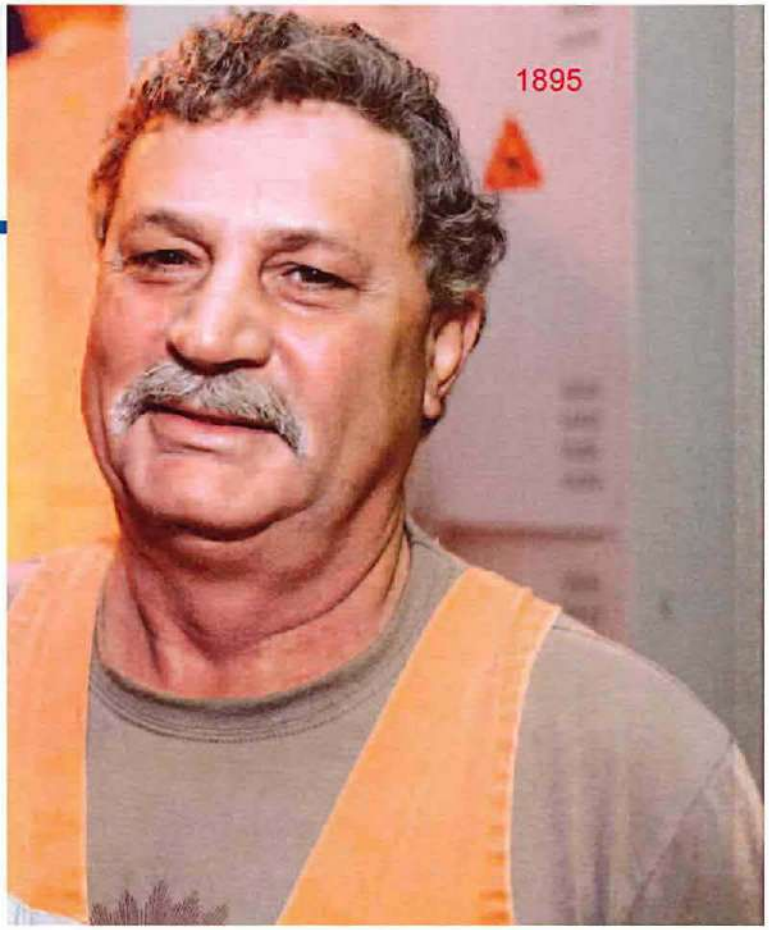
SUMMARY OF KEY FINANCIALS 2009-2011

Financial Years	2009	2010	2011
US$ Nickel Price (US$/lb) - avg	$6.06	$8.78	$10.89
US$:A$ FX Rate (US$) - avg	$0.7476	$0.8818	$0.9891
A$ Nickel Price (A$/lb) - avg	$8.11	$9.96	$11.01
Group Ni in Conc/Ore (dmt) - equity basis	17928	17458	17027
Group A$ Payable Cash Cost (inc. Royalties) (A$/lb) - avg	$4.58	$5.70	$6.25
Hedging gain/(loss) - realised (A$M)	44.3	40.0	-13.8
Total Net Revenue (Inc. Interest) (A$M)	228.7	287.8	249.6
EBITDA (before impairment adjs.) (A$M)	**81.7**	**124.1**	**82.8**
D&A (A$M)	49.5	52.7	46.1
Profit before Tax & Impairment (A$M)	31.7	71.1	36.3
Profit afterTax & before impairment (A$M)	24.0	51.2	26.2
Net profit after tax (A$M)	**5.6**	**56.2**	**22.3**
Income taxes paid (A$M)	-24.9	4.0	-20.8
Cash flow from operating activities before tax (A$M)	54.8	131.8	61.5
Net Cash Flows used in Investing Activities (exc. E&E) (A$M)	56.8	38.0	61.0
Exploration & evaluation expenditure (A$M)	3.1	8.8	10.4
Cash and current receivables (A$M)	95.9	158.3	126.4
Total assets (A$M)	**380.3**	**416.8**	**408.7**
Borrowings (A$M)	6.5	4.7	1.4
Total liabilities (A$M)	94.0	125.7	105.6
Shareholders equity (A$M)	286.3	291.1	303.1
Return on equity (%)	6%	55%	22%
Earnings per share (cents)	2.9	27.5	10.8
Dividend declared per share (cents)	3.0	16.5	6.0
Dividends Paid (A$M)	11.5	24.6	12.4
Dividend pay-out ratio (%)	103%	60%	56%

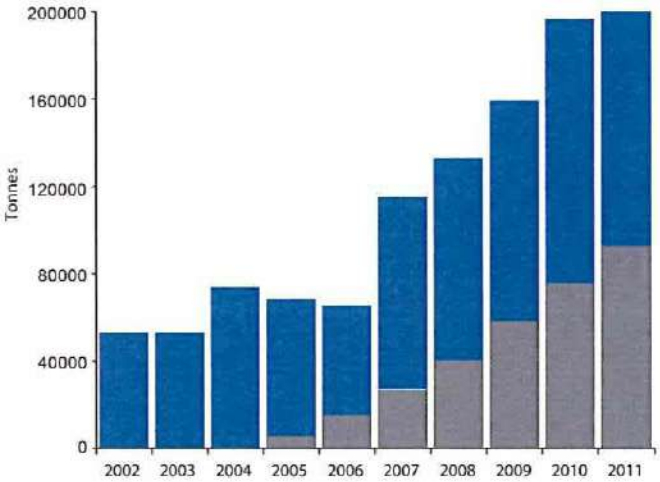
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PRODUCTION & SAFETY HIGHLIGHTS

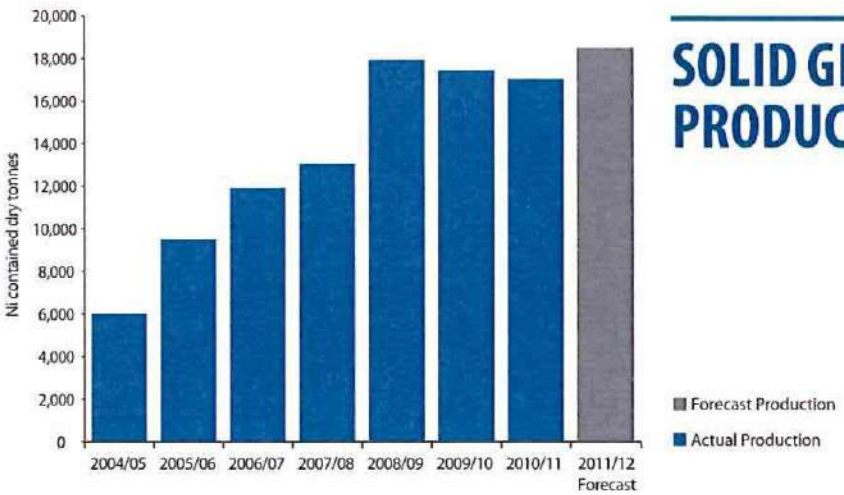
- Group metal production of 17,027t Ni, 4,514t Cu and 379t Co
- 42% reduction in LTIFR from 2007



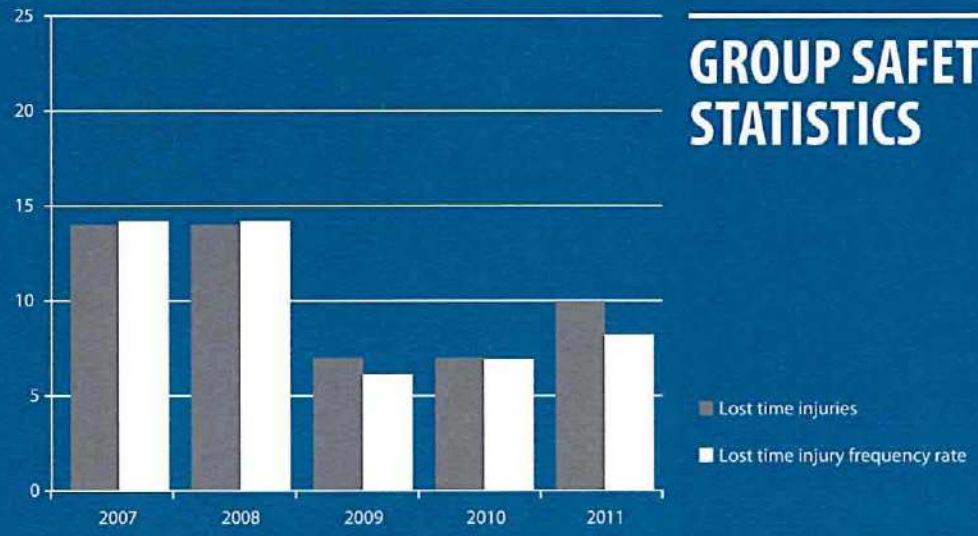
STRONG PRODUCTION & RESERVE GROWTH



SOLID GROUP NICKEL PRODUCTION

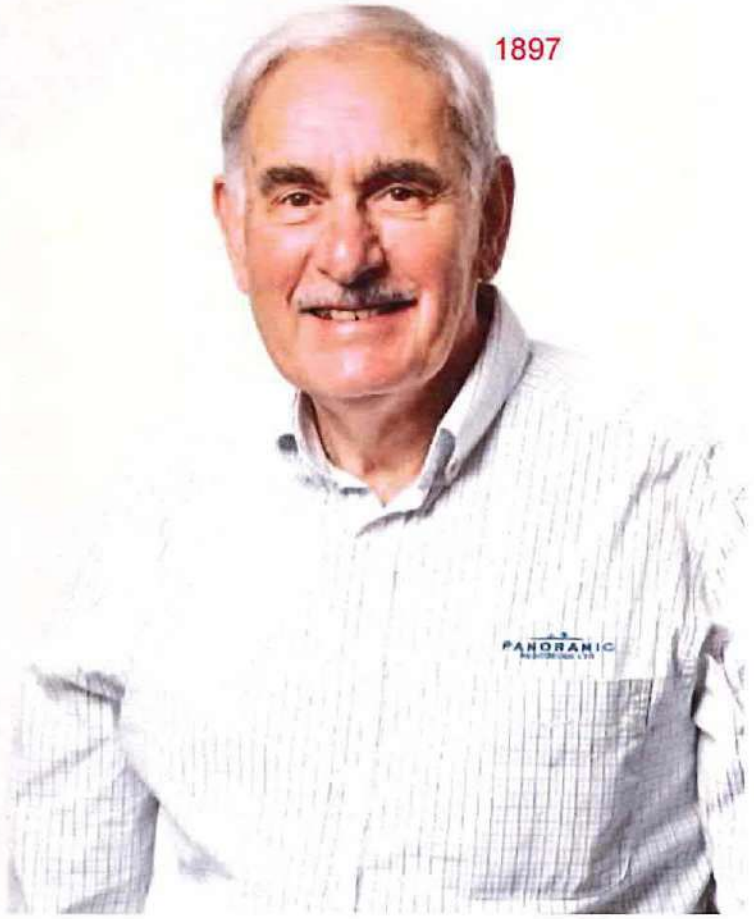


GROUP SAFETY STATISTICS





PANORAMIC
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CHAIRMAN'S REPORT

Dear Shareholders,

The past year has been challenging for some companies involved in exporting products from Australia. Whilst the nickel price traded in a US$8-13/lb range over the year the Australian dollar continued to strengthen against the US dollar, adversely impacting our Australian dollar earnings. The impact of the strong Australian dollar can be clearly seen in the nickel price graph below, which shows nickel prices in US dollar and Australian dollar terms over the past three years.

LME CASH NICKEL PRICE 1 JULY 2008 TO 31 AUGUST 2011



Group contained nickel production for the year was just over 17,000 tonnes which was about 8% below budget primarily due to mobile equipment availability issues at both our mines. Savannah also suffered a series of vent fan failures which limited access to various parts of the mine and required some re-scheduling of stope sequencing to maintain production levels, albeit at lower head grades. The ventilation issues have been resolved, our equipment availability has improved and we are on track for production of 17,500-18,500 tonnes contained nickel this year. The exploration efforts at Lanfranchi continue to produce positive results and during the year we added significant additional ore tonnes from the Helmut South Extension and at Cruickshank. We have also commenced a new program of exploration on some strong EM targets very close to the Savannah Orebody.

Our after tax profit of $22.3 million was a reasonable result, however it was impacted heavily by a combination of lower production, higher Australian

dollar, a hedging loss and a write-down of our investment in Magma Metals Limited (Magma). After generating over $80 million in hedging gains in the previous two financial years, our hedging program reported a loss of $13.8 million for the year. This was largely caused by the strong and unexpected rally in the nickel price over the second half of the year, which meant that while we benefited from the higher prices on our unhedged production, our production was down on budget and consequently we had less unhedged tonnes to price at the higher levels. This hedging was put in place some years earlier to protect our revenues and enabled us to get through the *GFC* without serious difficulties. To ameliorate the risks of the adverse effect from our hedging activities in the future, we have increased our use of nickel and currency options. Options do incur some upfront costs but provide protection against major reductions in revenue from adverse nickel price and exchange rate movements, while giving us full exposure to any price rallies and currency weakness.

China continues to dominate demand for nickel. They are increasingly looking to Nickel Pig Iron (NPI) to provide their stainless mills with low cost nickel feed. The latest estimates suggest China accounts for close to 50% of world nickel demand with NPI providing over 400,000 tonnes of nickel per annum. China continues to access supplies of nickel ore from Indonesia and The Philippines. Indonesia has recently announced restrictions on the export of ore to apply from 2014. As a result, there are moves by Chinese groups to install NPI processing plants in Indonesia.

Chinese stainless steel producers are now exporting finished product to Europe and the United States. Whilst there can be quality issues with this material, the pricing is attractive as a result of lower cost NPI feedstock. Perhaps the most encouraging aspect of the Chinese market is the reported move away from low nickel grade 200 series stainless steel back to the higher nickel grade 300 series. In part, this change has been caused by the move to have improved standards applied to building codes in China. It is interesting to note that a lift of 1% in the nickel tenor of stainless steel from the current level of around 7% to the more traditional level of 8% would increase nickel demand by approximately 300,000 tonnes per annum worldwide. This would be sufficient to put nickel supply into deficit again despite the planned supply increases from new laterite projects.

Jinchuan continues to take all of the concentrate we produce at Savannah and our relationship with them remains on the friendliest of terms. When Jinchuan had its 50th anniversary celebrations last year, we presented them with a Boab tree from the Kimberley as a token of our relationship. It is now a major attraction in that part of China and is living happily in its new abode in the wonderful, purpose built, indoor garden established by Jinchuan in Jinchang City, Gansu Province.

The Company continues to seek opportunities to acquire new resource projects both here and overseas. During the year we acquired the Gidgee Gold Project which has a known resource base over 300,000 ounces and has historical production of over 1 million ounces. We are now working hard to increase the resource base and have an aggressive target of at least 500,000 ounces of reserves to enable us to re-commission the existing mill and return Gidgee to a 100,000oz per annum gold producer within the next two to three years. The Company places great store in diversifying its revenue base by moving into gold production.

We are also looking to increase our exposure to promising mineral assets overseas. We have recently invested into an advanced copper exploration project in Chile, via our 4% shareholding in Hot Chili Limited and hope to grow our involvement in that company. We also have maintained our equity position in Canadian PGM explorer, Magma, at 9% via participation in a placement during the year, and have increased our exposure to base metals exploration in Scandinavia through our exploration JVs with Drake Resources Limited. We have done considerable due diligence on these projects and believe they offer unique opportunities for the Company.

The business environment in this country continues to be challenging. We believe that our job is to develop the resources we discover in an ethical manner for the benefit of shareholders, employees, and the country in general. It seems that the government sometimes loses sight of these objectives and seeks to put hurdles in the way of our success by imposing more and more bureaucratic and taxation imposts on the industry. Further regulation and policy uncertainty continues to impose additional cost on our operations, impacts on how we reward our people, who we employ and how we operate. These imposts will continue to erode our margins and in order that there be more debate and balanced discussion before policies are implemented, we continue to support the Minerals Council of Australia, the Association of Mining and Exploration Companies, the Chamber of Minerals and Energy of WA, and other organisations that support and promote our industry.

This is the last year that I will be reporting to you as Chairman. I have had the honour of serving you and working with a most co-operative and like-minded board and an extremely competent management team under Peter. It has been an exciting six years. When I joined the Company the share price was less than $1, since then it has been as high as $6.21 and as low as 70 cents post the *GFC*. This year we have seen unbelievable gyrations in the equity markets as the world markets went through convulsions because of the debt problems in Europe and the USA. While all of this happened, Peter and the team have worked hard to increase the resource base, maintain production rates, hold costs in check and manage the mines in a safe and efficient manner. Peter is probably the best team builder and motivator with whom I have ever had the pleasure of working with. He has built a team of willing and co-operative management who are creative and hard-working and able to extract the best out of our assets. The Company is well placed with over $100 million in cash and no debt to continue to grow.

I wish the Company well in its future endeavours and absolute success in both its exploration and acquisition ventures.



Christopher J G de Guingand
Chairman
20 September 2011

PANORAMIC
RESOURCES LTD

MANAGING DIRECTOR'S REPORT



Dear Shareholder,

I am pleased to report we had our 7th straight profitable year at Panoramic in our 10th year since incorporation.

Key milestones achieved during the year were as follows:

- Focus on safety maintained;
- Acquisition of the Gidgee Gold Project;
- Discovery of the Helmut South Extension Orebody containing 7,080t Ni;
- Total nickel contained production of 17,027 tonnes;
- Increase of the Cruickshank Resource to 33,560t Ni;
- Net cash flow of $61.5 million from operations before tax;
- Net profit after tax and write-downs of $22.3 million;
- Total fully franked dividends of 6 cents per share, a 56% pay-out ratio over the year; and
- Strong cash and current receivables of $126.4 million and no debt.

The number one value in our business is **Safety** and we continue to work hard on improving our safety culture and performance. Unfortunately we continue to have the occasional Lost Time Injury (LTI) at our operations, which is concerning and unacceptable. Our goal is to eliminate all LTIs. Our Group LTIFR was at 8.21 at the end of June. On a positive note we have seen a reduction in the number of "all injuries" reported and a reduction in the number of "all reported significant incidents". Importantly, the percentage of significant reported incidents compared to all other incidents was at, or below the Group target of 15% at year end.

We remain focused on all facets of safety in our business to ensure we improve our performance in accordance with our safety mission statement:

- **Vision** **safety is a value not just a priority**
- **Commitment** **safety improvement through leadership**
- **Results** **safely home every day**

On the production front, both operations missed their internal budgets, which was disappointing. Savannah mined and milled 595,944 tonnes of ore at an average grade of 1.35% nickel and produced 6,921 tonnes of nickel contained. Lanfranchi mined 412,403 tonnes of ore at an average grade of 2.45% nickel for 10,106 tonnes of nickel contained. Both operations suffered from poor equipment availability, however on a positive note the head grade at Savannah was up 8% year on year and Lanfranchi mined 3% more ore tonnes. Group production was still a respectable 17,027 tonnes of nickel for the year.

A significant highlight was the acquisition of the Gidgee Gold Project for $15.5 million. We have been looking for a quality gold asset for some time and believe we have found one in Gidgee. The project has produced over one million oz gold, mostly between 1987 and 2005, when the gold price ranged between US$250-450/oz and has known resources of 310,000oz

calculated at a US$500/oz gold price. The project has a 600,000tpa mill and associate infrastructure (150 person camp, airstrip, roads, tailings dam) and it is possible that the mill could be re-furbished for approximately $20 million. We have set a target of 500,000oz gold in reserve which would enable us to re-commence operations and believe this could be achieved within two to three years, subject to statutory approvals, a favorable gold price and identifying the reserves.

2011/12 promises to be another good year for Panoramic. On the production side we are forecasting to produce 17,500-18,500 tonnes nickel, significantly advance Gidgee and aggressively explore our own tenements and our JV ground. We have budgeted to spend a minimum of $12.5 million on exploration focused on the following projects:

- Undertaking significant AC and RC drill programs at Gidgee;
- Drill testing strong EM conductors around the Savannah Orebody;
- Drilling the strong EM response below the 900m level at Savannah;
- Continuing to drill test EM targets on the East Kimberley JV tenements;
- Down plunge drilling of the Deacon, Helmut South, Schmitz, Cruickshank and Lanfranchi orebodies; and
- Continuing to support Drake Resources in our strategic alliance for base and precious metals in Scandinavia.

On the acquisition front we remain well placed to make acquisitions with over $100 million in cash on the balance sheet. We will continue to assess opportunities to grow the business through quality acquisitions of projects and/or companies that would be complementary to the existing business, and to utilise management's experience and expertise. The primary focus is nickel, copper, PGMs and gold, and the preference is to buy assets at pre-feasibility stage through to operating mines. The Company is also interested in opportunities in other base metals and selected bulk commodities. While Australia remains the preferred location, the Company is actively pursuing opportunities in selected overseas countries which have mining friendly regulatory regimes and established infrastructure. We have taken a 4% equity interest in Hot Chili Limited which has an exciting copper project in Chile and hope to become more involved with that company.

The Board remains cautious about the short term outlook for the world economy and therefore commodity prices, however we expect a slow improvement in future business conditions. The Company will continue to actively manage its exposure to commodity prices, diesel and the US$/A$ exchange rate through its hedging program and will take opportunities when they present to protect margins and optimise revenue. The Board and management are committed to growing the Company in accordance with our 10 Year Plan which is to:

- **Improve our safety culture** so every employee believes that safety is our most important value in line with our safety mantra: **Vision, Commitment, Results;**
- **Optimise our metal production** to maximise our margins;
- **Grow the existing resource and reserve base** to extend the mine life of our operations;
- **Maintain dividend payments**; and
- **Acquire additional assets** to become a diversified mining house and an S&P/ASX Top 100 Company.

The success of our Company is mainly due to our dedicated workforce and I would like to personally thank the Board, all employees and contractors for their hard work and dedication again this year. I would also like to thank all our shareholders, other stakeholders and our two customers, the Jinchuan Group and BHP Billiton Nickel West for their ongoing support.

Our Chairman, Chris de Guingand, will be retiring in November after six years of distinguished service and I would like to personally thank Chris for his dedication. He has been a significant contributor to the success of Panoramic and we are all indebted to Chris. His enthusiasm, industry knowledge and contacts, approach to problem solving, leadership, mentoring skills and sense of humour are great qualities and he will be missed both in the Panoramic board room and at our operations. Chris will remain available as a consultant and we will continue to call on him as and when required.

As always, I urge all our staff and contractors to adopt and embrace our safety mission statement to ensure we get everybody "**home safely every day**".

Yours faithfully,



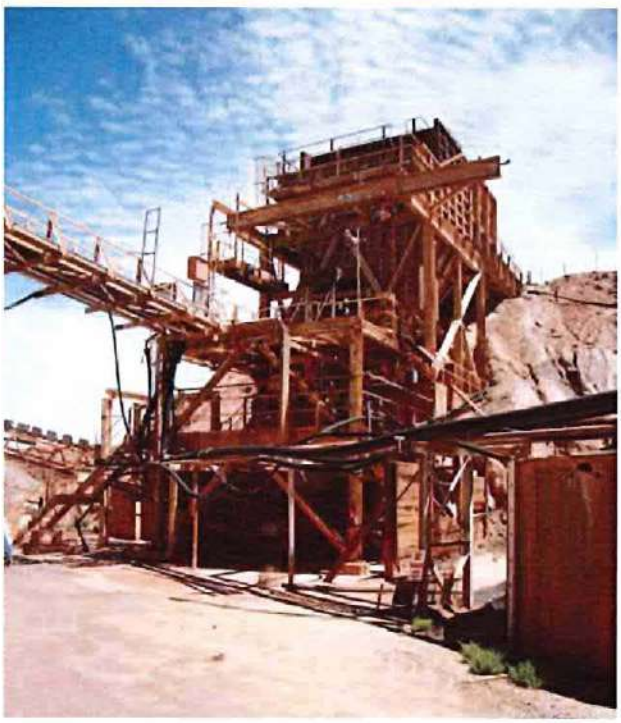
PETER HAROLD
Managing Director
20 September 2011

GIDGEE GOLD PROJECT

The Gidgee Gold Project is located 640 kilometres north-east of Perth in the Murchison region of Western Australia, and includes a 600,000tpa process facility (not in operation), a 150 person camp and airstrip.

The Gidgee Gold Project was mined almost continuously from 1987 to 2005 when it was placed on care and maintenance. Approximately twenty open pits were mined on near surface gold mineralisation and underground mining was conducted beneath the Swan Bitter and Kingfisher pits.

Product:	Gold
Mining:	Open pit & underground (historical)
Production:	Targeting 100,000oz per year
Resources:	310,000oz
Workforce:	Four employees



2011 HIGHLIGHTS

- The Project was acquired in February 2011 for $15.5 million as an advanced exploration project with **310,000oz of gold in resources**
- Refurbishment of the camp and associated infrastructure commenced in readiness for onsite exploration and evaluation work
- Exploration and production strategy formalised and announced, targeting **100,000oz per annum**
- Key personnel recruited
- Multiple drill targets identified from initial review of geological interpretations and historic drill results
- Initial air-core drilling program commenced

GIDGEE HISTORY

1926	gold discovered
1970/80's	base metal exploration
1983-1999	Arimco/Australian Resources consolidated the land holding and explored for gold over 15 years
1987	numerous oxide gold resources were discovered and exploited through a 300ktpa CIP plant
1999	Abelle acquired the project, upgraded the plant to 600ktpa and commenced underground mining at Swan Bitter combined with open pit mining of numerous orebodies (Kingfisher, Wilsons, etc.)
2003	Legend acquired Gidgee and continued mining until 2005
2007	Apex acquired Gidgee and focused on the Wilsons Refractory Resource
2011	Panoramic acquired Gidgee (excluding Wilsons)

CAPACITY TO FAST TRACK PRODUCTION

- Fully funded
- Excellent on-site infrastructure including mill, camp and airstrip
- Majority of resources on granted mining leases
- Infrastructure re-commissioning studies commenced



INVESTMENT PROPOSITION

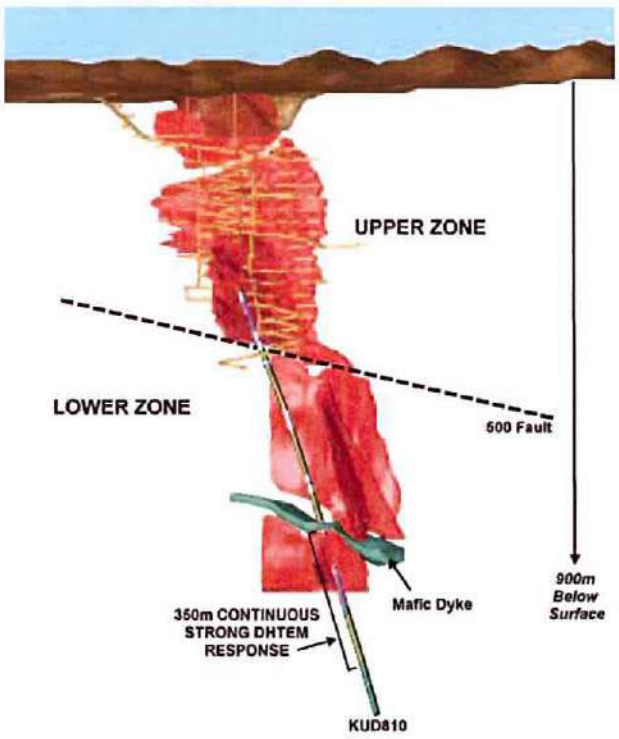
- The Right Location – large Western Australian greenstone belt (~1,200km^2) with historic production > 1 Million oz Au
- Under Explored – minimal exploration over the past 10 years
- Excellent Gold Price Environment - mining ceased when gold was < US$500/oz
- Prospective Mine Corridor – limited exploration around historical oxide pits and shallow underground operations, minimal drilling below 150 metres
- Quality Resource Targets – shallow, high grade resources with numerous high grade, gold intercepts open at depth and along strike
- Significantly Undervalued – in our view, existing resources are significantly undervalued compared with peers, despite attractive exploration potential
- Capacity – fully funded to expedite exploration, mine studies and mill refurbishment supporting a production target of 100,000oz pa



SAVANNAH PROJECT

The Savannah Project is located 240 kilometres south of Kununurra in the East Kimberley region of Western Australia, and consists of a nickel sulphide orebody, underground mine, process plant and associated infrastructure.

Product:	nickel, copper, cobalt concentrate
Mining:	open stoping with paste fill
Processing:	1.0Mtpa Crush, SAG, Float
Production:	7,500t Ni, 4,500t Cu, 400t Co per year
Life:	Ore Reserves to 2018
Offtake:	Contracted to Jinchuan until 2020
Workforce:	~300 including employees and contractors
Copernicus:	60% interest in the satellite Copernicus Project



SAVANNAH RESOURCE SUMMARY

	Nickel	Copper	Cobalt
Upper Zone	26,100t	14,000t	1,450t
Lower Zone	55,000t	28,000t	2,850t
Lower Zone Extension	Drilling in 2011		
Copernicus (60%)	6,000t	4,000t	200t

2011 HIGHLIGHTS

- 595,944 tonnes of ore milled at 1.35% Ni for a total of 6,921t Ni, 3,689t Cu and 379t Co in concentrate
- Ni recovery averaged 86%, well above the original feasibility study recovery of 78%
- Approval to construct a 3 metre tailings dam lift received and construction completed
- Approval to construct a new concentrate storage shed in Wyndham

FUTURE OUTLOOK

The Savannah Optimisation Program

- Increasing underground production rates
- Optimising cut-off grades and mining methods
- Drilling out the Savannah Lower Zone Extension Orebody below the 900m fault and commencing mine feasibility once mining inventory is in place
- Undertake conceptual studies on optimal haulage methods below the 900m fault
- The Savannah mill has spare capacity and could accommodate up to 40% more ore without modification


1904

LANFRANCHI PROJECT

The Lanfranchi Operations and associated Tramways Tenements (the Lanfranchi Project) are located 42 kilometres south of Kambalda, Western Australia.

Product:	nickel in ore
Mining:	open stoping with paste fill
Processing:	3rd party feed to Kambalda Concentrator
Production:	10-12,000t Ni & 1,000t Cu per year
Life:	Ore Reserves to 2016
Offtake:	Contracted to BHP Billiton until 2019
Workforce:	~300 including employees and contractors



LANFRANCHI RESOURCE SUMMARY

	Nickel
Deacon	47,800t
Helmut South & Helmut South Extension	9,300t
Cruickshank	33,600t
Gigantus	10,600t
McComish	14,800t

2011 HIGHLIGHTS

- 412,403 tonnes of ore mined containing 10,106t Ni and 825t Cu
- Approval and commencement of the construction of the Lanfranchi accommodation village
- A 26% increase of the Cruickshank Resource to 2.63Mt at 1.28% Ni for 33,560t Ni contained
- Discovery of a continuous zone of mineralisation extending over 300 metres down plunge of Helmut South resulting in the Helmut South Extension Resource estimate of 181,700 tonnes at 3.90% Ni for 7,080t Ni contained
- Underground diamond drilling in the Deacon/Schmitz areas results in several holes intersecting a new high grade of mineralisation to the east of the Helmut South Extension Resource

FUTURE OUTLOOK

The Lanfranchi Optimisation Program

- Following the discovery of high-grade nickel mineralisation down plunge of Helmut South, work has commenced to optimise production from all three producing channels (Helmut/Deacon, Schmitz and Lanfranchi)
- Feasibility work has commenced on the Cruickshank Orebody and further drilling down plunge is planned
- We will also focus on the lower grade orebodies such as Gigantus and McComish


1906


LANFRANCHI CHANNEL
SCHMITZ CHANNEL
HELMUT CHANNEL
WEST EDWIN
EDWIN
HAM
LANFRANCHI
WINNER
SKINNER
SCHMITZ
DEACON
HELMUT
HELMUT SOUTH
JOHN
McCOMISH
LANFRANCHI
1km

EXPLORATION

OVERVIEW

Panoramic has committed to spend at least $12.5 million on exploration in 2011/12 to advance new and ongoing programs at the Savannah and Lanfranchi Projects and to ramp up exploration activities at the recently acquired Gidgee Gold Project. The Company is also focused on actively developing diversified growth investments through exploration joint ventures with other companies in Australia and around the world.

GOLD

Gidgee Gold Project, Murchison, WA

The Company's initial strategy for Gidgee is to build up a resource and reserve profile and develop a mine plan which will support the re-commissioning of the existing Gidgee processing plant. The Company has set a target of establishing a Reserve of 500,000oz and to commence production at a rate of 100,000oz per annum within two years.

To achieve this goal, two concurrent exploration strategies have been implemented, the first focusing on Resource Definition Drill Testing and the second on Regional Greenfields Targeting. The Resource Definition Drill Testing Program includes a review of all historical mining areas, re-optimising existing high grade resources (see Figure 1) at a higher gold price and an initial 15,000 metre RC and diamond drill program planned to commence in the December 2011 quarter.

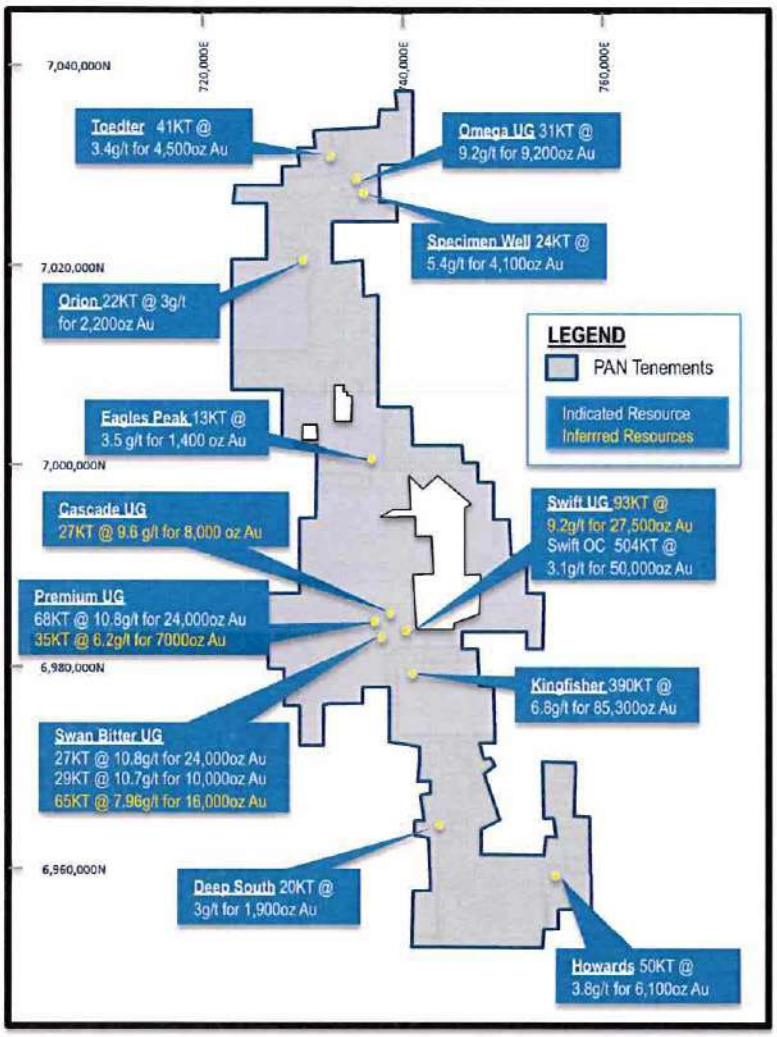


Figure 1- Gidgee Gold Project - Resources

The Regional Greenfields Targeting Program started in July 2011 with a 20,000 metre drill program to collect data to advance alteration analysis. Multi-element lithogeochemistry and hyperspectral data collected from this program will help identify and characterise gold mineralising alteration systems around the known deposits. This knowledge will then be applied to the discovery of new deposits elsewhere in the belt. This program is targeting a multi-million ounce discovery.

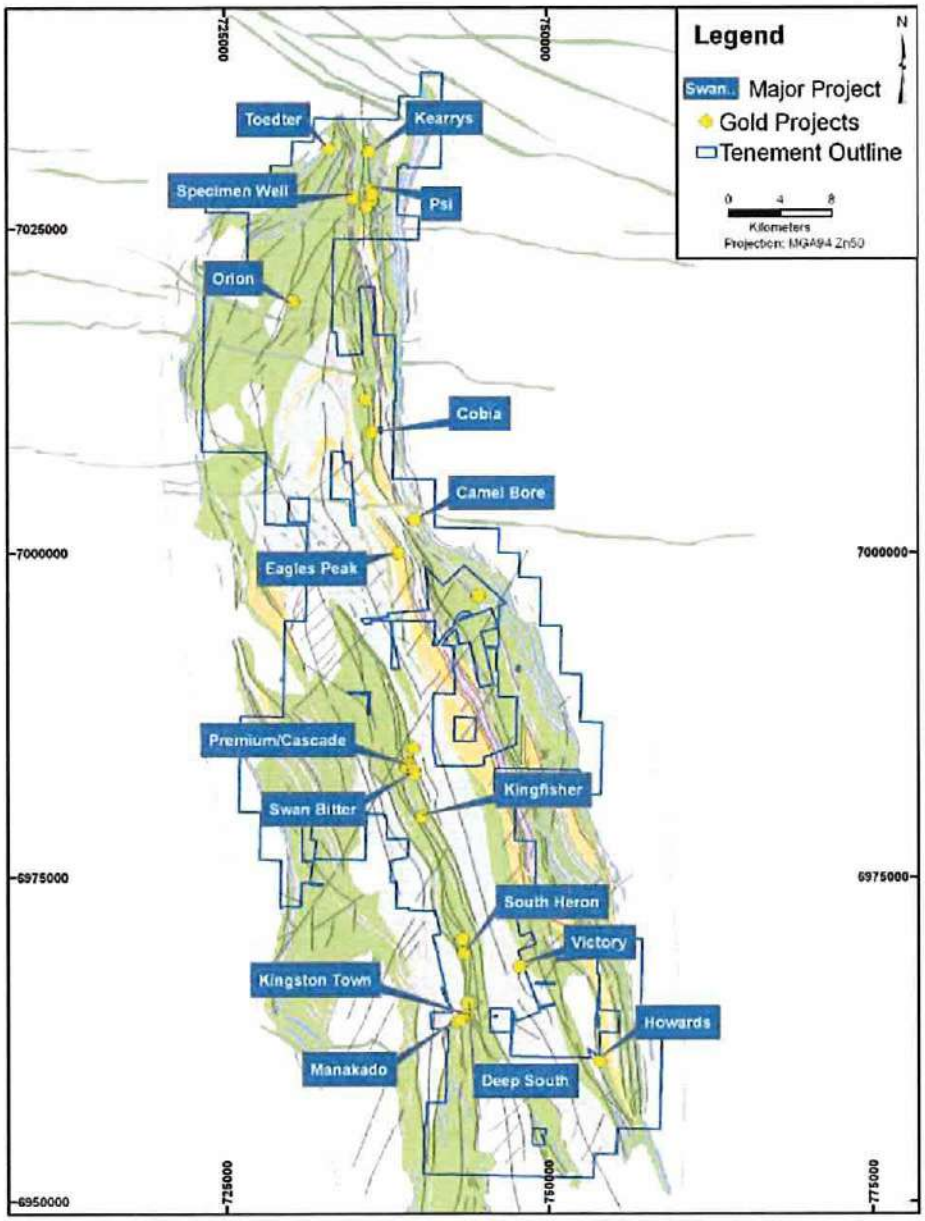


Figure 2 – Gidgee Gold Project prospect areas

A key opportunity at Gidgee is that both the historical and regional mining areas are under drilled. Less than 3% of all holes are drilled to a depth over 150 metres. There are a total of 210 exploration targets across the project, and as shown in Figures 2 and 3, there are multiple high grade targets around historical pits and underground operations.

The Swan Bitter and Swift prospective targets demonstrate significant potential as high grade extensions to existing resources. These targets are shown in detail in Figures 4 and 5.

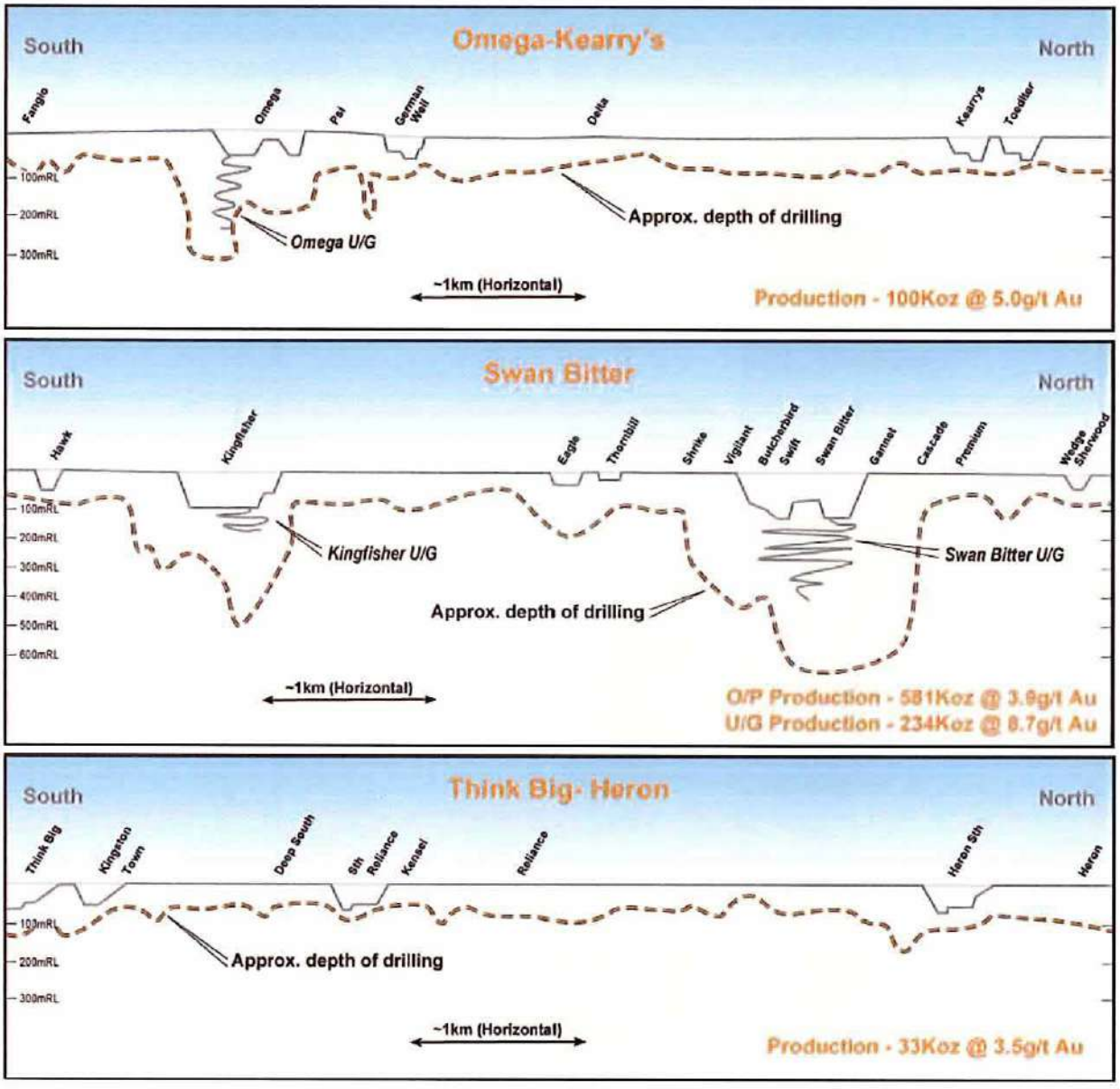


Figure 3- Gidgee Historical Mining Areas - Target Areas

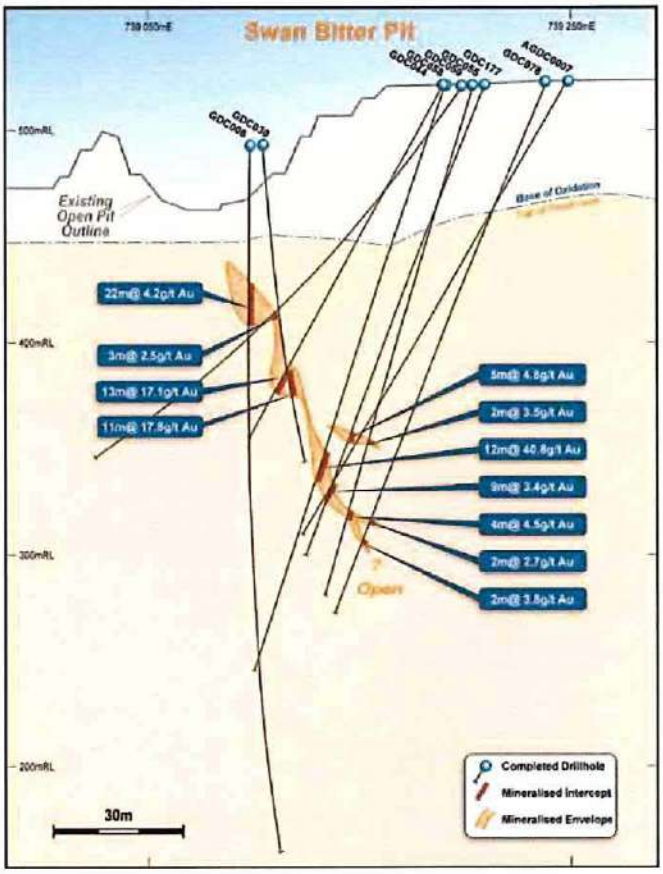


Figure 4- Gidgee Swan Bitter Pit - drill intersections below the pit

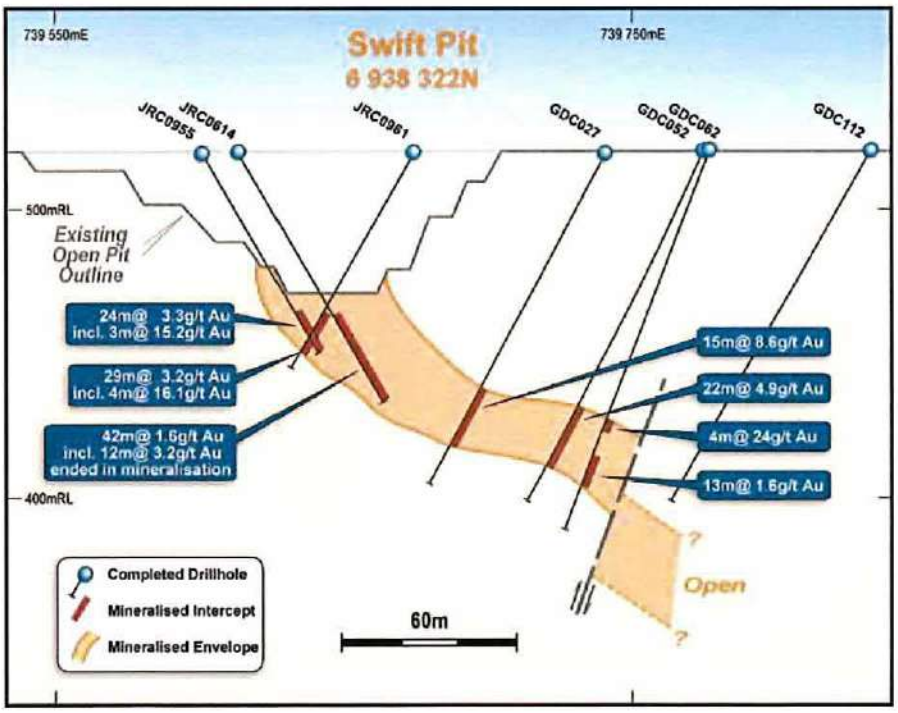


Figure 5- Gidgee Swift Pit - drill intersections below the pit

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NICKEL

SAVANNAH PROJECT, THE KIMBERLEY, WA

In 2009/10, Panoramic reported on the Lower Zone Ore Reserve below the 500 Fault and that the Lower Zone orebody was truncated by a fault structure (the 900 Fault) at approximately 900 metres below surface. In 2011/12, Panoramic intends to complete a new hanging wall drill drive from the Lower Zone decline and commence resource definition drill testing of the orebody below the 900 Fault (Figure 6).

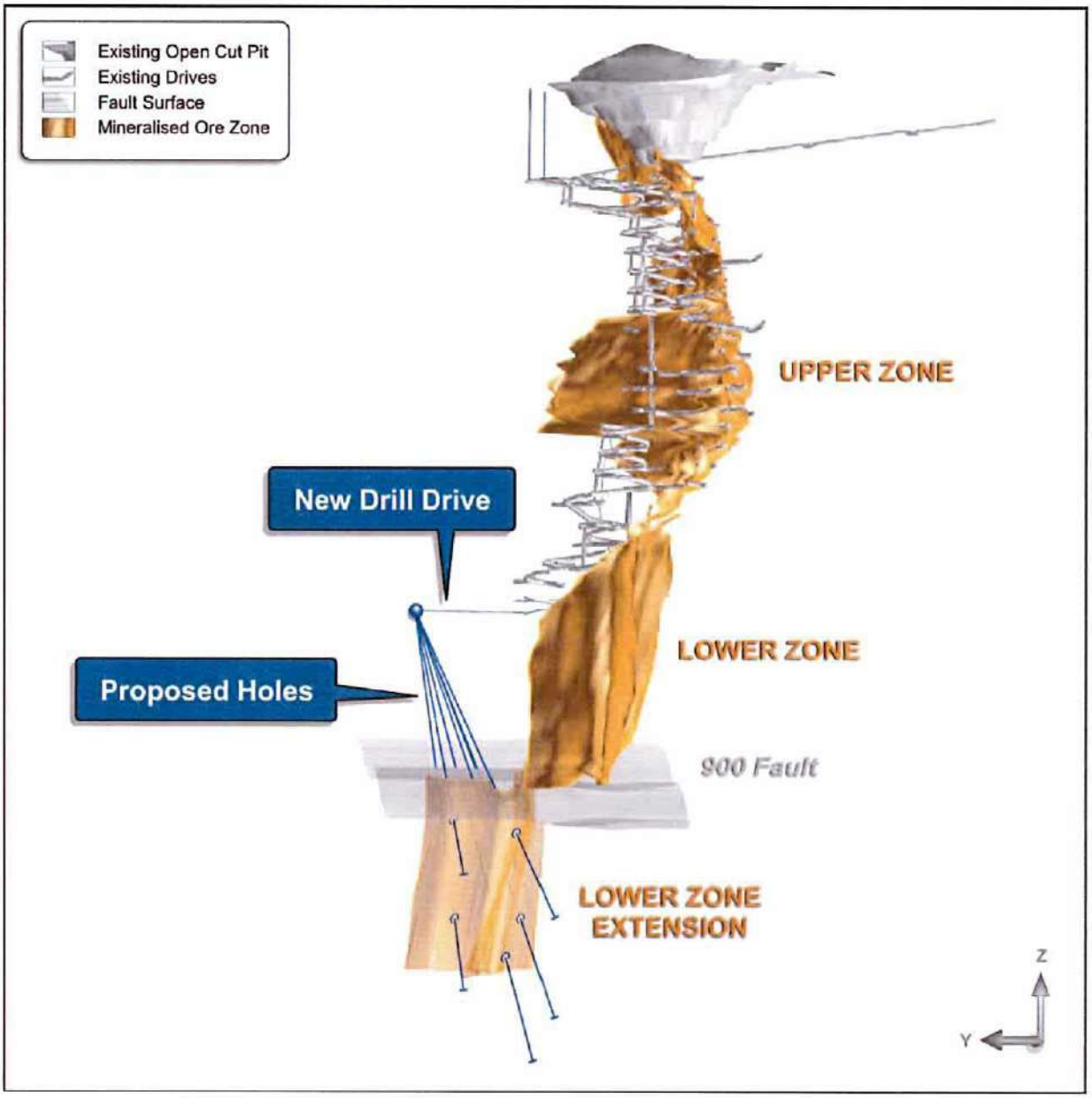


Figure 6 – 3D Schematic of the Savannah Mine showing proposed drilling below the 900 Fault

Surface exploration at Savannah during 2010/11 involved the completion of a series of deep penetrating fixed loop electromagnetic (FLEM) surveys comprising 30 line kilometres of survey around the mine site. Three deep electromagnetic anomalies were identified by these surveys and have been scheduled for diamond drill testing in early 2011/12 (Figure 7). Precollars for the three diamond drill holes (SMP176, 177 & 179) were completed during 2010/11.

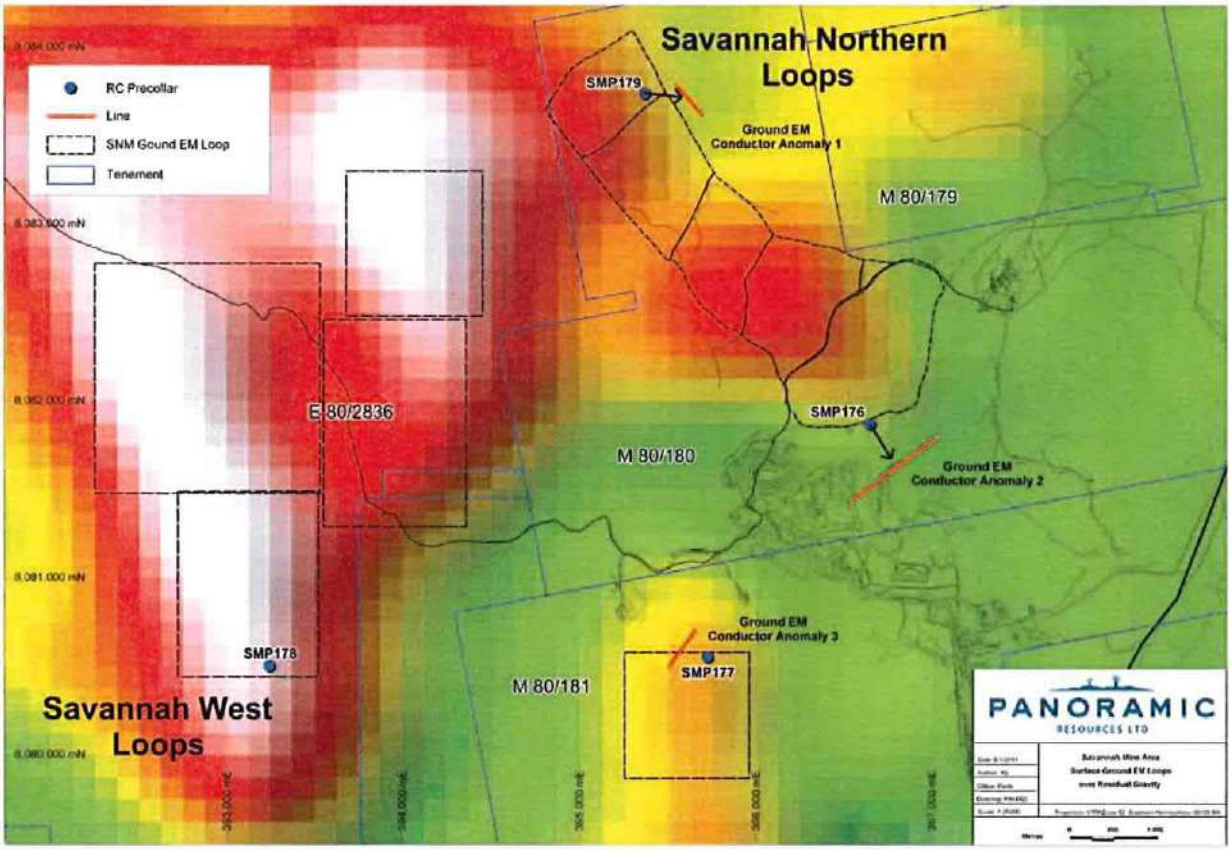


Figure 7 – Plan of the Savannah & Savannah West area showing FLEM loops & drill targets

EAST KIMBERLEY TENEMENTS JOINT VENTURE (EKJV) WITH THUNDELARRA EXPLORATION LTD

Since July 2009, Panoramic has spent over $3 million on the EKJV regional tenements, and under the terms of the farm-in agreement with Thundelarra Exploration Ltd we have now earned a 61% interest in these tenements.

In 2009/10 Panoramic commissioned a regional airborne gravity survey of the EKJV area and followed this during the year with a series of airborne electromagnetic (VTEM) surveys (5,307 line kilometres) over selected gravity anomalies. Processing and interpretation of these airborne data sets is now complete with numerous target areas having been identified. Follow-up FLEM ground surveys over these target areas has commenced in order to prioritise drill targets. To 30 June 2011, 22.5 line kilometres of FLEM survey had been completed at Springvale and 40 line kilometres at Savannah West. At Jenner, which is located just to the north of Copernicus, a strong EM conductor associated with a pyroxenite ultramafic body similar to the Copernicus Orebody was identified and drill tested in early 2011/12.

In 2011/12 Panoramic intends to complete the follow-up ground FLEM program and begin systematic drill testing of targets this year. In addition, the Company has commissioned two new VTEM surveys to be flown to identify new target areas.

NICKEL

LANFRANCHI PROJECT, KAMBALDA, WA

2010/11 was an exciting year for exploration at Lanfranchi with the discovery and subsequent delineation of the high-grade Helmut South Extension Orebody and the completion of the Cruickshank infill resource drill program and subsequent resource upgrade.

Helmut South Extension

The Helmut South Extension Orebody is ideally situated for mining, being located adjacent to existing Helmut South and Deacon underground development on the eastern flank of the Deacon channel, immediately down plunge of the Helmut South Orebody (Figure 8). Fifty seven drill holes were completed in early 2011 to define the Resource. The initial Mineral Resource contains 181,700 tonnes at 3.90% Ni for 7,080 tonnes contained nickel and is open at depth. Recently released drill intersections (HS610 and HS661), down-plunge of the Resource, demonstrate the ongoing potential of the eastern flank of the Deacon/Helmut channel system to host further discoveries of high-grade nickel sulphide mineralisation.

Mine planning studies are underway to convert the Indicated Resource to a maiden Reserve.

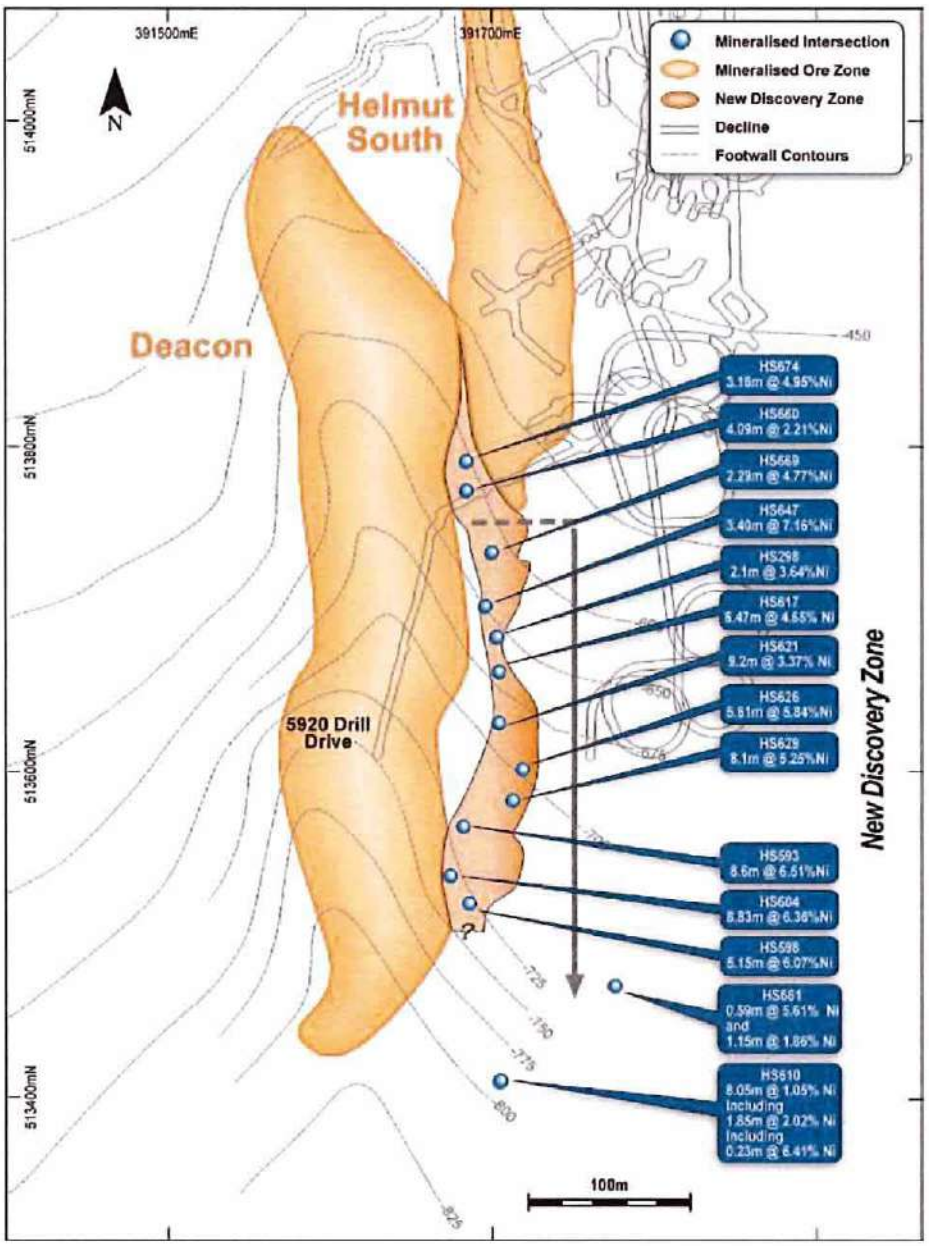


Figure 8 – Location plan of the Helmut South Extension showing representative drill hole intersections

Deacon/Schmitz Channel Extensions

Underground exploration drilling in the Deacon/Schmitz channel systems continued to return positive results in 2010/11 (Figure 9). On the eastern flank of the Deacon channel, holes HS610 and 661 were drilled to target previously defined EM anomalies. Both holes intersected significant mineralisation with the HS661 intersection possibly indicating the discovery of a new high-grade zone of mineralisation located to the east of the Helmut South Extension Orebody.

To the west of the Deacon channel, drill holes HS663 and HS664 were drilled targeting previously defined EM anomalies. Drill hole HS663 intersected graphitic sediments at the target depth which would explain the source of the anomaly, while drill hole HS664 intersected an un-mineralised ultramafic basalt contact on the southern edge of the EM anomaly. The source of this anomaly remains unexplained.

Three platform electromagnetic (EM) holes (SMT211, SMT212 & SMT213) were also drilled from the Deacon 5920 hanging wall drill drive south westwards across the theoretical down-plunge extension of the Schmitz Orebody. All three holes were successfully cased in preparation for EM surveying.

Towards the end of 2010/11, the exploration drill rig was relocated to the 7666 drill cuddy at the base of the Deacon decline to begin a series of long platform EM holes designed to explore 450 metres down-plunge from the Deacon and Helmut South Extension orebodies. The first two holes of this program (HS692 & HS693) have been completed. Both holes identified strong EM anomalies in the Deacon channel position and intersected broad zones of moderate grade nickel sulphide mineralisation (15.7 metres at 1.55% Ni in HS692) on the basalt ultramafic contact.

The drill program to test down-plunge of Deacon and Helmut South is ongoing and will continue well in to 2011/12. Based on the results to date, Panoramic has commenced development of the 7400 Deacon hanging wall drill drive, which when completed, will enable detailed resource definition drilling to commence.

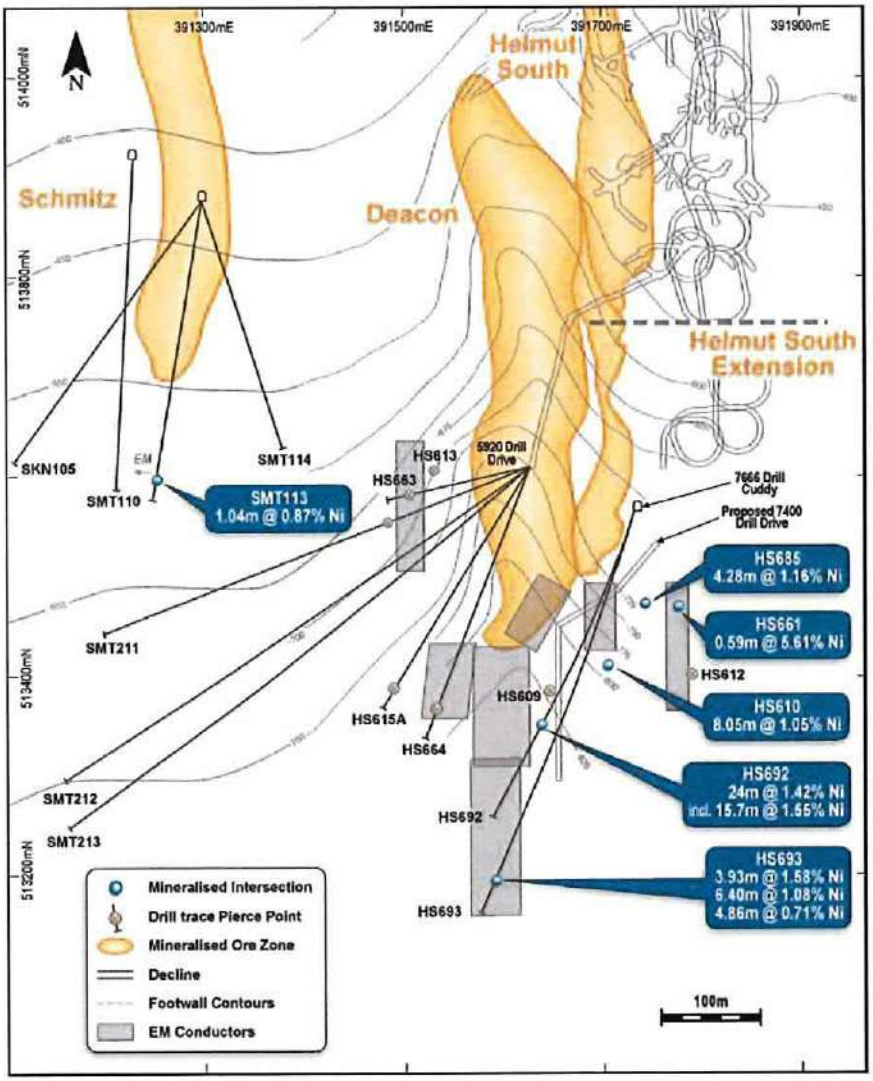


Figure 9 – Deacon Schmitz area showing recent exploration drill results

CRUICKSHANK/NORTHERN DOME

Cruickshank is an undeveloped nickel sulphide deposit located six kilometres north-east of Lanfranchi in the vicinity of the overturned Northern Dome. In 2010/11, Panoramic commenced a study of possible development scenarios for Cruickshank. As part of the study, an infill resource drill program was completed, involving fifty four drill holes (comprising 5,619 metres of RC and 7,888 metres of diamond core) for a program total of 13,508 drill metres. At the completion of the drill program, the Cruickshank Mineral Resource was re-estimated to contain a total resource of 2.63 million tonnes grading 1.28% Ni for 33,560 tonnes Ni contained, including an Indicated Resource total of 2.02 million tonnes grading 1.42% Ni for 28,720 tonnes Ni contained. The total resource figure of 2.63 million tonnes represents a 26% increase from the previous resource estimate completed in 2007.

During the course of the infill drill program, it was recognised that the Northern Dome overturned fold structure, located on the surface just to the north of Cruickshank, was beginning to interact with the deposit at depth (Figure 10). During the June quarter 2011, the Company deepened six recently completed infill resource drill holes (involving a total of 822 drill metres) to determine if the Cruickshank deposit was folded around the overturned fold structure and therefore not closed-off at depth. Although these six holes were not ideally situated to test the concept, all six deepened holes drilled through the fold axis and re-entered the Cruickshank ultramafic host lithologies on the underside of the fold. More importantly, five of the six deepened holes intersected encouraging zones of low-grade mineralisation on the underside of the fold as follows:

- 7.00 metres at 0.50% nickel in TD8174;
- 5.00 metres at 0.73% nickel in TD8195A;
- 1.00 metre at 0.54% nickel in TD8197A;
- 9.00 metres at 0.60% nickel in TD8199A; and
- 6.00 metres at 0.60% nickel in TD8205A.

The intersections of these broad zones of low-grade mineralisation on the underside of the Northern Dome overturned fold are highly encouraging. The intersections indicate that the Cruickshank deposit has been folded and that with new, better positioned surface drill holes, the central higher grade core of the Cruickshank deposit may be intersected further down-plunge on the underside of the Northern Dome overturned fold structure (Figures 11 and 12). More drilling from surface to test this concept is planned for 2011/12.

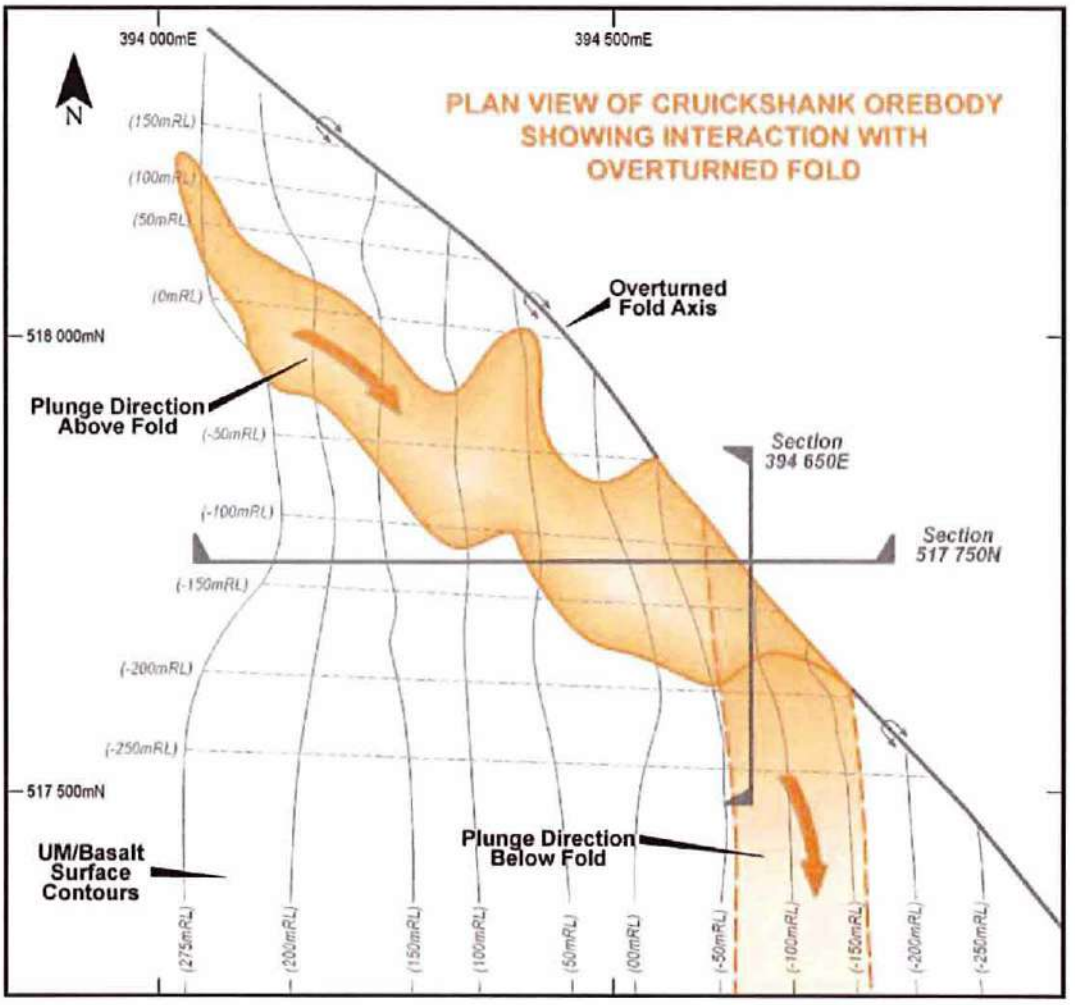


Figure 10 – Plan view of the Cruickshank Orebody showing overturned fold

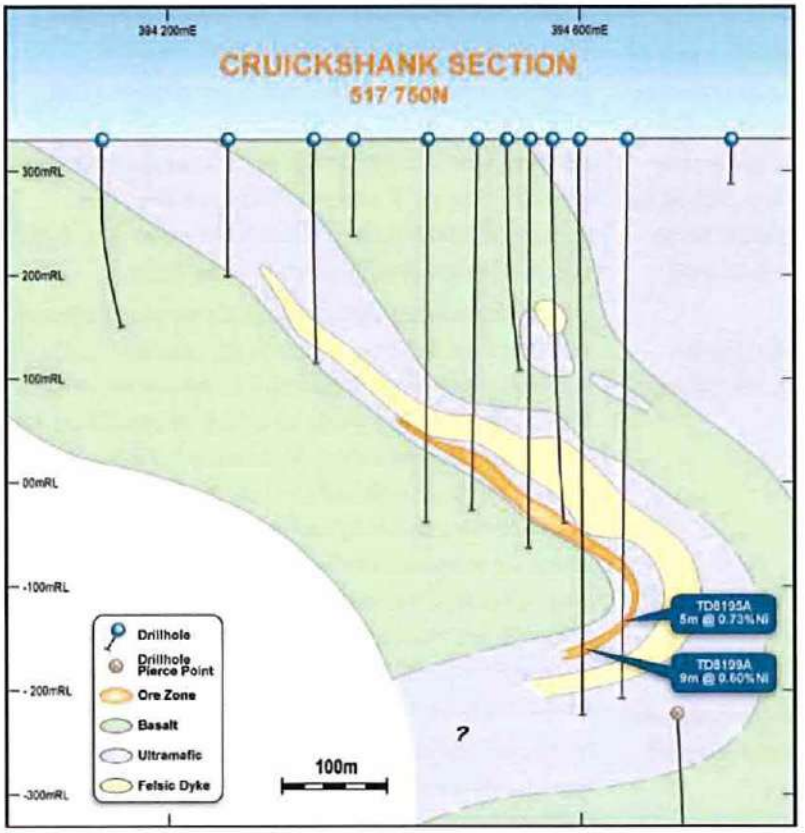


Figure 11 – Cruickshank geological cross section 517 750mN (looking north)

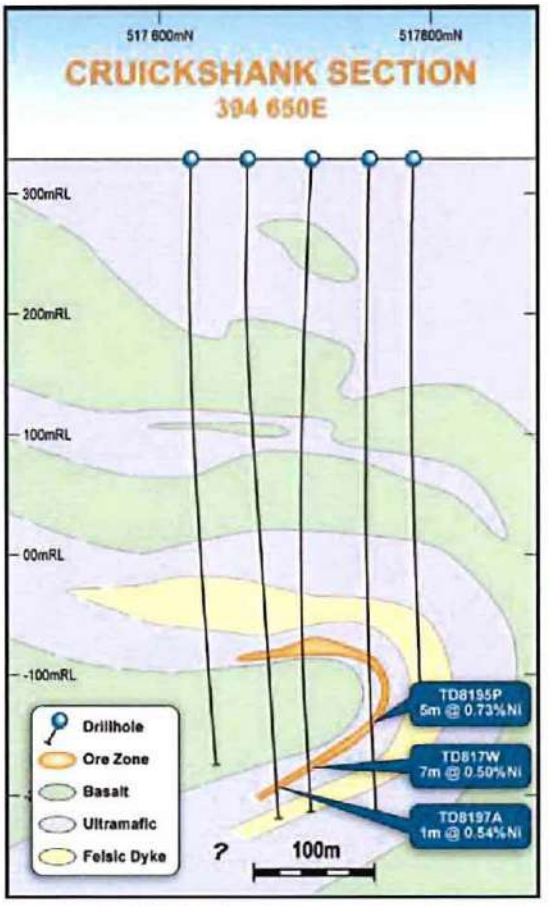


Figure 12 – Cruickshank geological cross section 394 650mE (looking west)

Lanfranchi Orebody

Development of the new, 150 metre long, hanging wall drill platform above the Lanfranchi Orebody was delayed during the year due to poor ground conditions. The drive is being developed to enable drill testing of the Lanfranchi deposit to 300 metres down-plunge of the existing resource and to follow-up on the exciting West Lanfranchi drill intersections from 2009/10. When completed, the drive will also enable several long platform EM drill holes to be drilled further down-plunge to test the continuation of the Lanfranchi system 500 to 600 metres below the base of the existing resource.

With the delay to the Lanfranchi drill drive in 2010/11, drilling from the drive was restricted to six drill holes. Four of the holes intersected significant mineralisation (Figure 13) including:

- 2.80 metres at 5.19% nickel (LAN212);
- 1.74 metres at 6.98% nickel (LAN214);
- 0.29 metres at 11.49% nickel (LAN215); and
- 3.69 metres at 1.19% nickel and 0.88 metres at 3.99% nickel (LAN200).

After completing these holes the drill rig was relocated to undertake the Helmut South Extension drill program.

Panoramic has re-designed the Lanfranchi hanging wall drill drive and intends to complete the drive in 2011/12 enabling the Lanfranchi down-plunge drill testing program to resume.

Cowan Project (100% Panoramic, Nickel Rights only)

The Cowan Nickel Project is located in the Widgiemooltha-Higginsville-Chalice-Democrat region of the Eastern Goldfields, Western Australia, and comprises two project areas over approximately 500 square kilometres. Panoramic commenced work on the Cowan Project in mid 2008.

Panoramic continued to explore the Cowan Project tenement package during 2010/11. Exploration activities to date have principally been via the application of moving loop electromagnetic (MLEM) surveys of prospective ultramafic horizons and RC drill testing of MLEM anomalies. MLEM techniques can detect the presence of buried nickel sulphide mineralisation and is an invaluable exploration tool to generate drill targets.

In the past twelve months, an additional 15 line kilometres of MLEM survey have been completed at Cowan, bringing the total completed to date to 383 line kilometres. In addition, a further nine MLEM anomalies were tested by RC drilling. Five of the nine anomalies were tested with graphitic black shales intersected on three of the five anomalies tested. All three drill holes that intersected the black shale horizons intersected the shales in the anticipated position of the EM anomalies explaining the source of the anomalies. One drill hole was terminated short of target due to excessive water, while the remaining hole failed to intersect a recognisable conductor. A subsequent EM survey of this hole returned a clear off-hole anomaly at the target depth. The remaining drill holes of the program were completed without intersecting any sulphide mineralisation.

In the second half of 2010/11, Panoramic initiated a review of the Cowan Project to determine what work remains to be completed. The review identified four prospective areas for MLEM coverage and fifteen RAB and MLEM anomalies that warrant drill testing. Panoramic is planning to complete this program of work in 2011/12.

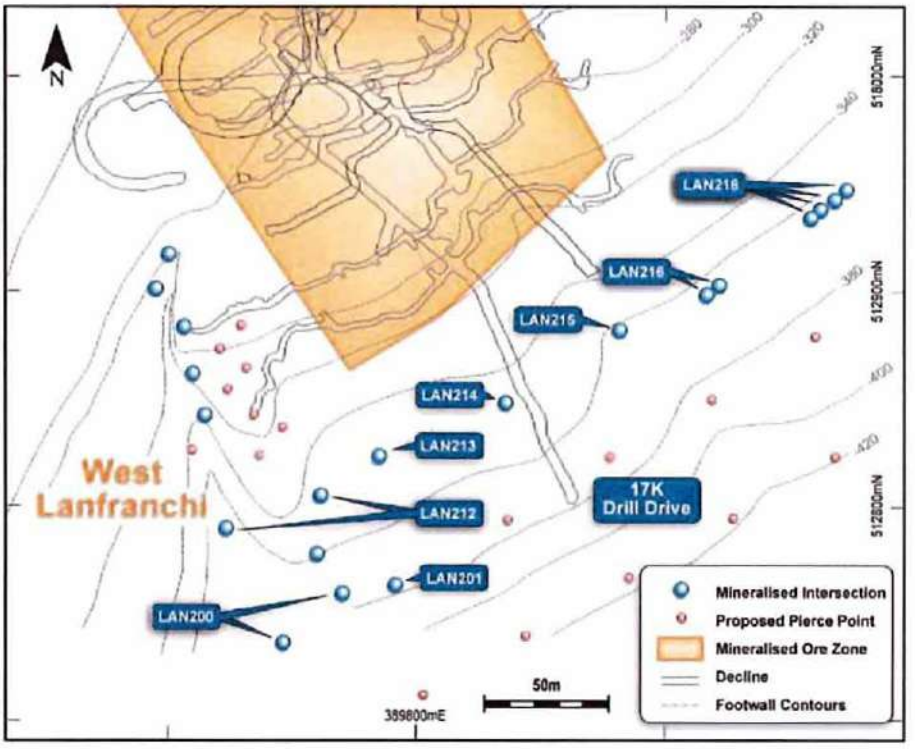


Figure 13 – Plan view of the Lanfranchi Orebody Extension

OTHER EXPLORATION ACTIVITIES

BLUEBUSH COPPER-GOLD EXPLORATION AGREEMENT WITH TUC RESOURCES LIMITED- CU/AU IN N.T.

In July 2009, Panoramic entered into a farm-in Agreement with TUC Resources Limited to earn up to 80% in the Bluebush copper-gold Project in the Tennant Creek region. The Bluebush Project is located between the Tennant Creek mineral field and the Rover mineral field where recent exploration success has provided renewed interest in an area that has historically produced over five million ounces of gold and 500,000 tonnes of copper.

The exploration focus of the Project was five well defined geophysical targets, considered analogous to Iron Oxide Copper Gold (IOCG) style systems. Three of the five targets were tested in 2009/10 without success. The remaining two targets were tested in the first half of 2010/11 without success. A 1,000 metre RAB/AC drill program targeting conceptual phosphate and copper/gold targets has been scheduled for completion in early 2011/12.

NORRLAND EXPLORATION JOINT VENTURE – NICKEL IN SWEDEN

Following the disappointing 2010 drill results and a review of the Project that concluded that no other targets existed to warrant further work, Panoramic formally withdrew from the Norrland Nickel JV in June 2011.

TUSHTENA GOLD PROJECT – GOLD IN ALASKA

Following disappointing drill results in 2010 and an unfavorable review of the remaining prospectivity of the Project, Panoramic formally withdrew from the Tushtena Gold Project in July 2011.

DRAKE RESOURCES EXPLORATION ALLIANCE – BASE METALS IN SCANDINAVIA

In July 2010, Panoramic and Drake Resources Limited formed an alliance to identify, explore and develop base and precious metal opportunities across Scandinavia.

Finland

As part of the alliance, two joint ventures areas were finalised in 2010 to explore for Palaeoproterozoic volcanic massive sulphide (VMS) style Cu-Zn mineralisation in Finland. The Kangasjarvi and Savia JV areas are located in the Pyhasalim-Vihanti region of the Fennoscandian Shield of Finland. The Fennoscandian Shield is one of the most intensely and varied mineralised Palaeoproterozoic terrains in the world, including VMS, iron oxide Cu-Au, orogenic gold and layered intrusions.

In August and September 2010 detailed airborne VTEM surveys, comprising a total of 2,740 line kilometres, were completed over both the Kangasjarvi and Savia JV areas. The final survey was received, processed and interpreted by January 2011. In total, 53 VTEM anomalies were identified and ranked, including 10 priority anomalies in six areas that were selected for drill testing. Three of the priority drill targets were deemed to require follow-up ground EM prior to drilling while the remainder were deemed ready to drill test.

Follow-up ground EM was completed on the three priority VTEM anomalies early in 2011. Drill testing of the priority VTEM targets commenced in May 2011. By early August, nine targets had been completed on the Kangasjarvi and Savia JVs for a cumulative total of 2,043 metres.

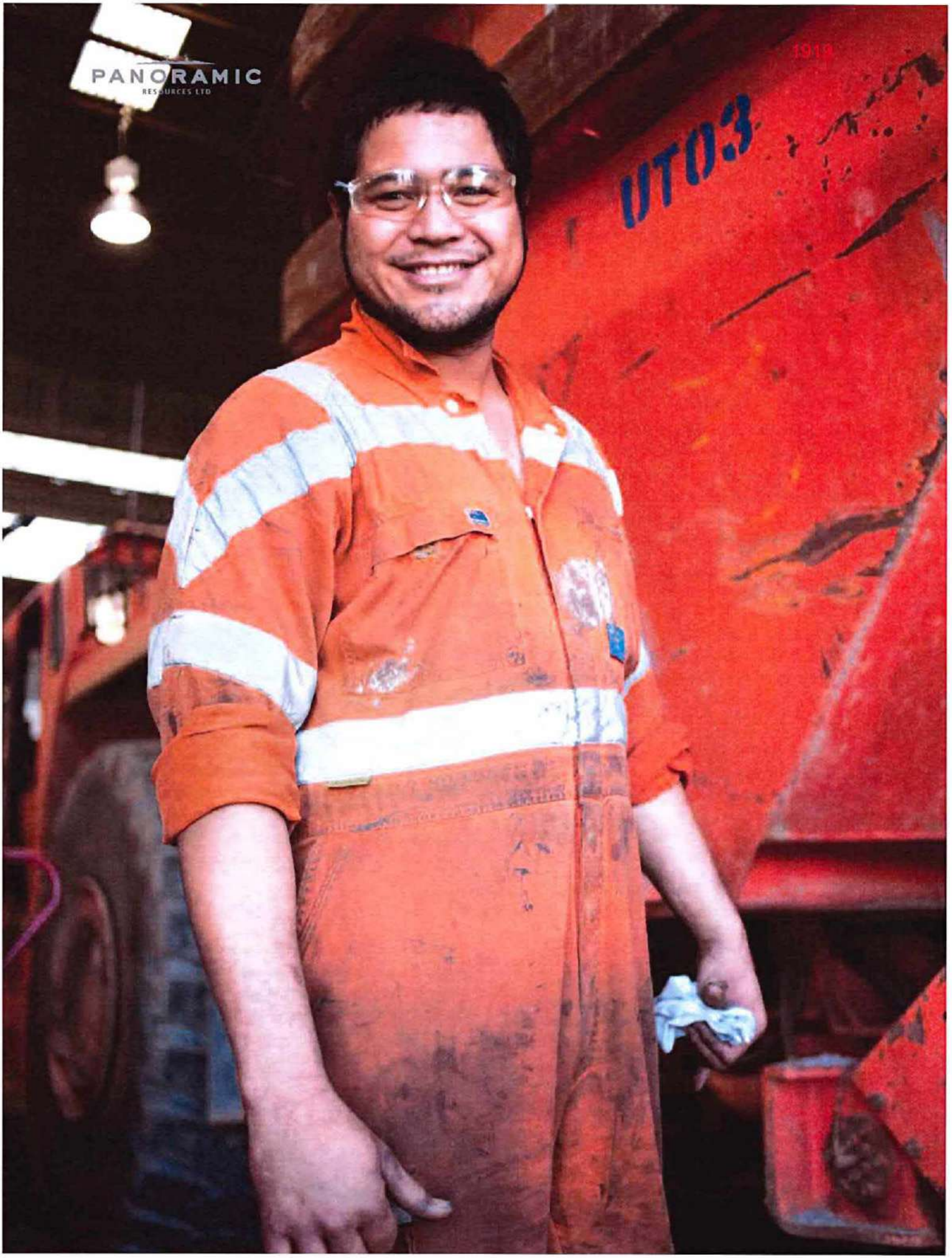
Assay results have been received from AKA 11 and AKA 7. At AKA 11, the best intercept was 1 metre at 242 ppm Cu, 552 ppm Ni. Target AKA 7 returned a number of anomalous results with up to 0.325 % Zn including several samples ranging between 0.1 to 0.3 % Zn and up to 717 ppm Cu. One sample assayed 0.129 % Ni.

A review of the drilling is currently underway and it is expected that some holes will require down-hole EM to test for "off-hole" conductors and to confirm that the target was intersected. This is expected to take place in the second quarter of 2011/12.

During August 2011, a review of the VTEM surveys identified a number of additional, highly prospective targets (including targets in the newly acquired Keitele survey). Seven of these targets will be followed up with ground geophysics in 2011/12. Drilling is planned for the 2011/2012 northern winter to access these targets, most of which are located in swampy areas.

Norway

In July 2011 as part of the ongoing Panoramic/Drake Generative Alliance, Panoramic agreed to establish three new joint ventures to explore for copper-rich massive sulphide mineralisation in Norway. The three Norway project areas are Lokken, Hersjo and Sulitjelma. Exploration on these areas is scheduled to commence in the first half of 2011/12.


PANORAMIC
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RESOURCES & RESERVES

NICKEL

Resource	Equity	Metal	Date of Resource	Measured		Indicated		Inferred		Total		Metal Tonnes
				Tonnes	Ni (%)	Tonnes	Ni (%)	Tonnes	Ni (%)	Tonnes	Ni (%)	
Savannah Project												
Savannah	100%	Nickel	Jul-11	1,483,000	1.57	3,782,000	1.52	-	-	5,265,000	1.53	80,800
		Copper			0.77		0.81		-		0.80	42,100
		Cobalt			0.08		0.08		-		0.08	4,200
Copernicus	60%	Nickel	Jul-11	233,000	1.08	240,000	1.38	14,000	1.01	487,000	1.23	6,000
		Copper			0.66		0.99		0.70		0.82	4,000
		Cobalt			0.04		0.05		0.03		0.04	200
Lanfranchi Project	100%	Nickel										
Cruikshank			Jul-11	-	-	2,018,000	1.42	611,000	0.79	2,629,000	1.28	33,600
Deacon			Jul-11	1,277,000	2.84	386,000	2.64	54,000	2.36	1,717,000	2.78	47,800
Gigantus			Jul-11	-	-	-	-	652,000	1.63	652,000	1.63	10,600
Helmut South			Jul-11	81,000	2.68	-	-	-	-	81,000	2.68	2,200
Helmut South Ext			Jun-11	-	-	166,000	4.10	15,000	1.67	181,000	3.90	7,100
John			Jul-11	-	-	-	-	291,000	1.42	291,000	1.42	4,100
Lanfranchi			Jul-11	4,000	4.35	64,000	5.85	11,000	5.24	79,000	5.69	4,500
Martin			Jul-11	-	-	71,000	3.10	7,000	2.48	79,000	3.04	2,400
McComish			Jul-11	-	-	-	-	992,000	1.49	992,000	1.49	14,800
Schmitz			Jul-11	24,000	4.75	55,000	4.72	0	3.68	79,000	4.73	3,700
Winner			Jul-11	-	-	14,000	4.40	-	-	14,000	4.40	600
Total (Equity)		Nickel										218,100
		Copper										46,100
		Cobalt										4,400

Reserve	Equity	Metal	Date of Reserve	Proven		Probable		Total		Metal Tonnes
				Tonnes	(%)	Tonnes	(%)	Tonnes	(%)	
Savannah Project										
Upper Zone	100%	Nickel	Jul-11	-	-	1,363,000	1.35	1,363,000	1.35	18,500
		Copper			-		0.64		0.64	8,700
		Cobalt			-		0.07		0.07	1,000
Lower Zone	100%	Nickel	Jul-11	-	-	3,216,000	1.25	3,216,000	1.25	40,300
		Copper			-		0.64		0.64	20,500
		Cobalt			-		0.06		0.06	2,000
Copernicus O/Pit	60%	Nickel	Jul-11	-	-	219,000	1.03	219,000	1.03	2,300
		Copper			-		0.63		0.63	1,400
		Cobalt			-		0.04		0.04	100
Lanfranchi Project	**100%**									
Deacon			Jul-11	-	-	1,840,000	2.20	1,840,000	2.20	40,400
Helmut South			Jul-11	28,000	2.01	-	-	28,000	2.01	600
Lanfranchi			Jul-11	-	-	90,000	3.12	90,000	3.12	2,800
Schmitz			Jul-11	-	-	71,000	2.73	71,000	2.73	1,900
Helmut Sth Ext			Jul-11	-	-	192,000	2.90	192,000	2.90	5,600
Total (Equity)		Nickel								112,300
		Copper								30,600
		Cobalt								3,100

- *Savannah project resource cutoff grades at 0.50% Ni.*
- *Lanfranchi project resource cutoff grades at 1.00% Ni.*
- *Cruikshank resource cutoff grade at 0.50% Ni.*
- *All resources are inclusive of reserves.*
- *Savannah reserve cutoff grade is 0.84% Ni*
- *Copernicus reserve cutoff grade is 0.50% Ni*
- *Lanfranchi project cutoff grade is 1.00% Ni except the Deacon Longhole stopes which are 0.80% Ni.*
- *Individual Project Resources and Reserves are stated on an equity basis*
- *Figures have been rounded and hence may not add up to the given totals.*

The information in this report that relates to Mineral Resources is based on information compiled by or reviewed by Paul Hetherington (MAusIMM) for the Savannah Project and Brad Robinson (MAusIMM) for the Lanfranchi Project. The aforementioned are full-time employees of Panoramic Resources Limited. The aforementioned have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2004 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. The aforementioned consents to the inclusion in the report of the matters based on their information in the form and context in which it appears.

Information in this report relating to Ore Reserves has been completed by or reviewed by Jonathon Bayley (MAusIMM) for the Lanfranchi Project and Lilong Chen (MAusIMM) for the Savannah Project. The aforementioned are full-time employees of Panoramic Resources Limited. The aforementioned have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2004 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. The aforementioned consents to the inclusion in the report of the matters based on his information in the form and context in which it appears.

RESOURCES & RESERVES

GOLD

Resource	Equity	Metal	Date of Resource	Measured		Indicated		Inferred		Total		Metal Tonnes
				Tonnes	Ni (%)	Tonnes	Ni (%)	Tonnes	Ni (%)	Tonnes	Ni (%)	
Gidgee Project	100%	Gold										
Open Pits			Jul-11	-	-	1,048,000	3.07	-	-	1,048,000	3.07	103,500
Premium/Cascade UG			Jul-11	-	-	68,000	10.80	62,000	7.70	130,000	9.32	39,000
Other UG			Jul-11	27,000	10.42	87,000	10.1	548,000	7.34	662,000	7.83	166,600
Total (Equity)		Gold		27,000	10.42	1,203,000	4.02	610,000	7.38	1,840,000	5.22	309,100

(i) Resources estimated by Apex Minerals NL (see www.asx.com.au ASX:AXM) at a 4.5g/t Au lower cut off and reported in their 2009 Annual Report

(ii) Resources estimated by Legend Mining Limited (see www.asx.com.au ASX:LEG) at a 3 g/t Au lower cut off and reported in their 2006 Annual Report

(iii) Resources estimated by Legend Mining Limited (see www.asx.com.au ASX:LEG) at a 1.3 g/t Au lower cut off and reported in their 2006 Annual Report

The information in this report that relates to Exploration Results is based on information reviewed by John Hicks. Mr Hicks is a member of the Australasian Institute of Mining and Metallurgy (AusIMM) and is a full-time employee of Panoramic Resources Limited. Mr Hicks has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which each person is undertaking to qualify as a Competent Person as defined in the 2004 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Mr Hicks consents to the inclusion in the report of the matters based on the information in the form and context in which it appears.

The information in this report that relates to Mineral Resources at Gidgee (Premium and Cascade) is based on studies commissioned and published by Apex Minerals NL which were compiled or reviewed by Mr Andrew Thompson as quoted in Apex Minerals' 2009 Annual Report.

The information in this report that relates to Mineral Resources at Gidgee (excluding Premium and Cascade) is based on studies commissioned and published by Legend Mining and its consultant Dr Spero Carras, as quoted in Legend Mining's 2006 Annual Report.

Mr Thompson is a member of and Dr Carras a Fellow of the Australasian Institute of Mining and Metallurgy (AusIMM) and have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which each person is undertaking to qualify as a Competent Person as defined in the 2004 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Mr Thompson was a full time employee of Apex Minerals NL in 2009 and is currently employed by Corazon Mining Limited. Dr Carras was a consultant to Legend in 2006 and currently Executive Director of Carras Mining Pty Ltd and consults to numerous mining and exploration companies. Mr Thomson and Dr Carras consent to the inclusion in the report of the matters based on the information in the form and context in which it appears.

SUSTAINABILITY

Panoramic has been committed to conducting its operations in a sustainable and responsible manner since the company was formed in 2001. We hold ourselves accountable for our actions and performance in the economic, governance, environmental and social areas of our business, which is central to this commitment.

To give our stakeholders a better sense of who we are as a business we publicly report on our sustainability performance via our Annual Report and our yearly Sustainability Report.

Panoramic's Approach to Sustainability Reporting

Over the past four years, Panoramic has invested greater time and effort in progressing the way we measure and communicate our sustainability endeavours.

As well as meeting the information needs of our stakeholders, we now aim to use the reporting process to achieve broader internal engagement and alignment around sustainability. This will not only produce a report that speaks to all our stakeholders, but one that also directs and integrates our efforts around sustainability with different parts of our business.

To achieve this goal we have engaged Banarra, a specialised Australian sustainability consultancy to drive the sustainability reporting process. Greater internal engagement and alignment around sustainability will assist in initiating and promoting change, particularly around areas that deliver clear value to our business, employees, shareholders, suppliers and to the communities in which we operate.

Ultimately, this will improve the way that we manage our sustainability performance and accountability to our stakeholders.





SUSTAINABILITY

Panoramic's Sustainability Report 2011

We continue to use the Global Reporting Initiative (GRI) Sustainability Reporting Framework as the basis of our reporting. Recognised as the world's most widely applied sustainability reporting framework, it enables readers to compare our performance to that of our peers, both in Australia and offshore.

The GRI approach also encourages transparency and relevance in terms of what is reported, so that our stakeholders gain a clearer picture of what we have achieved and the key issues still being addressed.

Panoramic's Materiality Approach and Key Impacts

Reporting on our key impacts requires us to first identify and understand them, and then to measure, monitor and set clear commitments and targets around how we respond to them.

In 2011, we have undergone a systematic process to identify those issues that are of most interest to, or that have the biggest impact upon, our stakeholders and on Panoramic as a business.

Having the full support and involvement of our senior management highlights the importance Panoramic places on understanding and accounting for our sustainability impacts. As a result, we have identified a number of impact areas spanning our economic performance, workforce, environment as well as the communities in which we operate.

Priority issues include a broad range of performance and management aspects around Panoramic's business growth and systems integration, mine life and impacts on employment, safety and manual handling, tailings storage and water management, and meeting community expectations.

These are the key issues that will be the focus of our Report, enhancing its value and relevance to our stakeholders. The Report will be available on our website www.panoramicresources.com in late 2011.



PANORAMIC
RESOURCES LTD

PANORAMIC BOARD OF DIRECTORS



Christopher J G de Guingand
Non-Executive Chairman



Peter J Harold
Managing Director



Christopher D J Langdon
Non-Executive Director



John Rowe
Non-Executive Director



Brian M Phillips
Non-Executive Director



Trevor R Eton
Company Secretary

CORPORATE DIRECTORY

AS AT 15 SEPTEMBER 2011

BOARD OF DIRECTORS

Christopher J G de Guingand
Non-Executive Chairman

Peter J Harold
Managing Director

Christopher D J Langdon
Non-Executive Director

Brian M Phillips
Non-Executive Director

John Rowe
Non-Executive Director

SENIOR MANAGEMENT

Trevor R Eton
Chief Financial Officer & Company Secretary

Christopher J Williams
General Manager Operations

John D Hicks
Exploration Manager

Wade J Evans
Business Development Manager

Simon A Jessop
Manager Gidgee & Projects

Jason B Grover
Financial Controller

David J Swain
Environmental Manager

Tracey M Ram
Human Resources Manager

Vera Waldby
Administration Manager & Asst Company Secretary

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LANFRANCHI PROJECT

Robert J Thorburn
Operations Manager

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www.panoramicresources.com

News Release

PJD-49

PANORAMIC RESOURCES LTD 1928

vision
commitment
results



Key Points

13 September 2013 **ASX: PAN**

Resources and Reserves at 30 June 2013 and Exploration Update

Nickel[1]

- Nickel Division Resources - **186,800t Ni**
- Nickel Division Reserves - **64,100t Ni**
- Lanfranchi Channel - upgraded Resource to **10,200t Ni, an increase of 117% from 2012**
- Jury-Metcalfe - maiden Resource of 6,500t Ni, previously released and now included

Gold[1]

- Gold Division Resources - **2.5 million oz Au, up 450,200 oz Au, a 22% increase**
- Gidgee Resources - **1.3 million oz Au, up 276,600 oz Au, a 26% increase**
  - 85% of Gidgee Resources are now in the Indicated Resource Category
- Mt Henry Resources - **1.7 million oz Au,** up by 248,000 oz Au, on a 100% basis
  - **1.2 million oz Au**, up by 173,600 oz Au, a 17% increase on Panoramic's 70% equity in the Project
  - 77% of the Mt Henry Resources are now in the Indicated Resource Category

PGMs[1]

- PGM Division Resources - **1.4 million oz Pt and 1.4 million oz Pd**

Commentary

Panoramic Resource Limited ("Panoramic") is pleased to disclose updated Group Resources and Reserves as at 30 June 2013.

General – The Company's geology and exploration teams have added significant value through exploration success across the Group in the past 12 months with the discovery of new orebodies and/or Resource upgrades at both nickel operations and gold projects.

Gold – A significant portion of the FY2013 Exploration Budget was directed towards the gold projects, which has resulted in the Gold Division's Resources increasing to 2.5 million oz Au. Importantly, there has also been a marked increase in the level of geological confidence, with a significant proportion of Resources now classified as Indicated Resources.

Nickel – The Nickel Division Resources have been maintained at close to 30 June 2012 levels with the upgrading of the Lanfranchi Resource and the inclusion of the recently Jury-Metcalfe Resource offsetting depletion from production in FY2013.

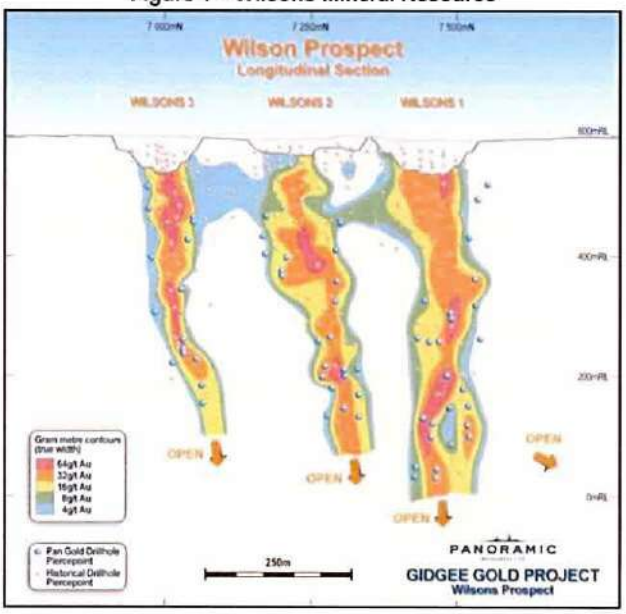
[1] Resource and Reserve tables, relevant disclosures and Competent Person Statements are provided in Tables 1-5.



Gold – Gidgee and Mt Henry Resource Upgrades

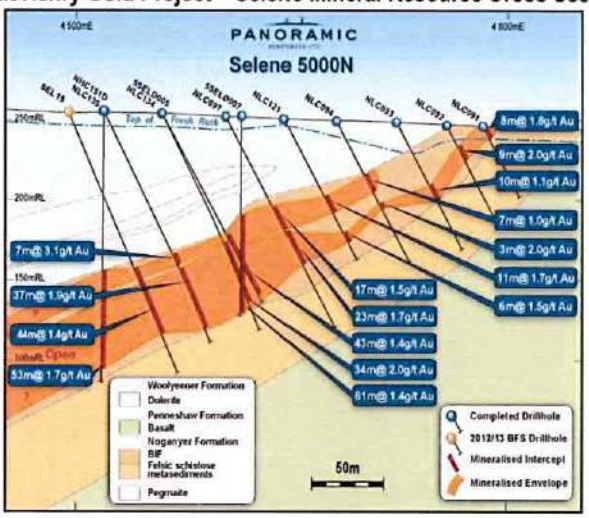
Gidgee – In FY2013, approximately 36,000m of drilling was carried out on the Mineral Resources at the Gidgee Gold Project, including 14,000m of reverse circulation drilling at Howards, Shiraz, Swift and Wilsons and a further 21,000m of diamond drilling at Wilsons, Howards and Swan Bitter. This significant drill program was very successful and resulted in the Gidgee Resources increasing by 26% and has allowed a significant proportion of the resources (85%) to be classified in the Indicated Resource Category. These results will further underpin and assist in de-risking the geological component of the Gidgee Bankable Feasibility Study (BFS), which is currently in progress.

Figure 1 – Wilsons Mineral Resource



Mt Henry – In December 2012, a drill program commenced at the Mt Henry Gold Project, with the principal aim to increase the Resource confidence via infill drilling and to reclassify the Mineral Resources from the Inferred to Indicated Resource Category. The program, which was completed in March 2013, also provided material for the metallurgical test work of the continuing Mt Henry BFS. Approximately 9,500m was drilled through a combination of reverse circulation pre-collars and diamond core tails. As a result of the program, there was a 17% increase in the Mt Henry Mineral Resources which now total 1.7 million oz Au (on a 100% basis). Panoramic's 70% equity in the Project equates to 1.2 million oz Au. Importantly, 77% of the Mt Henry Resources are now in the Indicated Resource Category.

Figure 2 – Mt Henry Gold Project – Selene Mineral Resource Cross Section 5000N

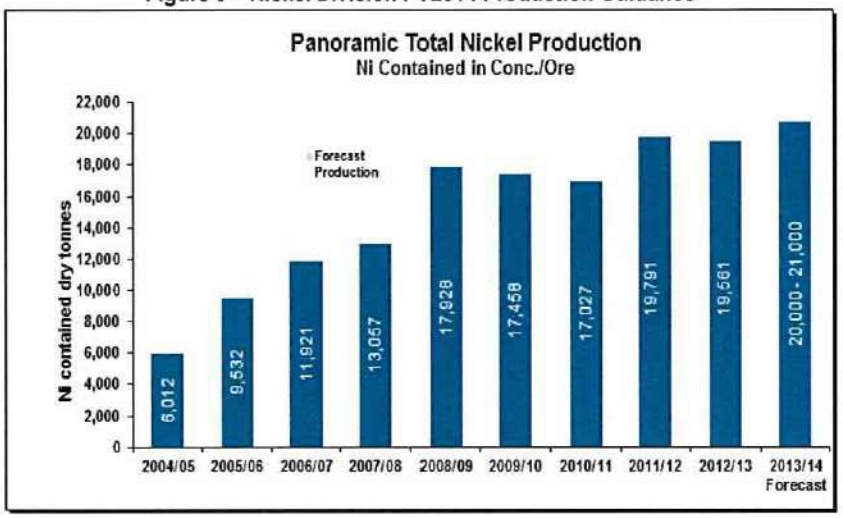




Nickel – FY 2014 outlook and exploration focus

Production – Both nickel operations have set themselves aggressive but realistic production budgets in order to achieve the FY2014 production guidance of 20,000-21,000 tonnes of nickel contained in concentrate/ore.

Figure 3 – Nickel Division FY2014 Production Guidance

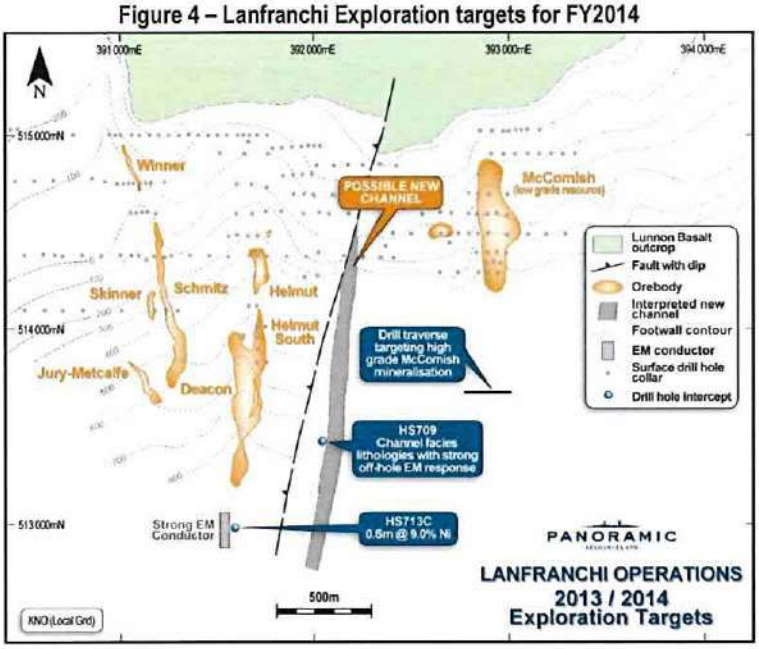


Exploration Update – The principal aim of the Group's FY2014 exploration programs is to **add mine life at both nickel operations**. The geological teams have identified various targets to be tested and the Board has approved a total of $6 million (excluding rents and rates of $1.5 million) to be spent on exploration activities in FY2014, with the majority of work on near mine nickel exploration.

At Lanfranchi, the following areas are to be explored:

- Up and down-plunge of the new Jury-Metcalfe Resource;
- Down-plunge of the Lanfranchi orebody – where the potential for high nickel grades is very promising;
- Down-plunge of Deacon – targeting the strong electromagnetic (EM) plate identified from previous drilling; and
- East of Deacon – targeting a potential new channel of mineralisation.

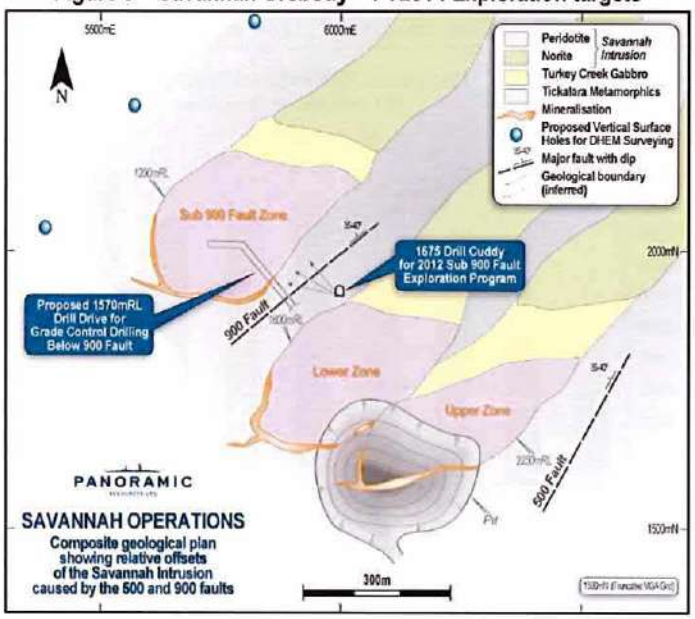
Figure 4 – Lanfranchi Exploration targets for FY2014





At Savannah, a small numbers of holes are to be drilled from surface targeting the mineralisation below the 900 Fault and then to use these holes to confirm the geometry of the mineralisation. Assuming the surface drilling confirms the right geometry, the next stage of the program is to drill a significant number of shorter holes from a dedicated exploration drill drive, most likely the 1570RL Drill Drive, to potentially prove up additional resources.

Figure 5 – Savannah Orebody – FY2014 Exploration targets



Panoramic and Drake Resources Limited ("Drake") have an alliance to identify, explore and develop base and precious metal opportunities across Scandinavia. Drake is about to commence drill testing targets identified from a detailed airborne EM survey at the Nordgruva Copper Zinc prospect in Norway. Firstly, a single hole will be drilled to a depth of approximately 600m targeting one of five exceptional targets identified from the airborne electromagnetic (VTEM) survey flown in 2012. These targets are considered to have a similar geophysical signature to the high grade historic copper mines in the district. The planned hole is targeting one of the largest modelled plates near Kongens with dimensions in the order of 1.5km by 750m, suggesting a potentially significant area of mineralisation in a well-endowed historical copper mining district.

Nordgruva is part of the Røros mining district in central Norway which has a copper/zinc mining history extending back over 300 years from multiple sites.

Figure 6 – Nordgruva Prospect, Norway





Resources, Reserves, Relevant Disclosures and Competent Persons Statements

Nickel Resources and Reserves

Table 1

Resource	Equity	Metal	Date of Resource	Measured		Indicated		Inferred		Total		Metal Tonnes
				Tonnes	Ni (%)	Tonnes	Ni (%)	Tonnes	Ni (%)	Tonnes	Ni (%)	
Savannah Project	**100%**											
		Nickel	Jul-13	2,175,000	1.52	1,508,000	1.54	-	-	3,684,000	1.53	56,400
		Copper			0.80		1.04		-		0.90	33,200
		Cobalt			0.08		0.07		-		0.08	2,900
Copernicus Open Pit	**79%**											
		Nickel	Jul-10	307,000	1.08	316,000	1.38	18,000	1.01	641,000	1.23	7,900
		Copper			0.66		0.99		0.70		0.82	5,300
		Cobalt			0.04		0.05		0.03		0.04	300
Lanfranchi Project	**100%**	**Nickel**										
Cruikshank			Apr-11	-	-	2,018,000	1.42	611,000	0.79	2,629,000	1.28	33,600
Deacon			Jul-13	918,000	2.64	229,000	2.60	105,000	1.66	1,252,000	2.55	32,000
Gigantus			Jul-07	-	-	-	-	652,000	1.63	652,000	1.63	10,600
Helmut South			Jul-12	28,000	3.00	-	-	-	-	28,000	3.00	900
Helmut South Ext			Jun-13	17,000	3.66	124,000	3.20	4,000	2.24	145,000	3.23	4,700
John			Jul-07	-	-	-	-	291,000	1.42	291,000	1.42	4,100
Lanfranchi			Jul-13	71,000	5.32	86,000	4.50	63,000	4.03	220,000	4.63	10,200
Martin			Feb-12	-	-	47,000	3.58	7,000	4.16	54,000	3.66	2,000
McComish			Jul-07	-	-	-	-	992,000	1.49	992,000	1.49	14,800
Metcalfe			Jul-13	-	-	237000	2.1	86,000	1.75	323,000	2.01	6,500
Schmitz			Jul-13	11,000	6.51	38,000	3.39	20,000	3.50	69,000	3.93	2,700
Winner			Jul-11	-	-	14,000	4.40	-	-	14,000	4.40	600
Total (Equity)		Nickel										186,800
		Copper										38,400
		Cobalt										3,200

Table 2

Reserve	Equity	Metal	Date of Reserve	Proven		Probable		Total		Metal Tonnes
				Tonnes	(%)	Tonnes	(%)	Tonnes	(%)	
Savannah Project										
Upper Zone	100%	Nickel	Jul-13	-	-	650,000	1.23	650,000	1.23	8,000
		Copper			-		0.57		0.57	3,700
		Cobalt			-		0.07		0.07	500
Lower Zone	100%	Nickel	Jul-13	-	-	2,041,000	1.32	2,041,000	1.32	26,900
		Copper			-		0.83		0.83	16,900
		Cobalt			-		0.06		0.06	1,300
Copernicus O/Pit	79%	Nickel	Jul-13	-	-	288,000	1.03	288,000	1.03	3,000
		Copper			-		0.63		0.63	1,800
		Cobalt			-		0.04		0.04	100
Lanfranchi Project	**100%**	**Nickel**								
Deacon			Jul-13	-	-	967,000	2.02	967,000	2.02	19,500
Lanfranchi			Jul-13	-	-	73,000	4.61	73,000	4.61	3,400
Helmut Sth Ext			Jul-13	-	-	158,000	2.13	158,000	2.13	3,400
Total (Equity)		Nickel								64,100
		Copper								22,500
		Cobalt								1,900

Savannah Project

• Savannah Project Resource cutoff grade at 0.50% Ni • Copernicus Project Resource cutoff grade at 0.50% Ni • Savannah Project Reserve cutoff grade is 0.85% Ni • Copernicus Project Reserve cutoff grade is 0.50% Ni

Lanfranchi Project

• Lanfranchi Project Resource cutoff grades at 1.00% Ni • Lanfranchi Project Reserve cutoff grade is 1.00% Ni except the Deacon Longhole stopes which are 0.80% Ni • All resources are inclusive of reserves • Individual Project Resources and Reserves are stated on an equity basis



The information in this report that relates to Mineral Resources is based on information compiled by or reviewed by Paul Hetherington (MAusIMM) for the Savannah Project Resource and Copernicus Project Resource and Bradley Robinson (MAusIMM) for the Lanfranchi Project Resources. The aforementioned are full-time employees of Panoramic Resources Limited. The aforementioned have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2004 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves ("the JORC Code"). The aforementioned consents to the inclusion in the report of the matters based on this information in the form and context in which it appears.

Information in this report relating to Ore Reserves has been completed by or reviewed by, Lilong Chen (MAusIMM) for both the Savannah Project and Lanfranchi Project and Jonathon Bayley (MAusIMM) for the Copernicus Project. Lilong Chen is a full-time employee of Panoramic Resources Limited. The aforementioned have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2004 Edition of the JORC Code. The aforementioned consents to the inclusion in the report of the matters based on this information in the form and context in which it appears.

Gold Resources

Table 3

Resource	Equity	Metal	Date of Resource	Measured		Indicated		Inferred		Total		Metal (Au oz)
				Tonnes	Au (g/t)	Tonnes	Au (g/t)	Tonnes	Au (g/t)	Tonnes	Au (g/t)	
Gidgee Project	**100%**	**Gold**										
Swan OC			Jun-12	-	-	3,399,000	2.40	327,000	3.51	3,726,000	2.49	298,600
Heron South			Oct-12	-	-	1,000,000	2.31	136,000	1.41	1,136,000	2.20	80,300
Howards			Jul-13	-	-	5,255,000	1.07	716,000	1.01	5,971,000	1.06	204,000
Specimen Well			Jun-12	-	-	289,000	2.08	72,000	1.79	361,000	2.00	23,200
Toedter			Jun-12	-	-	-	-	661,000	1.62	661,000	1.62	34,400
Eagles Peak			Mar-06	-	-	13,000	3.46	-	-	13,000	3.46	1,400
Orion			Mar-06	-	-	22,000	3.04	-	-	22,000	3.04	2,200
Deep South			Mar-06	-	-	20,000	3.02	-	-	20,000	3.02	1,900
Shiraz			Jul-13	-	-	2,476,000	0.84	440,000	0.78	2,916,000	0.83	77,600
Swan UG			Jun-12	-	-	207,000	8.71	125,000	9.02	332,000	8.83	94,200
Swift UG			Jun-12	-	-	-	-	72,000	9.23	72,000	9.23	21,400
Omega UG			Mar-06	-	-	31,000	9.20	-	-	31,000	9.20	9,200
Kingfisher UG			Mar-06	-	-	390,000	6.80	-	-	390,000	6.80	85,300
Wilsons UG			Jul-13	-	-	2,131,000	5.33	136,000	5.97	2,267,000	5.37	391,500
Mt Henry Project	**70%**	**Gold**										
Selene			Jul-13	-	-	12,258,000	1.17	3,614,000	0.92	15,872,000	1.11	568,100
Mt Henry			Jul-13	-	-	10,869,000	1.24	4,476,000	1.13	15,345,000	1.21	596,000
North Scotia			Jul-13	-	-	250,000	3.11	97,000	1.95	347,000	2.79	31,100
Total (Equity)		Gold		-	-	38,610,000	1.65	10,872,000	1.36	49,482,000	1.58	2,520,400

Gidgee Project

• Swan OC Resource cutoff grade is 0.7 g/t • Eagles Peak Resource cutoff grade is 1.2 g/t • Orion Resource cutoff grade is 1.3 g/t • Deep South Resource cutoff grade is 1.2 g/t • Swan UG Resource cutoff grade is 4.0 g/t for Indicated resources and 5.0 g/t for Inferred resources • Swift UG Resource cutoff grade is 5.0 g/t • Omega UG Resource cutoff grade is 3.0 g/t • Kingfisher UG Resource cutoff grade is 3.0 g/t. • Individual Project Resources and Reserves are stated on an equity basis.

The information in this report that relates to the Swan OC, Eagles Peak, Orion, Deep South, Swan UG, Swift UG, Omega, and Kingfisher Mineral Resources is based on information compiled by or reviewed by Dr Spero Carras (FAusIMM). Dr Carras is the Executive Director of Carras Mining Pty Ltd and was acting as a consultant to Legend Mining Ltd in 2006 and Panoramic Resources Limited in 2012. Dr Carras has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2004 Edition of the JORC Code. Dr Carras consents to the inclusion in the report of the matters based on this information in the form and context in which it appears.

• Heron South Resource cutoff grade is 0.5 g/t • Howards Resource cutoff grade is 0.5 g/t • Specimen Well Resource cutoff grade is 0.5 g/t
• Toedter Resource cutoff grade is 0.5 g/t • Wilsons Resource cutoff grade is 2.0 g/t. • Individual Project Resources and Reserves are stated on an equity basis

The information in this report that relates to the Heron South, Howards, Specimen Well, Toedter and Wilsons Mineral Resources is based on information compiled by or reviewed by Andrew Bewsher (AIG) and Ben Pollard (AIG & MAusIMM). The aforementioned are full time employees of BM Geological Services and have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2004 Edition of the JORC Code. The aforementioned consents to the inclusion in the report of the matters based on this information in the form and context in which it appears.



Mt Henry Project

• Mt Henry Project Resource cutoff grades are 0.4 g/t (previously reported at 1 g/t cutoff). • Individual Project Resources and Reserves are stated on an equity basis

The information in this report that relates to the Mt Henry Project Mineral Resources is based on information compiled by or reviewed by Andrew Bewsher (MAusIMM). Andrew Bewsher is a full time employee of BM Geological Services and has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2004 Edition of the JORC Code. Andrew Bewsher consents to the inclusion in the report of the matters based on this information in the form and context in which it appears.

PGM Resources

Table 4

Thunder Bay North Project				Grade									Metal (oz)	
Resource	Equity	Date of Resource	Tonnage	Pt	Pd	Rh	Au	Ag	Cu	Ni	Co	Pt-Eq	Pt	Pd
				(g/t)	(g/t)	(g/t)	(g/t)	(g/t)	(%)	(%)	%	(g/t)	(oz ,000)	(oz ,000)
Open Pit	**100%**	**Jan-11**												
Indicated			8,460,000	1.04	0.98	0.04	0.07	1.50	0.25	0.18	0.014	2.13	283	267
Inferred			53,000	0.96	0.89	0.04	0.07	1.60	0.22	0.18	0.014	2.00	2	2
Underground	**100%**	**Feb-12**												
Indicated			1,369,000	1.65	1.54	0.08	0.11	2.80	0.43	0.24	0.016	3.67	73	68
Inferred			472,000	1.32	1.25	0.06	0.09	2.10	0.36	0.19	0.011	2.97	20	19
Total (Equity)													377	355

Thunder Bay North

Open Pit Resource

The effective date of this estimate is 11 January 2011, which represents the cut-off date for the most recent scientific and technical information used in the report. The Mineral Resource categories under the JORC Code (2004) are the same as the equivalent categories under the CIM Definition Standards for Mineral Resources and Mineral Reserves (2010). The portion of the Mineral Resource underlying Current Lake is assumed to be accessible and that necessary permission and permitting will be acquired. All figures have been rounded; summations within the tables may not agree due to rounding.

The open pit Mineral Resource is reported at a cut-off grade of 0.59 g/t Pt-Eq within a Lerchs-Grossman resource pit shell optimized on Pt-Eq. The strip ratio (waste:ore) of this pit is 9.5:1. The contained metal figures shown are in situ. No assurance can be given that the estimated quantities will be produced. The platinum-equivalency formula is based on assumed metal prices and overall recoveries. The Pt-Eq formula is: Pt-Eq g/t = Pt g/t + Pd g/t x 0.3204 + Au g/t x 0.6379 + Ag g/t x 0.0062 + Cu g/t x 0.00011 + Total Ni g/t x 0.000195 + Total Co g/t x 0.000124 + Rh g/t x 2.1816. The conversion factor shown in the formula for each metal represents the conversion from each metal to platinum on a recovered value basis. The assumed metal prices used in the Pt-Eq formula are: Pt US$1,595/oz, Pd US$512/oz, Au US$1,015/oz, Ag US$15.74/oz, Cu US$2.20/lb, Ni US$7.71/lb, Co US$7.71/lb and Rh US$3,479/oz. The assumed combined flotation and PlatsolTM process recoveries used in the Pt-Eq formula are: Pt 76%, Pd 75%, Au 76%, Ag 55%, Cu 86%, Ni 44%, Co 28% and Rh 76%. The assumed refinery payables are: Pt 98%, Pd 98%, Au 97%, Ag 85%, Cu 100%, Ni 100%, Co 100% and Rh 98%.

The updated Resources do not include drilling conducted since 31 May 2010. The information in this report that relates to Mineral Resources compiled by AMEC Americas Limited was prepared by Greg Kulla P.Geo (APOG #1752, APEGBC #23492) and David Thomas, P.Geo, MAusIMM (APEGBC #149114, MAusIMM #225250), both full time employees of AMEC Americas Limited. The aforementioned have sufficient experience, which is relevant to the style of mineralisation and type of deposit under consideration and to the activities undertaken to qualify as Competent Persons as defined in the 2004 Edition of the JORC Code and independent qualified persons as this term is defined in Canadian National Instrument 43-101.

Underground Resource

The internal Underground Mineral Resource estimate for the East Beaver Lake extension was made by ordinary kriging methods using the same technical and financial parameters as those used by AMEC Americas Limited for the Underground Mineral Resource estimate reported by Magma Metals limited ("Magma") on 6 September 2010. The Underground Mineral Resource is reported at a cut-off grade of 1.94g/t Pt-Eq. The contained metal figures shown are in situ. The platinum equivalency formula is based on assumed metal prices and recoveries and therefore represents Pt-Eq metal in situ. The Pt-Eq formula is: Pt-Eq g/t = Pt g/t + Pd g/t x 0.2721 + Au g/t x 0.3968 + Ag g/t x 0.0084 + Cu g/t x 0.000118 + Sulphide Ni g/t x 0.000433 + Sulphide Co g/t x 0.000428 + Rh g/t x 2.7211. The assumed metal prices used in the Pt-Eq formula are: Pt US$1,470/oz, Pd US$400/oz, Rh US$4,000/oz, Au US$875/oz, Ag US$14.30/oz, Cu US$2.10/lb, Ni US$7.30/lb and Co US$13.00/lb. The assumed process recoveries used in the Pt-Eq formula are: Pt 75%, Pd 75%, Rh 75%, Au 50%, Ag 50%, Cu 90%, and Ni and Co in sulphide 90%. The assumed smelter recoveries used in the Pt-Eq formula are Pt 85%, Pd 85%, Rh 85%, Au 85%, Ag 85%, Cu 85%, Ni 90% and Co 50%. To account for a portion of the Ni and Co occurring as silicate minerals, Ni and Co in sulphide were estimated by linear regression of MgO to total Ni and total Co respectively. The regression formula for Ni in sulphide (NiSx) is: NiSx = Ni - (MgO% x 60.35 - 551.43). The regression formula for Co in sulphide (CoSx) is: CoSx = Co - (MgO% x 4.45 - 9.25). All figures have been rounded. Summations within the tables may not agree due to rounding. Magma undertook quality assurance and quality control studies on the mineral resource data and concluded that the collar, assay and lithology data are adequate to support resource estimation.



The Mineral Resource categories under JORC are the same as the equivalent categories under CIM Definition Standards (2005). The Mineral Resource has been estimated in conformity with both generally accepted CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice" (2003) guidelines and the 2004 Edition of the JORC Code. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

The information in this report that relates to Mineral Resources compiled internally was prepared by Guoliang Leon Ma P.Geo and Allan MacTavish P.Geo, both full time employees of Panoramic PGMs (Canada) Limited, a wholly owned subsidiary of Panoramic Resources Limited. Both the aforementioned have sufficient experience, which is relevant to the style of mineralisation and type of deposit under consideration and to the activities undertaken to qualify as Competent Persons as defined in the 2004 Edition of the JORC Code and qualified persons as this term is defined in Canadian National Instrument 43-101. The aforementioned consents to the inclusion in the report of the matters based on this information in the form and context in which it appears.

Table 5

Panton Project				Grade					Metal (oz)	
Resource	Equity	Date of Resource	Tonnage	Pt (g/t)	Pd (g/t)	Au (g/t)	Cu (%)	Ni (%)	Pt (oz ,000)	Pd (oz ,000)
Top Reef	**100%**	**Mar-12**								
Measured			4,400,000	2.46	2.83	0.42	0.28	0.08	348	400
Indicated			4,130,000	2.73	3.21	0.38	0.31	0.09	363	426
			1,560,000	2.10	2.35	0.38	0.36	0.13	105	118
Middle Reef	**100%**	**Mar-12**								
Measured			2,130,000	1.36	1.09	0.10	0.18	0.03	93	75
Indicated			1,500,000	1.56	1.28	0.10	0.19	0.04	75	62
			600,000	1.22	1.07	0.01	0.19	0.05	24	21
Total (Equity)									984	1,081

Panton

The information is in this release that relates to the Panton Project Mineral Resource is based on a resources estimate compiled by Ted Copeland who is a Director of Cube Consulting Pty Ltd. and is a Member of the Australian Institute of Mining and Metallurgy. Ted Copeland has more than 10 years' experience which is relevant to the style of mineralisation and type of deposit under consideration and in the activity which he is undertaking and qualifies as a Competent Person as defined in the 2004 Edition of the JORC Code. Ted Copeland consents to the inclusion in the release of the matters based on the information in the form and context in which they appear.

About the Company

Panoramic Resources Limited (ASX Code PAN, ABN 47 095 792 288) is an established Western Australian mining company operating two 100% owned underground nickel sulphide mines, the Savannah Project in East Kimberley, and the Lanfranchi Project near Kambalda, Western Australia. On a Group basis, Panoramic produced 19,561t of nickel contained in FY2013 and is forecasting to produce between **20,000 and 21,000t** of contained nickel in FY2014. Panoramic is an S&P/ASX 300 Index company with a solid balance sheet, no bank debt and a growing nickel, gold and PGM resource base, employing over 400 people (including contractors).

In early 2011, Panoramic acquired the Gidgee Gold Project, located near Wiluna, Western Australia. Panoramic subsequently purchased the high-grade Wilsons Project located within the Gidgee tenement package as well as a 70% interest in the Mt Henry Gold Project. Panoramic released a Scoping Study in August 2012 on the recommencement of gold production from Gidgee and released a positive Scoping Study on the Mt Henry Project in December 2012. Technical studies for the Mt Henry Bankable Feasibility Study have commenced. The Company has expanded into Platinum Group Metals (PGM) with the purchase of the Panton PGM Project located approximately 60km south of the Savannah Project in the East Kimberley and the Thunder Bay North PGM Project in Northern Ontario, Canada.

The Company's vision is to broaden its exploration and production base, with the aim of becoming a major, diversified mining house in the S&P/ASX 100 Index.

For further information contact:
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Managing Director
+61 8 6266 8600

News Release

PJD-50

6 December 2018

PANORAMIC RESOURCES LTD 1936



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ASX:PAN

Lanfranchi Sale Completed

Highlight

- **Settlement completed on the sale of the Lanfranchi Nickel Project to Black Mountain Metals LLC**

Details

On 13 September 2018, Panoramic Resources Limited ("Panoramic" or the "Company") announced that the Company had agreed to sell the Lanfranchi Nickel Project to a wholly owned subsidiary of Texas-based Black Mountain Metals LLC (**"Black Mountain"**) for a total cash consideration of A$15.1 million.

Under the terms of the Sale and Purchase Agreement (**"SPA"**), Black Mountain paid a deposit of A$1.51 million, with A$11.99 million payable upon completion.

The Company is pleased to advise that all conditions precedent to the sale have been satisfied or waived and final settlement has been concluded with Black Mountain with the payment of A$11.99 million being received by the Company.

Panoramic will receive a deferred cash consideration of A$1.6 million to be paid in 12 equal monthly instalments, commencing from the date that is 14 days from the first supply of ore under the current contract with BHP Nickel West Pty Ltd, the processing of ore in another commercial capacity or 1 January 2021, whichever is earlier.

The Company is delighted that Black Mountain has concluded this transaction and wishes Black Mountain every success with the Lanfranchi Nickel Project.

About the Company

Panoramic Resources Limited (**ASX code: PAN**) is a Western Australian mining company formed in 2001 for the purpose of developing the Savannah Nickel Project in the East Kimberley. Panoramic successfully commissioned the $65 million Savannah Project in late 2004 and then in 2005 purchased and restarted the Lanfranchi Nickel Project, near Kambalda. In FY2014, the Company produced a record 22,256t contained nickel and produced 19,301t contained nickel in FY2015. The Lanfranchi and Savannah Projects were placed on care and maintenance in November 2015 and May 2016 respectively pending a sustained recovery in the nickel price.

After delivering an updated feasibility study on the Savannah Project in October 2017, securing an offtake customer and putting in place project financing for up to A$40 million in July 2018, the Company has made the decision to restart operations at Savannah with first concentrate on schedule to be delivered early in the March 2019 quarter.

Apart from the nickel, copper and cobalt inventory at Savannah the Company has a diversified resource base including platinum group metals (PGM) and gold. The PGM Division consists of the Panton Project, located 60km south of the Savannah Project and the Thunder Bay North Project in Northern Ontario, Canada, in which Rio Tinto is earning 70% by spending up to C$20 million over five years. Following the ASX listing of Horizon Gold Limited (ASX Code: HRN) in December 2016, the Company's interest in gold consists of an indirect investment in the Gum Creek Gold Project located near Wiluna through its 51% majority shareholding in Horizon.

At 30 September 2018, Panoramic had $10.8 million in available and restricted cash, and no bank debt.

The Company's vision is to broaden its exploration and production base, with the aim of becoming a major, diversified mining company in the S&P/ASX 100 Index. The growth path will include developing existing resources, discovering new ore bodies, acquiring additional projects and is being led by an experienced exploration-to-production team with a proven track record.

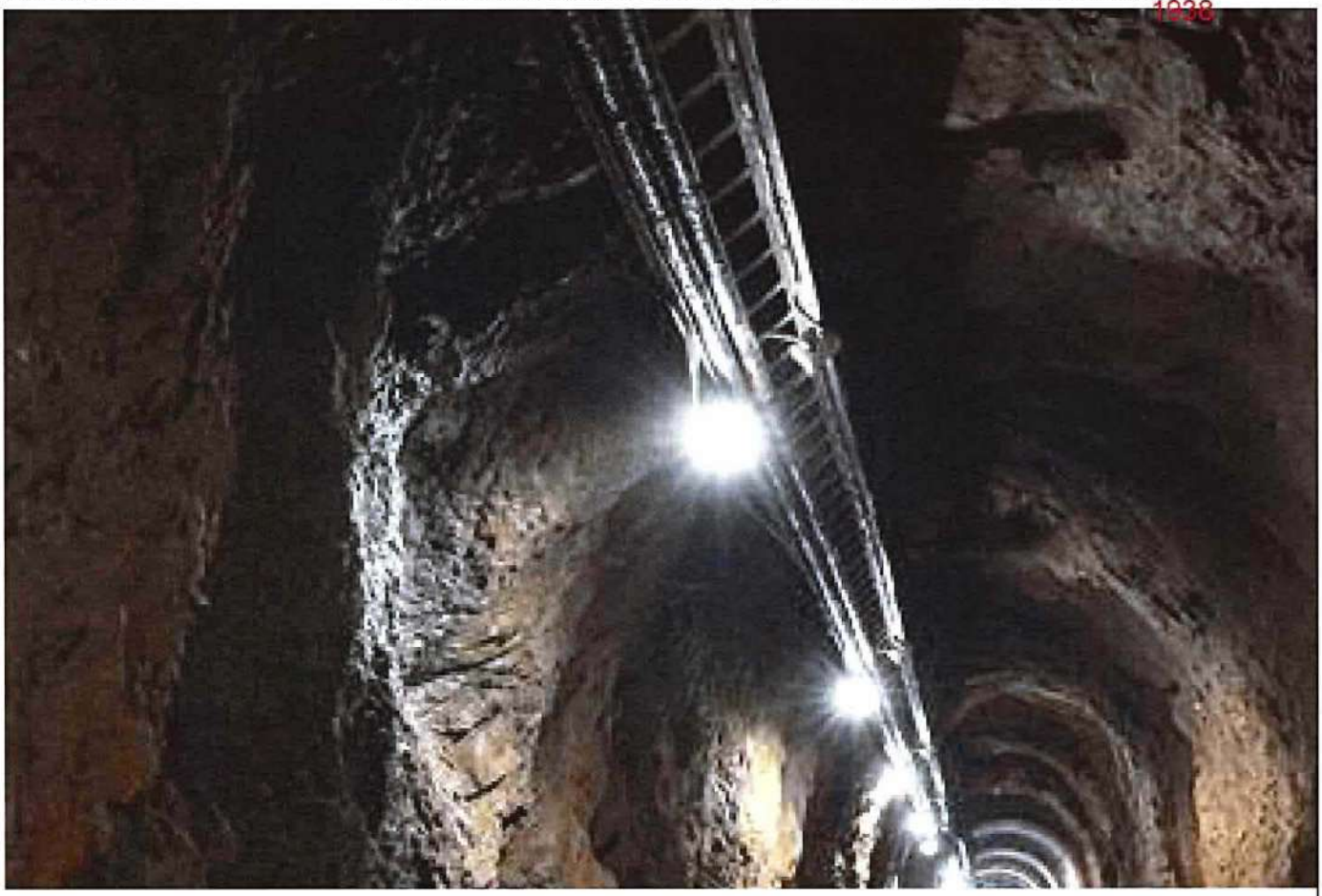
For further information contact:
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PJD-51

PRESS RELEASE

Black Mountain Metals and Tembo Capital Forge Battery Metal Partnership





PERTH, WESTERN AUSTRALIA— (October 05, 2021)

Black Mountain Metals ("Black Mountain") and the Tembo Capital Group ("Tembo Capital") are pleased to announce that the two groups have created a strategic partnership to jointly advance the Lanfranchi nickel sulphide mine in Western Australia.

As part of the transaction, Black Mountain sold a 50% ownership interest in Lanfranchi to Tembo Capital. AJ Saverimutto will be assuming the role of CEO of the new joint Black Mountain-Tembo development company.

Black Mountain President Ashley Zumwalt-Forbes said the agreement is an exciting endeavor for the team and industry. "Lanfranchi is an important future source of nickel ore for the battery metal industry.

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We are excited to bring Tembo on board to further extend the reserve base and advance Lanfranchi towards production to further support the new energy transition."

``We are very excited to be partnering with Black Mountain as significant investors in Lanfranchi. We have a strong focus on nickel as a critical metal in the battery supply chain and we view Lanfranchi as a very attractive, strategic asset.``

DAVID STREET

Tembo Capital CEO

The Lanfranchi Mine is located 40 kilometers south of Kambalda in Western Australia. Between 2005 and 2015, Lanfranchi produced a total of 3.85 million tonnes of high-grade nickel sulphide ore. About 11,000t of nickel per annum was produced at Lanfranchi when last in operation. Lanfranchi hosts a mineral resource of 4.5 million tonnes for over 80,000 tonnes of nickel metal. The mine has been on care and maintenance since 2015. Black Mountain acquired the mine in 2018 from Panoramic Resources Limited.

About Tembo Capital

Tembo Capital is a long-term, mining-focused investment group, which has a strong track record of identifying and supporting emerging resource companies globally. The Tembo Capital team has significant technical expertise, together with mining finance and private equity experience. Tembo Capital has a strong focus on investing in sustainable developments that meet the highest standards of ESG performance. For more details please visit www.tembocapital.com

About Black Mountain

Black Mountain Metals is a global battery metals mining company specializing in spearheading projects from development into production. Black Mountain's objective is to fuel the electric revolution by

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ensuring sustainable and ethical sourcing and mining practices.

Agile and visionary, Black Mountain Metals is well poised to bridge the gap between talent and junior mining asset opportunities, capitalizing on the growing need for battery metals in electric vehicles and other technologies.To learn more, please visit www.blackmountainmetals.com

Press Contacts

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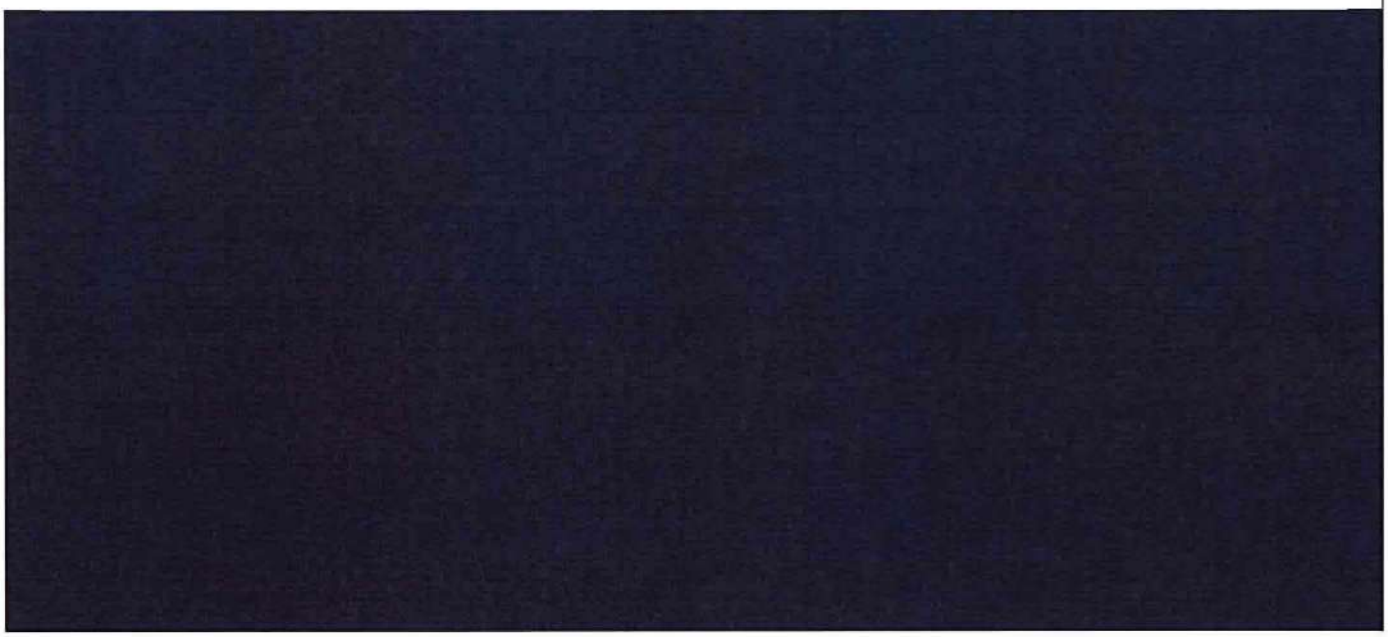
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COMPANY NEWSLETTER

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News Release

PJD-52

30 September 2015

1943 PANORAMIC RESOURCES LTD



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ASX:PAN

MINERAL RESOURCES AND ORE RESERVES AT 30 JUNE 2015

Summary

Panoramic Resources Limited (**ASX Code: PAN**) is pleased to announce the Group's Mineral Resources and Ore Reserves Statement as at 30 June 2015. Key points:

	FY2015	FY2014
Total Nickel Resources	218,600 tonnes	156,600 tonnes
Total Nickel Reserves	45,700 tonnes	53,300 tonnes
Total Copper Resources	68,300 tonnes	34,300 tonnes
Total Copper Reserves	24,600 tonnes	20,200 tonnes
Total Cobalt Resources	7,700 tonnes	2,800 tonnes
Total Cobalt Reserves	2,200 tonnes	1,700 tonnes
Total Gold Resources[1]	2.431 million ounces	2.483 million ounces
Total Platinum and Palladium Resources	2.797 million ounces	2.797 million ounces

[1] On 31 July 2015, the Company sold its 70% interest in the Mt Henry Gold Project. As a result of the sale, as at 30 September 2015, Total Gold Resources is 1.273 million ounces

Savannah Nickel Project (including Copernicus)

At Savannah, the change to the Mineral Resource and Ore Reserve position reflects the addition of the Western Splay mineralisation above the 900 Fault, the Mineral Resource estimate of 14,900t Ni for mineralisation below the 900 Fault and importantly, the Interim Mineral Resource estimate at Savannah North of 55,200t Ni. **The Savannah North Resource estimate at 30 June 2015 was based on 24 underground drill holes covering only approximately 50% of the planned maiden Resource test area.**

The 2015 Copernicus Mineral Resource and Ore Reserve Statement includes mining depletion of the Open Pit Reserve from December 2014.

Lanfranchi Nickel Project

At Lanfranchi, the Mineral Resource and Ore Reserve position reflects mining depletion and sterilisation where necessary. **It should be noted that the Lanfranchi Resource and Reserve Statements do not include any mineralisation from the recent Lower Schmitz discovery.** A maiden Resource estimate for Lower Schmitz will be compiled, interpreted and released during the December 2015 quarter.

Material Information Summary

In accordance with the ASX Listing Rules, a fair and balanced representation of the information provided in Appendix 1 must be presented in the body of the ASX announcement. That representation follows below. This information applies only to the Company's Savannah and Lanfranchi Nickel Projects where material changes to the Mineral Resource and Ore Reserve position occurred during the year due to additions, model updates, mining depletion and sterilisation.

The Mineral Resource and Ore Reserve position of the Company's Gidgee Gold Project reflects the upgrade of the Swan and Swift Resources to 2012 JORC. The 2015 Mineral Resource position for the Company's Panton PGM Project has also been upgraded to 2012 JORC.

Savannah Nickel Project

Drilling and Supporting Data

The Savannah Resource estimate is based almost entirely on data gathered from NQ2 or LTK60 underground diamond drill core. Holes are drilled on a nominal 25m x 25m grid spacing over the extent of mineralisation. Face and sludge-hole sampling data is used as well to refine resource and reserve stope shape outlines. All drill core is photographed, geologically logged, and then halved for sampling. All drill core is spatially orientated to the mine grid by survey control. Down-hole surveys are typically performed every 30m by using either Reflex EZ Shot™ or Flexit Smart Tool™.

Sampling and Assaying

Core sample lengths are typically between 0.2m to 1m long based on logged geological boundaries. Savannah has an on-site laboratory. For core and mining related samples, the standard analytical technique is a 3-acid digest with an AAS finish. This method best approaches total dissolution for most minerals. For exploration samples or other samples (QAQC) sent off-site that are included in the geological database, the analytical technique is 4-acid digest with either ICP OES or AAS finish (typically AAS for high grade ore samples).

Geology and Geological Interpretation

The Savannah Nickel Project mines ore associated with the Savannah Intrusion; a Palaeoproterozoic mafic/ultramafic intrusion (magma conduit). The Ni-Cu-Co rich massive sulphide mineralisation at Savannah occurs as "classic", readily recognisable magmatic breccia-textured ores developed about the more primitive MgO rich basal parts of the intrusion. Panoramic has been mining and exploring the Savannah orebody for over 10 years and has a sound knowledge and understanding of the geology and orientation of the orebody and a high level of confidence in the geological interpretation.

Database

The Savannah geological database is administered on a SQL Server by Panoramic's Database Manager in Perth. Data is captured on-site into Excel™ software templates using laptop computers and uploaded via "Datashed" to the site database, which in turn is automatically replicated to the SQL server in Perth. Regular exports of the database enable the Company's site and Perth based personnel to access the data. Validation of the database is undertaken regularly by Company geologists by plotting the data on plan and cross-sections and through visual 3D inspection using Surpac™ software.

Cut-off Grade

A 0.50% nickel cut-off grade with no minimum mining width is used to define mineralised shapes for resource modelling. This enables the entire mineralised part of the Savannah Intrusion to be encapsulated and available for conversion to Ore Reserve status once the appropriate mining and economic factors are applied. The 0.50% cut-off grade is a natural grade boundary between the magmatic breccia-textured ores and weaker disseminated mineralisation at Savannah.

The cut-off nickel grade used for inclusion in the Reserve was 1.0% Ni Equivalent (approximately 0.85% Ni). An economic assessment based on current costs and a nickel price ranging between ~A$16,890/tonne to ~A$26,450/tonne were used to determine if a mining area could be included as a Reserve. The nickel price ranges were derived after compiling recent forecasts from a cross section of external parties.

Metallurgical and Mining Assumptions

The metallurgical features of the Savannah ore are well understood and have not changed since mining began in 2004. The sulphide ore is processed via flotation to make a bulk Ni-Cu-Co concentrate, grading between 7-8 % Ni and free of any significant deleterious elements. The concentrate is transported to Wyndham and shipped to the Jinchuan Group's smelter/refinery in the Gansu province of north-west China.

The metallurgical nature of the Mineral Resource in this estimate has not changed. Metallurgical recoveries used for Ore Reserve estimations are 86% for nickel, 95% for copper and 88% for cobalt.

Due to the favourable geometry and availability of paste backfill, ore is extracted by sublevel stoping methods. Sublevel mining is restricted to a maximum height of 25m due to the variable nature of the orebody. Ore development is conducted under geological control with face and sludge sampling routinely performed to refine the geological interpretations and stope design. Blast hole drilling is designed to minimise over-break and is set out by survey control.

Mining dilution between 5-20% at zero grade is applied to stope and ore development depending on location in the mine and stope type (primary, secondary or sill pillar). The minimum underground mining width for development is 4.8m and 3.0m for stopes.

Estimation methodology

Ordinary Kriging techniques using Surpac™ software were used to estimate Ni, Co, Cu and density into the Savannah 3D Resource Block Model. Top-cut analysis was undertaken for each Resource domain using grade histograms, but generally no extreme values were detected. Variography was calculated for the domain with the largest sample population and the resultant variogram models adapted for the remaining domains. Check estimates by Panoramic staff using Inverse Distance Squared method yielded similar results to the Ordinary Kriged model. The Savannah Resource Model has been updated periodically since mining began in 2004, with differences in tonnage for successive updates accounted for by new drilling, mining depletion, sterilisation and new resource areas. Grade correlation between updated estimates has remained high.

Classification

The Resource classification system adopted at Savannah is based on the level of confidence as set out in the 2012 JORC Code guidelines. Measured Resources are defined by areas supported by strong drilling and confined up and down dip by mine development such that confidence in lode volume and continuity of grade is very high. Indicated Resources are defined by areas where geological confidence is high and drilling support is strong (equal to or less than 25m x 25m grid spacing). Inferred Resources are typically in areas where the drill spacing is greater than 25m x 25m and geological confidence is lower.

Savannah North Project

The Material Information Summary for Savannah North is essentially the same as other Savannah Resources. However, the Interim Savannah North Mineral Resource estimate contained in the 2015 Panoramic Resource and Reserve Statement differs from Savannah Resource estimates in that the Resource estimate is based on a 50m x 50m grid drill spacing and was estimated using ID^2 methodology. As further drilling is undertaken and future Resource estimates are made, these differences will no longer apply.

Lanfranchi Nickel Project

Drilling and Supporting Data

The Lanfranchi Resource estimates are based almost entirely on data gathered from NQ2 or LTK60 underground diamond drill core. Holes are typically drilled on a regular grid pattern that varies according to the size and consistency of the resource being drilled. Due to the low coefficient of variation of the Lanfranchi Resources nickel grades, Resource definition drilling is more about defining shapes and volume estimation purposes than grade estimation. All drill core was geologically logged and then halved for sampling. All drill core was spatially orientated to the mine grid by survey control. Down-hole surveys are typically performed every 30m by either Reflex Multi-Shot or single shot tools.

Sampling and Assaying

All sampling for Resource estimation purposes at Lanfranchi was based on underground diamond drill core. Sample selection was based on geological core logging with individual samples typically between 0.2m and 1.2m in length.

All Resource drill-hole samples were analysed by the Kalassay Group in their Perth laboratory. The laboratory process for Lanfranchi samples involved reducing each sample by crushing and pulverising to 90% passing 75um. A 0.2g assay aliquot was taken from the pulverised sample and digested by 4-Acid digest and analysed by an ICP-OES instrument. Lanfranchi Certified Reference Material (QAQC) samples are routinely inserted in all sample batches submitted to Kalassay.

Geology and Geological Interpretation

All Lanfranchi Resources belong to the "classic Kambalda style" komatiite hosted nickel sulphide deposit. This deposit type has been extensively studied and mined in the Kambalda area since the late 1960s. Due to this knowledge and history, there is a high level of confidence in the geological interpretation of the Lanfranchi Resources. The strongly contrasting character between mineralised and un-mineralised lithologies is readily apparent and easy to identify.

Database

The Lanfranchi geological database is administered on a SQL Server by Panoramic's Database Manager in Perth. All Lanfranchi drill hole and resource samples are logged and recorded using code restricted Excel™ software templates to ensure that only

approved data can be entered. The templates are uploaded to the SQL drill-hole database via the "Datashed" software. Uploads to the database on site are automatically replicated to the SQL server in Perth. Once Laboratory assay files have been scrutinised and finalised for QAQC, they are imported directly into the database.

Cut-off Grade

All Lanfranchi Resource models were constructed to a nominal 1.0% Ni cut-off grade. No minimum mining width assumptions were made during the resource wireframing or estimation process, but in some areas minor internal dilution was included to avoid over-complication of the wireframe shape and when it was obvious selective mining was not a realistic option. The 1.0% Ni cut-off grade is a natural grade boundary between Lanfranchi's low grade and high grade mineralisation.

The cut-off nickel grade used for inclusion in the Reserve was 1.0%. An economic assessment based on current costs and a nickel price of ~A$16,890/tonne was used to determine if a mining area could be included as a Reserve. The nickel price was derived after compiling recent forecasts from a cross section of external parties.

Metallurgical and Mining Assumptions

The metallurgical features of the Lanfranchi ore types are well understood as the ores have been processed at the Kambalda Nickel Concentrator since the1970s. As such, no new metallurgical studies were required. The Lanfranchi sulphide rich ores are suitable for processing via flotation to make a nickel concentrate. The Lanfranchi ore is processed under an Ore Tolling and Concentrate Purchase Agreement (OTCPA) with BHP Billiton Nickel West (BHPB) at the Kambalda Concentrator located about 40 kilometres to the north-west of Lanfranchi. Under the OTCPA, metallurgical recovery is determined by BHPB and is related to the average grade delivered on the monthly basis.

Due to favourable geometry and availability of paste backfill, sublevel stoping methods are employed to extract the thicker Lanfranchi orebodies, while air-leg mining methods are used in narrower, high-grade ore zones. The hanging wall ultramafic rocks at Lanfranchi are typically weak and therefore all open stopes are extensively cable bolted and all ore development is shotcreted. Ore development is conducted under geological control and is routinely mapped and sampled. Blast-hole drilling is designed to minimise over-break and is set out by survey control.

For all stopes and development drives, a mining dilution factor of 10% at zero grade was applied. The minimum stoping width is 3.0m and 1.8m for air-leg stopes. A 95% metal recovery factor was applied to all stopes and no Inferred Resources were included in the Ore Reserve.

Estimation methodology

The computer software package Surpac™ was used to develop all Lanfranchi Resource models. All resource models have been estimated using Inverse Distance Squared (ID2) methodology for simplicity and the amount of data available. Models that were run between 2007 and 2010 in parallel using ID2 and Ordinary Kriging methodology, produced very similar estimates, typically within ±4% on tonnes, ±0.3% in grade and <5% difference on a contained nickel basis.

Classification

The Resource classification system adopted at Lanfranchi is based on the level of confidence as set out in the 2012 JORC Code guidelines. The classification relies largely on drill density but with increased confidence in areas of ore development. Measured Resources are defined by areas of the resource with adjacent mining or development. Indicated Resources are assigned to areas of high geological confidence supported by a regular, systematic pattern of drilling.

NICKEL - MINERAL RESOURCES AS AT 30 JUNE 2015

Resource	Equity	Metal	Date of Resource	JORC Compliance	Measured		Indicated		Inferred		Total		Metal Tonnes
					Tonnes	Ni (%)	Tonnes	Ni (%)	Tonnes	Ni (%)	Tonnes	Ni (%)	
Savannah Project	100%												
Savannah (above 900 Fault)		Nickel	Jun-15	2012	2,346,000	1.46	927,000	1.67	-	-	3,273,000	1.52	49,700
		Copper				0.81		1.26		-		0.94	30,700
		Cobalt				0.08		0.08		-		0.08	2,700
Savannah (below 900 Fault)		Nickel	Jun-15	2012	780,000	1.64	125,000	1.72	-	-	905,000	1.65	14,900
		Copper				0.76		0.75		-		0.76	6,900
		Cobalt				0.10		0.09		-		0.10	900
Savannah North		Nickel	Jun-15	2012	-	-	-	-	3,155,000	1.75	3,155,000	1.75	55,200
		Copper				-		-		0.78		0.78	24,600
		Cobalt				-		-		0.12		0.12	3,800
Copernicus Open Pit		Nickel	Jun-15	2012	184,000	1.20	-	-	-	-	184,000	1.20	2,200
		Copper				0.74		-		-		0.74	1,400
		Cobalt				0.05		-		-		0.05	100
Copernicus Underground		Nickel	Jul-10	2004	-	-	508,000	1.30	25,000	0.98	532,000	1.29	6,800
		Copper				-		0.91		0.69		0.90	4,800
		Cobalt				-		0.05		0.02		0.05	300
Lanfranchi Project	100%	**Nickel**											
Cruikshank			Apr-11	2004	-	-	2,018,000	1.42	611,000	0.79	2,629,000	1.28	33,600
Deacon			Mar-14	2012	110,000	2.80	-	-	134,000	1.70	244,000	2.19	5,400
Gigantus			Jul-07	2004	-	-	-	-	652,000	1.63	652,000	1.63	10,600
Helmut South			May-14	2012	-	-	-	-	-	-	-		
Helmut South Ext			Apr-14	2012	32,000	3.59	29,000	2.87	-	-	61,000	3.25	2,000
John			Jul-07	2004	-	-	-	-	291,000	1.42	291,000	1.42	4,100
Lanfranchi			Apr-14	2012	50,000	4.12	55,000	4.40	63,000	3.49	167,000	3.98	6,700
Martin			Feb-12	2012	-	-	47,000	3.58	7,000	4.16	54,000	3.66	2,000
McComish			Jul-07	2004	-	-	-	-	992,000	1.49	992,000	1.49	14,800
Metcalfe			Jan-14	2012	-	-	286,000	1.98	111,000	1.35	397,000	1.80	7,200
Schmitz			Jul-13	2012	30,000	4.92	23,000	3.93	16,000	2.95	69,000	4.14	2,900
Winner			Jul-11	2004	-	-	14,000	4.40	-	-	14,000	4.40	600
Total (Equity)		Nickel											218,600
		Copper											68,300
		Cobalt											7,700

Notes:

- *Figures have been rounded and therefore may not add up exactly to the reported totals*
- *All resources are inclusive of reserves*
- *Savannah Project Resource cutoff grade is 0.50% Ni*
- *Copernicus Project Resource cutoff grade is 0.50% Ni*
- *Lanfranchi Project Resource cutoff grade is 1.00% Ni*

Competent Person Statement

The information in this report that relates to Mineral Resources is based on information compiled by or reviewed by Paul Hetherington (MAusIMM) for the Savannah Project Resource and Copernicus Project Resource and Bradley Robinson (MAusIMM) for the Lanfranchi Project Resources. The aforementioned are full-time employees of Panoramic Resources Limited. The aforementioned have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. The aforementioned consent to the inclusion in the release of the matters based on their information in the form and context in which it appears.

NICKEL - ORE RESERVE AS AT 30 JUNE 2015

Reserve	Equity	Metal	Date of Reserve	JORC Compliance	Proven		Probable		Total		Metal Tonnes
					Tonnes	(%)	Tonnes	(%)	Tonnes	(%)	
Savannah Project	**100%**										
Above 900 Fault		Nickel	Jul-15	2012	-	-	2,321,000	1.24	2,321,000	1.24	28,900
		Copper				-		0.79		0.79	18,300
		Cobalt				-		0.06		0.06	1,500
Below 900 Fault		Nickel	Jul-15	2012	-	-	883,000	1.22	883,000	1.22	10,800
		Copper				-		0.57		0.57	5,000
		Cobalt				-		0.08		0.08	700
Copernicus Open Pit		Nickel	Jul-15	2012	-	-	172,000	1.12	172,000	1.12	1,900
		Copper				-		0.74		0.74	1,300
		Cobalt				-		0.05		0.05	100
Lanfranchi Project	**100%**										
Deacon			Jul-15	2012	-	-	57,000	2.53	57,000	2.53	1,400
Metcalfe			Jul-15	2012	-	-	43,000	1.68	43,000	1.68	700
Lanfranchi			Jul-15	2012	-	-	25,000	2.89	25,000	2.89	700
Schmitz			Jul-15	2012	-	-	16,000	3.07	16,000	3.07	500
Helmut Sth Ext			Jul-15	2012	-	-	34,000	2.21	34,000	2.21	800
Total (Equity)		Nickel									45,700
		Copper									24,600
		Cobalt									2,200

Notes:

- *Figures have been rounded and therefore may not add up exactly to the reported totals*
- *All reserves are inclusive of resources*
- *Savannah Project Reserve cutoff grade is 1.0% Ni Equivalent (approximately 0.85% Ni)*
- *Copernicus Project Reserve cutoff grade is 0.50% Ni*
- *Lanfranchi Project Reserve cutoff grade is 1.00% Ni except for airleg mining which is 2.00% Ni*

Competent Person Statement

Information in this report relating to Ore Reserves has been compiled by or reviewed by, Owen Freeth (MAusIMM) for the Savannah Project and Copernicus Project and Lilong Chen (MAusIMM) for the Lanfranchi Project. The aforementioned are full-time employees of Panoramic Resources Limited. The aforementioned have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. The aforementioned consent to the inclusion in the release of the matters based on his information in the form and context in which it appears.

GOLD - MINERAL RESOURCES AS AT 30 JUNE 2015

Resource	Equity	Metal	Date of Resource	JORC Compliance	Measured		Indicated		Inferred		Total		Metal (Au oz)
					Tonnes	Au (g/t)	Tonnes	Au (g/t)	Tonnes	Au (g/t)	Tonnes	Au (g/t)	
Gidgee Project	**100%**	**Gold**											
Swan OC			Jun-15	2012	-	-	2,250,000	2.57	990,000	2.36	3,240,000	2.51	261,100
Heron South			Oct-12	2004	-	-	1,000,000	2.31	136,000	1.41	1,136,000	2.20	80,300
Howards			Jul-13	2012	-	-	5,255,000	1.07	716,000	1.01	5,971,000	1.06	204,000
Specimen Well			Jun-12	2004	-	-	289,000	2.06	72,000	1.79	361,000	2.00	23,200
Toedter			Jun-12	2004	-	-	-	-	661,000	1.62	661,000	1.62	34,400
Eagles Peak			Mar-06	2004	-	-	13,000	3.46	-	-	13,000	3.46	1,400
Orion			Mar-06	2004	-	-	22,000	3.04	-	-	22,000	3.04	2,200
Deep South			Mar-06	2004	-	-	20,000	3.02	-	-	20,000	3.02	1,900
Shiraz			Jul-13	2012	-	-	2,476,000	0.84	440,000	0.76	2,916,000	0.83	77,600
Swan UG			Jun-15	2012	-	-	207,000	8.71	77,000	11.25	284,000	9.40	85,800
Swift UG			Jun-15	2012	-	-	-	-	46,000	10.25	46,000	10.25	15,200
Omega UG			Mar-06	2004	-	-	31,000	9.20	-	-	31,000	9.20	9,200
Kingfisher UG			Mar-06	2004	-	-	390,000	6.80	-	-	390,000	6.80	85,300
Wilsons UG			Jul-13	2012	-	-	2,131,000	5.33	136,000	5.97	2,267,000	5.37	391,500
Mt Henry Project	**70%**	**Gold**											
Selene			Jul-13	2012	-	-	11,491,000	1.17	3,466,000	0.93	14,957,000	1.11	535,900
Mt Henry			Jul-13	2012	-	-	10,487,000	1.27	4,435,000	1.14	14,922,000	1.23	590,800
North Scotia			Jul-13	2012	-	-	250,000	3.11	97,000	1.95	347,000	2.79	31,100
Total (Equity)		Gold			-	-	36,312,000	1.66	11,272,000	1.37	47,584,000	1.59	2,431,000

Notes:

- *Swan OC resource cutoff grade is 0.7 g/t. The resources (both Indicated and Inferred categories) have been partially diluted over a minimum mining width of 2.5m and confined to a A$2,000 Whittle pit shell*
- *Eagles Peak Resource cutoff grade is 1.2 g/t*
- *Orion Resource cutoff grade is 1.3 g/t*
- *Deep South Resource cutoff grade is 1.2 g/t*
- *Swan UG Resource cutoff grade is 4.0 g/t for Indicated resource wireframes near historic workings and 6.0 g/t for Inferred resource wireframes away from historic workings. In transitioning the Swan UG resource from JORC2004 to 2012 in 2015 the Inferred resource cut-off grade has gone from 5.0 to 6.0 g/t Au. The resource is based on an approximate 2.5m minimum vertical mining width.*
- *Swift UG Resource cutoff grade is 6.0 g/t. In transitioning the Swift UG resource from JORC2004 to 2012 in 2015 the Inferred resource cut-off grade has gone from 5.0 to 6.0g/t Au*
- *Omega UG Resource cutoff grade is 3.0 g/t*
- *Kingfisher UG Resource cutoff grade is 3.0 g/t*
- *Individual Project Resources and Reserves are stated on an equity basis*

The information in this release that relates to the Swan OC, Eagles Peak, Orion, Deep South, Swan UG, Swift UG, Omega, and Kingfisher Mineral Resources is based on information compiled by or reviewed by Dr Spero Carras (FAusIMM). Dr Carras is the Executive Director of Carras Mining Pty Ltd and was acting as a consultant to Legend Mining Ltd in 2006 and Panoramic Resources Ltd in 2012. Dr Carras has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2004 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Dr Carras consents to the inclusion in the release of the matters based on their information in the form and context in which it appears.

- *Heron South resource cutoff grade is 0.5 g/t*
- *Howards resource cutoff grade is 0.5 g/t*
- *Specimen Well resource cutoff grade is 0.5 g/t*
- *Toedter resource cutoff grade is 0.5 g/t*
- *Wilsons resource cutoff grade is 2.0 g/t*
- *Individual Project Resources and Reserves are stated on an equity basis*

Competent Persons Statement

The information in this report that relates to the Heron South, Howards, Specimen Well, Toedter and Wilsons Mineral Resources is based on information compiled by or reviewed by Andrew Bewsher (AIG) and Ben Pollard (AIG & MAusIMM). Andrew Bewsher and Ben Pollard are full time employees of BM Geological Services and have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Andrew Bewsher and Ben Pollard consent to the inclusion in the report of the matters based on their information in the form and context in which it appears.

PLATINUM GROUP METALS (PGM) - MINERAL RESOURCES AS AT 30 JUNE 2015

Thunder Bay North Project

Resource	Equity	Date of Resource	JORC Compliance	Tonnage	Grade									Metal (oz)	
					Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Ni (%)	Co %	Pt-Eq (g/t)	Pt (oz ,000)	Pd (oz ,000)
Open Pit	**100%**	**Jan-11**	**2004**												
Indicated				8,460,000	1.04	0.98	0.04	0.07	1.50	0.25	0.18	0.014	2.13	283	267
Inferred				53,000	0.96	0.89	0.04	0.07	1.60	0.22	0.18	0.014	2.00	2	2
Underground	**100%**	**Feb-12**	**2004**												
Indicated				1,369,000	1.65	1.54	0.08	0.11	2.60	0.43	0.24	0.016	3.67	73	68
Inferred				472,000	1.32	1.25	0.06	0.09	2.10	0.36	0.19	0.011	2.97	20	19
Total (Equity)				10,354,000										377	355

Notes - Open Pit Resource:

The effective date of this estimate is 11 January 2011, which represents the cut-off date for the most recent scientific and technical information used in the report. The mineral resource categories under the JORC Code (2004) are the same as the equivalent categories under the CIM Definition Standards for Mineral Resources and Mineral Reserves (2010). The portion of the Mineral Resource underlying Current Lake is assumed to be accessible and that necessary permission and permitting will be acquired. All figures have been rounded; summations within the tables may not agree due to rounding.

The open pit Mineral Resource is reported at a cut-off grade of 0.59 g/t Pt-Eq within a Lerchs-Grossman resource pit shell optimized on Pt-Eq. The strip ratio (waste:ore) of this pit is 9.5:1. The contained metal figures shown are in situ. No assurance can be given that the estimated quantities will be produced. The platinum-equivalency formula is based on assumed metal prices and overall recoveries. The Pt-Eq formula is: Pt-Eq g/t = Pt g/t + Pd g/t x 0.3204 + Au g/t x 0.6379 + Ag g/t x 0.0062 + Cu g/t x 0.00011 + Total Ni g/t x 0.000195 + Total Co g/t x 0.000124 + Rh g/t x 2.1816. The conversion factor shown in the formula for each metal represents the conversion from each metal to platinum on a recovered value basis. The assumed metal prices used in the Pt-Eq formula are: Pt US$1,595/oz, Pd US$512/oz, Au US$1,015/oz, Ag US$15.74/oz, Cu US$2.20/lb, Ni US$7.71/lb, Co US$7.71/lb and Rh US$3,479/oz. The assumed combined flotation and PlatsolTM process recoveries used in the Pt-Eq formula are: Pt 76%, Pd 75%, Au 76%, Ag 55%, Cu 86%, Ni 44%, Co 28% and Rh 76%. The assumed refinery payables are: Pt 98%, Pd 98%, Au 97%, Ag 85%, Cu 100%, Ni 100%, Co 100% and Rh 98%.

The updated resources do not include drilling conducted since 31 May 2010.

The information in this report that relates to Mineral Resources compiled by AMEC Americas Limited was prepared by Greg Kulla P.Geo (APOG #1752, APEGBC #23492) and David Thomas, P.Geo, MAusIMM (APEGBC #149114, MAusIMM #225250), both full time employees of AMEC Americas Limited. Mr. Kulla and Mr. Thomas have sufficient experience, which is relevant to the style of mineralisation and type of deposit under consideration and to the activities undertaken to qualify as Competent Persons as defined in the 2004 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" (the JORC Code) and independent qualified persons as this term is defined in National Instrument 43-101.

Notes - Underground Resources:

Underground Mineral Resource Estimates: The internal mineral resource estimate for the East Beaver Lake extension was made by ordinary kriging methods using the same technical and financial parameters as those used by AMEC Americas Limited for the underground mineral resource estimate reported by the Company on 6 September 2010. The underground mineral resource is reported at a cut-off grade of 1.94g/t Pt-Eq. The contained metal figures shown are in situ. The platinumequivalency formula is based on assumed metal prices and recoveries and therefore represents Pt-Eq metal in situ. The Pt-Eq formula is: Pt-Eq g/t = Pt g/t + Pd g/t x 0.2721 + Au g/t x 0.3968 + Ag g/t x 0.0084 + Cu g/t x 0.000118 + Sulphide Ni g/t x 0.000433 + Sulphide Co g/t x 0.000428 + Rh g/t x 2.7211. The assumed metal prices used in the Pt-Eq formula are: Pt US$1,470/oz, Pd US$400/oz, Rh US$4,000/oz, Au US$875/oz, Ag US$14.30/oz, Cu US$2.10/lb, Ni US$7.30/lb and Co US$13.00/lb. The assumed process recoveries used in the Pt-Eq formula are: Pt 75%, Pd 75%, Rh 75%, Au 50%, Ag 50%, Cu 90%, and Ni and Co in sulphide 90%. The assumed smelter recoveries used in the Pt-Eq formula are Pt 85%, Pd 85%, Rh 85%, Au 85%, Ag 85%, Cu 85%, Ni 90% and Co 50%. To account for a portion of the Ni and Co occurring as silicate minerals, Ni and Co in sulphide were estimated by linear regression of MgO to total Ni and total Co respectively. The regression formula for Ni in sulphide (NiSx) is: NiSx = Ni - (MgO% x 60.35 - 551.43). The regression formula for Co in sulphide (CoSx) is: CoSx = Co - (MgO% x 4.45 - 9.25). All figures have been rounded. Summations within the tables may not agree due to rounding. Magma undertook quality assurance and quality control studies on the mineral resource data and concluded that the collar, assay and lithology data are adequate to support resource estimation. The mineral resource categories under JORC are the same as the equivalent categories under CIM Definition Standards (2005). The mineral resource has been estimated in conformity with both generally accepted CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice" (2003) guidelines and the JORC Code (2004). Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Competent Persons Statement

The information in this release that relates to Mineral Resources compiled internally by Panoramic was prepared by Mr. Guoliang Leon Ma P.Geo and Mr. Allan MacTavish P.Geo, both full time employees of Panoramic PGMs (Canada) Limited, a wholly owned subsidiary of Panoramic Resources Limited. Both Mr. Ma and Mr. MacTavish have sufficient experience, which is relevant to the style of mineralisation and type of deposit under consideration and to the activities undertaken to qualify as Competent Persons as defined in the 2004 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" (the JORC Code) and qualified persons as this term is defined in National Instrument 43-101. Mr. Ma and Mr. MacTavish consent to the inclusion in the release of the matters based on this information in the form and context in which it appears.

PLATINUM GROUP METALS (PGM) - MINERAL RESOURCES AS AT 30 JUNE 2015

Panton Project

Resource	Equity	Date of Resource	JORC Compliance	Tonnage	Grade					Metal (oz)	
					Pt (g/t)	Pd (g/t)	Au (g/t)	Ni (%)	Cu (%)	Pt (oz ,000)	Pd (oz ,000)
Top Reef	**100%**	**Mar-12**	**2012**								
Measured				4,400,000	2.46	2.83	0.42	0.28	0.08	348	400
Indicated				4,130,000	2.73	3.21	0.38	0.31	0.09	363	426
Inferred				1,560,000	2.10	2.35	0.38	0.36	0.13	105	118
Middle Reef	**100%**	**Mar-12**	**2012**								
Measured				2,130,000	1.36	1.09	0.10	0.18	0.03	93	75
Indicated				1,500,000	1.56	1.28	0.10	0.19	0.04	75	62
Inferred				600,000	1.22	1.07	0.10	0.19	0.05	24	21
Total (Equity)				14,320,000	2.19	2.39	0.31	0.27	0.08	984	1,081

Notes

Competent Persons Statement

The information is in this release that relates to the Panton Mineral Resource is based on a resources estimate compiled by Mr. Rick Adams who is a Competent Person and Member of the Australian Institute of Mining and Metallurgy. Rick Adams is a Director and full time Principal Consultant at Cube Consulting Pty Ltd ("Cube"). Mr. Adams has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and in the activity which he is undertaking and qualifies as a Competent Person as defined in the 2012 edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Mr Adams consents to the inclusion in the release of the matters based on his information in the form and context in which it appears.

It is the opinion of Cube that with the addition of the information required under the 2012 JORC Code, the estimated mineral Resources reported in 2003 can be re-stated in accordance with 2012 JORC.

Appendix 1

JORC Code 2012 Edition - Compliance Tables

Table 1 - Savannah Nickel Mine

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The deposit was sampled by diamond drilling techniques. Over 1500 holes have been drilled for a total in excess of 220,000m. The majority of holes were drilled from underground drill platforms. • The drillhole spacing is a nominal 25 x 25 metre grid spacing over the extent of the mineralization. • All drillhole collars were surveyed using Leica Total Station survey equipment by a registered surveyor. Downhole surveys were typically performed every 30 metres using either "Reflex EZ Shot" or "Flexit Smart Tools". • All diamond core was geologically logged with samples (typically between 0.2 metre to 1 metre long) defined by geological contacts. Analytical samples included a mix of full and sawn half core samples. Sample preparation included pulverising to 90% passing 75 µm followed by total 4 acid digest and analysis by ICP OES.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• A mix of LTK60 and NQ2 sized diamond drilling has been used to obtain >90% of the data used in the estimate. Some RC drilling has been used historically for the upper part of the resource.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• Diamond core recoveries are logged and recorded in the database. Overall recoveries are >99% and there are no apparent core loss issues or significant sample recovery problems. • Depths checked against core blocks, regular rod counts, driller breaks checked by fitting core together. • No relationship exists between sample recovery and grade
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• All holes have been geologically logged in full. Geotechnical logging was carried out on all diamond drillholes for recovery and RQD. Number of defects (per interval) and roughness was carried out around the ore zones. Structure type, alpha angle, infill, texture and healing is stored in the structure table of the database. • Logging of diamond core RC samples recorded lithology, colour, mineralisation, structural (DDH only) and other features. Core was photographed wet. • All drillholes were logged in full.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of	• Analytical core samples included a mix of full and sawn half core samples. • All samples from core • All core sampling and sample preparation followed industry best practice. • QC involved the addition of Savannah derived CRM assay standards, blanks, and duplicates. At least one form of QC was inserted in most sample batches. • Original versus duplicate assay results have always shown strong correlation due to massive sulphide rich nature of the orebody. • Sample sizes are considered appropriate to represent

Criteria	JORC Code explanation	Commentary
	the material being sampled.	the Savannah style of mineralisation.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.	• Total 4 acid digest (hydrochloric/perchloric/hydrofluoric/nitric) and analysis by ICP OES is the Savannah Nickel Mine (SNM) standard analytical technique. The method best approaches total dissolution for most minerals.
	• For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	• No other analytical tools or techniques are employed.
	• Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• The onsite laboratory carries out sizing checks, uses internal standards, duplicates, replicates, blanks and repeats. A selection of roughly 10% of pulps was sent to external laboratories for repeat analysis and sizing checks. No bias has been identified.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel.	• Drilling and sampling procedures at SNM have been inspected by many stakeholders since the project began.
	• The use of twinned holes.	• Throughout the life of the mine, there have been several instances where holes have been twinned, confirming intersections and continuity.
	• Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	• Holes are logged into Excel templates on laptops, data is then entered into MS Access database with user data entry front end built in. Data is ultimately transferred to SQL server from Perth office. Data periodically validated by site personnel.
	• Discuss any adjustment to assay data.	• No adjustments have been made to assay data.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	• All diamond drillhole collars were surveyed using Leica Total Station survey equipment by a registered surveyor. "Reflex EZ Shot" or "Flexit Smart Tool" was used for downhole surveys at approximately every 30m. Visual inspection in a 3D graphics environment using "Surpac" software failed to identify any obvious errors regarding the spatial position of drillhole collars or downhole surveys
	• Specification of the grid system used.	• The mine grid is a truncated 4 digit (MGA94) grid system.
	• Quality and adequacy of topographic control.	• Topographic control is of a high quality and is adequate for the resource estimation process
Data spacing and distribution	• Data spacing for reporting of Exploration Results.	• Nominal drillhole spacing of 25m (easting) by 25m (RL)
	• Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	• The mineralized domains delineated by the drill spacing show enough continuity to support the classification applied under the 2012 JORC Code.
	• Whether sample compositing has been applied.	• No sample compositing has been undertaken.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.	• Drillhole orientation was largely perpendicular to the orebody with the exception of the western extent where drill platform positions allowed only for oblique intersections.
	• If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• No orientation sampling bias has been identified.
Sample security	• The measures taken to ensure sample security.	• Samples transported to onsite lab by Panoramic staff. Samples sent off site are road freighted (Nexus transport) and tracked using spreadsheets onsite.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits/reviews of the sampling techniques have been undertaken in recent time. The procedures used are considered to be industry standard. Mine to mill reconciliation records throughout the life of the Savannah Project provide confidence in the sampling procedures.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. • Data validation procedures used.	• An Excel™ software logging template with lookup tables and fixed formatting is used for logging and data collection. • Data validation checks are performed every time a drillhole is entered to the database using a checklist.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The competent person is a site based, full time employee of Panoramic.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. • Nature of the data used and of any assumptions made. • The effect, if any, of alternative interpretations on Mineral Resource estimation. • The use of geology in guiding and controlling Mineral Resource estimation. • The factors affecting continuity both of grade and geology.	• The confidence of the geological interpretation is high which has been confirmed by mapping and 9 years of operational experience. • No other interpretations have been considered as the current model is demonstrably robust. • Geological controls were used to create the domains, namely, lithology, massive sulphide content, major structures • One of the main domains is affected by 2 major cross-cutting mafic dykes, the geometry and thickness of which are well understood.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The resource is 350m along strike (east), varies in thickness from 1 to 50m and averages 8m thick, from the surface to 900m depth.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. • The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • The assumptions made regarding recovery of by-products. • Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation). • In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. • Any assumptions behind modelling of selective mining units. • Any assumptions about correlation between variables. • Description of how the geological interpretation was used to control the resource estimates. • Discussion of basis for using or not using grade cutting or capping.	• Ordinary Kriging was employed using Surpac™ software to estimate Ni, Co, Cu and Density into a 3D block model. Top cut analysis was undertaken for each domain using grade histograms, no extreme values were detected and therefore no top cuts applied. Variography was calculated for the domain with the largest sample population and the resultant variogram models were adapted for the remaining domains. • Check estimates by Panoramic staff using Inverse Distance squared method has yielded similar results. The estimate has been updated periodically since mining began in 2004, differences in tonnage for each successive update have been accounted for by new drilling, depletion for mining, and new resource areas. Grade correlation between updated estimates has always remained high. • By-product credits for Copper and Cobalt form part of the off-take agreement between Panoramic and Jinchuan. • No deleterious elements have been modeled in the resource estimate; the Savannah orebody has low MgO and negligible Arsenic levels. • All block estimates were based on interpolation into 4m N x 20m E x 10m RL parent cells, sub celling to 0.5m N x 2.5m E x 1.25m RL. Block discretisation points were set to 2(Y) x 5(X) x 4(Z) points. The block dimensions are over half the average drill spacing of 25m. A search radius of 150m was used with a minimum of 8 samples and a maximum of 50 samples for all domains. • No selective mining units were assumed in the estimate. • Nickel and cobalt show a very strong correlation. Nickel and copper are much more variable. • The geological interpretation was used to derive the domains using massive sulphide content, lithology and structural boundaries. These were wireframed and used as hard boundaries to flag sample data for estimation. • Statistical analysis of the grade populations indicated no

Criteria	JORC Code explanation	Commentary
	• The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	extreme values and a low coefficient of variation. • Validation included comparing the raw data statistics to block estimates, volumes of wireframes to block model volumes, drillholes and block model value plots were produced for a visual check of the grades. Good reconciliation data exists between mined and milled figures.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• Tonnages estimated on a dry basis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• 0.5%Ni was used as a cut-off when defining the mineralised wireframes. Generally, this is the grade boundary between strongly disseminated sulphides and the ultramafic footwall unit.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• Mining at Savannah has been ongoing since 2004. Underground, sub-level open stoping is used effectively to extract the ore. No further assumptions were made on mining factors. Mining factors are applied during Ore Reserve conversion.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• Savannah ore has been successfully treated through a 1MTPA SAG mill and flotation circuit since commissioning in 2004. The metallurgical nature of the mineral resource in this estimate has not changed. Metallurgical factors are addressed in Ore Reserve conversion.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• SNM operates under the conditions set out by an environmental license to operate.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.	• Bulk density determinations are routinely performed. Most determinations involved calculating the core volume and weighing the core in air. Regular checks using the water immersion technique were also carried out. A regression analysis of measured density versus nickel is used to populate missing density values.
	• The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.	• Voids within the mineralized zones are non-existent
	• Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• The search parameters for density were the same as nickel for all domains. Waste material was assigned a value of 2.88, determined from the regression formula.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity	• The classification adopted is based on the level of confidence as set out in the JORC 2012 guidelines. • Measured Resources are defined by areas supported by strong drilling and confined up and down dip by mine development such that confidence in lode volume and continuity and grade is very high. Indicated Resources

Criteria	JORC Code explanation	Commentary
	and distribution of the data).	are defined by areas where geological confidence is high and drilling support is strong (equal to or greater than 25m x 25m grid spacing).
	• Whether the result appropriately reflects the Competent Person's view of the deposit.	• The estimate appropriately reflects the view of the competent person.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• The resource estimate has been peer reviewed on site and by Panoramic's corporate technical team.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.	• The relative accuracy of the resource estimate is considered robust as it has been compiled as per the guidelines of the 2012 JORC Code, and knowledge gained from extensive operational history of the mine.
	• The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.	• The statement relates to global estimates of tonnes and grade.
	• These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• Mine to mill reconciliation records throughout the life of the Savannah Project provide confidence in the accuracy of the resource estimate.

Section 4 - Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve. • Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.	• All resources that have been converted to reserve are classified as either an Indicated or Measured Resource. • Indicated Resources are only upgraded to Probable Reserves after adding appropriate modifying factors. • The Measured Resource is classified as Proved Mining Reserves once the access drives to the mineral resource is fully developed. • Mineral Resources are inclusive of Ore Reserves.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The competent person is a site based full time Panoramic employee on a rostered FIFO arrangement.
Study status	• The type and level of study undertaken to enable Mineral Resources to be converted to OreReserves. • The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.	• Underground Mining continues since January 2005. Following exploration and infill drilling activity, annual resource updates and economic assessment of the measured and indicated resources is completed using actual costs, operating parameters and modifying factors. • An annual update of Ore Reserves is completed on this basis.
Cut-off parameters	• The basis of the cut-off grade(s) or quality parameters applied.	• The cut-off grade used for inclusion in the Reserve is 1% Ni equivalent based on economic assessment and current operating and market parameters.
Mining factors or assumptions	• The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design). • The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc. • The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling. • The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate). • The mining dilution factors used.	• The Savannah mine predominantly applies an open stoping with paste to fill voids. • Mining dilution at zero grade is applied to the stopes and the ore development. • A minimum mining width of underground development is 4.8m. • A minimum width of stoping is 3.0m

Type	Dilution	Mining recovery
Development	10%	95%
Above 500 fault -Primary and Secondary stopes	10%	95%
-Sill pillar under paste	20%	95%

Criteria	JORC Code explanation	Commentary
	• The mining recovery factors used. • Any minimum mining widths used. • The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion. • The infrastructure requirements of the selected mining methods.	<table><tr><td>-Pit Pillar</td><td>20%</td><td>75%</td></tr><tr><td>Below 500 fault -Primary and Secondary stopes -Sill Pillars</td><td>10% 10% to 15%</td><td> 95% 75 to 90%</td></tr><tr><td>Below 900 fault -All stopes</td><td>10%</td><td>95%</td></tr></table>• No Inferred resources are included within either the Reserve or the mine plan.
Metallurgical factors or assumptions	• The metallurgical process proposed and the appropriateness of that process to the style of mineralisation. • Whether the metallurgical process is well-tested technology or novel in nature. • The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied. • Any assumptions or allowances made for deleterious elements. • The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole. • For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?	• The Savannah produces a Ni/Cu/Co concentrate of grade between 7 to 8 % Ni. The concentrate is transported to Wyndham, then shipped to Jinchuan's smelter/refinery in the Gansu province, northwest China. • Savannah ore has been successfully treated through a 1MTPA SAG mill and flotation circuit since commissioning in 2004. The metallurgical nature of the mineral resource in this estimate has not changed. • Metallurgical Recoveries used for Ore Reserve estimations are 86% for Nickel, 95% for Cu and 88% for Co.
Environmental	• The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.	• SNM operate under the conditions set out by an environmental license to operate. • Waste is generally placed on approved waste dumps, some amounts are placed underground in the mined voids when possible.
Infrastructure	• The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.	• The Savannah mine has substantial infrastructure in place including a paste fill plant, major electrical and pumping networks, a 1Mtpa throughput plant, a fully equipped laboratory, extensive workshop, administration facilities and a 215 person single person quarters camp, and tailings storage
Costs	• The derivation of, or assumptions made, regarding projected capital costs in the study. • The methodology used to estimate operating costs. • Allowances made for the content of deleterious elements. • The derivation of assumptions made of metal or commodity price(s), for the principal minerals and co-products. • The source of exchange rates used in the study. • Derivation of transportation charges. • The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc. • The allowances made for royalties payable, both Government and private.	• Costs are actual costs occurred to the mining, processing and transportation over the past 12 months; • Metal Prices and Exchange Rate are based on the latest future consensus median values forecast by a number of external market analysts. • All cost based on Australian dollars; • WA government royalty and Traditional Owner royalty included.
Revenue factors	• The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc. • he derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.	• Revenue is calculated by using the latest future consensus median prices of Ni, Cu and Co, and US$:A$ exchange rate forecast by a number of external market analysts
Market assessment	• The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future. • A customer and competitor analysis along with the identification of likely market windows for the product.	• The Concentrate is contracted for sale to the Jinchuan Group of China until April 2020. The Savannah concentrate will continue to be shipped from Wyndham to Jinchuan's smelter/refinery in the Gansu province, northwest China.

Criteria	JORC Code explanation	Commentary
	• Price and volume forecasts and the basis for these forecasts. • For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.	
Economic	• The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc. • NPV ranges and sensitivity to variations in the significant assumptions and inputs.	• As an operating mine, internal cash flow estimates and impairment models apply an implied 8% real discount rate for NPV analysis and only economically viable ores are considered for mining. The mine carries no external debt forces. • Sensitivity analysis of key financial and physical parameters is applied to future development project considerations and mine.
Social	• The status of agreements with key stakeholders and matters leading to social licence to operate.	• The Savannah mine is fully permitted and a major contributor to the local and regional economy. It has no external pressures that impact its operation or which could potentially jeopardise its continuous operation.
Other	• To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves: • Any identified material naturally occurring risks. • The status of material legal agreements and marketing arrangements. • The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	• No significant unresolved material matters relating to naturally occurring risks, third party agreements or governmental/statutory approvals currently exist.
Classification	• The basis for the classification of the Ore Reserves into varying confidence categories. • Whether the result appropriately reflects the Competent Person's view of the deposit. • The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).	• The classification adopted is based on the level of confidence as set out in the 2012 JORC guidelines • Proven Ore reserves are based on Measured Resources subject to economic viability. • Probable Ore Reserves are based on Indicated Resources subject to the economic viability. • The estimate appropriately reflects the view of the competent person. • No inferred material is included in the Reserve estimations.
Audits or reviews	• The results of any audits or reviews of Ore Reserve estimates.	• Site generated reserves and the parent data and economic evaluation data is routinely reviewed by the Savannah technical team.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage. • It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be	• The relative accuracy of the Resource and Reserve estimates are considered robust as it has been compiled as per the guidelines of the 2012 JORC Code, and knowledge gained from extensive operational history of the mine. • All currently reported Reserve estimations are considered representative on a global scale. • Mine to mill reconciliation records throughout the life of the Savannah Project provide confidence in the accuracy of the resource and reserve estimates.

Criteria	JORC Code explanation	Commentary
	compared with production data, where available.	

Table 1 - Savannah North

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• Exploration and resource definition holes at Savannah North are entirely diamond cored holes. Most are drilled from underground. The deposit to date has been defined by 24 surface and UG exploration holes, totalling 20,150m. UG resource definition holes completed to 30 June 2015 total 30 holes for 10,386m. • The Resource definition drill hole spacing is a nominal 50 x 50 metre grid spacing over the extent of the Resource reported in the release accompanying this Table. • All drill hole collars were surveyed using Leica Total Station survey equipment by a registered surveyor. Down-hole surveys are typically performed every 30 metres using either "Reflex EZ Shot" or "Flexit Smart Tools". • All diamond core is geologically logged with samples (typically between 0.2 metre to 1 metre long) defined by geological contacts. Analytical samples are dominantly sawn half core samples. Sample preparation includes pulverising to 90% passing 75 µm followed by either a 3 acid digest & AAS finish at the Savannah onsite laboratory or a total 4 acid digest with an ICP OES finish if the samples are analysed off-site.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• NQ2 sized diamond drilling has been used to obtain 100% of the data used in the estimate.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• Diamond core recoveries are logged and recorded in the database. Overall recoveries are >99% and there are no apparent core loss issues or significant sample recovery problems. • Depths checked against core blocks, regular rod counts, driller breaks checked by fitting core together. • No relationship exists between sample recovery and grade
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• All diamond holes have been geologically logged in full. Geotechnical logging is carried out on all diamond drill holes for recovery and RQD. Number of defects (per interval) and roughness was carried out around the ore zones. Structure type, alpha angle, infill, texture and healing is recorded in most holes and stored in the structure table of the database. • Recorded core logging attributes include lithology, colour, mineralisation, structural and other features. • All drill core is photographed.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half	• Analytical core samples are dominantly sawn half NQ2 samples. • All Resource definition samples are diamond core only. • All core sampling and sample preparation follow industry best practice. • QC involves the addition of purchased CRM and Savannah derived CRM assay standards, blanks, and duplicates. At least one form of QC is inserted in most sample batches. • Original versus duplicate assay results have always shown strong correlation due to massive sulphide rich nature of the Savannah North mineralisation.

Criteria	JORC Code explanation	Commentary
	sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• Sample sizes are considered appropriate to represent the Savannah North style of mineralisation.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. • For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• The Savannah Nickel Mine (SNM) onsite laboratory standard analytical technique is a 3-acid digest with an AAS finish. The method best approaches total dissolution for most minerals The onsite exploration sample analytical method for Ni,Cu,Co is AAS 22S. Exploration samples sent off-site are analysed using a 4-acid digest with either ICP OES or AAS finish (AAS for ore grade samples). • No other analytical tools or techniques are employed. • The onsite laboratory is run by SGS Laboratory Services • The onsite laboratory carries out sizing checks, uses internal standards, duplicates, replicates, blanks and repeats. A selection of roughly 10% of pulps was sent to external laboratories for repeat analysis and sizing checks. No bias has been identified.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel. • The use of twinned holes. • Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. • Discuss any adjustment to assay data.	• Drilling and sampling procedures at SNM have been inspected by many stakeholders since the project began. • The practice of twinning holes is not employed at Savannah North. • Holes are logged into Excel templates on laptops. The data is then entered into a SQL server database via a DataShed front end. Data is then replicated to the Perth office. Data periodically validated by site personnel. • No adjustments have been made to assay data.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. • Specification of the grid system used. • Quality and adequacy of topographic control.	• All diamond drill hole collars are surveyed using Leica Total Station survey equipment by a registered surveyor. "Reflex EZ Shot" or "Flexit Smart Tool" was used for down-hole surveys at approximately every 30m. • The mine grid is a truncated 4 digit (MGA94) grid system. • Conversion from local grid to MGA GDA94 Zone 52 is calculated by applying truncated factor to local coords: • E: +390000, N: +8080000N • Topographic control is well established, RL equals AHD + 2,000m.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. • Whether sample compositing has been applied.	• Exploration drill holes are spaced on a geological basis as opposed to a nominal drill hole spacing. • For the most part drilling is typically conducted on a regular spacing, sufficient to achieve the objectives of the drill program. For the current Savannah North Resource definition program the nominal spacing is 50m x 50m. • The mineralised domains delineated by the drill spacing show enough continuity to support the classification applied under the 2012 JORC Code. • No sample compositing has been undertaken.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• The geometry of the Savannah and Savannah North mineralisation to most drill positions is nearly always oblique. For this reason all SNM drill results are reported as down-hole intersection lengths and not true widths. • No orientation sampling bias has been identified.
Sample security	• The measures taken to ensure sample security.	• Samples transported to onsite lab by SNM staff. Samples sent off site are road freighted (Nexus transport) and tracked using spreadsheets onsite.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits/reviews of the sampling techniques have been undertaken in recent time. The procedures used are considered to be industry standard. Mine to mill reconciliation records throughout the life of the Savannah Project provide confidence in the sampling procedures.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. • Data validation procedures used.	• An Excel logging templates with lookup tables and fixed formatting is used for logging and data collection. • Data validation checks are performed every time a drill hole is entered to the database using a checklist.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The Competent Person is a site based, full time employee of the Company.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. • Nature of the data used and of any assumptions made. • The effect, if any, of alternative interpretations on Mineral Resource estimation. • The use of geology in guiding and controlling Mineral Resource estimation. • The factors affecting continuity both of grade and geology.	• The Savannah North mineralisation dips moderately (40-45 degrees) to the north-west and comprises two main domains, the first domain is associated with the basal contact of the North Olivine Gabbro, the second domain is remobilised massive sulphide mineralisation detached from the contact. Both domains are well defined by the drilling and the interpretation is considered robust. • No other interpretations have been considered as the current model is demonstrably robust. • Geological controls were used to create the domains. The interpretation has been defined by the presence of strong and continuous zones of massive sulphide mineralisation. • The detached domain of remobilised massive sulphide mineralisation is related to the NW dipping 500 Fault zone. • There are some instances where intervals of internal dilution have been included with the mineralized envelope- generally less than 0.5m
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The Savannah North mineralisation has been defined over a strike length of 1 kilometre. The interim resource reported herein relates to an area with a strike length of 300m from 5700mE to 6000mE. The average thickness of mineralisation is approximately 5 meters.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. • The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • The assumptions made regarding recovery of by-products. • Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation). • In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. • Any assumptions behind modelling of selective mining units. • Any assumptions about correlation between variables.	• Inverse Distance squared methodology was employed using Surpac™ software to estimate Ni, Co, Cu and Density into a 3D block model. Top cut analysis was undertaken for each domain using grade histograms, no extreme values were detected and therefore no top cuts applied. A search radius of 125m was used for the main basal domain, with a minimum of 3 samples and a maximum of 15 samples. • No check estimates have been performed to date. This estimate has yielded similar characteristics of previous Savannah estimates. • By-product credits for Cu and Co form part of the off-take agreement between SNM and Jinchuan. • No deleterious elements have been modelled in the resource estimate; the Savannah orebody has low MgO and negligible Arsenic levels. • A block model was created using Surpac software with parent cell dimensions of 4m N x 20m E x 10m RL parent cells, sub celling to 0.5m N x 2.5m E x 1.25m RL. Block discretisation points were set to 2(Y) x 5(X) x 4(Z) points. The block dimensions approach half the average drill spacing of 50m. • No selective mining units were assumed in the estimate. • Ni and Co show a very strong correlation. Ni and Cu are more variable.

Criteria	JORC Code explanation	Commentary
	• Description of how the geological interpretation was used to control the resource estimates.	• The geological interpretation was used to derive the domains using massive sulphide content, lithology and structural boundaries. These were wireframed and used as hard boundaries to flag sample data for estimation.
	• Discussion of basis for using or not using grade cutting or capping. • The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• Statistical analysis of the grade populations indicated no extreme values and a low coefficient of variation. • Validation included comparing the raw data statistics to block estimates, volumes of wireframes to block model volumes, drill holes and block model value plots were produced for a visual check of the grades. • Similar validation methods have been proven to be reliable at Savannah where good reconciliation data exists between mined and milled figures.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• Tonnages estimated on a dry basis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• 0.5%Ni was used as a cut-off when defining the mineralised wireframes.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• Mining at Savannah has been ongoing since 2004 • Underground, sub-level open stoping is used effectively to extract the ore at Savannah and a similar extraction method is likely for Savannah North. • As the interim Savannah North has been categorised entirely as **Inferred** no further mining assumptions are warranted. • Mining factors will be applied during Ore Reserve conversion.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• Savannah ore has been successfully treated through a 1Mtpa SAG mill and flotation circuit since commissioning in 2004. • The metallurgical nature of the mineral resource in this estimate has not changed. • Metallurgical factors are addressed in Ore Reserve conversion. • Preliminary testwork conducted on the Savannah North mineralisation has indicated that it has identical metallurgical characteristics as Savannah ore.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• The Savannah Project operates under the conditions set out by an environmental license to operate. • At this stage It is likely that extraction of the Savannah North Resource will be undertaken under the same license conditions
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples.	• Bulk density is determined using the water displacement method for all Savannah North assay samples.
	• The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit.	• Voids within the mineralised zones are non-existent
	• Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• The search parameters for density were the same as nickel for all domains. A default bulk density of 2.88 was assigned to waste material. The default density value was determined from a well-established regression formula.

Criteria	JORC Code explanation	Commentary
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).	• The classification adopted is based on the level of confidence as set out in the 2012 JORC guidelines.. • Because drilling is ongoing and additional infill holes may be drilled the entire Resource has been classified as 'Inferred'
	• Whether the result appropriately reflects the Competent Person's view of the deposit.	• The estimate appropriately reflects the view of the Competent Person.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• The Resource estimate has been peer reviewed on site and by Panoramic's corporate technical team.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.	• The relative accuracy of the Resource estimate is considered robust as it has been compiled as per the guidelines of the 2012 JORC Code, and knowledge gained from extensive operational history at Savannah.
	• The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.	• The statement relates to global estimates of tonnes and grade.
	• These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• Mine to mill reconciliation records throughout the life of the Savannah Project provide confidence in the accuracy of the resource estimate.

Table 1 – Lanfranchi Nickel Mine

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• All sampling for resource estimation purposes at Lanfranchi Nickel Mine (LNM) is based on diamond drill core. Sample selection is based on geological core logging. Individual samples typically vary between 0.2m and 1.2m in length.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• Diamond drilling at LNM is typically NQ2 or LTK60 size. Occasionally BQ and HQ core size holes have been drilled.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• All recovered diamond core is metre marked by on site geologists; any core loss is determined and recorded as part of the geological logging process. Core recovery is typically 100 percent. • No relationship exists between core recovery and grade.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• All core is geologically and geotechnically logged to a standard appropriate for mineral resource estimation purposes. Core is logged from start to end of hole without gaps. Core photography is not undertaken. Drillholes are logged using Excel templates that are code restricted to ensure that only approved data can be entered. The Excel templates are then uploaded to the Lanfranchi SQL Server drillhole database via Datashed.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• All diamond core is cut using a clipper brick saw and half core sampled for assay. Quarter core samples are sent as part of the LNM QAQC process for check assaying. Sample intervals typically vary between 0.2m and 1.2m and are positioned as to not cross geological boundaries.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.	• All LNM drillhole samples are analysed by Kalassay Group's Kalgoorlie laboratory. The Laboratory process for LNM samples involves: Crush sample to <3mm, pulverise to 90% passing 75um (lab blanks introduced and pulverised at this point). From the pulverised sample, a 0.2g assay aliquot is taken and weighed then

Criteria	JORC Code explanation	Commentary
		digested by 4-Acid digest and analysed by ICP-OES instrument. Laboratory QA/QC is performed on standards, blanks and duplicates. The LNM policy is to scrutinize the results for QA/QC standards and blanks when assay jobs are reported and to request re-runs if result are ± 1SD from the expected value.
	• For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	• No other geophysical or analytical tools have been used to estimate grade.
	• Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• Certified Reference Material (QAQC) samples are routinely inserted during all sampling at LNM. In addition samples are routinely sent for check analysis at a different Laboratory. The QAQC results indicate that the diamond core assays being used for resource estimation at LNM are a fair representation of the material that has been sampled.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel.	• Significant intersections are calculated by mine geologists and verified/reported on a monthly basis by the Geology Manager.
	• The use of twinned holes.	• Twinning of drillholes is not performed at LNM
	• Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	• Assay data are imported directly from the Kalassay assay files and QA/QC validated via Datashed to the LNM SQL drillhole database.
	• Discuss any adjustment to assay data.	• No adjustment to assay data is made.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	• Drillhole collars are accurately surveyed for X,Y,Z and azimuth & Dip by site Surveyors using "Total Station" control. Older holes may/may not have collar azimuth/dip measurements. Downhole surveys are generally conducted using single shot or reflex multishot tools at 15m, 30m and every 30m thereafter.
	• Specification of the grid system used.	• The LNM drillhole database contains both MGA94 and local mine grid (KNO) coordinates. All site geological and mine planning work is performed in the local KNO grid system. • Conversion from KNO grid to MGA GDA94 Zone 51 is based on a two point transformation: 389084.61E, 513790.88N = 389351.47E, 6513980.38N 389044.77E, 513543.54N = 389313.70E, 6513732.77N
	• Quality and adequacy of topographic control.	
Data spacing and distribution	• Data spacing for reporting of Exploration Results.	• LNM resource estimation drill holes are typically drilled on a regular grid spacing that varies according to the size and consistency of the resource being drilled. Due to the consistent grade and low Coefficient of Variation of nickel mineralisation generally, resource definition drilling at LNM is more for volume estimation purposes than grade estimation.
	• Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	• Data spacing is deemed to be sufficient for Mineral Resource estimation and reporting.
	• Whether sample compositing has been applied.	• No sample compositing is undertaken; all core samples are logged and analysed in full.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• Underground drill sites are not always ideally positioned for resource definition drilling however no sampling orientation bias is evident. The Ni grade is typically very consistent within individual resource domains and therefore drill orientation is not a determinant for reliable grade estimation
Sample security	• The measures taken to ensure sample security.	• All diamond core samples are taken directly from site to Kalassay for analysis via a local courier service. Sample security is considered adequate.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• All LNM Mineral Resource estimates are audited by independent consultants BM Geological Services. Minor adjustments to model dimensions, geostatistical analysis

Criteria	JORC Code explanation	Commentary
		and application of top-cuts (where required) and adjustments to search parameters have been made on occasions following this audit process.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. • Data validation procedures used.	• All LNM drillhole and resource samples are logged and recorded using MS Excel templates that are code restricted to ensure that only approved data can be entered. The Excel templates are uploaded to the LNM SQL drillhole database via Datashed software, this also ensures only approved data can be entered into the database. Once Laboratory assays files have been scrutinised and finalised for QAQC they are imported directly into database to ensure there are no transcription errors.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The competent person is a site based Panoramic employee on a rostered, FIFO arrangement • BM Geological Service personnel have visited LNM on numerous occasions.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. • Nature of the data used and of any assumptions made. • The effect, if any, of alternative interpretations on Mineral Resource estimation. • The use of geology in guiding and controlling Mineral Resource estimation. • The factors affecting continuity both of grade and geology.	• There is a high level of confidence in the geological interpretation of all LNM resources due to; the extensive operating experience, and the readily recognizable, strongly contrasting mineralised and un-mineralised lithologies. Composites are individually selected for each drillhole based on logging and cut-off grade boundaries rather than using an intercept method. This method ensures where drillholes skim in and out of mineralisation along a resource edge the mineralised grades for the hole are used in the estimation process even though due to the complexity of the wireframe interpretation they may fall just outside the wireframe shape. The standard composite length is 1m. Wireframes are based on drillhole intercepts, survey pick-ups, face mapping and sludge sampling where available. Although rock chip, grab and sludge hole data is also available, they are not used in the estimation process to ensure clustering of lower quality does not bias the estimation process, as such only diamond drilling samples are used for estimation. Wireframes are used to constrain the estimation process to ensure rigid geological boundaries are adhered to. All wireframes are constructed to a 1.0% Ni cut-off grade
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. • The availability of check estimates, previous estimates	• The computer software package "Surpac" was used for all resource models. All resource models have been estimated using ID2 methodology for simplicity and the amount of data available. Previous models run between 2007 and 2010 had run ID2 and OK models in parallel. This approach produced very similar estimates that were, typically within ±4% on tonnes, ±0.3% in grade and <5% difference on a contained nickel basis. Confidence in the

Resource	Length	Width	Depth below surface (limit)	
			Upper	Lower
Deacon	915	115	850	1315
Helmut South Extension	185	40	935	1160
Lanfranchi	185	80	630	820
Metcalfe	285	40	810	935
Martin	160	40	275	440

Criteria	JORC Code explanation	Commentary
	and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • The assumptions made regarding recovery of by-products. • Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation). • In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. • Any assumptions behind modelling of selective mining units. • Any assumptions about correlation between variables. • Description of how the geological interpretation was used to control the resource estimates. • Discussion of basis for using or not using grade cutting or capping. • The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	modelling process is also high due to good reconciliation with production data over many years, especially within the Deacon resource. Block model parent cell sizes are matched to drillhole spacing for each resource model with search ellipses aligned parallel to mineralisation trends. Estimated elements in all resource models are as follows: Ni, As, Cu, Co, Fe, MgO, S. Domaining within the model is based upon primary lithology types and separate estimation passes are conducted to ensure both composites and search ellipses are realistic for each rock type. In the case of the Helmut South Extension (HSE) resource, geostatistical analysis of the ore population identified two distinct sample populations. Due to the complexity of the two populations within the main ore shape, they were unable to be domained separately. In this case a top-cut was applied to ensure that overestimation of the resource did not occur. The top-cut was established following an audit of the HSE resource model by BM Geological Services. Minimal stoping has been completed in the HSE resource to adequately study the estimation versus actual data. No other top-cuts were applied across the other resource models. Validation of the resource estimate was completed by onscreen visual validation of block grades vs. drillhole assays in sectional view and via Swanson plots of composites grades vs. block grades. In the case of the Deacon resource, the model composites and block model reports were compared in 100m increments along strike with very good correlation between "data in" and "data out".
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• Tonnages are estimated on a dry basis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• All LNM resource models are modelled to a nominal cut-off grade of 1% Ni. In some cases, minor internal dilution was included in the intercept to avoid over-complication of the wireframe shape.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• No minimum mining width assumptions are made during the resource wireframing or estimation process. Mining parameters, including minimum mining width assumptions are applied during the conversion to Ore Reserves. Internal dilution is included during the resource estimation process when it is obvious selective mining is not a realistic option.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• No metallurgical factors or assumptions are made during the Resource estimation process. These matters are addressed during conversion to Ore Reserve.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these	• No environmental factors or assumptions are made during the Resource estimation process.

Criteria	JORC Code explanation	Commentary
	potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• For all LNM Resources a nickel vs. SG correlation plot is generated using all available SG data. All samples without an SG measurement are assigned a calculated SG value based on the regression analysis. During the estimation process, actual SG measurements were given priority over calculated values. LNM has an extensive SG database, generated over many years of operating experience. Determinations have typically been performed using the water immersion technique. The technique is adequate due to low core porosity, fresh rock underground environment of the LNM operation.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• Classification of the LNM resource models are based primarily on drill density in conjunction with increased confidence from existing ore development. Significant ore development has been completed within the Deacon Resource which enables a Measured category to be applied to most parts of this Resource.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• In March 2013, BM Geological Services (BMGS) conducted an independent review of all LNM Resource estimates. The review covered the building and interpolation of grades in the relevant block models and their representation of grades based on the composite files. Overall BMGS concluded the resource models grades compared favourably with drill hole composite grades. Concern was expressed however about the representivity of the Helmut South Extension (HSE) Resource estimation, and the potential for overcalling of grades within the BM. To alleviate this issue, top-cuts were applied by BMGS and the model re-run.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• A high level of confidence exists for all LNM Resource estimates. The estimates are based on many years of operating experience. Mine production is currently derived from 3 of the 6 underground resources with good reconciled agreement between Resource/Reserve grades and mine production. • All estimates are global estimates of tonnes and grade. • **Deacon** – In production with good reconciled agreement between Resource/Reserve grades and mine production. • **Helmut South Extension** - Estimated grades generally reflect composite grades, grade trends are under reported within the Swanson plot due to the application of Ni top-cuts. 2013/14 production numbers will assist in reconciling performance of the block model against composites. • **Lanfranchi** - In production with good agreement between Resource/Reserve grade and mine production. • **Martin** - estimated grades closely reflect grades within the Swanson plot, ore too thin for proper analysis of composites in cross section. • **Metcalfe** - Estimated grades generally reflect composite grades, grade trends appear consistent with composites in cross section. • **Schmitz** - In production with good reconciled agreement between Resource/Reserve grade and mine production.

Section 4 - Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve. • Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.	• All resources used in the estimation of ore reserves are classified as either Indicated or Measured. • Indicated Resources are only upgraded to Probable Reserves after adding appropriate modifying factors. • Measured Resources are classified as Proven Mining Reserves once the access drives are fully developed. • Mineral resources are inclusive of the Ore Reserves.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The Competent Person is a site based full time LNM employee on a rostered FIFO arrangement.
Study status	• The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves. • The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.	• Mining operations have continued since Panoramic acquired the Lanfranchi project from WMC Resources Limited in 2004. • Annual Resource updates and economic assessment of the Measured and Indicated Resources is completed using actual costs, operating parameters and modifying factors. • An annual update of Ore Reserves is completed on this basis.
Cut-off parameters	• The basis of the cut-off grade(s) or quality parameters applied.	• An initial cut-off grade of 1% Ni is used except for airleg mining which is 2% Ni • Economic analysis is carried out for each planned stope, and only stopes with a positive return are included in the Ore Reserves.
Mining factors or assumptions	• The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design). • The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc. • The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling. • The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate). • The mining dilution factors used. • The mining recovery factors used. • Any minimum mining widths used. • The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion. • The infrastructure requirements of the selected mining methods.	• The Deacon and HSE Ore has been extracted by sub-level stoping due to favourable geometry and the availability of paste backfill. • Narrow high grade ore zones are mined using air-leg mining methods. • The hanging wall rocks are ultramafic and very weak, therefore all open stopes are extensively cable-bolted, all ore development is shotcreted and stope voids are filled with paste fill. Regular inspections are made by appropriately qualified geotechnical staff. A seismic monitoring system is in place and routinely checked. • Stress models are regularly updated and results used in determining stable stope sizes. • Ore development is routinely mapped and sampled. • Mining dilution is 10% at zero grade and is applied to all stopes; • Mining dilution is 4% at zero grade and is applied to the jumbo ore development; • 95% metal recovery is assigned to all open stopes. • 99% metal recovery is assigned to all jumbo ore development. • Minimum stoping width is 3.0m. • Minimum air-leg stope width is 1.8m. • No Inferred Resources are included in the Ore Reserves. • The paste fill plant is designed to produce suitable quality and quantities of paste to match the underground production rate.

Criteria	JORC Code explanation	Commentary
Metallurgical factors or assumptions	• The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.	• LNM ore is a sulphide and suitable for processing via flotation to make a concentrate. As such the ore is processed under an Ore Tolling and Concentrate Purchase Agreement (OTCPA) with BHP Billiton Nickel West (BHP). The Kambalda Concentrator is located about 40 kilometres to the north-west of LNM by road. The offtake agreement with BHP operates until 2019. Recovery is determined by BHP and is related to the averaged grade delivered on the monthly basis. The ore from LNM has been processed by previous mine operators at the Kambalda Concentrator since the 1970s.
	• Whether the metallurgical process is well-tested technology or novel in nature. • The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.	
	• Any assumptions or allowances made for deleterious elements. • The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole. • For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?	• There are insignificant amounts of deleterious elements in the current Resources.
Environmental	• The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.	• LNM operates under the conditions set out by an environmental license to operate. • There are no outstanding current environmental approvals requiring supporting environmental studies. • Waste rock is inert basalt and classified as NAF. • Waste is placed on approved surface waste dumps, with some amounts placed underground when possible.
Infrastructure	• The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.	• LNM is an operating mine, and has substantial mine infrastructure in place, including a paste fill plant, major electrical and pumping networks, administration facilities and a 150 person accommodation village.
Costs	• The derivation of, or assumptions made, regarding projected capital costs in the study.	• Most capital costs are related to the renewal of existing mining equipment and are based on quotes from equipment suppliers.
	• The methodology used to estimate operating costs.	• The forecast of operating costs is based on the detailed operating history from the last 12 months. • Processing costs are based on the OTCPA with BHP.
	• Allowances made for the content of deleterious elements. • The derivation of assumptions made of metal or commodity price(s), for the principal minerals and co-products. • The source of exchange rates used in the study.	• Metal prices and US$:A$ Exchange rate are the latest future consensus median values forecast by a number of external market analysts.
	• Derivation of transportation charges.	• All costs are in Australian dollars; • The forecast of transportation costs is based on detailed transportation costings from the last 12 months.
	• The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc.	• Payability and penalties for delivering off-spec ore are defined in the OTCPA. Based on operating history and the extremely low-levels of deleterious elements any occurrence of such penalties is considered highly unlikely.
	• The allowances made for royalties payable, both Government and private.	• WA government royalties are included in cost estimates.
Revenue factors	• The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc. • The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.	• The head-grade is based on the mine plan and the geological model (with suitable modifying factors applied) • Revenue is calculated using the latest future consensus median prices for Ni and Cu, and the US$:A$ exchange rate forecast by a number of external market analysts
Market assessment	• The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to	• The OTCPA to purchase the ore is in place until 2019.

Criteria	JORC Code explanation	Commentary
	affect supply and demand into the future. • A customer and competitor analysis along with the identification of likely market windows for the product. • Price and volume forecasts and the basis for these forecasts. • For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.	
Economic	• The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc. • NPV ranges and sensitivity to variations in the significant assumptions and inputs.	• As an operating mine internal cash flow estimates and impairment models apply an implied 8% real discount rate for NPV analysis and only economically viable ores are considered for mining. LNM has no external debt. • Sensitivity analysis of key financial and physical parameters is undertaken as part of the Ore Reserve process
Social	• The status of agreements with key stakeholders and matters leading to social licence to operate.	• Licence to operate from WA State Government. • Pre native title mining tenements for current Reserves. • Good relationships with local community and strong cooperation with neighbouring mining operations.
Other	• To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves: • Any identified material naturally occurring risks. • The status of material legal agreements and marketing arrangements. • The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	• No significant unresolved material matters relating to either naturally occurring risks, third party agreements or governmental/statutory approvals, currently exist. • Third party off-take agreement in place with BHP
Classification	• The basis for the classification of the Ore Reserves into varying confidence categories. • Whether the result appropriately reflects the Competent Person's view of the deposit. • The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).	• The classification adopted is based on the level of confidence as set out in the 2012 JORC guidelines • Proven Ore Reserves are based on Measured Resources subject to economic viability. Probable Ore Reserves are based on Indicated Resources subject to economic viability. • The LNM Ore Reserve reflects the view of the competent person(s). • No inferred resources are used for Ore Reserve estimation.
Audits or reviews	• The results of any audits or reviews of Ore Reserve estimates.	• Site generated reserves and resources and economic evaluation data is routinely reviewed by Panoramic's in-house technical mining team.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the	• The relative accuracy of the resource and reserve estimates are considered robust as they has been compiled as per the guidelines of the 2012 JORC Code, and knowledge gained from extensive operational history at the mine, including good production reconciliation history. There is extensive data available due to the requirements of the OTCPA under which all ore is required to be weighed, and 25% of the ore is processed in a separate sampling circuit, including crushing and assaying. • All currently reported reserve estimations are considered representative on a global scale.

Criteria	JORC Code explanation	Commentary
	current study stage. • It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• Mine to mill reconciliation records throughout the life of the Lanfranchi Project provide confidence in the accuracy of the resource and reserve estimates.

Table 1 – Gidgee Gold Project

Howards

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The Howards deposit has been extensively sampled using Reverse Circulation (RC) and Diamond drilling (DD) techniques. The Howards (Gidgee Gold Project) resource database subset contains 237 RC & DD holes for a total of 19,730m. Of this total 75 RC & DD holes totalling 11,086m were drilled by Panoramic as part of a Project feasibility study. In addition the database contains 329 historical RAB holes (totalling 3,173m which have not been used for resource estimation. • The drill spacing is typically 20m * 30m and 40m * 40m grid spacing over the extent of the mineralisation. • RC holes were sampled by collecting 1m samples and splitting these down to a ~3kg assay sample using either automated on-board rig cone splitters or manual riffle splitters. • Diamond holes were either NQ2 or HQ size and were sampled by cutting the core in half to honour geologically logged intervals between 30cm and 1m in length. • All (7,056) recent Panoramic resource assay samples were submitted to ALS Laboratories in Perth for gold analysis by FA30 (Fire Assay) technique. Of the 5,261 historical RC & DD gold assays in the Howards database, 3,108 (59%) have an un-known technique. 2,565 of these assays are described as "unknown digest, AAS finish" and are believed to be Fire Assay results. In addition, results for 6,423 (52% of the entire analytical database) QAQC samples are recorded in the database.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• The drilling methods use to evaluate the deposit are RC and DD drilling. The RC drilling was typically completed utilising a 5 ¼ inch hammer. Face sampling RC hammers were used by Panoramic. The DD drilling was either NQ/NQ2 (47.6mm /50mm) or HQ (63.5mm) diameter core. • HQ size core was typically used for geotechnical holes cored from surface by Panoramic.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• RC sample recovery was monitored by Panoramic by recording visual estimates of the sampling bags. Typical recoveries for RC were greater than 90%. • Core recovery is noted during the drilling and geological logging processes as a percentage recovered vs. expected drill length. Core was reconstructed into continuous runs on lengths of angle iron to enable accurate geological logging and estimation of core recovery. Recovery was typically 100%. • No apparent relationships were noted between sample recovery and grade.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• All drill holes in the Howards resource database have been geologically logged. • Both chip and core samples in recent Panoramic drill holes have been logged using geological legends at detail to support geological confidence in Mineral Resource estimates. • Logging details lithology, weathering, oxidation, veining, mineralisation and structural features where noted in drill core. • All mineralised intersections and associated samples have been logged in full.

Criteria	JORC Code explanation	Commentary
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• All diamond core was half core sampled using an electric diamond core saw. • All RC samples were collected in 1m intervals through the drill rig cyclone system and reduced to a ~3kg assay sample by either automated on-board cone splitters or manually by riffle splitting. • The sample preparation process for all samples submitted for analysis followed industry standards, including oven drying for a minimum of 8 hours, crushing and pulverising to 85% passing 75 microns. • Quality control procedures included the insertion of standards, blanks and field duplicates to monitor sampling and analytical processes. In addition ALS Laboratories in Perth conducted their own internal QAQC system. • The sample sizes used are industry accepted standards used extensively throughout the goldfields and are appropriate to the style of deposit.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. • For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• The analytical technique used is Fire Assay (30g charge). Where other element determinations were made it was generally by 4 acid digest and either ICP OES or AAS technique. • No other geophysical or analytical tools have been used to estimate grade. • QAQC has been routinely completed during all sampling. The QAQC results indicate the assays being used for resource estimation are a fair representation of the material being sampled. The Panoramic QAQC process was to insert 1 Certified Reference Material (CRM) or blank for every 20 RC samples and between 1 in 15 and 1 in 20 for core samples. The CRM quartz wash blank was also inserted at the beginning of each core assay batch and where possible immediately prior to the mineralised intervals. • Quarter core field duplicates were submitted at a rate of 1 in 20 samples. For RC drilling, field duplicates were inserted at a rate of 1 in 25 samples. Coarse crush laboratory split duplicates were also inserted at the rate of 1 in 20 samples for both RC and DD drilling.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel. • The use of twinned holes. • Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. • Discuss any adjustment to assay data.	• The deposit is continuous in terms of mineralisation and grade. The continuity and consistency of the grade intercepts down dip and along strike provide strong confidence in the verification of the grade and style of deposit. • No twin holes were completed. Verification holes were completed by Panoramic to test continuity of mineralisation in selected sections. The drilling confirmed expected geological and mineralogical interpretations. • Logging was completed in logging code protected excel templates and loaded into Panoramic's SQL database for validation. Sections were then generated and visual validation was completed to ensure integrity of the data. • No adjustments were made to assay data except for replacing negative (below detection reported results) with half detection limit numerical values.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	• All drill hole set-outs, pickups and collar alignments were undertaken by TEAMS Surveying using DGPS equipment with a horizontal accuracy of ±10 mm and a vertical accuracy of ±15 mm. Down-hole surveys were routinely performed every 30m using a range of single shot, electronic multi-shot and north seeking gyro tools.

Criteria	JORC Code explanation	Commentary
		Panoramic validated all down hole survey data to correct anomalous readings due to magnetic interference. Recent gyroscopic surveys undertaken by Panoramic confirmed the reliability of earlier single and multi-shot readings.
	• Specification of the grid system used.	• The grid system used in the resource estimate is MGA_GDA94 Zone 50. All historic drilling positions were originally located on the Howards truncated AMG grid system that was constructed by Dalrymple in 1989. Panoramic has adopted MGA94 as the survey system for the Howards Project. The Howards database contains both sets of coordinates, but for the purpose of this estimate the MGA94 grid coordinates have been used. • Conversion from local grid to AMG AGD84 Zone 50 is calculated by applying truncated factor to local coords: • E: +700000, N: +6900000
	• Quality and adequacy of topographic control.	• The Howards topographic layer was created by Panoramic using a 2006 Landgate aerial survey and modified by DGPS pickups of historical and current drill-hole collars.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. • Whether sample compositing has been applied.	• The drilling density is on a nominal 20m by 30m and then 40m by 40m spacing through the majority of the deposit. This spacing is sufficient to provide strong geological and mineralogical confidence in the style of the deposit being estimated. • Outside of the mineralised zones and through RC pre-collars, 3 metre composite spear samples were collected and submitted for assay. If any of these returned anomalous gold values (> 0.2g/t) then the original 1 metre (~3kg) cone split drill-rig samples were submitted from the respective composites. All cored intervals selected for analysis were cut and sampled accordingly and sent directly to the laboratory. No core sample compositing was undertaken.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• All drilling has been completed roughly perpendicular to the main strike of the deposit geometry and at angle to intercept mineralisation as close to perpendicular as possible. • No sampling bias is apparent from the direction of drilling.
Sample security	• The measures taken to ensure sample security.	• Samples were kept secure on site until dispatched direct to the ALS laboratory in Perth.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits or review of the Panoramic sampling procedures and protocols has been completed.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.	• Logging was completed in logging code protected MS Excel templates and loaded into Panoramic's SQL database, with a "Datashed" software frontend, for validation and storage. • Geological and assay information was printed on hard copy plans and visually validated against original logs and assay results to ensure the digital copy agrees with the original format.
	• Data validation procedures used.	• Data validation was completed internally in SQL Server by setting allowable and expected values. Automated queries are run as the data is imported to ensure it meets specified criteria. • A subset of the SQL database, restricting the data to the Howards Resource area, was exported into MS Access database. Additional data checks were run to ensure appropriate data robustness for the Resource Estimation.

Criteria	JORC Code explanation	Commentary
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits.	• No site visits were completed by BMGS. LNM Panoramic staff managed the 2013 drill program and were integral in the development of mineralisation interpretations used in the Ordinary Kriged model.
	• If no site visits have been undertaken indicate why this is the case.	• Site visits were not required as the documented procedures on the recent drilling were deemed appropriate for the style of deposit and the work was undertaken to industry accepted standards.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.	• There is a high degree of confidence in the geological interpretation of the Howards deposit. The deposit is confined to a basalt hosted shear, which has good continuity at a 0.3g/t cut off. The uncut coefficient of variation (COV) of the dataset was 4.31, however this was heavily skewed by the 6 extreme values. By top-cutting the dataset a reduction of 1.39 to the COV was realised, which suggests the domains are acceptable.
	• Nature of the data used and of any assumptions made.	• The data used for the resource estimate was from RC and diamond drilling. Raw assays, typically 1m were composited to 2m to provide equal sample weights and reduce grade variance.
	• The effect, if any, of alternative interpretations on Mineral Resource estimation.	• Two separate interpretations were undertaken which were used for two differing estimation techniques. An interpretation of 0.3g/t continuity was created for the OK estimate with a slightly broader approach at 0.2g/t (including larger zones of internal dilution) for the MIK estimate.
	• The use of geology in guiding and controlling Mineral Resource estimation.	• The geological and mineralisation models are based on detailed geological logging which confirms the concentration of Au mineralisation within a broad basalt hosted shear zone. • With 14 diamond holes in the deposit, combined with detailed geological logs on all other hole types, it is thought that there is sufficient detail to support the geological model (framework).
	• The factors affecting continuity both of grade and geology.	• The geological and grade continuity is typical of most gold deposits where the continuity at a lower grade cut-off is far greater than the higher grade thresholds. There is a presence of localised higher grade zones within the mineralised domains. The continuity of these high grade zones vary from good continuity in the heart of the deposit and dissipate on the margins of the deposit.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The Howards mineralised domain is approximately 780m long and has a down dip extent of 200m and is open at depth. The deposit consists of a main lode that varies between 2m and 30m thick with numerous parallel & sub-parallel lodes at various stages along the length of the deposit.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.	• Grade estimation of Howards deposit was completed using two separate methods; (1) Ordinary kriging (OK) in Surpac software and (2) Multiple Indicator Kriging (MIK) using GS3 software. Variogram analysis and modelling for the OK estimate was completed using Supervisor software. Two meter composites were generated from the drillhole database and then tagged to mineralised wireframes generated at a 0.3g/t gold grade cut-off. The wireframe modelling conditions included, a minimum downhole width of 2 meters of mineralisation and internal dilution of up to 3m downhole could be included if the entire intercept graded above 0.3 g/t. If dilution was greater than 3m then separate lodes were generated if geological/grade continuity was permissible (typically used trouser legs on the fringes of the main lode). • The data was reviewed through disintegration analysis and reviewing the raw statistics to determine whether applying a top-cut was necessary. It was decided a top-cut was required to reduce the high CV and limit the effect of these higher grades on the estimate. Only the

Criteria	JORC Code explanation	Commentary
		OK estimate was run using a top-cut, with a top-cut of 10g/t being applied (removing the top 6 outliers from the dataset). The dataset was then normal scores transformed to generate variogram models. The variogram models had moderate to low nuggets with a range of maximum continuity along the main axis of 32m. Separate variograms were also generated for the MIK estimate, based on the different ranked grade thresholds; metal variograms were also generated.
	• The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	• A previous estimate of the Howards deposit was completed in 2012. This was completed by BMGS on behalf of Panoramic Resources and was not classed as being 2012 JORC compliant. The 2012 estimate was created using a slightly higher cut-off envelope at 0.4g/t Au and is not therefore directly comparable with the 2013 estimates. Comparisons were made between two recent estimates using different estimation methods. The comparisons demonstrate that the metal accumulation between models is generally comparable with a slight fall in the MIK estimate. This is most likely a result of the broader domain boundaries used in the MIK estimate. The two new estimates are comparable at a 0.6 & 0.7g/t Au cut off.
	• The assumptions made regarding recovery of by-products.	• No assumptions have been made about gold grade recovery or the recovery of related by products. Recent metallurgical test work had been performed as part of a feasibility study of the greater Gidgee Gold Project being undertaken by Panoramic. No recovery issues have been identified.
	• Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).	• A review of deleterious elements or other non-grade variables was undertaken. Out of the available 12,271 raw samples 9,833 had been analysed for Copper (Cu). A correlation study was undertaken to determine if there was any correlation between analysed elements, there appeared to be no relationship between Cu and Au. Given that 80% of the dataset has Cu values, it was decided to estimate Cu. The Cu was only estimated in the OK model using id2 estimation methods with similar search criteria as used for the Au.
	• In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	• The OK block model was constructed with a parent cell size of 20m Y 20m X and 10m Z with subcelling to 5m Y 5m X and 1.25m Z. All estimations completed at the parent cell resolution. Data spacing is typically on 20m sections by northing and 10m on section by easting. The size of the initial search ellipse was based on the variography with a cascading five pass estimation used to populate cells. The first estimation pass utilized a minimum of 12 and maximum of 32 samples, using a major distant search of 35m without octant constraints. The search criteria were then changed for the remaining estimation passes.
	• Any assumptions behind modelling of selective mining units.	• The composites were generated at 2m downhole and the ore wireframes were maintained at a minimum width of 2m downhole to simulate a minimum mining width, assuming an open pit mining operation using 120 tonne excavators.
	• Any assumptions about correlation between variables.	• No correlations between grade variables have been assumed.
	• Description of how the geological interpretation was used to control the resource estimates.	• The geology of the deposit is classified as an Archean orogenic shear hosted deposit. The mineralisation is hosted within a sheared basalt unit.
	• Discussion of basis for using or not using grade cutting or capping.	
	• The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• Validation of the Resource estimate was completed by onscreen visual validation of block grades vs. drill hole assays in sectional view. Line graphs were also generated to show comparison between composite input

Criteria	JORC Code explanation	Commentary
		grades and block output grades over 50m intervals in the northing direction and 20m intervals in the easting direction throughout the entire deposit to ensure the composite data was accurately reflected in the model. The OK model was also compared to the MIK estimate to ensure the estimates were sensible.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• The tonnages are reported as dry tonnes. Sample preparation process involved drying the samples for 8hrs prior to analysis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• The mineralisation wireframes for the OK estimate were modelled on a gold grade cut off of 0.3g/t. This value was determined by visual assessment of grade continuity. The mineralised envelope adopted for the MIK estimate was slightly broader with a continuity of 0.2g/t Au.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• The Howards deposit has been modelled under the assumption that it will be mined by conventional open pit mining methods, utilising excavators and trucks. This would typically entail 5m bench heights with 2.5m flitches and the use of 120 tonne excavators with a bucket width of approximate 2m to mine the majority of the pit. The potential then exists to engage a smaller 30 tonne excavator to mine thinner higher-grade ore zones in order to maximise head grade and reduce dilution. Mineralisation wireframes were constructed to a minimum downhole length of 2m to replicate the smallest possible mining selectivity.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• No metallurgical assumptions have been made in respect to the generation of the estimate however recent metallurgical test work had been performed as part of a feasibility study of the greater Gidgee Gold Project being undertaken by Panoramic. This work has shown that recoveries greater than 90% Au can be achieved using conventional CIL extraction methods.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• The current data available doesn't suggest there are any high-level environmental risks with mined waste by-products. If mining were to commence all statutory requirements would be implemented to comply with waste by-product management.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• Six hundred and fifty nine density determinations were calculated for Howards, based on 5 diamond holes drilled as part of the greater Gidgee Gold Project Feasibility Study. The determination methodology was by water immersion technique. • The host rock type for mineralisation and surrounding mafic material is non-porous and void space porosity is not considered to be of relevance to the measurements. • Given the localised spatial distribution of the density measurements, average densities were assigned to the various domains within both models (OK & MIK). The values applied were: Oxide - 2.0, Transitional - 2.4 and Fresh - 2.8.
Classification	• The basis for the classification of the Mineral Resources	• The classification of the OK resource has been weighted

Criteria	JORC Code explanation	Commentary
	into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).	by strong geological continuity within the Indicated resource (>80% of the resource) with weaker continuity observed in the Inferred resource (<20% of the resource). The classification of the MIK estimate is heavily influenced by the search parameters applied. A proportion of the northern extent of Howards is densely drilled 5m*10m spaced RC (at the surface expression). The structural understanding of the Howards deposit is enhanced by the 14 diamond drill holes within the deposit. • The remainder of the deposit is drilled at 20m*20m, cascading to 20m*40m spaced drilling. The gold mineralisation is highly continuous over a 780m strike length and is structurally (shear controlled) bounded. The bulk of the resource has been classified as Indicated (>80%) with the remaining amount (<20%) inferred and unclassified.
	• Whether the result appropriately reflects the Competent Person's view of the deposit.	• The Mineral Resource estimate reflects the Competent Person's view of the deposit.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• No audits or reviews of the Mineral Resource estimate have been completed.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.	• The resource classification is based on standard practices and guidelines as prescribed in the 2012 JORC Code.
	• The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.	• The Resource estimate relates to a global estimate of tonnes and grade.
	• These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• No mining currently exists at Howards, therefore there is no production data available for comparison.

Shiraz

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The deposit has been extensively sampled using Reverse Circulation (RC) drilling techniques. The Shiraz (Gidgee Gold Project) Resource database subset contains 142 RC & 2 diamond (DD) drill holes for a total of 12,656m. Of this total, 20 RC holes totalling 2,614 were drilled by Panoramic in 2013 as part of a Gidgee Gold Project Feasibility Study. In addition the database contains 196 historical RAB holes (totalling 5,676m which have not been used for resource estimation. • The drill spacing is typically 20m * 20m grid spacing over the extent of the mineralisation. • RC holes were sampled by collecting 1m samples and splitting these down to a ~3kg assay sample using either automated on-board rig cone splitters or by manual riffle splitting. • Only two historical diamond holes have been drilled at Shiraz. The core size of both holes is unknown. • All (1,670) recent Panoramic resource assay samples were submitted to ALS Laboratories in Perth for gold analysis by FA30 (Fire Assay) technique. Of the 5,871 historical RC & DD gold assays in the Shiraz database, 3,566 (61%) have an un-known technique. In addition,

Criteria	JORC Code explanation	Commentary
		results for 1,836 (24% of the entire analytical database) QAQC samples are recorded in the database.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• The drilling method used to evaluate the deposit is predominantly RC. The historic RC drilling was typically completed using 5 ¼ inch hammers. Face sampling, 5 ¼ inch RC hammers were used by Panoramic.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• RC sample recoveries were monitored by Panoramic by recording visual estimates of the sampling bags. Typical recoveries for RC were greater than 90%. • No apparent relationships were noted between sample recovery and grade.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• All drill holes in the Shiraz resource database have been geologically logged. • RC samples in recent Panoramic drill holes have been logged using geological legends in sufficient detail to support geological confidence in Mineral Resource estimates. • Logging details lithology, weathering, oxidation, veining, and mineralisation. • All mineralised intersections and associated samples have been logged in full.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• For the two historical diamond holes no sampling information is recorded • All RC samples were collected in 1m intervals through the drill rig cyclone system and reduced to a ~3kg assay sample by either automated on-board cone splitters or manually by riffle splitting. • The sample preparation process for all samples submitted for analysis followed industry standards, including oven drying for a minimum of 8 hours, crushing and pulverizing to 85% passing 75 microns. • Quality control procedures included the insertion of standards, blanks and field duplicates to monitor sampling and analytical processes. In addition ALS Laboratories in Perth conducted their own internal QAQC system. • All sampling by Panoramic was conducted according to accepted industry practices. • The sample sizes used are industry accepted standards used extensively throughout the goldfields and are appropriate to the style of deposit.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. • For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• The analytical technique used is Fire Assay (30g charge). Where other element determinations were made it was generally by 4 acid digest and either ICP OES or AAS technique. • No other geophysical or analytical tools have been used to estimate grade. • QAQC has been routinely completed during all sampling. The QAQC results indicate the assays being used for resource estimation are a fair representation of the material being sampled. The Panoramic QAQC process was to insert 1 Certified Reference Material (CRM) or blank for every 20 RC samples. A CRM quartz wash blank was also inserted at the beginning of each RC assay batch and where possible immediately prior to the mineralised intervals. • For RC drilling, field duplicates were inserted at a rate of 1 in 25 samples. Coarse crush laboratory split duplicates were also inserted at the rate of 1 in 20 samples.

Criteria	JORC Code explanation	Commentary
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel.	• The deposit is continuous in terms of mineralisation and grade. The continuity and consistency of the grade intercepts down dip and along strike give strong confidence in the verification of the grade and style of deposit.
	• The use of twinned holes.	• No twin holes were completed. Verification holes were completed by Panoramic to test continuity of mineralisation in selected sections. The drilling confirmed expected geological and mineralogical interpretations.
	• Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	• Logging was completed in logging code protected excel templates and loaded into Panoramic's SQL database for validation. Sections were then generated and visual validation was completed to ensure integrity of the data.
	• Discuss any adjustment to assay data.	• No adjustments were made to assay data except for replacing negative (below detection reported results) with half detection limit numerical values.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	• All drill hole set-outs, pickups and collar alignments were undertaken by TEAMS Surveying using DGPS equipment with a horizontal accuracy of ±10 mm and a vertical accuracy of ±15 mm. Down-hole surveys were routinely performed every 30m using a range of single shot, electronic multi-shot and north seeking gyro tools. Panoramic validated all down hole survey data to correct anomalous readings due to magnetic interference. Recent gyroscopic surveys undertaken by Panoramic confirmed the reliability of earlier single and multi-shot readings.
	• Specification of the grid system used.	• All recent planned drill hole locations were positioned by hand-held global positioning satellite (GPS) in MGA GDA94 zone 50 and subsequently set-out and picked up by differential GPS. A total of 41 historical drill holes located on a former Shiraz grid were also picked up by Panoramic in MGA. All recorded Shiraz drill positions have been converted to the "Mt Townsend" local grid by Panoramic. The database contains both MGA and local coordinates, but for the purpose of this estimate the local grid coordinates have been used. • Conversion from Mt Townsend grid to MGA GDA94 Zone 50 is based on a two point transformation: • 10000E, 7080N = 744294.62E, 6996202.93N • 10000E, 8560N = 743940.53E, 6997640.12N • Conversion from Shiraz grid to MGA GDA94 Zone 50 is based on a two point transformation: • 50000E, 50707N = 743469.32E, 6997462.19N • 50000E, 49293N = 744679.63E, 6996730.39N
	• Quality and adequacy of topographic control.	• The Shiraz topographic layer was created by Panoramic using a 2006 Landgate aerial survey and modified by DGPS pickups of historical and current drill-hole collars as well as two control tie lines set out across the project area.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	• The drilling density is on a nominal 20m by 20m grid spacing through the majority of the deposit. This spacing is sufficient to provide strong geological and mineralogical confidence in the style of the deposit being estimated.
	• Whether sample compositing has been applied.	• Outside of the mineralised zones and through RC pre-collars, 3 metre composite spear samples were collected and submitted for assay. If any of these returned anomalous gold values (> 0.2g/t) then the original 1 metre (~3kg) cone split drill-rig samples were submitted from the respective composites.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.	• All drilling has been completed roughly perpendicular to the main strike of the deposit geometry and at angle to intercept mineralisation as close to perpendicular as possible.
	• If the relationship between the drilling orientation and the	• No sampling bias is apparent from the direction of

Criteria	JORC Code explanation	Commentary
	orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	drilling.
Sample security	• The measures taken to ensure sample security.	• Samples were kept secure on site until dispatched direct to the ALS laboratory in Perth.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits or review of the Panoramic sampling procedures and protocols has been completed.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.	• Data Logging was completed in logging code protected Excel™ templates and loaded into Panoramic's SQL database, with a "Datashed" software front-end, for validation and storage. • All geological and assay information was printed on hard copy plans and visually validated against original logs and assay results to ensure the digital copy agrees with the original format.
	• Data validation procedures used.	• Data validation was completed internally in SQL Server by setting allowable and expected values. Automated queries are run as the data is imported to ensure it meets specified criteria. • A subset of the SQL database, restricting the data to the Shiraz Resource area, was exported into MS Access database. Additional data checks were run to ensure appropriate data robustness for the Resource Estimation.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits.	• No site visits were completed by BMGS. Panoramic staff managed the 2013 drill program and was integral in the development of the Shiraz geological/mineralisation model.
	• If no site visits have been undertaken indicate why this is the case.	• Site visits were not required as the documented procedures employed by Panoramic were deemed appropriate for the style of deposit and the work was undertaken to industry accepted standards.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.	• There is a high degree of confidence in the geological interpretation of the Shiraz deposit. Shiraz is classified as an Archean orogenic shear hosted deposit. The mineralisation is typically associated with the presence of arsenopyrite, pyrrhotite and trace pyrite.
	• Nature of the data used and of any assumptions made.	• The data used for the resource estimate was from RC and diamond drilling. Raw assays typically 1m in length were composited to 2m to provide equal sample weights and reduce grade variance.
	• The effect, if any, of alternative interpretations on Mineral Resource estimation.	• Two resource estimates were performed. The first was performed using ordinary kriging within a grade envelope interpretation of 0.4g/t Au, whilst allowing for internal dilution of up to 3metres. A second broader interpretation was undertaken for a comparable MIK estimate. The MIK estimate ensured a broader continuity at a lower grade cut-off of 0.2g/t Au.
	• The use of geology in guiding and controlling Mineral Resource estimation.	• The geological and mineralisation models are based on detailed geological logging which restricts the concentration of Au mineralisation to an altered sequence within the Shiraz Dolerite. The altered sequence is characterized by 5-25% veining, and the presence of associated blue quartz and arsenopyrite, pyrrhotite and trace pyrite mineralisation.
	• The factors affecting continuity both of grade and geology.	• The geological and grade continuity is typical of most gold deposits where the continuity at a lower grade cut-off is far greater than the higher grade thresholds. There is a presence of localised higher grade zones within the mineralised domain. The continuity of these high grade zones vary from good continuity in the heart of the deposit and dissipate on the margins of the deposit.

Criteria	JORC Code explanation	Commentary
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The Shiraz mineralised domain is approximately 700m long and has a down dip extent of 150m in the southern end of the deposit and is open at depth. The deposit consists of a main lode that varies between 2m and 25m thick with numerous parallel and sub-parallel lodes at various stages along the length of the deposit.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.	• Grade estimation of Shiraz deposit was completed using two separate methods: (1) Ordinary Kriging (OK) in Surpac™ software and (2) Multiple Indicator Kriging (MIK) using GS3 software. Variogram analysis and modelling for the OK estimate was completed using Supervisor software. Two meter composites were generated from the drillhole database and then tagged according mineralized wireframes generated at a 0.4g/t gold grade cut-off. The wireframe modelling conditions included, a minimum down-hole mineralisation width of 2 meters, internal dilution of up to 3m downhole could be included if the entire intercept graded above 0.4 g/t. If dilution was greater than 3m then separate lodes were generated if geological/grade continuity was permissible (typically used trouser legs on the fringes of the main lode). • The data was review through disintegration analysis and reviewing the raw statistics to determine whether applying a top-cut was necessary. It was decided a top-cut was unwarranted, therefore all estimates were run as Uncut. The dataset was then normal scores transformed to generate variogram models. The variogram models had moderate to low nuggets with range of maximum continuity along the main axis of 26m. Separate variograms were also generated for the MIK estimate, based on the different ranked grade thresholds; metal variograms were also generated.
	• The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	• A previous estimate of the Shiraz deposit was completed in 2000 by Abelle Mining. The historical estimate appears to have been constructed at a much higher grade cut-off and is non-comparable with either of the two BMGS estimates. Therefore comparisons can only be made between the two recent estimates. The comparison demonstrates that the metal accumulation between models is generally comparable with a slight fall in the MIK estimate. This is most likely a result of the broader domain boundaries used in the MIK estimate.
	• The assumptions made regarding recovery of by-products.	• No assumptions have been made about gold grade recovery or the recovery of related by products. However it is understood that there is a refractory component, as mineralisation is associated with a sulphide assemblage including arsenopyrite.
	• Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).	• A review of deleterious elements or other non-grade variables was undertaken. Out of the available 8,644 samples 5,329 had been analysed for arsenic (As). It was established that an arsenic-gold correlation was present and with the As refractory component being responsible for expected lower recoveries using traditional CIL processing methods and given that almost 60% of the dataset was assayed for As, As was then estimated. The arsenic was estimated using id2 estimation methods, with similar search criteria used for the Au estimation. The blocks which did not estimate a value were then allocated the block mean grade by domain (Ore- 2815ppm and Waste- 823ppm). Given that the majority of the available As data was evenly spread in the southern extent of the deposit the arsenic estimate itself is considered representative. The northern extent of the deposit was not assayed for arsenic and this proportion of the estimate is considered inaccurate.

Criteria	JORC Code explanation	Commentary
	• In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	• The OK block model was constructed with a parent cell size of 20m Y 20m X and 10m Z with subcelling to 2.5m Y 2.5m X and 1.25m Z. All estimations completed at the parent cell resolution. Data spacing is typically on 20m sections by northing and 10m on section by easting. The size of the initial search ellipse was based on the variography with a cascading five pass estimation used to populate cells. The first estimation pass utilized a minimum of 12 and maximum of 32 samples without octant constraints. The search criteria were then changed for the remaining estimation passes. The MIK model was constructed using a block size of 20m Y 20m X and 10m Z.
	• Any assumptions behind modelling of selective mining units.	• The composites were generated at 2m downhole and the ore wireframes were maintained at a minimum width of 2m downhole to simulate a minimum mining width assuming an open pit mining operation using excavators of approximately 120 tonnes.
	• Any assumptions about correlation between variables.	• Although a semi correlation between As and Au exist at Shiraz, no correlations between grade variables have been assumed.
	• Description of how the geological interpretation was used to control the resource estimates.	• The geology of the deposit is classified as an Archean orogenic shear hosted deposit. The mineralisation is confined to an altered sequence of the Shiraz Dolerite. The alteration sequence is characterized by 5-25% veining, and the presence of associated blue quartz and arsenopyrite, pyrrhotite and trace pyrite mineralisation.
	• Discussion of basis for using or not using grade cutting or capping. • The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• Validation of the resource estimate was completed by onscreen visual validation of block grades vs. drill hole assays in sectional view. Line graphs were also generated to show comparison between composite input grades and block output grades over 25m intervals (in both northing and easting directions) through the entire deposit to ensure the composite data was accurately reflected in the model. The OK model was also compared to the MIK estimate to ensure the estimates were sensible.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• The tonnages are reported as dry tonnes. Sample preparation process involved drying the samples for 8hrs prior to analysis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• The mineralisation wireframes for the OK estimate were modelled on a gold grade cut off of 0.4g/t. This value was determined by visual assessment of grade continuity. The mineralised envelope adopted for the MIK estimate was slightly broader with a continuity of 0.2g/t Au.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• The Shiraz deposit has been modelled under the assumption that it will be mined by conventional open pit mining methods using excavators and trucks. This would typically entail 5m bench heights with 2.5m flitches and the use of 120 tonne excavators with a bucket width of approximately 2m to mine the majority of the pit. The potential then exists to engage a smaller 30 tonne excavator to mine thinner higher-grade ore zones in order to maximise head grade and reduce dilution. Mineralisation wireframes were constructed to a minimum downhole length of 2m to replicate the smallest possible mining selectivity.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported	• No metallurgical assumptions have been made in respect to the generation of the estimate. This will be undertaken in the advent that the Resource is converted to an Ore Reserve. Metallurgical testwork is currently being undertaken by Panoramic as part of the broader Gidgee Gold Project Feasibility Study.

Criteria	JORC Code explanation	Commentary
	with an explanation of the basis of the metallurgical assumptions made.	
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• The potential environmental implication of mining Shiraz relates to the presence of elevated arsenic levels. This issue will need to be considered in any future mining proposal.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• No Bulk density (BD) determinations have been completed on the Shiraz Project, however determinations were performed on the nearby Wilsons Project, which is hosted in the same stratigraphic dolerite sequence. The densities applied to the Shiraz Deposit are based the Wilsons density data. The assigned density values are; Oxide - 2.0g/cm³, Transitional - 2.4 g/cm³ and Fresh - 2.8 g/cm³.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• The classification of the OK resource has been weighted by strong geological continuity within the Indicated resource area (>80% of the resource), with weaker continuity observed in the Inferred resource area (<20% of the resource). The classification of the MIK estimate is heavily influenced by the search parameters applied. • Two diamond drill holes are present at Shiraz; located at the northern and southern limits of the deposit. The southern extent of the deposit has a higher drill density spacing of 20x10m compared with the northern extent which is less densely drilled at 40x20m. • The gold mineralisation is highly continuous over a 750m strike length within an altered sequence of the Shiraz Dolerite. The bulk of the resource has been classified as Indicated, including the less densely drilled northern extent of the deposit (although intuitively the northern extent is of lower confidence). • The Mineral Resource estimate reflects the Competent Person's view of the deposit
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• No audits or reviews of the Mineral Resource estimate have been completed.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• The resource classification is based on standard practices and guidelines as prescribed in the 2012 JORC Code. • The Resource estimate relates to a global estimate of tonnes and grade. • A small shallow oxide pit is present at Shiraz and is approximately 100m in strike and 30m in vertical depth. Detailed reconciliation data specific to Shiraz pit is unavailable and no mine specific comparisons have been undertaken. The pit volume has been excluded from the Shiraz Mineral Resource estimate.

Wilsons

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The deposit was sampled using Reverse Circulation (RC), Aircore (AC) and Diamond drilling (DD) techniques. A total of 213 RC holes for a total of 19,400m, 4 AC holes for 195m and 162 diamond holes for 53,328m. • The drill spacing was nominally 40m * 40m grid spacing over the extent of the mineralisation. • RC holes were sampled by collecting 1m samples and splitting these down using a (cone and riffle splitter) to approximately 3kg sample sizes. • Diamond holes were typically NQ in diameter and were sampled by cutting the core in half over geologically logged intervals that typically ranged between 30cm and 1.2m. • All samples were submitted to ALS Wangara for analysis by Fire Assay. Field and laboratory duplicates and analytical standards were routinely inserted to quantify QAQC performance.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• The drilling methods used on this deposit are RC drilling and DD drilling. The RC drilling was completed utilizing a 5 ¼ inch face sampling hammer. • The Diamond drilling was typically NQ2 (50mm) diameter core • All core was oriented where possible using "Ori-Mark" system.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• RC sample recoveries were monitored by recording visual estimates of the sample bags prior to sampling. Typical recoveries for RC were >90% • Core recovery is noted during drilling process and geological logging process as a percentage recovered vs. expected drill length. Core was reconstructed into continuous runs on a length of angle iron to enable accurate geological logging and estimation of core recovery. • No apparent relationships were noted in relation to sample recovery and grade.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• All drill holes were geologically logged. • Both chip and core samples have been logged in sufficient detail using Panoramic's lithological codes to support geological confidence in Mineral Resource Estimates. • Logging detailed lithology, alteration, mineralisation, weathering, oxidation, veining and structural features if available.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.	• All diamond core was half core sampled using an electric diamond core saw. The minimum sample length was 0.3m. • All RC samples were collected in 1m intervals through drill rig cyclone system and then split via (riffle and cone splitters) to produce a ~3kg assay sample. • Sample preparation process for all samples submitted followed industry standards, including oven drying sample for a minimum of 8 hours, crushing and pulverizing the sample to 85% passing 75 microns. • Quality control procedures included the insertion of standards, blanks and field duplicates to monitor sampling and analytical processes.

Criteria	JORC Code explanation	Commentary
	• Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• The sample sizes used are those typically used throughout the goldfields and are considered appropriate to this style of deposit.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. • For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• The analytical technique used is Fire Assay (30g charge) • All analytical data generated by direct laboratory assaying. No field estimation devises were employed. • QAQC has been routinely completed during all sampling. The QAQC results indicate that the assays being used in the estimate are a fair representation of the material that has been sampled. The Panoramic QAQC process was to insert 1 Certified Reference Material (CRM) or blank for every 20 RC samples and between 1 in 15 and 1 in 20 for diamond core samples. The CRM quartz wash blank was also inserted at the beginning of each diamond core assay batch and where possible immediately prior to the mineralised intervals. • Quarter core field duplicates were submitted at a rate of 1 in 20 samples. For RC drilling, field duplicates were inserted at a rate of 1 in 25 samples. Coarse crush laboratory split duplicates were also inserted at a rate of 1 in 20 samples for both RC and DD drilling.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel. • The use of twinned holes. • Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. • Discuss any adjustment to assay data.	• The deposit is very continuous in terms of mineralisation and grade. The continuity and consistency of the grade intercepts down dip and along strike give strong confidence in the verification of the grade and style of deposit. • No twin holes were completed. Verification holes were completed by Panoramic to test continuity of mineralisation in selected sections. Virtually all drilling confirmed expected geological and mineralogical interpretations. • Logging was completed in excel templates and loaded into Panoramic's SQL database for validation. Sections were then generated and visual validation was completed to ensure integrity of the data. • No adjustments were made to assay data except for replacing negatives with half detection limit numerical values.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. • Specification of the grid system used. • Quality and adequacy of topographic control.	• All drill hole set-outs, pickups and collar alignments were undertaken by TEAMS Surveying using DGPS equipment with a horizontal accuracy of ±10 mm and a vertical accuracy of ±15 mm. Down hole surveys were routinely performed every 30m using a range of electronic multi-shot (EMS) tool. Gyroscopic surveys were completed as verification on the EMS surveys on all Panoramic drill holes or 95% of the total drilling. The gyroscopic data confirmed the reliability of the EMS surveys. • The grid system used in the resource estimate is a local grid system which is rotated 13.5 degrees to the west of MGA_GDA94 Zone 50. • Conversion from local grid to MGA GDA94 Zone 50 is based on a two point transformation: • 10000E, 7080N = 744294.62E, 6996202.93N • 10000E, 8560N = 743940.53E, 6997640.12N • A Wilsons surface topography DTM was acquired with the purchase of the Project from Apex. The origin of the DTM is unclear, but accurately surveyed drill hole collar

Criteria	JORC Code explanation	Commentary
		RLs agree closely with the DTM.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. • Whether sample compositing has been applied.	• The drilling density is on a nominal 40m by 40m spacing through the majority of the deposit. This spacing is sufficient to give strong geological and mineralogical confidence in the style of the deposit being estimated. • Sample compositing to 1m intervals has been completed to try and represent selective mining units that would be typical in an underground environment.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• All drilling has been completed roughly perpendicular to the main strike of the deposit geometry and at angle to intercept mineralisation as close to perpendicular as possible. • No sampling bias is apparent from the direction of drilling.
Sample security	• The measures taken to ensure sample security.	• Samples were kept secure on site until dispatched direct to the ALS laboratory in Perth.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• All sampling techniques were by accepted industry standards. No audits or reviews have been undertaken.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. • Data validation procedures used.	• Data is collected in excel templates and imported into Panoramic's SQL database using Datashed import and validation software to ensure appropriate values are being imported into correct fields. All geological and assay information is printed on hard copy plans and visually validated against original logs and assay results to ensure the digital copy agrees with the original format. • Data validation is completed internally in SQL by setting allowable and expected values. Automated queries are run as the data is imported to ensure it meets specified criteria.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The author has visited the site previously, though not specifically for this round of work. Panoramic staff managed the 2013 drilling program and were integral in the development of the geological model and mineralisation interpretations.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. • Nature of the data used and of any assumptions made. • The effect, if any, of alternative interpretations on Mineral Resource estimation. • The use of geology in guiding and controlling Mineral Resource estimation. • The factors affecting continuity both of grade and geology.	• There is high confidence in the geological understanding of the deposit. There is a strong relationship between grade and logged alteration. The gold mineralisation is of a consistent grade and 1m composites of the mineralised zone indicate a very good, low coefficient of variation of < 1.1. • The data used for the resource estimate was from RC and diamond drilling. Raw assays were composited to 1m to provide equal sample weights. Short composites were normalised to 1m via accumulation with grade value. • No alternative interpretations were considered. • The geological controls relate strongly with the mineralisation interpretation. The deposit is essentially strata hosted within a shear zone (Wilsons shear) adjacent to the contact with a dolerite sill (Wilsons Dolerite). • Geological and grade continuity is strong within the Wilsons shear.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan	• The greater Wilsons mineralisation is approximately 0.6km long and has a down dip extent of >700m (open at

Criteria	JORC Code explanation	Commentary
	width, and depth below surface to the upper and lower limits of the Mineral Resource.	depth). The deposit consists of a main lode containing three distinct shoots (Wilsons 1, 2 & 3) that vary between 1m and 12m thick. Wilsons 1 contains several footwall lodes.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.	• Grade estimation of the Wilsons deposit was completed using Ordinary kriging (OK) in Surpac software. Variogram analysis and modelling was completed using supervisor software. 1m composites were generated from the drill hole database and then tagged according to mineralized wireframes generated at a 1g/t lower gold grade for low grade domains and 2g/t gold grade for high grade domains. The wireframe modelling conditions included, minimum down hole width of 1 meters of mineralisation, internal dilution of up to 2m down hole could be included if the entire intercept graded above 1 or 2g/t respectively. If dilution was greater than 2m then separate lodes were generated. • The coded composites were reviewed in supervisor; top cut analysis was completed using disintegration analysis and use of coefficient of variation. Domain grades were top cut. The cut dataset was then log transformed to review variograms and generate variomodels. Variomodels generated confirmed geological continuity. The variomodel had a moderate nugget with range of maximum continuity along main axis of 234m.
	• The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	• A previous estimate of the Wilsons deposit was completed in 2008. This was used as a guide and comparative tool for validation purposes with the current estimate.
	• The assumptions made regarding recovery of by-products.	• No assumptions have been made about gold grade recovery or the recovery of related by-products.
	• Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).	• Other elements including S and As have been estimated where data was present. Only recent 2013 drilling has results relating to these elements and as such the estimates of the elements are considered to be of low confidence.
	• In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	• The block model was constructed with a parent cell size of 20m Y 5m X and 5m Z with sub-celling to 5m Y 1.25m X and 1.25m Z. All estimation is completed at the parent cell resolution. Data spacing is typically on 20m sections. Drill spacing within sections is typically 40m. The size of the search ellipse was based on the variography with 2 estimation passes used to populate cells. The first estimation pass utilised a minimum of 4 samples and maximum of 16 samples with 4 octants of the ellipse requiring data to allow estimation.
	• Any assumptions behind modelling of selective mining units.	• The composites were generated at 1m down hole and the ore wireframes were maintained at a minimum width of 1m down hole to try and represent a minimum selectable size assuming narrow vein underground mining operations.
	• Any assumptions about correlation between variables.	• No correlations between grade variables have been assumed.
	• Description of how the geological interpretation was used to control the resource estimates.	• The geology of the deposit consists of a sheared mafic metasedimentary unit adjacent to a dolerite sill. The mineralisation interpretation is consistent with a shear zone in the metasedimentary unit.
	• Discussion of basis for using or not using grade cutting or capping.	• Top cut analysis was completed using disintegration analysis and use of coefficient of variation. Estimates were generated using cut and uncut grades to demonstrate the influence of outliers.
	• The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• Validation of the resource estimate was completed by visual validation of block grades versus drill hole assays in sectional view on computer. Line graphs were generated to show comparison between composite input grades and output block grades over 20m RL intervals through the entire deposit to ensure the composite data

Criteria	JORC Code explanation	Commentary
		was being accurately reflected in the model. The model was also compared with historical estimate to ensure report figures were reasonable.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• The tonnages are reported as dry tonnes. Sample preparation process involves drying the sample for 8hrs prior to analysis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• The mineralisation wireframes were modelled on a gold lower grade cut-off of 1g/t and 2g/t. These values were determined by visual assessment of grade continuity in Surpac™.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• The Wilsons Resource has been modelled under the assumption that it will be mined by narrow vein underground methods. This would typically involve some configuration of open stoping to extract down to 1 or 1.5m wide mineralisation. Mineralisation wireframes were constructed based on minimum thickness of 1m downhole intercepts to resolve smallest possible mining selectivity.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• Assumptions based on typical metallurgical recoveries for a deposit such as Wilsons have not been made in respect to the generation of this Mineral Resource estimate. • Metallurgical assumptions (based on test-work results) will be applied during the mine planning and conversion of resource to ore reserve stage of the Project BFS.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• No material environmental concerns have been identified. Wilsons is located on a brownfields site with existing environmental disturbance.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• Bulk density (BD) determinations have been completed on core samples by Archimedes water immersion method to determine BD of in-situ material. A total of 3,553 BD determinations are recorded in the Wilsons database, 1,140 historic and 2,413 by Panoramic. • The host rock type for mineralisation and surrounding mafic material is non-porous and void-space porosity is not considered to be of relevance to the measurements. • An average BD for each of the main lithological rock types was calculated using the recorded measurements. The assay table in the database was tagged with the actual BD or the average value based on rock type grouped averages. The density value was then extracted with the gold grade in the 2m composite file and composited based on the underlying rock type. The densities were then estimated using the Variogram models and search parameters for the gold waste domains. Average fresh rock density was calculated as 2.92g/cm^3, oxide and transitional materials were assigned values of 2.00g/cm^3 and 2.30g/cm^3 respectively.

Criteria	JORC Code explanation	Commentary
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories.	• The classification of the Resource has been based on strong geological confidence with 40m*40m spaced RC and diamond drilling. Gold mineralisation is highly continuous over its strike length and is effectively strata bound.
	• Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• All appropriate data and factors have been addressed and taken into account for this estimate. • The Mineral Resource reflects the Competent Person's view of the deposit
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• This work has been peer reviewed by BMGS personnel other than the author.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.	• The resource classification is based on standard practices and guidelines as prescribed in the 2012 JORC Code.
	• The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.	• The resource estimate relates to a global estimate of tonnes and grade.
	• These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• Good correlation exists between the estimated resources constrained within the historical Wilsons 1, 2 & 3 open pits and production data for the same volume.

Swan Open Pit and Swift Underground

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The Resources stated in this report covers both an Open Pit and Underground component. Reverse Circulation Drilling (RC) and Diamond Drilling (DD) were the techniques used. Drilling into the Open Pit was mostly by RC whereas the Underground was mostly DD. Drillholes used in this study range from holes drilled in 1984 to 2011. Mining has occurred in both the Open Pits and Underground and as a result the behavior of the ore is reasonably well known in a general sense. However locally the orebody can show high variability. • Sampling has involved 1m RC cuttings using riffle splitter in dry materials and a wedge splitter or rotary splitter in wet materials. Usually 2kg was retained. • DD has involved HQ and NQ. Some PQ holes have been drilled. • Sampling of diamond core has involved 1m sampling in early work to sampling over geological intervals (down to 0.1m) in more recent holes. The diamond core has generally been half cored with some holes split at whole core and some at three quarter core. Where it has been suspected that drillholes were drilled down dip, cross holes have been drilled. (This is particularly the case in Swift where drilling down dip had been suspected.) • Initially assaying utilized the Aqua Regia process but most assays used in this study have been by fire assay with an AAS finish using the site laboratory or off-site laboratories. A 50g charge has been used. After 2000, samples were assayed at the on-site laboratory at

Criteria	JORC Code explanation	Commentary
		Gidgee using the Leachwell method. • Visible gold occurs.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• RC and DD were the only types of drilling used in the Resource estimate. • RC drilling up until 1989 used an Open Face hammer. After 1989 this was changed to a downhole enclosed hammer. Drilling using an Open Face hammer had the potential to smear data. An analysis of drillholes pre and post 1989 showed that only approximately 5,000 tonnes of the Indicated Resources stated may have been affected by smearing. Other holes are either in the Inferred category or are supported by later drilling.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• Most drilling showed good recovery with the exception of some holes drilled in 1989. • All RC samples were thoroughly mixed in the riffling process. • There is no stated evidence of there being sample bias due to preferential sampling. • There is no relationship between sample recovery and grade.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• Drill core was photographed and appropriately logged. • Mining has been carried out and the metallurgical characteristics of the ore are well known. • Logging is qualitative in nature and was completed on all drillholes.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• Sampling has involved 1m RC cuttings using riffle splitter in dry materials and a wedge splitter or rotary splitter in wet materials. Usually 2kg was retained. • DD has involved HQ and NQ. Some PQ holes have been drilled. • Sampling of diamond core has involved 1m sampling in early work to sampling over geological intervals (down to 0.1m) in more recent holes. The diamond core has generally been half cored with some holes split at whole core and some at three quarter core. Where it has been suspected that drillholes were drilled down dip, cross holes have been drilled. (This is particularly the case in Swift where drilling down dip had been suspected.) • Samples were submitted to off-site laboratories with check assays carried out in 1988. Further check assays were carried out in other years, however this data has not been analysed. There are indications of Standards and Blanks having been submitted prior to 2002 however there is insufficient information to complete an accurate analysis. There are lists of Standards and Blanks having been submitted post 2002 and an analysis of these shows good correlation. No evidence has been found in the mining process that there was suspected issues with assaying. An analysis of Duplicates showed that in general the precision of samples was adequate. • The analytic techniques were appropriate with approximately 30g of sample pulverized to 85% passing -200 mesh. Where coarse gold occurred screen fire assaying was carried out using a 105 micron sieve.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. • For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory	• Most of the assaying is by fire which is total. • Post 2002 there exists a complete list of Standards and Blanks. This data has been analysed and shows no bias. Prior to 2002 checks were carried out however that data has not been appraised due to difficulty. However there has been no evidence of any comment to the effect that mining showed that assays had been biased.

Criteria	JORC Code explanation	Commentary
	checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel. • The use of twinned holes. • Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. • Discuss any adjustment to assay data.	• Some significant Intersections had been re-assayed and cross holes had been drilled into areas where drilling down dip had been suspected. • There have been no adjustments made to assay data.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. • Specification of the grid system used. • Quality and adequacy of topographic control.	• Accurate surveying was carried out of drillhole collars. Prior to 2002 the method of down hole survey is not recorded. There is no evidence to the effect that mining found drillholes in incorrect positions however in 2000 some RC holes > 75 degrees tended to lift and holes < 75 degrees tended to drop. There is a full description of down hole survey methods post 2002.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. • Whether sample compositing has been applied.	• Drilling is generally on a 25m grid spacing but there are large areas of 12.5m drilling. This drilling together with the fact that the orebody has been mined in both Open Pit and Underground makes it appropriate for the classification of Resource reporting. • Samples have been composited to provide Intersections which reflect Open Pit and Underground mining.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• Drillholes have been drilled both to the East and to the West to allow for the orebody dip. • Where drilling has been suspected down dip, cross holes have been available to assess this.
Sample security	• The measures taken to ensure sample security.	• There is no evidence to suggest inadequate sample security.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• An Audit was carried out in 2003 by Resource Evaluations Pty Ltd. The issue raised was that half core Kempe Diamond was used for Underground sample assaying and may have been too small. Underground drilling has been used in this work.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. • Data validation procedures used.	• The database used in this work was obtained by the Competent Person's site visit in December 2004 and has been kept at the Competent Person's Perth office since that time. The data was validated by plotting on plans and sections and having the complete involvement of Legend's previous Geologist in all interpretive work.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The Competent Person visited the site in 2004 and was responsible for the Closure Report in 2005. This involved time spent underground looking at Lodes which were being mined at the time.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. • Nature of the data used and of any assumptions made. • The effect, if any, of alternative interpretations on Mineral Resource estimation. • The use of geology in guiding and controlling Mineral Resource estimation. • The factors affecting continuity both of grade and geology.	• The Gidgee orebodies have been mined over a long period of time and are well understood in general, however locally there can be large discrepancies due to the nature of the controlling structures. Independent Geological studies have been carried out by SRK and Fractal Graphics. Locally, gold grades can exhibit very high variability. • There is only minimal scope for alternative Lode interpretations. However there is short scale variability within Lodes. • Known geology has been used as the basis of the interpretation. • Drilling is relatively close (up to 12.5m) and together with the understanding from mining a very reasonable

Criteria	JORC Code explanation	Commentary
		interpretation exists.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The Open Pit Resource is constrained to the optimized Aus $2,000 pit and covers an area of approximately 1.5km long, 1.1km wide and 200m deep. • The Underground Resource is centered around existing workings and covers an area of approximately 1.1km long, 800m wide and is up to 300m below the optimized Aus $2,000 pit.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. • The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • The assumptions made regarding recovery of by-products. • Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation). • In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. • Any assumptions behind modelling of selective mining units. • Any assumptions about correlation between variables. • Description of how the geological interpretation was used to control the resource estimates. • Discussion of basis for using or not using grade cutting or capping. • The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• Intersection Selection was carried out using the following parameters for Open Pit: Cut-off Grade: 0.7g/t Minimum Mining Width: 4m Down hole Internal Dilution: 2m Down hole Edge Dilution: 1m Either Side Down hole • Intersection Selection was then used to create wireframes. • Block Modelling was carried out for Resources using the following parameters: Block Size: 2.5m North South, 2m East West, 1m RL Block Discretisation: 1 East, 2 North, 1 RL Search Type: Elliptical Octant Maximum Number of Samples: 64 Interpolation: Inverse Distance Cubed Search Size: 60m Down dip, 30m Along strike, 3m Across strike [These were obtained from historical variography.] For reporting purposes material within the wireframes contains the reported Resource. Note: Reporting is not carried out on block cut-off grades but within wireframed shapes which are at least 2,000 tonnes in size. • Intersection Selection was carried out using the following parameters for Underground: Cut-off Grade: 2.0g/t Minimum Mining Width: 3m Down hole • For the Underground, the average of the samples within the wireframe were used to give each wireframe a value. • The following high grade cuts have been used after examination of the sampling distributions: o Premium: ▪ Oxide: 10g/t ▪ Transition: 12g/t ▪ Fresh: 60g/t o Bitter: ▪ Oxide: 10g/t ▪ Transition: 20g/t ▪ Fresh: 200g/t o Swift: ▪ Oxide: 20g/t ▪ Transition: 30g/t ▪ Fresh: 30g/t Note: Swan comprises Premium and Bitter
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• Tonnages are estimated on a dry basis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• For Open Pit a cut-off grade of 0.7g/t has been used based on milling cost. • The cut-off grade for Underground Resources is a marginal cut-off grade based on mining costs, overheads and milling. It does not include the cost of development or refurbishment if required. This results in a higher cut-off grade of 6g/t for material distal to present

Criteria	JORC Code explanation	Commentary
		Underground workings and 4g/t for wireframes close to existing infrastructure.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• Conventional Open Pit mining on 5m benches is applicable to the deposits. Mining can be selective and grade control via blasthole sampling is an applicable method. Geotechnical work has been undertaken to determine suitable slope angles and berm and batter designs. The existing excavations provide an insight into suitability of previous designs. There are no spatial constraints on Open Pit footprints (i.e. existing infrastructure, tenement boundaries and/or heritage values)
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• Conventional gravity/CIL gold extraction and recovery is applicable to these deposits. Testwork has been undertaken to determine and optimise grind size, recovery and reagent use. There is a substantial water supply of good quality readily available. Historical records of plant performance treating similar ores are available which support the metallurgical testwork.
Environmen-tal factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• Site investigation of the existing tailings storage facility revealed no deleterious materials from treating similar ores in the past. A Mining Proposal has been granted by DMP in 2014 for mining and processing Swift ore. There are no known issues which would affect this approval being extended to include Swan ore.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• Based on historic mining the following bulk densities have been used: o Fill: 1.4 tonnes per cubic metre o Oxide: 1.8 tonnes per cubic metre o Transition: 2.3 tonnes per cubic metre o Fresh: 2.8 tonnes per cubic metre
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• In general for the Open Pit, the shapes are extended a maximum of 15m along strike from an Intersection and 20m down dip. Intersections that were able to be wireframed into a shape that was on 2 or more sections as well as the cross structures were classified as Indicated. If singular Intersections were part of a structure over more than 1 section but it was too difficult to produce a wireframe then a cylinder was drawn around these areas which were also classified as Indicated. All other Intersections, as well as blocks more than 15m away from a drillhole were then classified as Inferred. • For the Underground, a bounding volume was used to define an Indicated category and an Inferred category of material. The Indicated boundary enveloped areas where there were either underground workings or a higher drilling density. Material outside of this envelope was defined as Inferred. The Inferred carries a higher cut off grade due to it being further from infrastructure, thus requiring it to carry a higher capital cost. This was used only as a guide in selecting Indicated material as

Criteria	JORC Code explanation	Commentary
		distance from existing workings was also used.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• The only audits and reviews of these estimates has been by Carras Mining Pty Ltd in 2004, 2006, 2007 and 2012.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• In an overall sense, the estimates should be accurate. However, locally estimates can vary due to the complex nature of the geology. Geological interpretation at the local scale remains the biggest source of potential error. However the previous mining of both Open Pit and Underground has resulted in reasonable understanding of geological control. • Local estimates can be difficult to quantify. • Overall the estimates should be reasonable if taken over large tonnages. This is typical of all Eastern Goldfields gold deposits.

Table 1 – Mt Henry Gold Project

Mt Henry

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The deposit has been extensively sampled using Reverse Circulation (RC) and Diamond drilling (DD) techniques. The Mt Henry (MH) Resource database subset contains 688 RC & DD holes for a total of 55,428m. Of this total 44 RC & DD holes totalling 6,522m were drill in 2013 by Panoramic as part of a Project Feasibility Study. In addition the database contains 221 RAB & Aircore holes (totalling 2,607m) which have not been used for resource estimation. • The grid drill spacing is typically 25m * 25m over the extent of the mineralisation. • RC holes were sampled by collecting 1m samples and splitting then down using either on-board rig or manual riffle splitters to produce an assay sample of ~3kg size. • Diamond holes are typically NQ2 (NQ for some historical holes) & occasionally HQ size and were sampled by cutting the core in half or quarter for the HQ core over geologically logged intervals between 20cm and 1m in length. • All (2,793) recent Panoramic resource assay samples were submitted to SGS Laboratories in Perth for gold analysis by FA50 (Fire Assay) technique. Of the 36,028 historical RC & DD gold assays in the database, 841 (2.3%) have an unrecorded technique or are by a technique other than Fire Assay. In addition results for 2,744 (7.1%) QAQC samples are recorded in the database.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• The drilling methods used on this deposit are predominantly RC and DD drilling. The RC drilling was typically completed using 5 ¼ inch hammers and recently 5 ¼ inch face sampling hammers. • The DD drilling was typically NQ (47.6mm), and more recently NQ2 (50mm) and HQ (63.5mm) diameter core • HQ size core was typically drilled as geotechnical holes from surface by Panoramic.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• RC sample recoveries were monitored by Panoramic by recording visual estimates of the sample bags prior to sampling. Typical recoveries for RC were greater than 90%. • Core recovery is noted during drilling and geological logging processes as a percentage recovered vs. expected drill length. Core was reconstructed into continuous runs on a length of angle iron to enable accurate geological logging and estimation of core recovery. Core recovery is typically 100 percent. • No apparent relationships were noted in relation to sample recovery and grade.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.	• All drill holes in the MH resource database subset have been geologically logged. • Both chip and core samples in recent Panoramic drill holes have been logged using geological legends at detail to support geological confidence in Mineral Resource estimates. • Logging details lithology, weathering, oxidation, veining, mineralisation and structural features where noted in drill core.

Criteria	JORC Code explanation	Commentary
	• The total length and percentage of the relevant intersections logged.	• All mineralised drill intersections and associated samples have been logged in full.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken.	• Both historical NQ and recent NQ2 core was typically sawn in half and half core sampled. Recent HQ geotechnical core was quarter core sampled where mineralised. Core sample lengths typically varied between 0.2 and 1.0 metre.
	• If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.	• The standard RC sample length is 1 metre with samples collected directly from the rig cyclone system. The individual 1m RC samples are then reduced to a 3-5kg assay sample by either automated on-board rig splitters or manually by riffle splitting.
	• For all sample types, the nature, quality and appropriateness of the sample preparation technique.	• The sample preparation process for all samples submitted for analysis follow accepted industry standards, including oven drying sample for a minimum of 8 hours, crushing and pulverising to 85% passing 75 microns.
	• Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.	• Quality control procedures have included the insertion of standards, blanks and duplicates to monitor the sampling and analytical process.
	• Whether sample sizes are appropriate to the grain size of the material being sampled.	• The sample sizes used are accepted industry standard sizes used extensively throughout the goldfields and are appropriate for the style of deposit.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.	• The standard analytical technique used is Fire Assay, mostly by AAS finished. Of the 38,821 RC & DD Au assays in the MH resource database subset, 841 historical assays (2.2%) do not have a recorded technique or are by technique other than Fire Assay. Where non gold analyses exist they are either by AAS or ICP OES determination.
	• For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	• No other geophysical or analytical tools have been used to estimate grade.
	• Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• QAQC has been completed routinely during all sampling throughout the life of the Project; though less so historically than more recently. The QAQC results indicate that the RC and DD assays being used for resource estimation are a fair representation of the material that has been sampled.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel.	• The deposit is very continuous in terms of mineralisation and grade intercepts. The continuity and consistency of the grade intercepts in section and along strike provides strong confidence in the verification of the grade and style of deposit. The similarity and consistency of intersections reported by past Project owners over many years is further verification of the reliability of the data.
	• The use of twinned holes.	• No recent twin holes were completed. Historical twin holes verified mineralisation continuity. In-fill verification holes were completed by Panoramic to test both geological and mineralisation continuity on selected sections. In each instance the expected geological and mineralogical interpretation was confirmed and no major discrepancies were identified.
	• Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	• Logging was completed in logging code protected MS Excel templates on laptops and then imported into the Project SQL database for validation. Sections were then generated and visual validation completed to ensure integrity of the data.
	• Discuss any adjustment to assay data.	• No adjustments were made to assay data.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	• All recent drill collars and where possible historical drill collars have been accurately located by differential GPS. A range of down-hole survey instruments, including

Criteria	JORC Code explanation	Commentary
		single shot, electronic multi-shot and gyroscopic tools have been used. Gyroscopic surveys undertaken by Panoramic and previous companies demonstrate that holes do not deviate significantly from design.
	• Specification of the grid system used.	• The MH drill hole database contains local, AMG and MGA coordinates. The resource has been estimated in local grid which is rotated +1.079 degrees from MGA GDA94 zone 51. • Conversion from local grid to AMG AGD84 zone 51 is based on a two point transformation: • 5000E, 14000N = 385844.34E, 6421899.31N • 5000E, 6400N = 385701.32E, 6414302.52N
	• Quality and adequacy of topographic control.	• Fugro 2.5m topographic contour data was the primary topographical control. In places this was modified by differential GPS height data.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. • Whether sample compositing has been applied.	• The drilling density is on a nominal 25m by 25m spacing through the majority of the deposit. This spacing is sufficient to provide strong geological and mineralogical confidence in the style of deposit being estimated. • As a general rule sample compositing has not be used. Sample compositing of RC precollars outside the main mineralised zone was undertaken at times.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• Virtually all drilling has been completed perpendicular to the main strike of the deposit geometry and angled to best intercept the west dipping mineralisation. • No sampling bias is apparent from the direction of drilling.
Sample security	• The measures taken to ensure sample security.	• Little is known about the sample security practices adopted by previous companies. Panoramic samples were freighted in sealed bulka-bags direct from site to the SGS Laboratory in Perth.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits or review of the Panoramic sampling procedures and protocols has been completed.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.	• Logging was completed in logging code protected MS Excel templates and loaded into Panoramic's SQL Server database, with a "Datashed" software frontend, for validation and storage. • Geological and assay information was printed on hard copy plans and visually validated against original logs and assay results to ensure the digital copy agrees with the original format.
	• Data validation procedures used.	• Data validation was completed internally in SQL Server by setting allowable and expected values. Automated queries are run as the data is imported to ensure it meets specified criteria. • For resource estimation a subset of the SQL database, restricting the data to the Mt Henry Resource area was exported into an MS Access database. Additional data checks were run to ensure appropriate data robustness for the Resource Estimation.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits.	• No site visits were completed by BMGS. Panoramic staff managed the 2013 drilling program and were integral in the development of the geological and mineralisation models.
	• If no site visits have been undertaken indicate why this is the case.	• Site visits were not required as the documented procedures on the recent drilling were deemed appropriate for the style of deposit and the historical drilling had been previously covered in a report released

Criteria	JORC Code explanation	Commentary
		under 2004 JORC guidelines.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.	• There is high confidence in the geological understanding of the deposit. There is a strong relationship between grade and a particular stratigraphical unit. The gold mineralisation is of a consistent grade and 2m composites generated of the mineralised zone showed a very good coefficient of variation (<1.5 using a 0.4g/t grade shell envelope)
	• Nature of the data used and of any assumptions made.	• The data used for the resource estimate was from RC and diamond drilling. Raw assays, typically representing a 1m sample length, were composited to 2m to provide equal sample weights and reduce grade variance.
	• The effect, if any, of alternative interpretations on Mineral Resource estimation.	• Three separate mineralised envelopes were considered; grade shells at 0.4, 0.6 and 1.0g/t Au were developed and respective estimates run for each grade shell. It was established through validation of the models that the 0.4g/t Au model was the most representative of grade expected in the mining environment. .
	• The use of geology in guiding and controlling Mineral Resource estimation.	• There is a strong geological control to the mineralisation interpretation. The deposit is essentially strata hosted within a sheared Banded Iron Formation (BIF). The shear is essentially contiguous along the upper contact of the BIF and an overlying mafic unit. There is some interpreted supergene mineralisation in the northern extents of the deposit that is controlled by weathering horizons and typically cross cuts stratigraphy at shallow levels.
	• The factors affecting continuity both of grade and geology.	• The geological and grade continuity is strata controlled, there are series of late pegmatites that cross cut mineralisation and lithology. These pegmatites are generally unmineralised and have been modelled into the resource as background 0.1g/t gold grade.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The Mt Henry mineralised domain is approximately 2km long and has a down dip extent of 280m and is open at depth. The deposit consists of a main lode that varies between 3m and 40m thick with numerous parallel lodes at various stages along the length of the deposit.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.	• Grade estimation of Mt Henry deposit was completed using Ordinary kriging (OK) in Surpac software. Variogram analysis and modelling was completed using Supervisor software. Two metre composites were generated from the drill hole database and then tagged according to mineralised wireframes generated at a 0.4g/t lower gold grade. The wireframe modelling conditions included a minimum downhole mineralisation width of 2m; internal dilution of up to 3m could be included if the entire intercept graded above 0.4 g/t. If dilution was greater than 3m then separate lodes were generated if geological/grade continuity was supported by nearby holes. The coded composites were reviewed in Supervisor; top cut analysis was completed using disintegration analysis and the impacts the cuts made to the coefficient of variation. One outlying grade at 1810g/t was cut to 100g/t. The cut dataset was then normal scores transformed to generate variogram models. Variogram models generated confirmed geological stratigraphy as the key controlling factor on mineralisation. The variogram models had moderate to low nuggets with a range of maximum continuity along main axis of 73m.
	• The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	• A previous estimate of the Mt Henry deposit was completed in 2009. This was used as a guide and comparative tools for validation purposes with the current estimate.
	• The assumptions made regarding recovery of by-products.	• No assumptions have been made about gold grade recovery or the recovery of related by products.

Criteria	JORC Code explanation	Commentary
	• Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).	• A review of deleterious elements or other non-grade variables was undertaken. It was decided that there was insufficient data on other elements to reliably estimate their distribution. Regression analysis conducted between Au, and Fe, Cu, S and As found no correlation.
	• In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	• The block model was constructed with a parent cell size of 20m Y, 10m X, and 10m Z with subcelling to 10m Y, 2.5m X, and 2.5m Z. All estimations completed at the parent cell resolution. Data spacing is typically on 25m sections for the southern section of the deposit with the northern extents stepping down to 20m sections. Drill spacing across sections is typically 20m. The size of the initial search ellipse was based on the variography with a cascading four pass estimation used to populate cells. The first estimation pass used a minimum of 12 and maximum of 32 samples with no octants constraints of the ellipse preventing estimation. The search criteria were then changed for the remaining estimation passes.
	• Any assumptions behind modelling of selective mining units.	• The composites were generated at 2m downhole and the ore wireframes were maintained at a minimum width of 2m downhole in order to represent a likely minimum mining width, assuming open pit mining operations using 120 tonne excavators.
	• Any assumptions about correlation between variables.	• No correlations between grade variables have been assumed.
	• Description of how the geological interpretation was used to control the resource estimates.	• The geology of the deposit is similar in many respects to the nearby Selene Deposit. Mineralisation is confined to a series of shears within the silicate facies, Banded Iron Formation (Noganyer Formation). The footwall to the BIF is a metamorphic sedimentary schist unit and the hanging wall is defined by mafic flows and dykes of the Woolyeener Formation. The vast majority of the mineralisation is confined to a single shear running contiguously along the upper contact of the BIF Noganyer Formation and the overlying Woolyeener Formation. The mineralisation interpretation is consistent with the shape and continuity of this BIF unit contact zone.
	• Discussion of basis for using or not using grade cutting or capping.	
	• The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• Validation of the resource estimate was completed by onscreen visual validation of block grades vs. drill hole assays in sectional view. Line graphs were also generated to show comparison between composite input grades and output block grades over 50m intervals on Northing spacings and 10m bench heights in elevation, through the entire deposit to ensure the composite data was being accurately reflected in the model. The model was also compared with historical estimate to ensure report figures were sensible.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• The tonnages are reported as dry tonnes. Sample preparation process involved drying the samples for 8hrs prior to analysis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• The mineralisation wireframes were modelled on a gold lower grade cut-off of 0.4g/t Au. This value was determined by visual assessment of grade continuity in Surpac™. Models were also generated at 0.6 g/t and 1.0g/t Au cut-off grades for comparison.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be	• The Mt Henry deposit has been modelled under the assumption that it will be mined by conventional open pit mining methods, using excavators and trucks. This would typically entail 5m bench heights with 2.5m flitches and use of 120 tonne excavators with a bucket of approximately 2m width. Mineralisation wireframes were constructed based on minimum thickness of 2m downhole in order to replicate the smallest possible

Criteria	JORC Code explanation	Commentary
	rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	mining selectivity.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• No metallurgical assumptions have been made in respect to the generation of the estimate however recent metallurgical test work had been performed as part of a feasibility study of the greater Mt Henry gold project being undertaken by Panoramic. This work has shown that recoveries greater than 90% Au can be achieved using conventional CIL extraction methods.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• The current data available doesn't suggest there are any high-level environmental risks with mined waste by-products. If mining were to commence all statutory requirements would be implemented to comply with waste by-product management.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• A total of 2,501 bulk density (BD) determinations are recorded in the Mt Henry Resource database subset. Panoramic completed most of these with measurements on 2,104 whole core samples by Archimedes water immersion method. There are a small number of historical measurements by pycnometer (7HENC115 & 7HENC116 for 54 samples) and down hole geophysical tool (NHC127, NHD120 and NHD121 for 343 one metre intervals). This data was used to generate a default SG for all lithological types. The default was then assigned to unmeasured intervals, and the density was estimated. • The host rock type for mineralisation and surrounding mafic material is non-porous and void space porosity is not considered to be of relevance to the measurements. • BD estimation for the resource was generated by grouping the 2501 recorded measurements by rock type to provide an average SG for each of the main lithological rock types. The assay table in the database was tagged with the actual BD or an average value based on rock type grouped average. The BD value was then extracted with the Au grade in the 2m composite file. The densities were estimated using the Variogram models and search parameters for the various domains.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• The classification of the Resource has been based on the Competent Person's strong confidence in the geological model; weighted by the strong geological confidence with 25*25m spaced RC and diamond drilling and 20m*20m spaced drilling through northern extents of deposit and the demonstrable consistency and continuity of the mineralisation (gold mineralisation is highly continuous over a 2.0km strike length and is strata bound). • The Mineral Resource reflects the Competent Person's view of the deposit.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• Apart from BMGS's own internal audit process no audits or reviews of the Mineral Resource estimate have been completed.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of	• The Resource classification is based on standard practices and guidelines as prescribed in the 2012 JORC Code.

Criteria	JORC Code explanation	Commentary
	statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• The resource estimate relates to a global estimate of tonnes and grade. • No reliable production data exists for the small open pit operated within the confines of the Mt Henry Resource by Australis Mining in the 1980s to compare with this resource estimate.

North Scotia

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The deposit has been extensively sampled using Reverse Circulation (RC) and Diamond drilling (DD) techniques. The North Scotia (NS) database resource subset contains 193 RC & DD holes for a total of 13,573m. Of this total 16 RC & DD holes totalling 933.5m were drill in 2013 by Panoramic as part of a Mt Henry Project Feasibility Study. In addition the database contains 155 Aircore (totalling 2,915m) which have not been used for resource estimation. • The drill spacing is typically 20m *20m grid spacing over the extent of the mineralisation. • RC holes were sampled by collecting 1m samples and splitting these down using either on-board rig or manual riffle splitters to produce an assay sample of~3kg. • Diamond holes were typically NQ2 & occasionally HQ in size and were sampled by cutting the core in half or quarter for the HQ core over geologically logged intervals between 20cm and 1m in length. • All (579) recent Panoramic resource assay samples were submitted to SGS Laboratories in Perth for gold analysis by FA50 (Fire Assay) technique. Of the 11,753 historical RC & DD gold assays in the database, 1,430 (12.2%) have an unrecorded technique or are by a technique other than Fire Assay. In addition, results for 1,353 (11.0%) QAQC samples are recorded in the database.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• The drilling methods used on this deposit are predominantly RC and DD drilling. The RC drilling was typically completed using a 5 ¼ inch hammer and recently 5 ¼ inch face sampling hammers. • The DD drilling was typically NQ (47.6mm), and more recently NQ2 (50mm) and HQ (63.5mm) diameter core • HQ size core was typically drilled as geotechnical holes from surface by Panoramic.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples.	• RC sample recoveries were monitored by Panoramic by recording visual estimates of the sample bags prior to sampling. Typical recoveries for RC were greater than 90%. • Core recovery is noted during drilling and geological logging processes as a percentage recovered vs. expected drill length. Core was reconstructed into continuous runs on a length of angle iron to enable accurate geological logging and estimation of core recovery. Core recovery is typically 100 percent.

Criteria	JORC Code explanation	Commentary
	• Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• No apparent relationships were noted in relation to sample recovery and grade.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.	• All drill holes in the NS resource database have been geologically logged. • Both chip and core samples in recent Panoramic drill holes have been logged using geological legends at detail to support geological confidence in Mineral Resource estimates. • Logging details lithology, weathering, oxidation, veining, mineralisation and structural features where noted in drill core.
	• The total length and percentage of the relevant intersections logged.	• All mineralised drill intersections and associated samples have been logged in full.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken.	• Both historical NQ and recent NQ2 core was typically sawn and half core sampled. Recent HQ geotechnical core was quarter core sampled where mineralized. Core sample lengths typically varied between 0.2 and 1.0 metre.
	• If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.	• The standard RC sample length is 1 metre with samples collected directly from the rig cyclone system. The individual 1m RC samples are then reduced to a 3-5kg assay sample by either automated on-board rig splitters or manually by riffle splitting.
	• For all sample types, the nature, quality and appropriateness of the sample preparation technique.	• The sample preparation process for all samples submitted for analysis follow accepted industry standards, including oven drying sample for a minimum of 8 hours, crushing and pulverising to 85% passing 75 microns.
	• Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.	• Quality control procedures have included the insertion of standards, blanks and duplicates to monitor the sampling and analytical process.
	• Whether sample sizes are appropriate to the grain size of the material being sampled.	• The sample sizes used are industry standard sizes used extensively throughout the goldfields and are appropriate for the style of deposit.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.	• The standard analytical technique used is Fire Assay, mostly by AAS finish. Of the 12,332 RC & DD Au assays in the NS resource database 1,430 historical assays (11.6%) do not have a recorded technique or are by technique other than Fire Assay. Where non gold analyses exist they are either by AAS or ICP OES determination.
	• For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	• No other geophysical or analytical tools have been used to estimate grade.
	• Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• QAQC has been completed routinely during all sampling throughout the life of the Project; though less so historically than more recently. The QAQC results indicate that the RC and DD assays being used for resource estimation are a fair representation of the material that has been sampled.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel.	• The North Scotia deposit is a classic "Norseman style quartz vein" type. Gold mineralisation is nuggetty within the veins with coarse visible gold not uncommon.
	• The use of twinned holes.	• No recent twin holes were completed. In-fill verification holes were completed by Panoramic to test both geological and mineralisation continuity on selected sections. In most instances the expected geological and mineralogical interpretation was confirmed and no major discrepancies were identified.
	• Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic)	• Logging was completed in logging code protected MS Excel templates on laptops and then imported into the

Criteria	JORC Code explanation	Commentary
	protocols.	Project SQL database for validation. Sections were then generated and visual validation completed to ensure integrity of the data.
	• Discuss any adjustment to assay data.	• No adjustments were made to assay data.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	• All recent drill collars and where possible historical drill collars have been accurately located by differential GPS. A range of down-hole survey instruments, including single shot, electronic multi-shot and gyroscopic tools have been used. Gyroscopic surveys undertaken by Panoramic and previous companies demonstrate that holes do not deviate significantly from design.
	• Specification of the grid system used.	• The NS drill hole database subset contains local, AMG and MGA coordinates. The resource has been estimated in local grid which is rotated +1.079 degrees from MGA_GDA94 zone 51. • Conversion from local grid to AMG AGD84 zone 51 is based on a two point transformation: • 5000E, 14000N = 385844.34E, 6421899.31N • 5000E, 6400N = 385701.32E, 6414302.52N
	• Quality and adequacy of topographic control.	• Fugro 2.5m topographic contour data was the primary topographical control. In places this was modified by differential GPS height data.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	• The drilling density is on a nominal 20m by 20m spacing through the majority of the deposit. This spacing is sufficient to provide strong geological and mineralogical confidence in the style of deposit being estimated.
	• Whether sample compositing has been applied.	• As a general rule, sample compositing has not be used. Sample compositing of RC precollars outside the main mineralised zone was undertaken at times.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• Virtually all drilling has been completed perpendicular to the main strike of the deposit geometry and angled to best intercept the west dipping mineralised vein system. • No sampling bias is apparent from the direction of drilling.
Sample security	• The measures taken to ensure sample security.	• Little is known about the sample security practices adopted by previous companies. Panoramic samples were freighted in sealed bulka-bags direct from site to the SGS Laboratory in Perth.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits or review of the Panoramic sampling procedures and protocols has been completed.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.	• Logging was completed in logging code protected MS Excel templates and loaded into Panoramic's SQL Server database, with a "Datashed" software frontend, for validation and storage. • All geological and assay information was printed on hard copy plans and visually validated against original logs and assay results to ensure the digital copy agrees with the original format.
	• Data validation procedures used.	• Data validation was completed internally in SQL Server by setting allowable and expected values. Automated queries are run as the data is imported to ensure it meets specified criteria. • For resource estimation a subset of the SQL database, restricting the data to the North Scotia Resource area was exported into MS Access database. Additional data checks were run to ensure appropriate data robustness for the Resource Estimation.

Criteria	JORC Code explanation	Commentary
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits.	• No site visits were completed by BMGS. Panoramic staff managed the 2013 drilling program. The new interpretation was constructed by using the original Cube 2009 wireframes and adjusting the shapes based on the new drill results.
	• If no site visits have been undertaken indicate why this is the case.	• Site visits were not required as the documented procedures on the recent drilling were deemed appropriate for the style of deposit and the historical drilling had been previously covered in a report released under 2004 JORC guidelines.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.	• There is moderate confidence in the deposit. The deposit is a structurally controlled auriferous quartz vein system. The deposit has a high grade nuggetty nature; the nuggetty gold mineralisation is disseminated throughout one to five metre wide laminated quartz veins that dip around 70° towards the west. Minor pyrite and galena is also present in the veins.
	• Nature of the data used and of any assumptions made.	• The data used for the resource estimate was from RC and diamond drilling. Raw assays were composited to 2m to provide equal sample weights and reduce grade variance.
	• The effect, if any, of alternative interpretations on Mineral Resource estimation.	• No alternative interpretations were considered.
	• The use of geology in guiding and controlling Mineral Resource estimation.	• The geological logging was used to generate the interpretation. Vein/mineralisation wireframes were allowed to pass through sub-mineralised drill hole zones where the Quartz veining was logged but returned little to barren grades. It is believed this method is acceptable, as long as the block data is reflective of the raw data on a localised scale.
	• The factors affecting continuity both of grade and geology.	• The mineralisation is vein hosted, typically around 2 to 5metre wide zones, continuity along strike appears to be reasonable and there appears to be some slight dilational jogs, which is observed within the geology of the historical Scotia pit (commentary provided by Panoramic Geologist).
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The North Scotia mineralized domain is approximately 450m long and has a down dip of extent of 110m and is open at depth. The deposit consists of multiple NNE trending quartz lodes that vary between 1m and 5m in true thickness with numerous thinner parallel lodes at various stages along the length of the deposit.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.	• Grade estimation of North Scotia deposit was completed using Ordinary Kriging (OK) in Surpac™ software. Variogram analysis and modelling was completed using supervisor software. Two meter composites were generated from the drill hole database and then tagged according to mineralized wireframes generated at a 0.8g/t gold grade. The wireframe modelling conditions included, a minimum downhole width of 2 meters of mineralisation, internal dilution of up to 2m downhole could be included if the entire intercept graded above 0.8 g/t. If dilution was greater than 2m then separate lodes were generated. • The coded composites were reviewed in Supervisor; top-cut analysis was completed using disintegration analysis and the use of coefficient of variation statistics. Top-cuts were applied on a domain basis with a maximum top-cut of 30g/t on FW1 lode and 7g/t & 4g/t on the HW1 and HW2 lodes respectively (remaining lodes were left un-cut). The cut dataset was then transformed by normal scores to review variograms and generate variogram models. The variogram models had high to low nuggets and varied for each domain, with a range of maximum continuity along the main axis varying for each domain between 20 to 40m.

Criteria	JORC Code explanation	Commentary
	• The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	• A previous estimate of the North Scotia deposit was completed in 2009. This was used as a guide and comparative tools for validation purposes with the current estimate.
	• The assumptions made regarding recovery of by-products.	• No assumptions have been made about gold grade recovery or the recovery of related by products.
	• Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).	• Only gold was estimated; no other elements were estimated. Insufficient multi-element data was available and was deemed to be of no relevance and to have a spatial distribution too sparse for adequate estimation.
	• In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	• The block model was constructed with a parent cell size of 10m Y, 5m X, and 10m Z with subcelling to 2.5m Y, 1.25m X, and 1.25m Z. All estimations were completed at the parent cell resolution. Data spacing is typically on 20 x 20m spacings for the majority of the deposit. The size of the search ellipse was based on the variography with two estimation passes used to populate all cells. The first estimation pass used a minimum of 3, and maximum of 15 samples, with an ellipsoid based approach. Two separate estimation methods were employed, an Ordinary Kriged and Inverse Distance ^2 (ID2) was also performed on the top-cut dataset.
	• Any assumptions behind modelling of selective mining units.	• The composites were generated at 2m downhole and the ore wireframes were maintained at a minimum width of 2m downhole to try and represent a minimum selectable size assuming open pit mining operations using 120 tonne excavators.
	• Any assumptions about correlation between variables.	• No correlations between grade variables have been assumed.
	• Description of how the geological interpretation was used to control the resource estimates.	• The geology of the deposit consists of Archaean mafics of the Woolyeenyer formation (predominately Mafic Basalt/Dolerite), with late stage cross-cutting pegmatite dykes and a very shallow weathering profile of Quaternary saprolite clays. There is no evidence of supergene mineralisation at North Scotia. All mineralisation is associated with the primary quartz vein system.
	• Discussion of basis for using or not using grade cutting or capping.	• Top-cut analysis was completed using disintegration analysis and use of coefficient of variation statistics. Fourteen outlying grades were cut which differed on a domain basis (FW1- number of samples top-cut 3, top-cut applied 30g/t; HW1- number of samples top-cut and top-cut applied was 7; HW2- number of samples top-cut and top-cut applied was 4). Only cut estimates were provided.
	• The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• Validation of the Resource estimate was completed by visual validation of block grades vs. drill hole assays in sectional view on computer. Given the thin nature of the deposit a volume check was performed to ensure the block volume was reflective of the 3dm volume. With a volume variance less that 1%, the coding in the model was deemed acceptable. Validation comparisons were made by comparing the mean composite grades to the block estimate grades on a domain basis. This was to ensure the estimate was reflective of the data used to generate it.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• The tonnages are reported as dry tonnes. Sample preparation process involved drying the samples for 8hrs prior to analysis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• The mineralisation wireframes were modelled on a gold grade cut-off of 0.8g/t. The cut-off selection was left unchanged from that used in the 2009 resource.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to	• The North Scotia deposit has been modelled under the assumption that it will be mined by conventional open pit mining methods, using excavators and trucks. This would typically entail 5m bench heights with 2.5m flitches and use of 120 tonne excavators with a bucket width of

Criteria	JORC Code explanation	Commentary
	consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	approximately 2m. Mineralisation wireframes were constructed based on minimum downhole intercept thickness of 2m in order to replicate the smallest possible mining selectivity.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• No metallurgical assumptions have been made in respect to the generation of the estimate. • The understanding is that the anticipated recovery from metallurgical test work completed by Panoramic as part of a broader Mt Henry Gold Project feasibility study is in the order of 95% recovery using a conventional CIL process.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• If mining were to commence appropriate measures would be implemented to ensure correct containment of waste by-products.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• Bulk density (BD) determinations have been completed on 279 core samples by Archimedes water immersion method to determine BD of insitu material. The measurements were performed on whole core samples by Panoramic from recently drilled diamond holes (SCO* series holes). A small number of historical pycnometer measurements (7SSC* series for 58 samples) also exist. • Both the waste and host rock type for mineralisation is typically non-porous and void space porosity is not considered to be of relevance to the measurements. • Default densities were assigned to the model based on the specific gravities used in the previous models. The default values were verified by the recent BD work. The densities applied were Alluvium 1.8 g/cm³, Qtz Reef 2.65 g/cm³, Pegmatite 2.7g/cm³, Oxide Mafic 1.8 g/cm³, Transitional Mafic 2.1 g/cm³ and Fresh Mafic 3 g/cm³.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• The classification of the Resource has been based on strong geological confidence with 20*20m spaced RC and diamond drilling. The gold mineralisation is moderately continuous over a 450m strike length and is laminated vein hosted. The classification was applied to the model using the estimation pass and a string constraint applied to limit the potential overcall in classification down dip. • The Mineral Resource reflects the Competent Person's view of the deposit.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• Apart from BMGS's own internal audit process no audits or reviews of the Mineral Resource estimate have been completed.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the	• The resource classification is based on standard practices and guidelines as prescribed in the 2012 JORC Code.

Criteria	JORC Code explanation	Commentary
	relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• The Resource estimate relates to a global estimate of tonnes and grade. • No mining data is available for comparison.

Selene

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The deposit has been extensively sampled using Reverse Circulation (RC) and Diamond drilling (DD) techniques. The Selene (Mt Henry Project) resource database subset contains 284 RC & DD holes for a total of 25,830m. Of this total 14 RC & DD holes totalling 2,005m were drill in 2013 by Panoramic as part of a Project feasibility study. In addition the database contains 10 RAB & Aircore holes (totalling 133m) which have not been used for resource estimation. • The drill grid spacing is typically 40m * 40m over the extent of the mineralisation. • RC holes were sampled by collecting 1m samples and splitting these down using either on-board rig or manual riffle splitters to produce an assay sample of ~3kg size. • Diamond holes were typically NQ2 (NQ for some historical holes) & occasionally HQ size and were sampled by cutting the core in half or quarter for the HQ core over geologically logged intervals between 20cm and 1m in length. • All (847) recent Panoramic resource assay samples were submitted to SGS Laboratories in Perth for gold analysis by FA50 (Fire Assay) technique. Of the 15,528 historical RC & DD gold assays in the database, 457 (2.9%) have an un-recorded technique or are by a technique other than Fire Assay. In addition data on 1,059 (6.5%) QAQC samples are recorded in the database.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• The drilling methods used on this deposit are predominantly RC and DD drilling. The RC drilling was typically completed using 5 ¼ inch hammers and recently 5 ¼ inch face sampling hammers. • The DD drilling was typically NQ (47.6mm), and more recently NQ2 (50mm) and HQ (63.5mm) diameter core • HQ size core was typically drilled as geotechnical holes from surface by Panoramic.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• RC sample recoveries were monitored by Panoramic by recording visual estimates of the sample bags prior to sampling. Typical recoveries for RC were greater than 90%. • Core recovery is noted during drilling process and geological logging process as a percentage recovered vs. expected drill length. Core was reconstructed into continuous runs on a length of angle iron to enable accurate geological logging and estimation of core recovery. Core recovery is typically 100 percent. • No apparent relationships were noted in relation to sample recovery and grade.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.	• All drill holes in the Selene resource database have been geologically logged. • Both chip and core samples in recent Panoramic drill holes have been logged using geological legends at

Criteria	JORC Code explanation	Commentary
	• Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.	detail to support geological confidence in Mineral Resource estimates. • Logging details lithology, weathering, oxidation, veining, mineralisation and structural features where noted in drill core.
	• The total length and percentage of the relevant intersections logged.	• All mineralised drill intersections and associated samples have been logged in full.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken.	• Both historical NQ and recent NQ2 core was typically sawn in half and half core sampled. Recent HQ geotechnical core was quarter core sampled where mineralised. Core sample lengths typically varied between 0.2 and 1.0 metre.
	• If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.	• The standard RC sample length is 1 metre with samples collected directly from the rig cyclone system. The individual 1m RC samples are then reduced to a 3-5kg assay sample by either automated on-board rig splitters or manually by riffle splitting.
	• For all sample types, the nature, quality and appropriateness of the sample preparation technique.	• The sample preparation process for all samples submitted for analysis follow accepted industry standards, including oven drying sample for a minimum of 8 hours, crushing and pulverising to 85% passing 75 microns.
	• Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.	• Quality control procedures have included the insertion of standards, blanks and duplicates to monitor the sampling and analytical process.
	• Whether sample sizes are appropriate to the grain size of the material being sampled.	• The sample sizes used are accepted industry standard sizes used extensively throughout the goldfields and are appropriate for the style of deposit.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.	• The standard analytical technique used is Fire Assay, mostly by AAS finished. Of the 16,886 RC & DD Au assays in the Selene resource database, 457 historical assays (2.7%) do not have a recorded technique or are by technique other than Fire Assay. Where non gold analyses exist they are either by AAS or ICP OES determination.
	• For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	• No other geophysical or analytical tools have been used to estimate grade.
	• Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• QAQC has been completed routinely during the life of the Project; though less so historically than more recently. The QAQC results indicate that the RC and DD assays being used for resource estimation are an accurate representation of the sampled material.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel.	• The deposit is very continuous in terms of mineralisation and grade intercepts. The continuity and consistency of the grade intercepts in section and along strike provides strong confidence in the verification of the grade and style of deposit. The similarity and consistency of intersections reported by past Project owners over many years is further verification of the reliability of the data.
	• The use of twinned holes.	• No recent twin holes were completed. Historical twin holes verified mineralisation continuity. In-fill verification holes were completed by Panoramic to test both geological and mineralisation continuity on selected sections. In each instance the expected geological and mineralogical interpretation was confirmed and no major discrepancies were identified.
	• Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	• Logging was completed in logging code protected excel templates on laptops and then imported into the Project SQL database for validation. Sections were then generated and visual validation completed to ensure integrity of the data.

Criteria	JORC Code explanation	Commentary
	• Discuss any adjustment to assay data.	• No adjustments were made to assay data.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	• All recent drill collars and where possible historical drill collars have been accurately located by differential GPS. A range of down-hole survey instruments, including single shot, electronic multi-shot and gyroscopic tools have been used. Gyroscopic surveys undertaken by Panoramic and previous companies demonstrate that holes do not deviate significantly from design.
	• Specification of the grid system used.	• The Selene drill hole database contains local, AMG and MGA coordinates. The resource has been estimated in local grid which is rotated +1.25 degrees from MGA GDA94 zone 51. • Conversion from local grid to AMG AGD84 zone 51 is based on a two point transformation: • 4400E, 6000N = 385096.84E, 6413919.03N • 4400E, 2000N = 385009.80E, 6409920.95N
	• Quality and adequacy of topographic control.	• Fugro 2.5m topographic contour data was the primary topographical control. In places this was modified by differential GPS height data.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	• The drilling density is on a nominal 40m by 40m spacing through the majority of the deposit. This spacing is sufficient to provide strong geological and mineralogical confidence in the style of deposit being estimated.
	• Whether sample compositing has been applied.	• As a general rule sample compositing has not be used. Sample compositing of RC precollars outside the main mineralised zone was undertaken at times.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• Virtually all drilling has been completed perpendicular to the main strike of the deposit geometry and angled to best intercept the west dipping mineralisation. • No sampling bias is apparent from the direction of drilling.
Sample security	• The measures taken to ensure sample security.	• Little is known about the sample security practices adopted by previous companies. Panoramic samples were freighted in sealed bulka-bags direct from site to the SGS Laboratory in Perth.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits or review of the Panoramic sampling procedures and protocols has been completed.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.	• Logging was completed in logging code protected MS Excel templates and loaded into Panoramic's SQL Server database, with a "Datashed" software frontend, for validation and storage. • Geological and assay information was printed on hard copy plans and visually validated against original logs and assay results to ensure the digital copy agrees with the original format.
	• Data validation procedures used.	• Data validation was completed internally in SQL Server by setting allowable and expected values. Automated queries are run as the data is imported to ensure it meets specified criteria. • For resource estimation a subset of the SQL database, restricting the data to the Selene Resource area was exported into an MS Access database. Additional data checks were run to ensure appropriate data robustness for the Resource Estimation.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits.	• No site visits were completed by BMGS. Panoramic staff managed the 2013 drilling program and were integral in the development of the geological and mineralisation

Criteria	JORC Code explanation	Commentary
		models.
	• If no site visits have been undertaken indicate why this is the case.	• Site visits were not required as the documented procedures on the recent drilling were deemed appropriate for the style of deposit and the historical drilling had been previously covered in a report released under 2004 JORC guidelines.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.	• There is high confidence in the geological understanding of the deposit. There is a strong relationship between grade and a particular stratigraphical unit. The gold mineralisation is of a consistent grade and 2m composites generated of the mineralised zone showed an extremely good coefficient of variation (< 1.1), such coefficients are rarely encountered with gold deposits.
	• Nature of the data used and of any assumptions made.	• The data used for the resource estimate was from RC and diamond drilling. Raw assays, typically representing a 1m sample length, were composited to 2m to provide equal sample weights and reduce grade variance.
	• The effect, if any, of alternative interpretations on Mineral Resource estimation.	• Three separate mineralised envelopes were considered; grade shells at 0.4, 0.6 and 1.0g/t Au were developed and respective estimates run for each grade shell. It was established through validation of the models that the 0.4g/t Au model was the most representative of grade expected in the mining environment.
	• The use of geology in guiding and controlling Mineral Resource estimation.	• There is a strong geological control to the mineralisation interpretation. The deposit is essentially strata hosted within a sheared Banded Iron Formation. There is some interpreted supergene mineralisation in the northern extents of the deposit that is controlled by weathering horizons and typically cross cuts stratigraphy at shallow levels.
	• The factors affecting continuity both of grade and geology.	• The geological and grade continuity is strata controlled, with a series of late pegmatites that cross cut mineralisation and lithology. These pegmatites are generally unmineralised and have been modelled into the resource as background 0.1g/t gold grade.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The Selene mineralised domain is approximately 1.3km long, has a down dip of extent of up to 440m and is open at depth. The deposit consists of a main lode that varies between 3m and 25m thick with numerous parallel lodes at various stages along the length of the deposit.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.	• Grade estimation of Selene deposit was completed using Ordinary Kriging (OK) in Surpac™ software. Variogram analysis and modelling was completed using Supervisor software. Two metre composites were generated from the drill hole database and then tagged according to mineralised wireframes generated at a 0.4g/t lower Au grade. The wireframe modelling conditions included a minimum downhole mineralisation width of 2m; an internal dilution of up to 3m could be included if the entire intercept graded above 0.4 g/t. If dilution was greater than 3m then separate lodes were generated if geological/grade continuity was supported by nearby drilling. The coded composites were reviewed in Supervisor, top cut analysis was completed using disintegration analysis and the impact cuts made to the coefficient of variation. Seven outlying grades ranging from 11.79 to 21.56g/t were cut to 11g/t. The cut dataset was then transformed by normal scores to review variograms and generate variomodels. Variomodels generated confirmed geological stratigraphy as the key controlling factor on mineralisation. The variomodels had moderate to low nuggets with a range of maximum continuity along the main axis of 70m.
	• The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	• A previous estimate of the Selene deposit was completed in 2008. This was used as a guide and comparative tools for validation purposes with the current estimate.

Criteria	JORC Code explanation	Commentary
	• The assumptions made regarding recovery of by-products.	• No assumptions have been made about gold grade recovery or the recovery of related by products.
	• Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation).	• It was established that there was insufficient multiple element data to enable adequate estimation of deleterious elements or other non-grading variables.
	• In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	• The block model was constructed with a parent cell size of 20m Y, 10m X, and 10m Z, with subcelling to 5m Y, 2.5m X, and 2.5m Z. All estimations were completed at the parent cell resolution. A very regular pattern of holes cover the Selene Deposit. Drill data is typically on sections 40m apart on northings, with holes as little as 10m apart on eastings. The size of the search ellipse was based on the variography with five estimation passes used to populate cells. The first estimation pass used a minimum of 12, and maximum of 32 samples, with no octant constraints of the ellipse requiring data to allow estimation.
	• Any assumptions behind modelling of selective mining units.	• The composites were generated at 2m downhole and the ore wireframes were maintained at a minimum width of 2m downhole in order to represent a likely minimum mining width, assuming an open pit mining operations using 120 tonne excavators.
	• Any assumptions about correlation between variables.	• No correlations between grade variables have been assumed.
	• Description of how the geological interpretation was used to control the resource estimates.	• The Selene Deposit is similar in many respects to the nearby Mt Henry Deposit. Mineralisation is confined to a series of shears within the silicate facies, Banded Iron Formation (Noganyer Formation). The footwall to the BIF is a metamorphic sedimentary schist unit and the hanging wall is defined by mafic flows and dykes of the Woolyeener Formation. The mineralisation interpretation is consistent with the shape and continuity of the BIF unit.
	• Discussion of basis for using or not using grade cutting or capping.	
	• The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• Validation of the Resource estimate was completed by visual validation of block grades vs. drill hole assays in sectional view on computer. Line graphs were also generated to show comparison between composite input grades and block output grades over 80m intervals – Northings, and 40m internals –Eastings, throughout the entire deposit to ensure the composite data was being accurately reflected in the model. The model was also compared with historical estimate to ensure report figures were sensible.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• The tonnages are reported as dry tonnes. Sample preparation process involved drying the samples for 8hrs prior to analysis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• The mineralisation wireframes were modelled on a gold lower grade cut-off of 0.4g/t. This value was determined by visual assessment of grade continuity in Surpac™. Models were also generated at 0.6 g/t and 1.0g/t Au cut-off grades for comparison.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• The Selene deposit has been modelled under the assumption that it will be mined by conventional open pit mining methods, utilising excavators and trucks. This would typically entail 5m bench heights with 2.5m flitches and use of 120 tonne excavators with a bucket width of approximately 2m. Mineralisation wireframes were constructed based on minimum thickness of 2m downhole in order to replicate the smallest possible mining selectivity.
Metallurgical factors or	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part	• No metallurgical assumptions have been made in respect to the generation of the estimate however recent

Criteria	JORC Code explanation	Commentary
assumptions	of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	metallurgical test work had been performed as part of a feasibility study of the greater Mt Henry gold project being undertaken by Panoramic. This work has shown that recoveries greater than 90% Au can be achieved using conventional CIL extraction methods.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• The current data available doesn't suggest there are any high-level environmental risks with mined waste by-products. If mining were to commence all statutory requirements would be implemented to comply with waste by-product management.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• A total of 1087 bulk density (BD) determinations are recorded in the Selene resource database subset. Panoramic completed most of these with measurements on 727 core samples by Archimedes water immersion method. There are a number of historical measurements by pycnometer (7SEL* series for 181 samples) and down hole geophysical tool (NLC151D & NLC158 for 179 one metre intervals). • The host rock type for mineralisation and surrounding mafic material is non-porous and void space porosity is not considered to be of relevance to the measurements. • The estimate of BD through the resource was generated by grouping the 1087recorded measurements by rock type to provide an average SG for each of the main lithological rock types. The assay table in the database was tagged with the actual BD or an average value based on rock type grouped averages. The density value was then extracted with the gold grade in the 2m composite file. The BD was then estimated using the same Variogram models and search parameters for the various domains.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• The classification of the resource has been based on the Competent Person's strong confidence in the geological model; derived largely from the high density of drilling (40*10m spaced RC and selective diamond drilling in the central extent of the deposit); and the demonstrable consistency and continuity of the mineralisation (gold mineralisation is highly continuous over a 1.3km strike length and is strata bound). • A large weighting of the classification related to the estimation pass, string constraints were also used to override the estimation pass allocation and reflect the competent person's view where necessary. • The Mineral Resource reflects the Competent Person's view of the deposit.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• Apart from BMGS's own internal audit process no audits or reviews of the Mineral Resource estimate have been completed.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate,	• The Resource classification is based on standard practices and guidelines as prescribed in the 2012 JORC Code.

Criteria	JORC Code explanation	Commentary
	a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• The Resource estimate relates to a global estimate of tonnes and grade. • No mining currently exists at Selene therefore there is no production data available for comparison.

Table 1 – Panton PGM Project

Section 1 - Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The database of sampling for the project resource definition is comprised of a number of different sampling methods. It contains historic drilling diamond drilling (30 holes= 9,524.4m) diamond drilling (including RC pre-collar holes) undertaken by Platinum Australia Limited (PLA) between 2001 and 2003 (166 holes = 34,410.2m), RC drilling undertaken by PLA between 2001 and 2003 (29 holes=2,366.3m) and channel sampling of surface and underground trenches and faces (1,391m). • Diamond Drill Core, RC chips and surface trench channel sampling are the three primary sample types. Relatively small numbers of samples are from decline, wall and face sampling undertaken in an exploration decline. • Diamond core the predominant sampling (HQ, HQ3, NQ and NQ2 sizes) was orientated, geologically logged and sampled to lithological contacts or changes in the nature of mineralisation. Nominal samples lengths of 1.0m with a minimum sample length of 0.25m. NQ and NQ2 core was half core sampled. For NQ and NQ2 core the left hand (looking down the hole) half of the core is sampled. HQ and HQ3 core was quarter core sampled. For HQ and HQ3 core the right hand (looking down the hole) half of the core is cut again to provide quarter core. The lower quarter was sent for analysis. • RC chips sampled at 1m or 0.5m intervals. The total chipped material for each interval is collected off the drill cyclone; it is riffle split on site to produce a sample of approximately 2-3kg to be sent to the laboratory for analysis. Wet sample has been left to dry before splitting. Remaining reject is stored at the site facility. • Trench channel chip sampling was undertaken from the base or as close to the base of the trench as possible. Each trench was sampled continuously over the entire length. Sample lengths varied from 0.15m to 2m. Sample boundaries were based on geological contacts and changes in nature of mineralisation. Where the material was soft a channel was cut with a geological pick and in harder material chip sampling using a geological hammer. Samples of 2-3kg were collected for analysis. • Decline sampling, wall and face sampling was undertaken on geologically marked up channels approximately 1.5m from the floor. Sampled intervals varied from 0.25 to 0.5m across the full width of mineralisation. Sample material was chipped out using a geological hammer in most instances, the resulting sample weights were consistently less than those for drilling.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• RC drilling has been used predominantly to provide pre-collars for diamond drill holes and to provide testing of areas outside the resource, sterilisation and metallurgical test holes. RC pre-collar depths range from 2 to 200m. Where sampled, the RC drilling used face sampling hammers. • Diamond core drilling (NQ, NQ2, HQ and HQ3 diameters with one BQ hole) is the predominant sample source. Standard tube was used. • Where possible diamond core has been oriented based on manual orientation spearing methods. A distinction is routinely made between oriented and non-oriented core.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery	• Diamond drill core loss (in metres) was measured in the core trays and core loss and recovery (%) recorded in geotechnical records. • Measures taken to maximize sample recovery and ensure representative nature of the samples are not known.

Criteria	JORC Code explanation	Commentary
	and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• No analysis on relationship between sample core recovery and grade has been undertaken.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography. • The total length and percentage of the relevant intersections logged.	• Diamond core and RC chips have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation. • Underground face data logging and wall mapping have been used to support mineralisation interpretation and Mineral Resource estimation. • Total length of geologically logged data is 36,622m which represents 74% of the total 44,306.89m drilled or channelled. • Logging has been conducted both qualitatively and quantitatively – full description of lithologies, alteration and comments are noted. • Qualitative structural measurements (9,956 individual measurements) have been undertaken on oriented diamond core holes.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• Diamond Core was half core and quarter core sampled depending on the core size. The core was cut so as to divide the mineralisation consistently down the holes. A minimum sample size of 0.25m and a maximum size of 1.0m • RC drillholes are typically sampled on 1.0m intervals. Pre-collar samples were typically sampled at either 2m or 4m intervals. The drill cuttings are riffle or cone split to produce a final sample of approximately 2-3 kg. • Sample size of 2-3 kg is appropriate for grain size of material. • A small number of decline, wall and face samples have been used with sample weights consistently less that the drilling. The impact of these smaller weights has been mitigated by the used of full interval compositing for estimation as described below. • As a result for all sample types, the nature, quality and appropriateness of the sample preparation technique is industry standard.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. • For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• Genalysis Assay laboratories in Perth were the primary facility used for assaying, with UltraTrace in Perth used for check assaying. • For drilling prior to 2001, original laboratory identify is unknown. • The PLA standard assaying techniques used were lead or nickel collection fire assay with a Mass Spectrometry (MS) finish for Au, Pd, Pt and peroxide fusion using HCl acid to dissolve the melt with an Optical Emission Spectrometry (OES) finish for As, Co, Cr, Cu, Ni, S. These methods are considered total digestion methods. Fire assay nickel sulphide collection technique was preferred (for samples containing chromite) to lead collection as it is efficient in collecting all PGEs and gold from a sample. • Sample preparation for the Genalysis Lab were: whole sampled dried at 140°, whole sample crushed (LM2/LM5) to 90% passing 75micron, 150g collected for pulp split and reject stored. • Analysis – Chromite reef – Genalysis method: NiS/*MS for Au(5ppd); Pt(2ppb); Pd(2ppb), Ru(2ppb), Os(2ppb), Ir(2ppb) and Rh(1ppb). • Analysis – Low grade dunite - Genalysis method: FA/*MS for Au(1ppd); Pt(5ppb) and Pd(10ppb). • Additional elements – Genalysis method: D/OES for As(0.01%), Co(20ppm), Cr(50ppm), Cu(20ppm), Ni(20ppm) and S(0.01%). • Descriptions of quality control procedures are based on previous resource reports and historical documents. • Ultra Trace, Perth was engaged to provide services for low level detection work (RC samples only) and check assaying on the Genalysis results; sample preparation was

Criteria	JORC Code explanation	Commentary
		analogous to that used by Genalysis. • Analysis – Low grade dunite (northern exploration) – ltra Trace method: FA002/FA003 for Au(1ppb), Pt(1ppb) and Pd(1ppb). • Analysis – Sterilisation RC – Ultra Trace methods: AR001 for Au(1ppb), AR101 for Cr(5ppm), Cu(0.5ppm), Ni(1ppm) and S(20ppm), AR102 for As(0.2ppm), Co(0.2ppm), Pd(10ppb) and Pt(5ppb). • Analysis – Chromite Reefs – Ultra Trace method NSF01 for Au(5ppb), Pt(2ppb), Pd(2ppb), Ru(2ppb), Os(2ppb), Tr(2ppb) and Rh(1ppb). • Additional elements – Ultra Trace method ICP104 for As(100ppm), Co(20ppm), Cr(50ppm), Cu(20ppm), Ni(20ppm) and S(0,01%). • QAQC consisted of systematic submission of field duplicates for RC samples (1 in 50); barren flushes (all drilling and trenching samples) after samples from top and middle reef chromite intervals; certified local reference standards (CRM) prepared by Gannet Holdings Pty (3 PGE+Au) at a rate of 1 in 50 in all drilling samples and two programmes of inter lab check assays (89 checks in total). • Review of documented reports noted analysis of duplicates and standards assaying information showed good correlation with original results for duplicates; good correlation with the certified standards expected results with incorrect sample identification errors noted on a small number of occasions; assaying of blanks showed acceptable results with rare incorrect submissions of CRM for blank material; and inter lab check assaying showed acceptable correlation between labs for all elements with the exception of Co in the first programme of 53 samples. The Ultra Trace assays for Co were biased low (-14%) compared to the original Genalysis assays. This issue was thoroughly investigated and shown to be a lab specific issue at Ultra Trace. Programme two (36 samples) Co assays correlated well with the original Genalysis assays. • Based on the reported information samples show acceptable levels of accuracy and precision.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel. • The use of twinned holes. • Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. • Discuss any adjustment to assay data.	• No independent sampling has been undertaken by Cube. • Drillhole assay data has not been checked against the original hardcopy laboratory assay reports. Recent drilling and face sampling assay records in digital format have been checked for significant intervals within the resource area. • No twinning of holes has been identified in the drillhole data. Underground face data has been compared against the surface diamond drilling in close proximity and shows very good correlation with the drillhole logging and the significant intersections. • Data entry and verification was completed by the PLA and its data management consultant Maxwell Geoservices of Perth WA (Maxwell) who have maintained the database. • No adjustment to assay data has been undertaken. • Pt, Pd, Au, Cu and Cr are assayed for all estimated domain intervals (total of 180.07m); there are a small number of un-assayed intervals (As - 98% assayed and Co – 79% assayed). • Samples not received or missing have had the interval left blank in the database.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. • Specification of the grid system used. • Quality and adequacy of topographic control.	• PLA Diamond and RC drillhole collars were routinely surveyed for collar location and RL using GPS methodology. Within the estimation database, drilling prior to 2001 (29 DD holes) no information is recorded for the methodology used. Of the 166 PLA DD holes, 45 have no recorded survey method, 111 were surveyed by differential GPS and 10 were estimated. Of the 29 RC holes 14 were surveyed by differential GPS and 15 have no recorded

Criteria	JORC Code explanation	Commentary
		survey method. • Work by Snowden in 2001 indicated that all drill collar locations for holes used in their estimate have been surveyed by GPS or other survey instrument. From their available data they estimated that the collar accuracy was within 1.5m in three dimensions. • Additionally validation of collar RL positions against the digital terrain model provided by PLA confirmed this level of accuracy for most holes. • The estimation has been undertaken in a local grid co-ordinates. Grid transformation conversion data from Local Grid to UTM (AMG84_52) was provided in the data set. • Downhole surveys have been routinely undertaken for all drilling 2001 to 2003. Historic drillholes (pre- PS059 were surveyed using a compass for the first survey at 0m and a single shot Eastman camera at 30m down hole intervals. The post PS059 holes (PLA holes) were surveyed with either with an Eastman single shot survey tool (with a compass reading at 0m depth) or using a Gyroscopic Deviation Tool (Surtron Technologies) every 10m down hole. • Surface tranches were surveyed by compass using a surface dtm for RL. Underground trench and face sample alignments were manually calculated. • Of the 294 holes in the database, 39 or 13% used Gyro; 174 or 59% used camera; 70 or 24% were trenches or underground face sample type; and 11 or 11% have no method listed for down hole survey method. Reports from PLA indicate that there was good correspondence between camera and gyro measurements where duplicate surveys were undertaken. Additionally analysis by the PLA of magnetic susceptibility results indicates no adverse effects on camera measurements due to the host rock magnetic field. A small number of historic holes with acid etch down hole surveys have been excluded from the database used for this estimate. • Topography was provided as a DTM file, converted from DOLA, 1:25,000 scale aerial photography with ground survey control by Whelan Survey and Mapping Pty Ltd in Broome WA. The inferred accuracy of this DTM surface was +-1.25m. • This topography is adequate for resource estimation. • Visual inspection in 3D graphics did not identify any inaccuracies with the spatial position of the drillholes.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. • Whether sample compositing has been applied.	• Drill intercept pierce point data spacing is variable ranging from 25m (along strike) x 25m (down dip) in the shallower parts to 250m below surface; increasing to 100m x 200m to a depth of 1000m below surface. Below 1000m pierce point spacing is on a 500mx500m grid approximately. • Given the nature of the mineralisation (a chromite reef deposit) this drill and sampling spacing is adequate and appropriate to determine the geological and grade continuity for reporting of Mineral Resources and the classifications applied to represent risk. • Sample compositing was over the full length of the drillhole intervals within the mineralised domains. These grade composites were weighted by length and density for estimation purposes.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• Both drilling and underground face sampling is orientated normal to the dip and plunge of the mineralisation as far as possible. • The orebodies are interpreted to be a series of separate sheets that make up a plunging synclinal feature that is in parts faulted. • Each drill hole has been oriented with the intention of intersecting the mineralisation as close to orthogonal as possible.

Criteria	JORC Code explanation	Commentary
		• Given the different orientations of drilling there is no significant risk of an introduced sampling bias resulting from drilling orientation.
Sample security	• The measures taken to ensure sample security.	• No active drilling or sampling work is currently taking place. Routine sampling, submission and storage procedures are described in PLA drilling reports. The procedures described indicate industry standard practices were followed during the drilling and sampling of all holes drilled between 2001 and 2003. • No information was available regarding the historical data from holes pre PS059. • Routine data input, validation, QAQC and laboratory follow up are described in PLA reports. The procedures described indicate industry standard practices were followed during the drilling and sampling of all holes drilled between 2001 and 2003. • Again, no information was available regarding the historical holes pre PS059.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• Several reviews have been undertaken by previous PLA company staff and independent data management consultants Maxwell, detailed in PLA reports. These audits have not revealed any material issues. • Cube conducted a data compilation review and validation prior to resource estimation which involved checks for duplicate surveys, downhole surveys errors, assays and geological intervals beyond drillhole total depths, overlapping intervals, and gaps between intervals. No significant errors were found.

Section 2 - Reporting of Exploration Results

(Criteria listed in the preceding section also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings. • The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	• The Panton PGM Project – is located in the Halls Creek Shire of Western Australia, locality of Rose Bore on three granted Mining Leases; • Mining Lease M80/103, 859.4 HA Rent paid in full till 16/3/2017 and Expiring 16/3/2028; • Mining Lease M80/104, 570.3 HA Rent paid in full till 16/3/2016 and Expiring 16/3/2028;. • Mining Lease M80/105, 828.3 HA Rent paid in full till 16/3/2017 and Expiring 16/3/2028; • The three Mining Leases are held 100% by Panton Sill Pty Ltd, which in turn is a wholly owned subsidiary of Panoramic Resources Ltd. • There are no third parties or joint venture partners involved in the Project. • A royalty is payable to Helix (now residing with Goldfields) of 2% NSR. • The three Mining Leases were granted pre Native Title Act and so are free of native title claim. • There is according to the Company no conflict with any other tenure in the vicinity and no known impediment to operations. • The previous owners have done a substantial amount of work understanding the baseline conditions for flora, fauna, hydrology, waste characterisation. This work would require updating but it is reasonable to assume environmental approvals will be obtained.
Exploration done by other parties	• Acknowledgment and appraisal of exploration by other parties.	• A number of exploration drill holes (59) have been completed by previous owners and are considered historic data. • A significant number of these historical holes have been excluded from the estimation – of those used, the impact of unknown quality is considered to be a minor risk when they are compared to the significant number of holes and data gathered by PLA between 2001 and 2003 for which

Criteria	JORC Code explanation	Commentary
		complete QA/QC is available. • The historical data & database has been appraised and is considered to be of fair to good quality.
Geology	• Deposit type, geological setting and style of mineralisation.	• The Panton deposit is interpreted as a differentiated layered intrusion that has been folded into a south plunging syncline. A series of late stage shears offsets and disrupts the layered sequence through the model area. • The differentiated stratigraphy comprises a series of narrow chromite seams within dunite units. The focus of the resource estimate has been three chromite layers known as the Top, Middle and Bottom reefs. PGE mineralisation is associated with sulphides within the chromite seams. Both the grade and thickness of the reefs decrease down the stratigraphic order. • PLA identified the reef mineralisation by a 2ppm Pt+Pd+Au threshold within zones of elevated chromite grade. • At this cut-off most of the upper reef comprises a seam of one to two metres thick.
Drill hole Information	• A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes: o easting and northing of the drill hole collar o elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar o dip and azimuth of the hole o down hole length and interception depth o hole length. • If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	• Complete drill hole collar location survey data and mineralised intercept tables are included as Appendix II of this release.
Data aggregation methods	• In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated. • Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. • The assumptions used for any reporting of metal equivalent values should be clearly stated.	• All public reporting of exploration results is historic in nature and was undertaken by PLA during the drill out of the project between 2001 and 2003. This information is available in the public domain and has no material impact on the information contained in this release.
Relationship between mineralisation widths and intercept lengths	• These relationships are particularly important in the reporting of Exploration Results. • If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. • If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').	• In the majority of cases, the drill intercept lengths approximate the mineralisation widths as the drilling has been deliberately targeted to test the mineralisation true width. The mineralisation is variable in dip and azimuth across the project and as a result a constant orthogonal drill orientation to mineralisation is difficult to achieve.
Diagrams	• Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	• Drill hole plans and orthogonal views of the mineralisation and drilling were undertaken by PLA where appropriate.
Balanced reporting	• Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.	• All public reporting of exploration results is historic in nature and was undertaken by PLA during the drill out of the project between 2001 and 2003. This information is available in the public domain and has no material impact on the information contained in this release.
Other substantive exploration data	• Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical	• All meaningful and material exploration data has been reported historically by the operating company PLA.

Criteria	JORC Code explanation	Commentary
	survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	
Further work	• The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling). • Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	• Planned further work is expected to include the infill and confirmation of the Indicated and Inferred portions of the mineralisation.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. • Data validation procedures used.	• Database was maintained by PLA with assistance from Maxwell Consultants. Maxwell conducted validation and audit services on behalf of PLA over the period 2001 to 2003. • Cube completed validation checks on the database comparing collar points to the topography, maximum hole depths checks between tables and the collar data. Cube also verified the data using visual inspection of the drillholes in 3D to identify inconsistencies of drill hole traces.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The Competent Person did not conduct a site visit and they relied on information from the PLA company geologist Tony Greenaway who has been to site on numerous occasions. At the time of the original resource estimate (2001) undertaken under the JORC 2004 code PLA deemed a site visit unnecessary as the geological interpretation was undertaken by PLA geologists. As no active drilling or sampling is underway a site visit at this time is considered un-productive by Cube and the current operators. • In the advent of further drilling and a Resource up-grade a site visit by the relevant CP is strongly recommended.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. • Nature of the data used and of any assumptions made. • The effect, if any, of alternative interpretations on Mineral Resource estimation. • The use of geology in guiding and controlling Mineral Resource estimation. • The factors affecting continuity both of grade and geology.	• The confidence in the interpretation is high as a result of a predominance of core logging and underground mapping information from surface sampling, drilling and exploration mining activity. • Underground exploration development and wall and face mapping of the mineralisation confirm earlier drill hole logging and surface mapping. • The current geological interpretation is based on the logged and assayed chromite content within the host dunite sequence. Significant sulphide percentage was also used in the criteria to identify reef mineralisation defined by a 2PGE + Au cut off of 2ppm. • Wireframe models of the mineralised volumes have been made by independent consultants ECS using a seam modelling approach. While alternate models of the mineralisation may be possible Cube consider the current interpretation to be a valid representation of the factual drill hole and underground data available. • The mineralised dunite is interpreted to be a south plunging synclinal feature, this geological interpretation is based on geological logging of drill hole data. See Appendix I. • A series of four major shears are interpreted to cut-off or offset the mineralisation and separate the mineralisation into a series of discrete blocks.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The Mineral Resource estimate contains six major moderately to steep dipping mineralised domains. The orientation of each domain changes gradually with the progression south to north around the synclinal structure.

Criteria	JORC Code explanation	Commentary
		• The two interpreted mineralised chromite reefs have a downhole intercept thickness of between 0.1 and 4m (average 0.4m) and an unfolded strike extent 3,500m with an unfolded depth extent of 1,750m. Mineralisation extends from surface to approximately 1,800m vertical depth below surface. Mineralisation is open at depth.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. • The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • The assumptions made regarding recovery of by-products. • Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation). • In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. • Any assumptions behind modelling of selective mining units. • Any assumptions about correlation between variables. • Description of how the geological interpretation was used to control the resource estimates. • Discussion of basis for using or not using grade cutting or capping. • The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• Due to the narrow width of the mineralisation, interval composites were generated for the two mineralised lodes, using an intercept table in the database to control compositing. • The interval composites were then weighted by their respective horizontal width lengths and density to result in an 'accumulation variable'. • The accumulation variable for all attributes estimated was then used for variogram analysis and 2D interpolation of grades. • Each of six estimation domains (for each of two lodes upper - 101 and middle -201) has been analysed and interpolated separately. • The estimated 2D block values were then exported back into 3D space. • The use of a 2D accumulation method is considered by Cube to be appropriate for this style of mineralisation. • Grade items, Pt, Pd, Au, Ni, Cr, Cu and Co were estimated for each domain in both lodes. At the completion of the estimate a regression formula has been used to assign grades for the rare PGE's Os, Ir, Rh and Ru. These assigned values are an indication of the expected grades and should not be used in any economic evaluation. • The estimation methodology used was Ordinary Kriging as implemented in Surpac Mining Software (Ver 4.1H). • Variogram ranges and search distances were defined in the vertical plane, ranges for all attributes estimated significantly exceeded the data spacing in all domains. • A search radius has been optimised for each domain based on the special statistics of the variogram model. The initial search radius was 300m for all domains with a second pass search of either 750m or 900m applied if required to fill un-estimated volumes. The estimations generally used a minimum of 4 and maximum of 16 samples; in domain 101A and 201A the maximum was set to 6; and to 8 in domains 101D and 201D. • The search orientation and anisotropy were based on the modelled variogram for each domain. • No by-product recoveries were considered. • Estimations of any deleterious elements were not completed for the Mineral Resource. • Estimation block size used was 50m x 50m in long section projection. • No assumptions of specific selective mining units were made as a it has been assumed that full seam width mining will be undertaken. • The mineralised domain acted as a hard boundary to control the Mineral Resource volume and estimate. • To limit the effects of extreme grades three high grade limits were applied to raw grade values prior to the calculation of the accumulation variable; gold in 101 A, B and C was cut to 1,200ppb and in 201 A, B and C to 600ppb; copper in domain 201 A, B and C was cut to 1,750ppm. A manual declustering of wedge holes was undertaken in domain 201BC. • Block model validation was undertaken using the comparison of block model estimate to drill hole data composites of horizontal width and density weighted mean grades. These showed acceptable correspondence for all estimated attributes in domains A, AB, B and C. The comparison within D domains shows higher variation due

Criteria	JORC Code explanation	Commentary
		to the data clustering and wider data spacing of this domain. • A validation estimate was undertaken using inverse distance squared and compared to the OK estimate. The two estimates show some local differences due to the weighting strategies but on balance align well on a domain by domain comparison. • A final visual validation involved comparing contoured raw composite grade against estimated block grades. This validation did not highlight ant significant errors with estimated block closely matching the underlying raw grade trends.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• Moisture was not considered in the density assignment and has been estimated as dry tonnes.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• No Cut-off grade has been used for reporting. The mineralisation has been defined using a combination of geological information and grade criteria and the reported estimated grades represent a total metal content of mineralised material – all of which is expected to be mined, without selectivity due to the thin vein nature and high value of the mineralisation.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• Mining of the Panton deposit is envisaged to be by open pit and underground methods. An assumption of non-selective total vein width mining has been made in the estimation, no other mining factors were considered during the interpretation and 2D modelling of the mineralisation however mining dilution and mining loss are likely to be material factors in combination of small open pit and underground exploitation. • Minimum mining widths were not considered during the interpretation and 2D modelling of the mineralisation.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• No specific metallurgical factors were considered during the interpretation and 2D modelling of the mineralisation. • Two studies containing an assessment of the metallurgical amenability of the mineralisation have been undertaken; the first as part of the 2003 BFS by Lycopodium and an updated project review in March 2012 by Tetratech; a further technical review is required and planned by the current owners. • The assumed extraction methodology is by floatation to produce a concentrate which is further treated on site by hydrometallurgical processes to produce separate PGM and base metal concentrates. • Alternative processing options such as Kell that offer reductions in capital and operating costs, and/or improvements in metal recovery may have a material impact on economics, and should be investigated. • Synergies with the nearby Savannah operations have been factored into the model, and could offer additional project upside once confirmed by relevant studies. • Studies of the reasonable prospects for eventual economic extraction have been undertaken. Financial modelling has been recently updated by the owners to reflect current broker/bank long-term metal pricing and A$:US$ exchange rate forecasts. The modelled scenario assumes production of approximately 600kt/year for an initial seven years. Platinum is the dominant contributor to revenue, at 50% in the base case. Other significant revenue metals are Pd 36%, Au 7%, and Ni 5% for the base case model. In this updated base case scenario, which Cube considers is based on reasonable cost and revenue factors, the project demonstrates a positive NPV and modest IRR. The project is sensitive to exchange rate and commodity prices. Improvements in operating costs and metal recoveries offer significant opportunities to the Company.

Criteria	JORC Code explanation	Commentary
		• It is strongly recommended that such studies be undertaken on these two key items as part of the planned work.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• No assumptions were made regarding environmental restrictions.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• Density determinations were undertaken in the top reef (117) across all chromite domains, within the middle and lower reef a lessor number (56) of determinations were undertaken and within non-chromite material 516 determinations were made. • A database of bulk density data for every assayed chromite reef interval was calculated based on a regression formula (derived from measurements by PLA's consultants Geostokos Ltd, Budge, 2002) and using actual bulk density measurements where they were undertaken by PLA on HQ and NQ core. • Density measurements were undertaken using a core cylinder measurement technique, with 10% being determined by water emersion methods. Given the shallow weathering profile of the project area these density measurements on competent core are considered representative of the mineralised material. • The dry bulk density has been estimated into blocks by back calculation of ordinary kriged horizontal width x density accumulation composite data.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• Resource blocks have been classified as Measured, Indicated or Inferred on the basis of a range of criteria. • The Resource classification applies to the estimated block grade items only (Pt, Pd, Au, Ni, Cr, Cu and Co). Cube considers that the regressed grades for rare PGE's Os, Ir, Rh and Ru are an indication of the grade and should not be used in definitive economic analysis. • The key criteria considered by Cube were geological continuity and confidence in reef volume; data spacing and distribution; appropriateness of the modelling technique; and estimation quality parameters such as search strategy, number of informing composite data, average distance from informing composites and kriging variance. • Within the reef domains the key estimated items Pt, Pd, and Au exhibit relatively low nugget (10 to 30%) variogram models with ranges of 100 to 500m. With the exception of Au the estimated items are shown to be statistically of low variability with CV's of less than 1. These factors combined with the geological nature of the reef estimation domains establish a moderate to high confidence in the metal continuity within the reefs. • Data spacing within the most densely drilled area of the project range from 25x25 to 50x100 metres; this area extends from surface at approximately 1450m RL to approximately the 1200m RL. Below this zone drilling density widens to between 50x100 and 100x200m spaced intersections to a depth of 1000m RL, while from 1000 to 800mRL data spacing is in excess of 200m and clustered. See Illustration long sections in Appendix I, below. • Cube contends that the use of the 2D accumulation estimation method has a number of risk minimising

Criteria	JORC Code explanation	Commentary
		advantages: it simplifies the complexity of undulating reef geometry which can yield uncertain search outcomes; it eliminates the need for multiple orientation defined domains which have no geological significance and allows more informing composites to be used; and it allows an optimised estimation block size to be chosen independently from the volume definition model requirements, minimising over smoothing due to a small and inappropriate block size choice. • Measured Resource are defined where geological continuity risk is considered low, confidence in metal continuity is considered high due to the data spacing; and where the estimation quality is high as indicated by a low estimation block variance (within the first 30th percentile). Generally within the Measured part of the Mineral Resource blocks have been estimated using 10 or more composite data at an average distance of less than 200 metres (within the modelled range of most variograms). • Indicated resources are defined where geological and metal continuity risk is considered moderate to low. Generally within the Indicated part of the Mineral Resource block have been estimated using 6 or more composite data at an average distance of less than 300 metres (within the modelled range of some of the variograms). • Inferred Resources are defined by that area of the Mineral Resource with moderate confidence in the continuity of the geological model and metal where drill spacing is wider than 200m by 200m. • The Mineral Resource estimate appropriately reflects the Competent Person's view of the deposit.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• Several reviews have been undertaken for the Mineral Resource estimate. An external review was completed by a third party consultant, and an internal peer review of the estimation methodology was conducted. • The interpreted mineralisation wireframe has been reviewed by PLA and other qualified professionals in Cube.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• Due to the wide spaced drilling in areas, local variations can be expected within the narrow undulating chromite reefs. The orientation of the chromite reefs may be affected by regular structural offsets and bifurcations. • Given the geostatistically determined nature of the mineralisation, the use of OK has assisted in reducing the risk associated with the metal estimates. The additional benefit of OK is it inherently assists in declustering the data during the estimate. • No specific analysis or estimate of the relative accuracy or to establish confidence limits has been undertaken. However previous estimations by different methodology have not resulted in significantly different tonnages or contained metal estimates. • The Mineral Resources constitute a global resource estimate. • Underground exploration sampling data from the decline confirms the nature and grade tenor of the mineralisation as intersected by the drilling. • No large scale production data is available.

PJD-53

2027

PANORAMIC RESOURCES LTD

28 April 2016



VISION | COMMITMENT | RESULTS

ASX:PAN

Quarterly Report for the period ending 31 March 2016

Significant Points

GROUP

- Safety – **Group LTI Frequency Rate at zero after recording 397 days without an LTI**
- Liquid Assets – **$27 million (including ~$6 million of nickel in concentrate)**

NICKEL

Savannah

- Production – **2,861t Ni in concentrate, a new quarterly production record**
- Costs – aggregate site costs down, payable cash costs down to A$3.48/lb Ni (inclusive of royalties), C1 Cash Cost A$2.09/lb
- Operational changes – **decision made to stop capital development and to put the project onto care and maintenance in May 2016 due to low US$ nickel price**
- Savannah North – **positive Scoping Study released**
- Exploration – 2016 Savannah North drill programs commenced, with excellent intersections in seven of the first eight holes

Lanfranchi

- Lower Schmitz – **maiden Resource of 131,000t @ 5.1% Ni for 6,700t Ni contained**

GOLD

Gum Creek

- **Positive Scoping Study released**
- Optimisation work continuing in conjunction with the sale process
- Fourteen new geophysical targets identified along the Wilsons shear zone

PGM

Panton

- Evaluation and research studies continuing

Thunder Bay North (TBN)

- Rio completed its winter drilling program at TBN as part of the C$20 million farm-in over 5 years to earn 70%
- Semi-airborne HeliSAM™ magnetics survey to be undertaken over the project area in June 2016

CORPORATE

- Renounceable Pro-rata Entitlement Offer – closed on 26 April 2016 with a take-up from eligible shareholders of approximately 77.8% - shortfall fully underwritten
- Hedging – nickel hedging in place to cover quotational period (QP) pricing risk on remaining concentrate shipments

Group Summary

Safety

No lost time injuries (LTI) were recorded at the operations.

The 12 month moving average Group LTI Frequency Rate (LTIFR) **has dropped to zero (down from 0.93).** This is after recording 397 days without an LTI across the Group. Figure 1 shows the Group actual LTIFR against the 2014/15 WA Nickel Industry Average LTIFR of 3.30, as published by the WA Department of Mines and Petroleum (DMP). Figure 2 shows the Group Hazard and Incident Reports over the last 12 months. Hazards reported and the number of incidents fell during the quarter.

Figure 1 – Group LTIFR Performance (12 month rolling average)

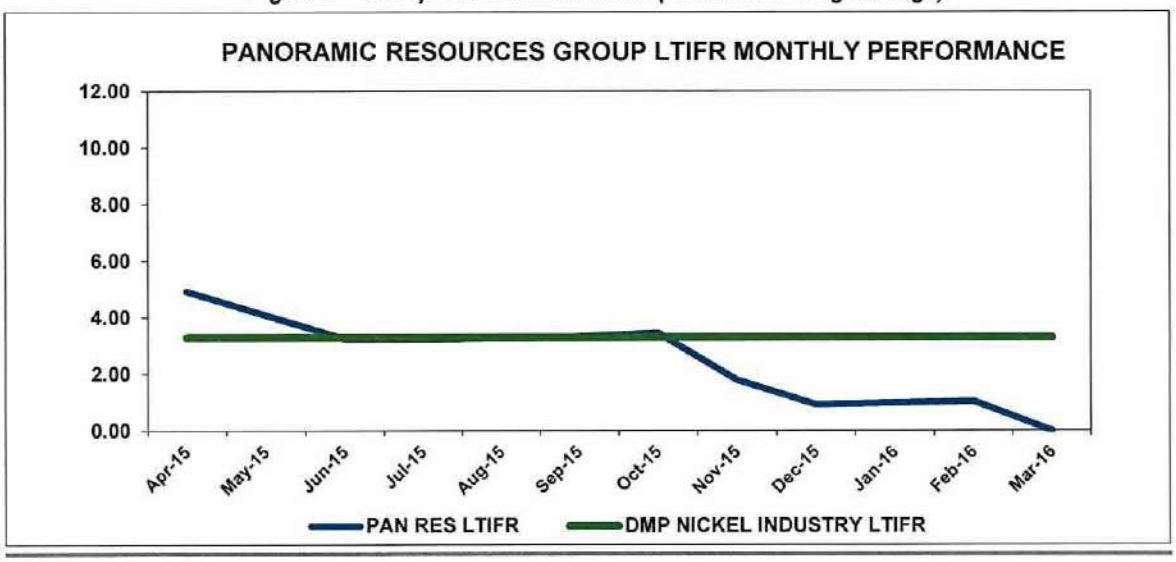
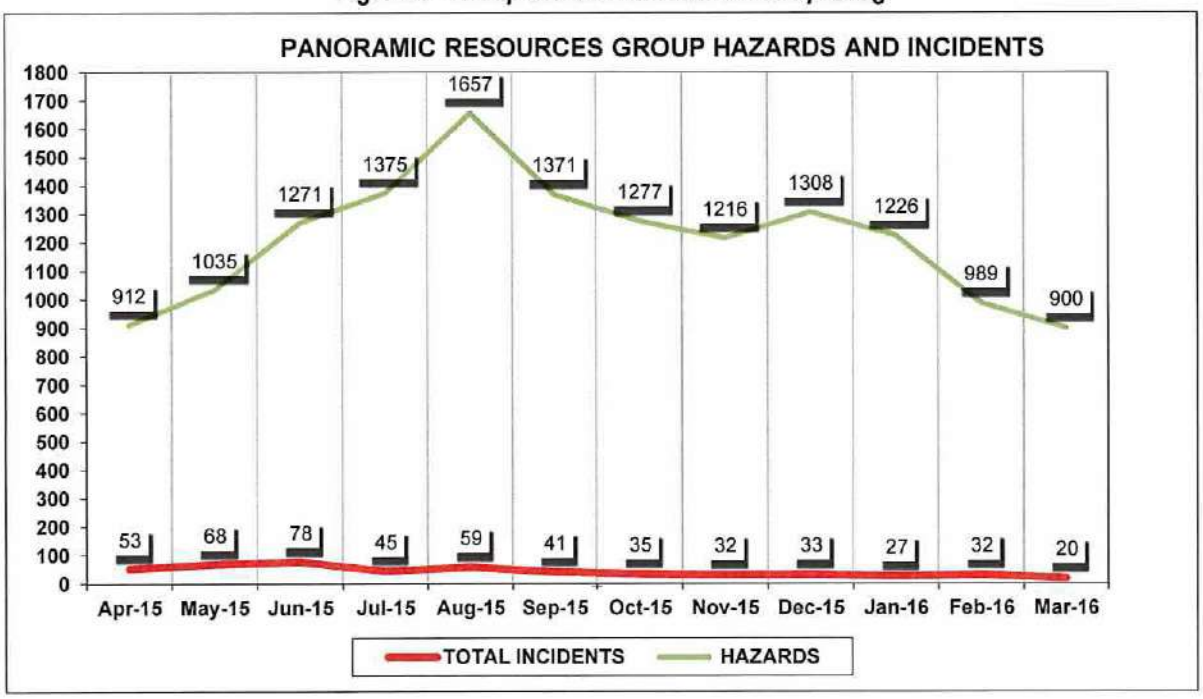


Figure 2 – Group Hazards and Incidents Reporting



Environment

There were no significant environmental incidents recorded and the operations were conducted within all statutory, regulatory and licence conditions.

Nickel – Savannah Project

General

The Savannah Project produced 2,861t Ni, 1,816t Cu and 139t Co contained in concentrate. **This result was a new quarterly production record.**

Total ore mined dropped by 9% to 236,370t, as a result of completing mining the Copernicus open pit early in February. Both total ore milled and nickel head grade increased by 5% to 251,456t and 1.33% respectively.

The record nickel production, lower site aggregate costs (excluding redundancy costs) and the higher average milled nickel head grade resulted in a 38% decrease in the quarterly payable cash costs to A$3.48lb (*Table 2*).

Four concentrate shipments for a combined 2,901t contained nickel were exported. At 31 March 2016, there was 570t of contained nickel in concentrate on hand valued at ~$6 million.

Table 1 – Savannah Project Operating Statistics (including Copernicus)

Area	Details	Units	3 mths ending 31 Mar 2016	3 mths ending 31 Dec 2015	2015/16 YTD	2014/15 Full Year
Mining	Ore mined	dmt	**236,370**	**259,581**	735,313	865,660
	Ni grade	%	**1.34**	**1.26**	1.28	1.18
	Ni metal contained	dmt	**3,172**	**3,273**	9,407	10,258
	Cu grade	%	**0.79**	**0.67**	0.73	0.66
	Co grade	%	**0.06**	**0.06**	0.06	0.06
Milling	Ore milled	dmt	**251,456**	**238,637**	717,479	854,794
	Ni grade	%	**1.33**	**1.26**	1.28	1.18
	Cu grade	%	**0.78**	**0.67**	0.73	0.66
	Co grade	%	**0.06**	**0.06**	0.06	0.06
	Ni Recovery	%	**85.5**	**86.0**	86.0	86.4
	Cu Recovery	%	**92.7**	**92.7**	93.2	94.1
	Co Recovery	%	**89.4**	**87.8**	88.4	88.5
Concentrate Production	Concentrate	dmt	**38,141**	**36,387**	109,449	119,084
	Ni grade	%	**7.50**	**7.14**	7.21	7.33
	Ni metal contained	dmt	**2,861**	**2,599**	7,887	8,726
	Cu grade	%	**4.76**	**4.05**	4.48	4.46
	Cu metal contained	dmt	**1,816**	**1,475**	4,905	5,314
	Co grade	%	**0.36**	**0.34**	0.35	0.37
	Co metal contained	dmt	**139**	**122**	380	443
Concentrate Shipments	Concentrate	dmt	**39,498**	**38,285**	105,509	122,262
	Ni grade	%	**7.34**	**7.12**	7.25	7.31
	Ni metal contained	dmt	**2,901**	**2,727**	7,649	8,936
	Cu grade	%	**4.57**	**4.20**	4.48	4.39
	Cu metal contained	dmt	**1,807**	**1,608**	4,732	5,368
	Co grade	%	**0.34**	**0.33**	0.34	0.36
	Co metal contained	dmt	**136**	**125**	357	445

Payable Cash Costs

Table 2 – Savannah Project Payable Cash Costs (including Copernicus)

	Units	Savannah 3mths ending 31 Mar 2016	Savannah 3mths ending 31 Dec 2015
Costs Per Pound Payable Nickel			
Mining	A$ per lb	2.25	3.52
Milling	A$ per lb	1.22	1.46
Administration	A$ per lb	0.73	1.20
Payable Operating Cash Costs (Mine Gate)	**A$ per lb**	**4.20**	**6.18**
Haulage	A$ per lb	0.35	0.31
Port Charges/Shipping	A$ per lb	0.26	0.35
Ore Treatment	A$ per lb	-	-
Net By-product Credits	A$ per lb	(1.67)	(1.59)
Royalties	A$ per lb	0.34	0.36
Total Payable Operating Cash Costs[a]	**A$ per lb**	**3.48**	**5.61**
Total Payable Operating Cash Costs [b]	***US$ per lb***	***2.51***	***4.04***

(a) *Savannah capital development cash cost for the quarter was A$0.16/lb. This cost is not included in Table 2. Capital development costs represent capitalised mining cash costs for deposits in production. These costs do not include any pre-production costs for deposits being developed for future mining.*

(b) *Average March 2016 quarter RBA US$/A$ settlement rate of US$0.7209 (Average December 2015 quarter exchange rate was US$0.7202).*

Care and Maintenance

On 27 January 2016, the Company announced that due to the weak US$ nickel price and uncertainty around the timing of a price recovery, the Project was to (1) immediately stop developing new stoping areas and (2); begin a transition onto care and maintenance. On 26 February 2016, the Company updated on discussions with key stakeholders and the work on site to maximise ore production and mill throughput in an efficient and safe manner until the operation is put onto care and maintenace in mid-May 2016.

The net cost to place Savannah onto care and maintenance will depend on revenue received from production until mid-May and the ability to run down inventories of consumables and extinguish outstanding creditor payments over that period. The remaining employee redundancy costs to place the Savannah site onto care and maintenance are approximately $8.6 million

By taking this decision, the remaining Savannah Resource will be preserved until the US$ nickel price returns to a sustainable level where the mining of the Savannah orebody, mostly likely in conjunction with the development of Savannah North, provides an acceptable return to shareholders.

Restart Plan

The remaining Mining Inventory at Savannah, after the operations are placed onto care and maintenance, can be readily accessed with little new capital development. The Company believes that the free cash flow generated from the ore remaining in the Savannah orebody could underpin commencing the Savannah North capital investment.

The plan during the care and maintenance period is to complete the Feasibility Study on Savannah North and to undertake additional exploration to continue testing the strike and width dimensions of Savannah North, and grow the Resource base.

Savannah North Scoping Study

As previously reported (*refer to the Company's ASX announcements of 27 January 2016 and 26 February 2016*), the Savannah North Scoping Study demonstrates there is potential to add significant mine life at Savannah through the development of Savannah North. The key operating metrics from the Scoping Study are summarised in Table 3.

Table 3 – Savannah North - Resource and Production Summary

Operating Metric	Result
Mineral Resource	**6.88Mt** @ 1.59%Ni, 0.77% Cu, 0.11% Co containing **109,600t Ni, 52,900t Cu, 7,800t Co**
Mining Inventory	**6.07Mt** @ 1.26% Ni, 0.64% Cu, 0.09% Co containing **76,500t Ni, 38,600t Cu, 5,300t Co**
Mine Life	**7.75 years**
Life-of-mine production (metal in concentrate)	**66,200t Ni, 36,700t Cu, 5,000t Co**
Annual production (metal in concentrate)	**9,500tpa Ni, 5,300tpa Cu, 700t Co** at full production

Cautionary Statement

The Savannah North Scoping Study referred to in this announcement is based on low-level technical and economic assessments, and is insufficient to support the estimation of Ore Reserves or to provide assurance of an economic development case at this stage, or to provide certainty that the conclusions of the Scoping Study will be realised.

There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target itself will be realised.

All material assumptions underpinning the production target, and forecast financial information derived from the production target, in the Company's ASX announcements of 27 January 2016 and 26 February 2016, continue to apply and have not materially changed.

The Scoping Study is based on mining Savannah North via conventional long-hole open stoping with paste fill, processing the ore through the existing Savannah plant to produce a bulk nickel/copper/cobalt concentrate. The Scoping Study demonstrates a relatively low initial capital investment of $42 million to achieve full production (on a stand-alone basis), with the existing mine, processing plant and supporting infrastructure of the Savannah operation. Forecast average C1 cash costs (nickel-in-concentrate after by-product credits) are at the lower end of the industry cost curve at US$2.20/lb Ni over the life of the project. Table 4 summarises the financial outcomes of the Scoping Study at various US$ nickel prices and a flat US$:A$ 0.70 exchange rate.

Table 4 – Savannah North - Financial summary for a range of US$ nickel prices (US$:A$ = 0.70)

Financial Metric	Units	US$5.00/lb	US$6.00/lb	US$7.00/lb	US$8.00/lb
Revenue	**A$M**	892	1,032	1,179	1,319
Initial Capital *(Pre-production and ramp-up)*	**A$M**	42	42	42	42
LOM Capital *(inclusive of initial capital)*	**A$M**	137	137	137	137
Operating costs plus royalties	**A$M**	700	708	715	722
Pre-tax cashflow	**A$M**	54	187	327	460
Pre-tax NPV (11% discount rate)	**A$M**	6	80	158	232
IRR	**%**	14	47	82	118
C1 cash cost *(Ni in concentrate basis)*	**A$/lb Ni**	3.14	3.14	3.14	3.14
	US$/lb	2.20	2.20	2.20	2.20
Payable Ni cash costs	**A$/lb Ni**	5.19	5.26	5.29	5.36
	US$/lb	3.63	3.68	3.70	3.75

As part of the Scoping Study, Panoramic identified the following enhancement opportunities which could add significant additional value to the Project:

- **Future Resource growth** – less than 30% of the potential 2km mineralisation footprint has been tested by drilling;
- **Mining Inventory upgrade** – potential to convert further Resources into Mining Inventory by optimising the mine plan;
- **Optimised mine plan** – opportunity to lower unit costs, increase production rates and/or improve mined grades by optimising mining methods, stope shapes and cut-off grades;
- **Improved mine scheduling** – opportunities to bring forward zones of high grade mineralisation and defer some development to a "just-in-time" schedule;
- **Increased production rates** – potential to increase production rates through refinement of mining methods, further additions to the Mining Inventory, and shaft versus decline haulage;
- **Optimisation of plant throughput and recoveries** – a number of opportunities have been identified to improve metallurgical performance of Savannah ore that are expected to be directly applicable to the Savannah North mineralisation;
- **Alternative products** – assess the amenability of Savannah North mineralisation to produce separate nickel and copper concentrates, co-processing of Panoramic's Panton PGM mineralisation to produce a PGM-rich bulk concentrate, and matte production via mini-smelting technology (e.g. top submerged lance); and
- **Power** – the Scoping Study is based on utilising the existing diesel fired power station. If an alternative fuel source such as gas is available, or solar/wind systems are viable, power costs could be reduced.

The Company is targeting the completion of a Feasibility Study on Savannah North during the December 2016 quarter.

Nickel – Lanfranchi Project

General

The Lanfranchi Project has been on care and maintenance since November 2015. FY2016 production before mining ceased is shown in Table 5.

Table 5 – Lanfranchi Project Operating Statistics

Area	Details	Units	3mths ending 31 Dec 2015	3mths ending 30 Sep 2015	2015/16 YTD	2014/15 Full Year
Mining	Ore mined	dmt	-	43,692	43,692	468,491
	Ni grade	%	-	2.33	2.33	2.26
	Ni metal contained	dmt	-	1,019	1,019	10,575
	Cu grade	%	-	0.18	0.18	0.20
Ore Delivered	Ore delivered	dmt	-	46,279	46,279	470,322
	Ni grade	%	-	2.27	2.27	2.26
	Ni metal contained	dmt	-	1,051	1,051	10,611
	Cu grade	%	-	0.18	0.18	0.20

Lower Schmitz

The maiden Lower Schmitz Resource Statement was completed and is summarised in the Exploration Section of this report.

Native Title

In November 2014, the Federal Court made a Determination of native title in favour of the Ngadju People, the consequence of which was that the Company's tenements at the Lanfranchi Nickel Project were invalid to the extent that they were inconsistent with the continued existence, enjoyment or exercise of native title rights held by the Ngadju People. The Determination was subsequently appealed by some of the Respondents to the Determination, and the Company joined as a non-participating Respondent Party to the appeal.

On 29 March 2016, the Full Federal Court handed down its decision in the appeal, which overturned the initial decision and confirmed the validity of the relevant tenements. The Company understands that the Ngadju People have filed an application for special leave to appeal to the High Court.

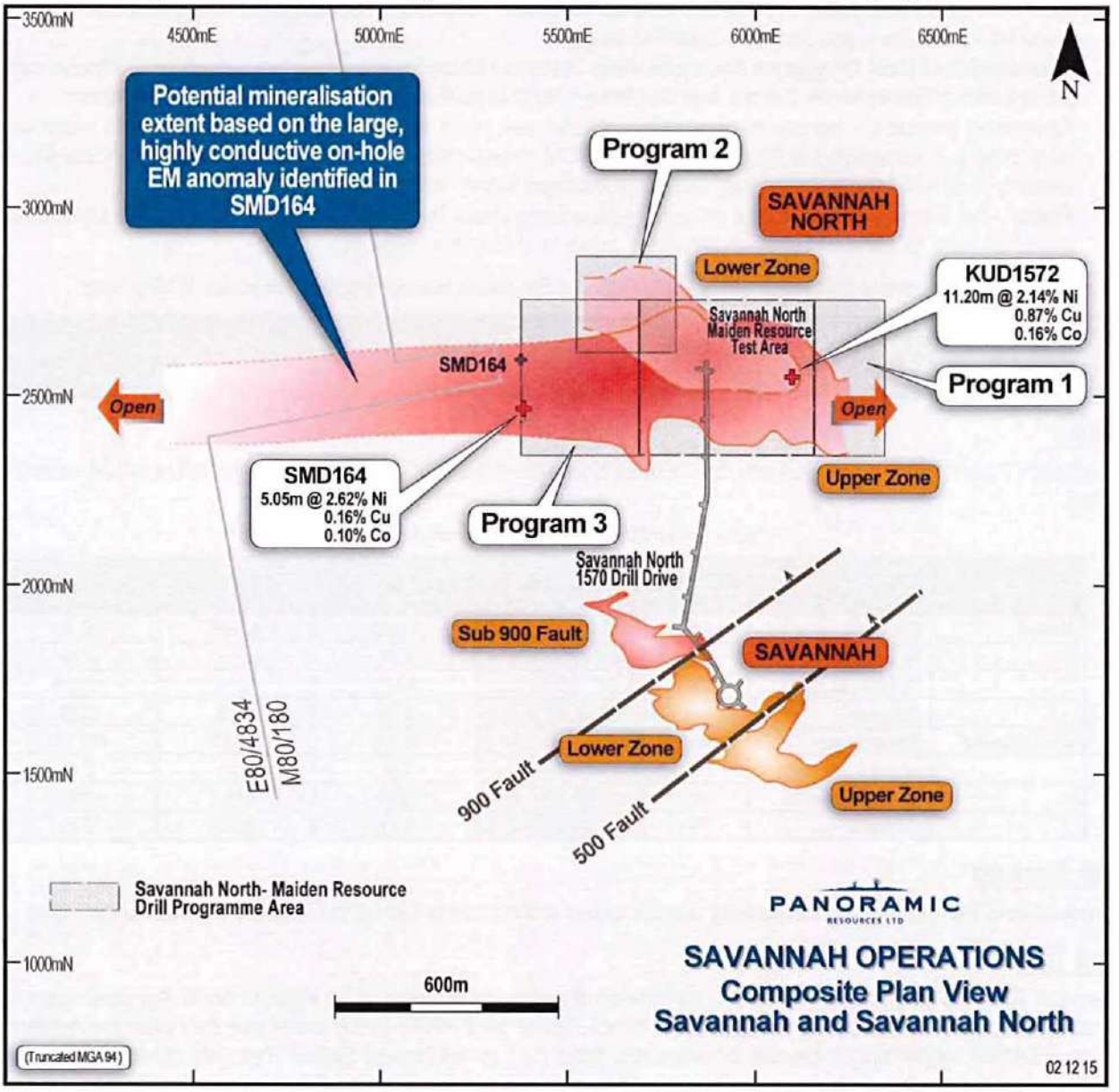
Base Metal Exploration

Savannah and East Kimberley Regional

Savannah and Savannah North Project

As previously reported (*refer to the Company's ASX announcements of 26 February 2016 and 21 March 2016*), the resumption of Resource drilling at Savannah North from late February involves three programs (*Figure 3*).

Figure 3 – Plan View showing position of the Savannah North resource drill programs



- *Program 1 – Savannah North Up-Dip to the East (High Priority)*
  The Scoping Study has highlighted that the time and capital development required to access the first ore at Savannah North could be reduced significantly if the Savannah North Resource extends up-dip towards the existing Savannah mine levels. A program of up to 20 diamond holes comprising approximately 8,000m is in progress to determine the proximity of the Savannah North mineralisation to the existing Savannah mine levels. If successful, it should lead to an increase in the Resource.

- *Program 2 – Lower Zone infill high grade Inferred area (High Priority)*
  Preliminary Savannah North mine designs completed as part of the Scoping Study have highlighted the significant contribution delivered by the deeper levels of the high-grade Savannah North Lower Zone Resource. This zone is essentially constrained by the limited number of drill intersections. In order to de-risk the mine designs in this area, greater Resource confidence is required and a ten hole program comprising approximately 7,300m is planned to infill this area.

- *Program 3 – Upper Zone western extension*
  Prior to ceasing the Savannah North Maiden Resource drill program in August 2015, it had been intended to infill (on a 100m by 100m spacing) the area between the western edge of the Maiden Resource test area (5700mE) and surface hole SMD164, a distance of ~300m *(Figure 4)*. A significant portion of the existing Inferred Resource and Unclassified mineralisation in this area could be converted to Indicated category and a 10 hole drill program comprising 5,800m is now planned. The program is also designed to provide a clearer picture of the shape and grade of the Resource in this area, which in turn could be applied to the unclassified mineralisation that has been modelled to extend for a further 1km to the west from SMD164. The results should provide a better understanding of the resource potential of Savannah North.

Results for the first seven holes of Program 1, which were targeted at Sections 6100mE and 6200mE, were very positive with significant widths and grades of mineralisation. The results are summarised below (*refer to Figure 4 for hole locations and Figures 5 and 6 for sectional views*):

- **22.97m @ 1.79% Ni, 0.75% Cu, 0.13% Co(KUD1573);**
- **15.20m @ 1.76% Ni, 0.60% Cu, 0.13% Co (KUD1574);**
- **16.40m @ 1.78% Ni, 0.96% Cu, 0.14% Co (KUD1575);**
- **23.03m @ 2.09% Ni, 1.10% Cu, 0.14% Co and 4.95m @ 2.29% Ni, 0.67% Cu, 0.15% Co (KUD1576);**
- **9.80m @ 1.33% Ni, 0.34% Cu, 0.09% Co (KUD1579);**
- **3.38m @ 0.72% Ni, 0.22% Cu, 0.05% Co (KUD1580); and**
- **12.35m @ 1.32% Ni, 0.28% Cu, 0.09% Co (KUD1581).**

Note: all intersections are down-hole lengths and not true-widths. The drill hole details and JORC Compliance Tables are contained in Appendix 2.

Figure 4 – Plan View showing position of the Savannah North Resource drill holes

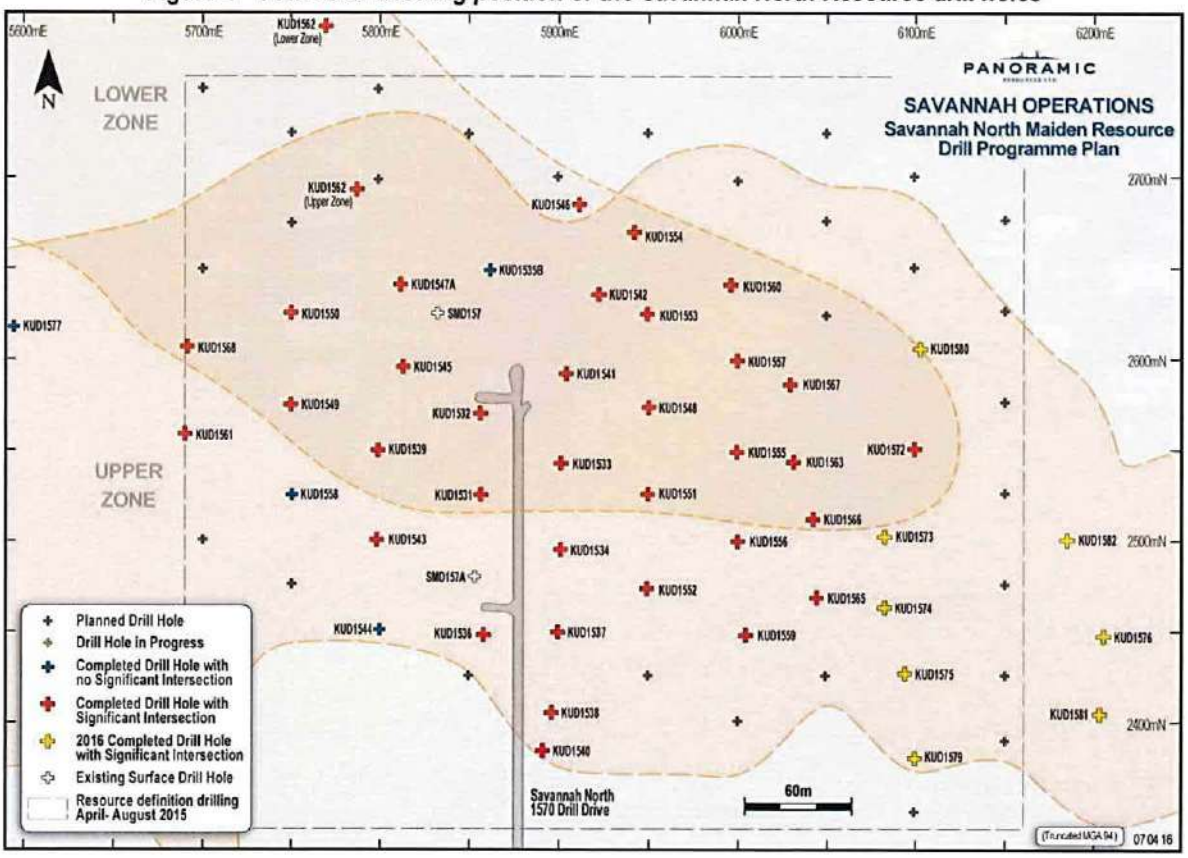


Figure 5 – Savannah North Section 6100mE showing latest drill results

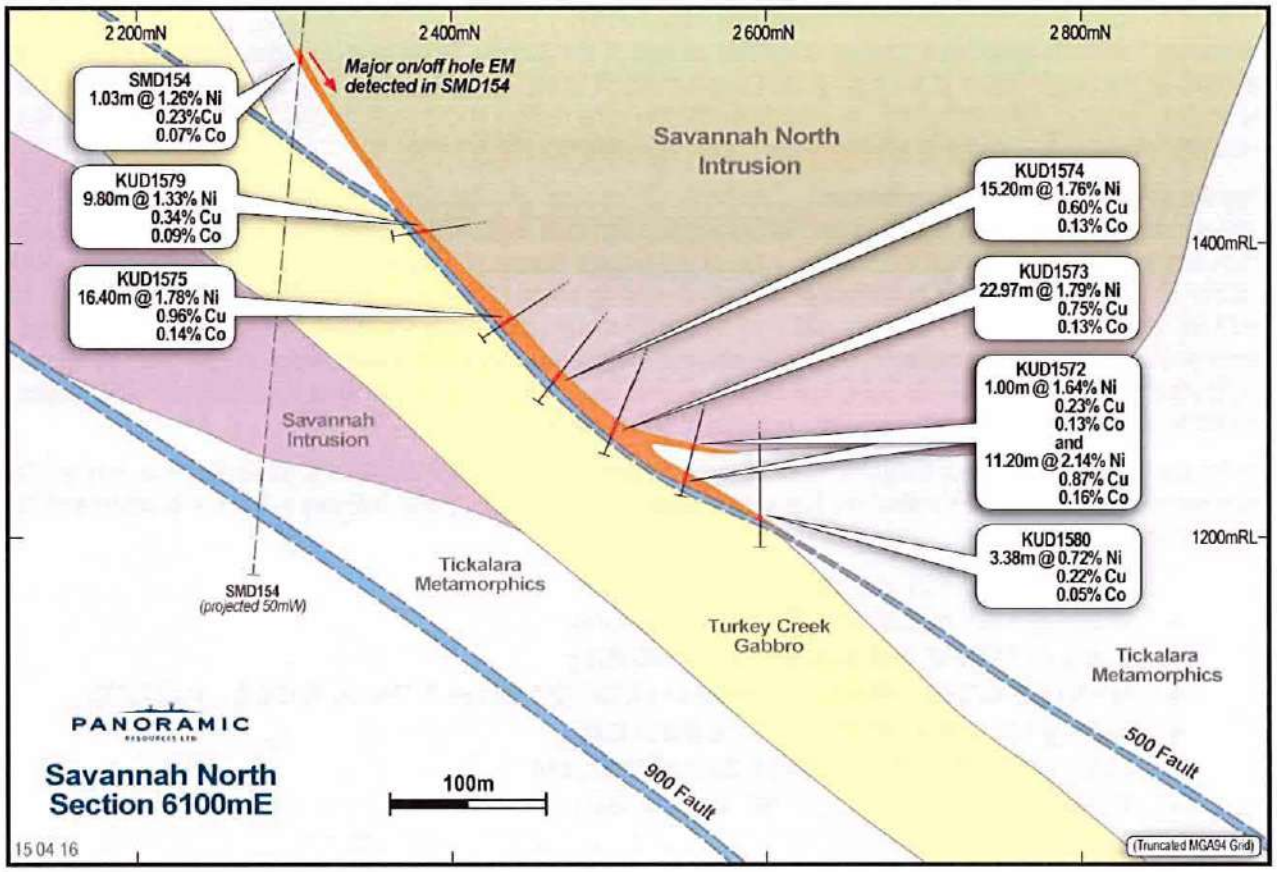
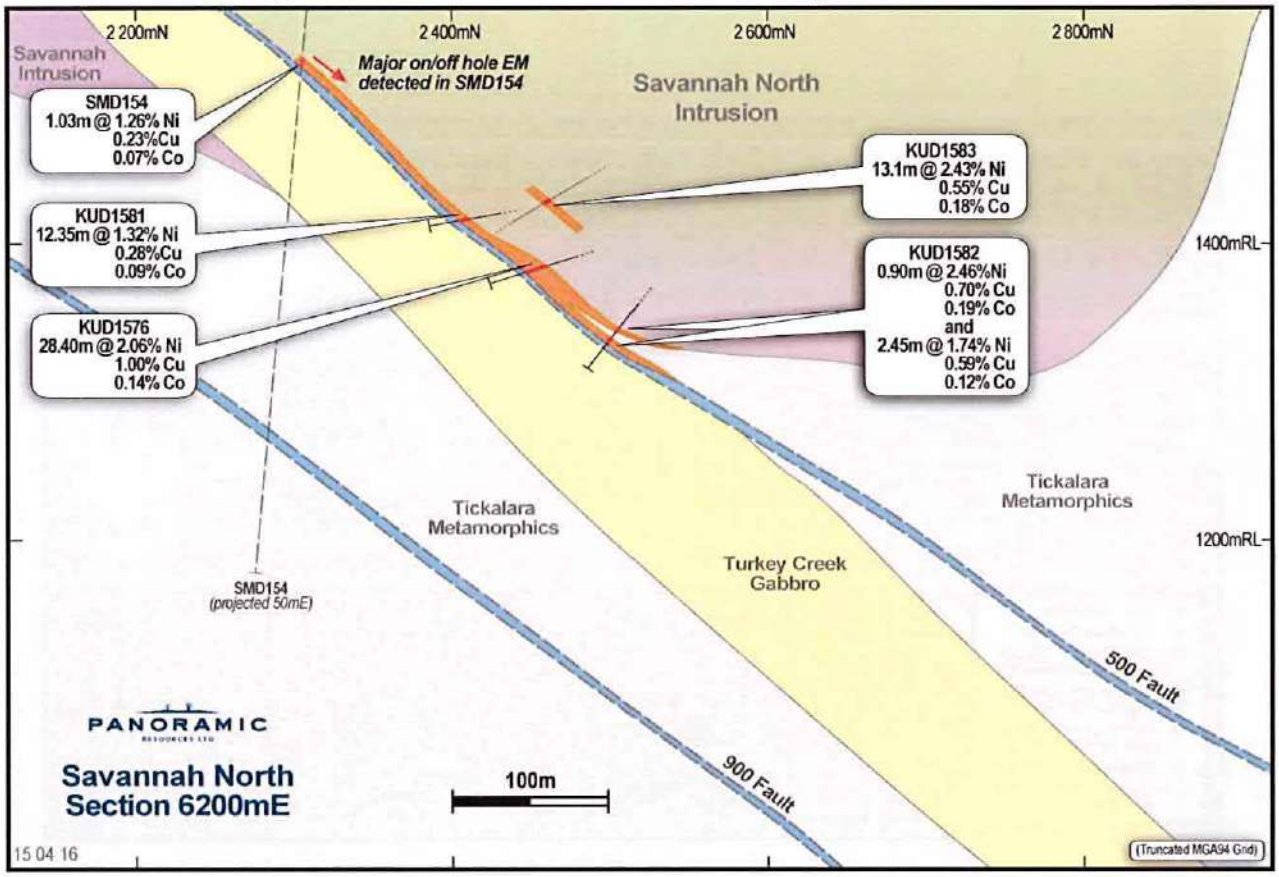


Figure 6 – Savannah North Section 6200mE showing latest drill results



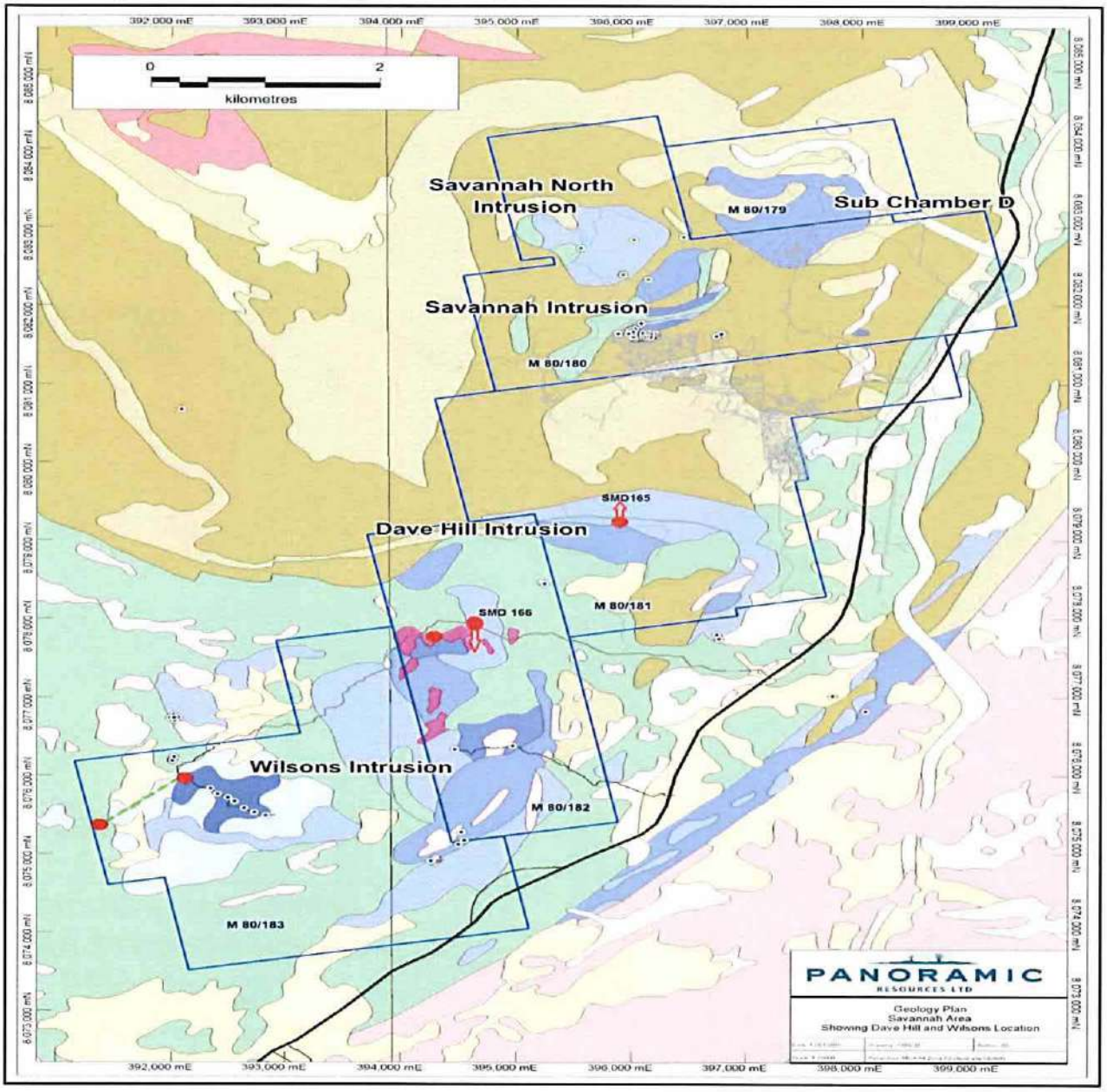
The intersections in KUD1576, KUD1581 and KUD1582 are significant as they are located 50m to the east of the existing Savannah North Resource thereby confirming the continuation of the Savannah North mineralisation up dip to the east, as predicted by the geological model.

Dave Hill and Wilsons Intrusions

An initial drill test of the Dave Hill and Wilsons Intrusions located to the south of Savannah was completed during February. Two holes, SMD165 and 166 were completed (*Figure 7*). The program was in part funded by a State Government Co-funded drilling grant of up to $150,000. The first hole of the program (SMD165) was drilled in to the Dave Hill intrusion and was completed at 751m depth. The second hole of the program, SMD166 targeting the Wilsons intrusion, was completed at 697m. Both holes encountered broad zones of mafic to ultramafic lithologies, similar in many respects to the Savannah and Savannah North intrusions.

Encouragingly, low tenor disseminated sulphide mineralisation was intersected in both holes, confirming the potential of these intrusions to host nickel sulphide mineralisation. Both holes were also EM surveyed but no significant off-hole conductors were identified. Further exploration work on the Dave Hill and Wilsons areas is justified and will be considered in the future.

Figure 7 – Savannah Geology Plan showing Dave Hill and Wilsons Intrusions and drill hole locations



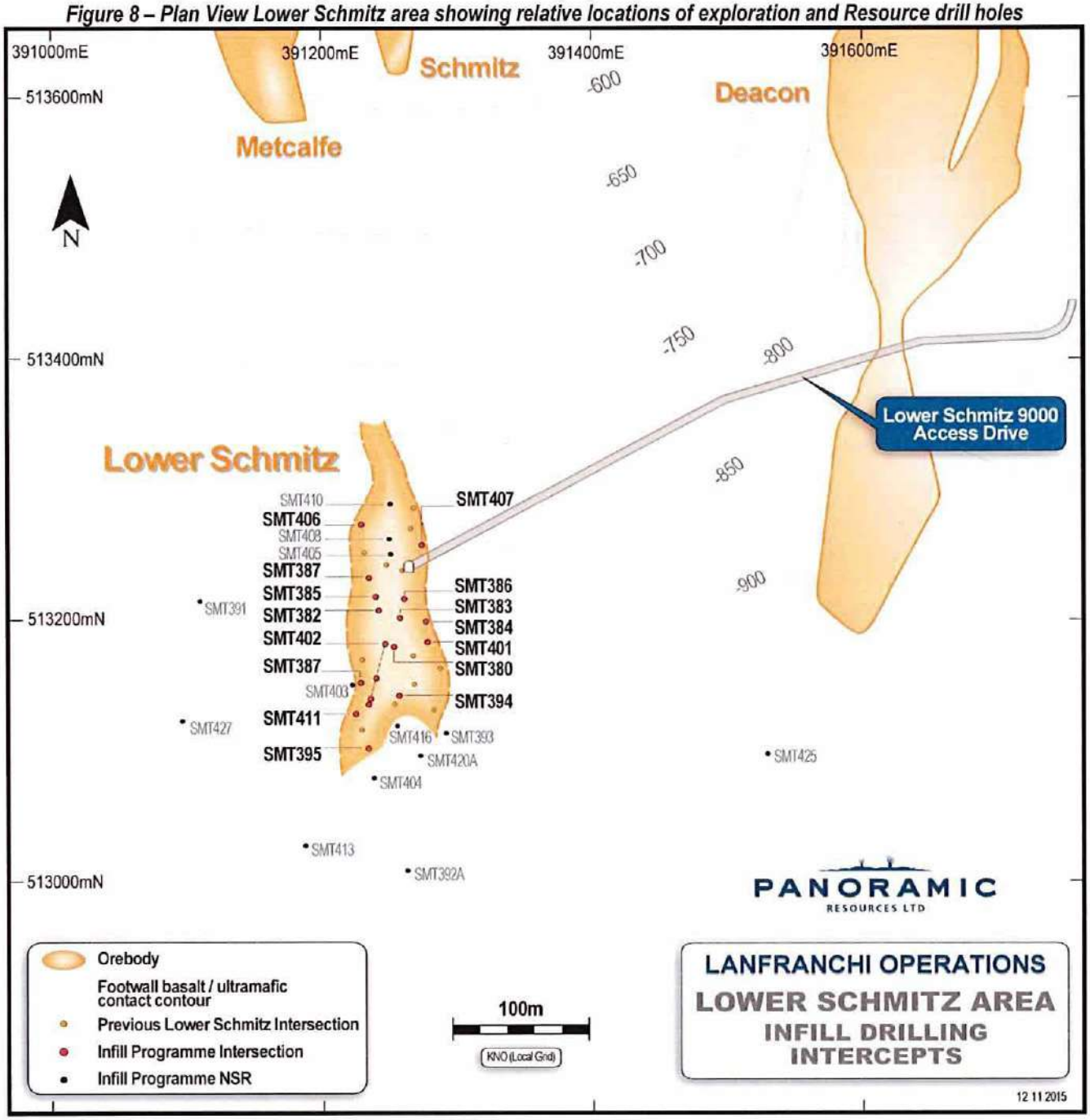
Lanfranchi

Overview

In January 2015, the Company announced the discovery of high-grade mineralisation at Lower Schmitz in drill hole SMT373A (*refer to ASX announcements of 21 and 23 January* 2015). Since the discovery, the sole exploration focus at Lanfranchi has been directed towards evaluating the Lower Schmitz area. Drilling to evaluate Lower Schmitz was initially conducted from the base of the Schmitz Decline and the Schmitz 4510 drill drive. This program finished in late July 2015 with completion of drill hole SMT380. A second, smaller infill Resource definition program was undertaken following the completion of the 9000 exploration/access drive in September 2015 (*Figure 8*).

The infill Resource drill program, which involved a total of 28 drill holes for 4,765 drill metres was completed in mid-November 2015.

Figure 8 – Plan View Lower Schmitz area showing relative locations of exploration and Resource drill holes



Lower Schmitz – Maiden Mineral Resource Estimate

Following the completion of the infill program in November 2015, work commenced on the estimation of a maiden Mineral Resource for Lower Schmitz.

The maiden Lower Schmitz Mineral Resource Estimate was completed by Payne Geological Services Pty Ltd (PayneGeo). The results (*Table 6*) and following summary of the estimation process is taken from an extract from the PayneGeo Resource Report. A plan and sectional view, showing the shape of the Resource are presented in Figures 9 and 10. Details of the drill holes used to estimate the Resource and JORC Compliance Tables are contained in Appendix 1.

Summary of Payne Geological Services Report

"The Lower Schmitz ("LS") nickel deposit lies within the Lanfranchi Nickel Mine ("LNM") which is wholly owned by Panoramic Resources Limited ("PAN"). PAN acquired the mine in 2004 and operated it until late 2015 before placing the operation on care and maintenance.

The deposit lies approximately 400m to the south, and 200m vertically below the previously mined Schmitz orebody. It was discovered in 2015 by drill testing of an EM conductor down plunge of the Schmitz orebody and has now been delineated by 30 drill holes which have been drilled from a number of underground development headings. The Mineral Resource estimate is summarised in Table 6.

Table 6: Lower Schmitz – February 2016 Mineral Resource Estimate @ 1.0% Ni Cut-Off

Classification	Tonnes (t)	Ni (%)	Cu (%)	Co (%)	Ni (t)	Cu (t)	Co (t)
Measured	-	-	-	-	-	-	-
Indicated	51,000	5.6	0.4	0.10	2,900	200	50
Inferred	79,000	4.8	0.3	0.09	3,800	300	70
Total	**131,000**	**5.1**	**0.4**	**0.09**	**6,700**	**500**	**120**

The LS mineralisation is typical of many Kambalda nickel sulphide ("NiS") deposits, with massive and matrix sulphides occurring within a ribbon-like channel on the basal contact of the ultramafic unit. The true thickness of the LS deposit averages 2.6m, but is up to 6.4m thick in places.

The Mineral Resource has been defined by underground diamond drill holes over a length of 240m with an average width of 70m. The upper limit of the defined mineralisation is at a depth of 1,100m below surface and extends over a vertical extent of 170m.

The typical intersection spacing is approximately 20m by 20m in the central portion of the deposit. Initial holes drilled from the Schmitz mine area have intersected the LS mineralisation at a close angle. Later holes were drilled from a dedicated drill drive and have more optimal intersection angles.

A single mineralisation wireframe was prepared by PAN geologists based on detailed geological interpretation. A nominal cut-off grade of 1.0% Ni was used, but the majority of contacts are based on logged sulphide accumulations displaying sharply defined contacts. Due to highly variable intersection angles of the drilling, vein composites were prepared for each drill hole, to provide a single composite per hole. Length and density weighting of the raw sample data was used to prepare the composites. Grade estimation was carried out using inverse distance squared ("ID2") interpolation with the ellipsoid oriented down the plunge of the mineralisation.

Density in the block model was based on a regression formula derived from the raw sample data within the wireframes.

The portion of the deposit defined by drill spacing of 20m or less and lying in the highly tabular base of the interpreted channel was classified as Indicated Mineral Resource. The remainder of the deposit was classified as Inferred Mineral resource.

The deposit remains open down plunge. If economic potential of the defined Mineral Resource can be demonstrated, further drilling is warranted to test the extent of the high grade mineralisation at depth. Future estimates should consider the use of separate estimation domains for massive/matrix and disseminated sulphide mineralisation".

Figure 9 – Simplified Lower Schmitz Geological Cross Section 513175mN

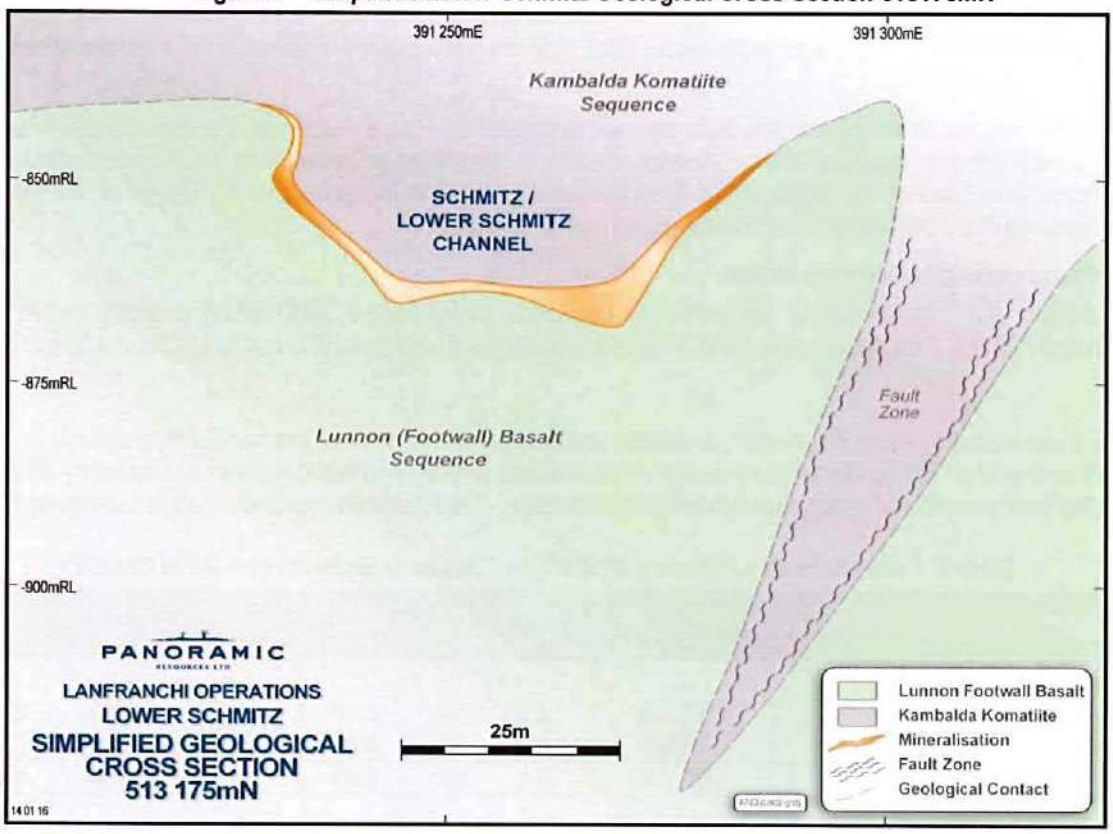
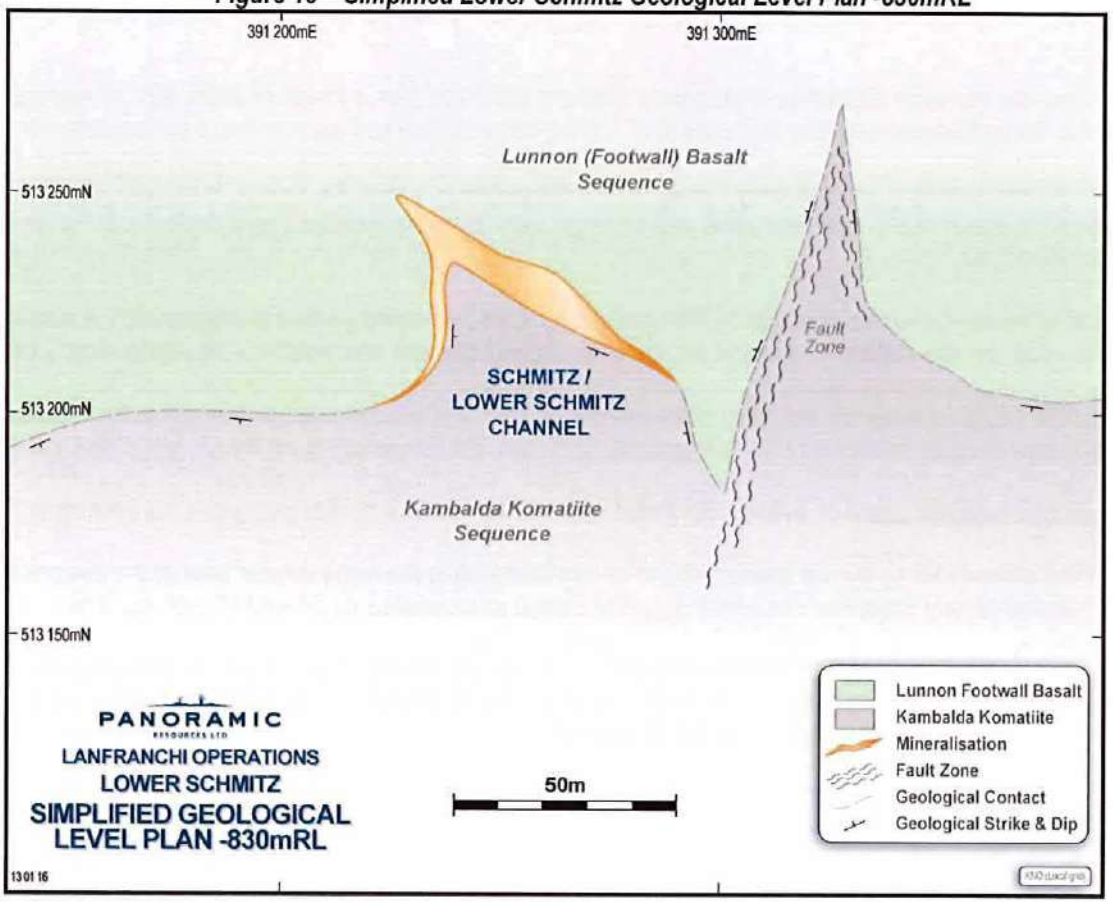


Figure 10 – Simplified Lower Schmitz Geological Level Plan -830mRL



Cowan Nickel Project, WA (Panoramic holds 100% nickel rights)

The Company withdrew from Cowan Nickel Project during the quarter.

Drake Resources Exploration Alliance - Scandinavia

During the quarter, Panoramic withdrew from the alliance with Drake Resources Limited ("Drake") that was set up to identify, explore and develop base and precious metal opportunities across Scandinavia.

Gold – Gum Creek Project

Updated Scoping Study

An updated Scoping Study on the Gum Creek Gold Project (previously known as the Gidgee Gold Project) was released in March and is based on the processing of free milling open-pit Resources only (*refer to the Company's ASX announcements of 18 March 2016 and 22 March 2016*). The Study demonstrates a robust Project at an A$1,700/oz Base Case gold price (*Table 7*), with the potential for further enhancement from adding other free milling and/or refractory material to the production profile, reduced capital and operating costs via an optimisation of the flowsheet, and more competitive mining contract and plant construction costs.

The Study is based on mining open pit Resources from the Swan, Swift and Howards Resources and processing it through a new 800ktpa Carbon in Leach ("CIL") processing plant to produce approximately 290koz gold over an initial six year mine life at an all in sustaining cost of A$1,209/oz gold. The Study assumes a maximum cash drawdown of A$62 million which includes construction of a new processing facility, associated infrastructure and open pit pre-development. Based on A$1,700/oz gold (equivalent to US$1,275/oz gold and an US$:A$ FX rate of US$0.75), the Project generates a pre-tax cashflow of $82 million (after royalties).

The aim of the Study was to focus on production from free milling Resources only to reduce the initial construction capital whilst retaining optionality of production from the Wilsons refractory Resource at a later date. Historic production of gold from free milling sources was in excess of one million ounces at high metallurgical recoveries. This information, together with testwork done by the Company, provides support for the metallurgical recoveries used in the Study.

The results from the Study demonstrate a Project with attractive economic outcomes and future optionality for production from either other free milling or refractory Resources.

Table 7- Key Outcomes of the Study

Operating Metric	Assumption/Result
Gold Price and Exchange Rate	**A$1,700/oz** (US$1,275/oz and A$1:US$0.75)
Mining inventory	4.9Mt @ 1.94g/t Au for 309,000oz contained gold
Mining Method	Open Pit
Project life (processing)	5.8 years
Life of Mine ("LOM") production	290,000oz Au (average ~60,000oz for years 1-4)
Pre-production capital cost	$62M
Average LOM All-in sustaining costs ("AISC")	A$1,209/oz Au
LOM processing recovery	95%
LOM strip ratio	9.4:1
Pre-tax IRR	30%
LOM Revenue	A$496M
LOM EBITDA	A$149M
LOM Pre-tax cash flow	A$82M
LOM Pre-tax NPV	A$37M (11% real), A$47M (8% real)
Construction period	~12 months

Cautionary Statement

The Gum Creek Scoping Study referred to in this announcement is based on low-level technical and economic assessments, and is insufficient to support the estimation of Ore Reserves or to provide assurance of an economic development case at this stage, or to provide certainty that the conclusions of the Scoping Study will be realised.

There is a low level of geological confidence associated with Inferred Mineral Resources and there is no certainty that further exploration work will result in the determination of Indicated Mineral Resources or that the production target referred to above will be realised.

Table 8- Sensitivity to A$ Gold Price and Discount Rate

Pre-tax cashflows and NPVs at various gold prices	A$/oz Gold Price				
	1,600	1,700	1,800	1,900	2,000
Cashflow after royalties (A$M)	53	82	110	139	168
NPV 11% after royalties (A$M)	17	37	57	77	97

Project NPVs at various discount rates	5%	6%	7%	8%	9%	10%	11%
Pre Tax NPV (A$M) at A$1,700/oz	58	54	51	47	43	40	37

The Mineral Resources underpinning the above production target have been prepared by a Competent Person or persons in accordance with the requirements of the JORC Code, (*refer to the Company's ASX announcement of 30 September 2015).*

The Company is continuing to review the option of enhancing shareholder value through the sale of Gum Creek, or by retaining and developing the Project in-house.

Exploration

In 2015, Panoramic completed detailed ground gravity and airborne electromagnetic (EM) survey over the Gum Creek Project. The geophysical data collected was subsequently integrated with existing magnetic surveys, geological mapping and the drill hole database to identify new exploration targets. A limited drill program on these targets has recently commenced.

Concurrently with this work, the shear zone associated with the Wilsons gold deposit has been defined geo-physically with 14 new targets along this zone identified for follow up exploration.

Gold – WA Exploration Projects (ex-Magma)

Under the Laverton Farm-in Agreement between Poseidon Nickel Limited (Poseidon) and Magma Metals Pty Ltd (100% owned by Panoramic), Poseidon has the sole and exclusive right to earn a 60% interest in the tenements by sole funding an additional $2,700,000 in expenditure on the tenements within the period of three years commencing on 3 June 2014. At least 75% of the $2,700,000 must be incurred on activities within the Target Area. Poseidon is required to contribute not less than $700,000 to expenditure after 14 July 2014 before it is able to withdraw from the Agreement.

PGM – Thunder Bay North Project

The Thunder Bay North (TBN) Project is located near Thunder Bay in northwest Ontario, Canada. The advanced exploration project claims cover an aggregate area of 40,816 hectares. The TBN Project Resource contains **10.4Mt at 1.13g/t Pt and 1.07g/t Pd for ~0.4Moz Pt and ~0.4Moz Pd** (*refer to the Company's ASX announcement of 30 September 2015 for disclosures on the TBN Resource*) with exploration potential at depth and along strike.

On 30 July 2014, Panoramic announced that its wholly owned subsidiary, Panoramic PGMs (Canada) Limited (PANP), had signed an Earn-in with Option to Joint Venture Agreement (Agreement) with Rio Tinto Exploration Canada Inc. (RTEC), a wholly owned subsidiary of Rio Tinto, to consolidate their respective Platinum Group Metal (PGM) projects in northwest Ontario, Canada. RTEC holds a single tenement called Escape Lake (EL) within the core of the TBN tenement package. PANP and RTEC have recognised that the best way of realising value from both Projects is to combine TBN and EL into a single project (Consolidated Property).

In January 2015, the Company announced that RTEC had exercised its right under the Agreement by electing to move into the Earn-in Option Phase (Phase 2) of the Agreement. Under the terms of Phase 2, RTEC can earn a 70% interest in the TBN Project by sole funding C$20 million of expenditure over a five year period (minimum spend of C$5 million before RTEC can withdraw). During this period, RTEC will be responsible for managing the Consolidated Property and ensuring the tenements are kept in good standing. If RTEC does not earn its 70% interest, PANP has certain rights to purchase 100% of EL.

The 2015 RTEC diamond drilling program on the Project commenced in July and was completed in early November after drilling 11 holes, totalling 4,955 drill metres. The results of the 2015 program were encouraging and a follow-up winter drill program using two diamond drill rigs commenced in mid-January 2016. **The winter program was completed in April after drilling 10 holes, totalling 4,832 drill metres. RTEC are still waiting to receive a large number of assay results from the drill hole samples.**

In June 2016, Gap GeoPhysics is to undertake a semi-airborne HeliSAM™ magnetics survey over the Escape Lake and Beaver Lake portions of the Project. If the results of the HeliSAM™ survey are positive, the survey could be expanded to cover the 025 Intrusion, Steepledge Lake Intrusive Complex and northern Current Lake Intrusive Complex areas.

PGM – Panton Project

Panton is located 60km south of the Savannah Nickel Project in the East Kimberley region of Western Australia. **Panton is a significant PGM Resource containing ~1.0Moz Pt at 2.2g/t and ~1.1Moz Pd at 2.4g/t** (*refer to the Company's announcement of ASX Announcement of 30 September 2015 on "Mineral Resources and Ore Reserves at 30 June 2015" for disclosures on the Resource*) with exploration potential at depth and along strike.

Panoramic considers the Panton Project to be a quality PGM development asset which fits within the Company's commodity diversification and growth strategy. In March 2012, the previous owner announced the results of a review of the 2003 Bankable Feasibility Study Review (2012 BFS Review).

During the quarter, the Company met with representatives from the Minerals Research Institute of Western Australia (MRIWA) as part of the Company's sponsored research with Curtin University on alternative PGM leaching methods applicable to Panton ore.

Corporate

Liquid Assets and Debt

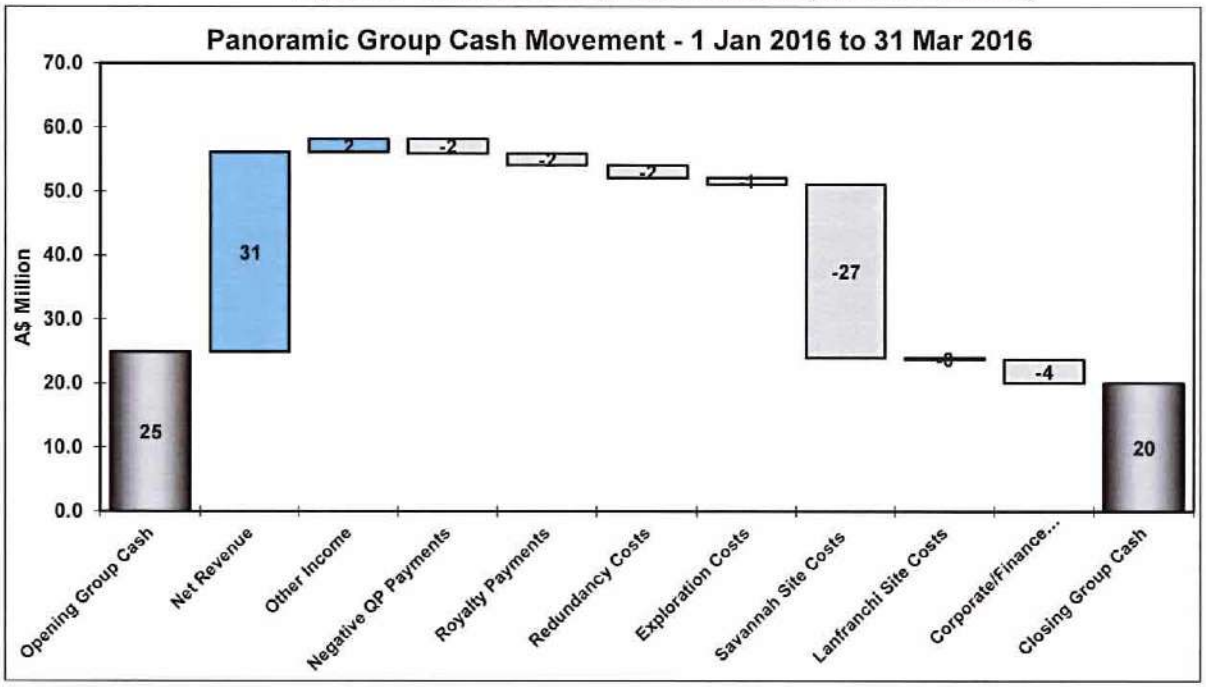
Liquid assets at the end of the quarter totalled $27 million, comprising $20 million in cash, $1 million of trade receivables and ~$6 million of nickel in concentrate which was ready to be shipped at 31 March.

The movement in liquid assets was primarily due to:

- ~$2 million redundancy costs at Lanfranchi, Savannah;
- the continuing weakness in the US$ nickel price impacting on sales revenue and requiring the Company to refund ~$2 million for final invoice quotational period (QP) pricing adjustments; and
- ~$6 million in proceeds from the sale of MLX shares (total of ~$15 million received in the half year).

Aggregate movements in the Group Cash balance over the quarter are shown in Figure 11.

Figure 11 – Panoramic Group Cash Movement (March 2016 Quarter)



Group finance leases on mobile equipment and insurance premiums at 31 March 2016 totalled $1.8 million.

Renounceable Pro-Rata Entitlement Offer

On 31 March 2016, the Company announced a fully underwritten, pro-rata renounceable one (1) for three (3) rights issue at $0.10 per share to raise $10.7 million before costs (*refer to the Company's ASX announcements of 31 March 2016 and 28 April 2016*). The Entitlement Offer ("Offer") closed on Tuesday, 26 April 2016.

Entitlements not taken up by shareholders on a pro-rata basis or under the Top-Up Facility totalled 23,790,653 shares ("Shortfall"). The Shortfall is fully underwritten by Zeta Resources Limited and GMP Securities Australia Pty Ltd. The quotation of new shares issued under the Offer and the mailing of holding statements are both expected to occur on 4 May 2016. The funds raised under the Offer will be used primarily to support the Company's objectives for Savannah North, evaluation studies on other projects and for working capital.

Hedging

During and since the end of the quarter, the Company has added the following short term nickel hedging to cover the US$ quotational period (QP) price risk on the remaining concentrate shipments:

- Sold forward on an unsecured basis, 130t of nickel hedge contracts at a forward price of US$8,500/t (US$3.85/lb) for delivery in April 2016;
- Sold forward on an unsecured basis, 300t of nickel hedge contracts at an average weighted forward price of US$8,400/t (US$3.85/lb) for delivery in May 2016;
- Purchased 181t of nickel put options at US$8,310/t (US$3.77/lb) for delivery in April 2016 if exercised by the Company; and
- Purchased 200t of nickel put options at US$8,288/t (US$3.76/lb) for delivery in May 2016 if exercised by the Company.

Table 9 – Group Hedge Book – A$ Mark-to-Market Valuation as at 31 March 2016

Commodity	Mark-to-Market 31 Dec 2015
US$ Nickel Forwards	-
Bought us$ Nickel Put Options	-
Bought US$ Diesel Call Options	-
Total Mark-to-Market	**-**

Table 10 – Group Hedge Book – Delivery Profile as at 30 April 2016

Commodity	Quantity 30 Apr 2016	Average Price/Rate 30 Apr 2016
Diesel – **Bought Diesel Call Options** (delivery April 2016-June 2016)	500,000litres/mth	US$0.55/litre
Nickel – **Nickel Forwards** (delivery April 2016)	130t	US$8,500/t US$3.85/lb
Nickel Forwards (delivery May 2016)	300t	US$8,400/t US$3.81/lb
Bought Nickel Put Options (delivery April 2016)	181	US$8,310/t US$3.77/lb
Bought Nickel Put Options (delivery May 2016)	200t	US$8,288/t US$3.756/lb

About the Company

Panoramic Resources Limited (**ASX code: PAN**) is a Western Australian mining company formed in 2001 for the purpose of developing the Savannah Nickel Project in the East Kimberley. Panoramic successfully commissioned the $65 million Savannah Project in late 2004 and then in 2005 purchased and restarted the Lanfranchi Nickel Project, near Kambalda. In FY2014, the Company produced a record 22,256t contained nickel and produced 19,301t contained nickel in FY2015. The Lanfranchi Pproject was placed onto care and maintenance in November 2015.

Following the successful development of the nickel projects, the Company diversified its resource base to include gold and platinum group metals (PGM). The Gold Division consists of the Gum Creek Project located near Wiluna. The PGM Division consists of the Panton Project, located 60km south of the Savannah Project and the Thunder Bay North Project in Northern Ontario, Canada, in which Rio Tinto is earning 70% in the project by spending up to C$20 million over five years.

Panoramic has been a consistent dividend payer and has paid out a total of $114.3 million in fully franked dividends since 2008. At 31 March 2016, Panoramic had $20 million in cash and no bank debt.

The Company's vision is to broaden its exploration and production base, with the aim of becoming a major, diversified mining company in the S&P/ASX 100 Index. The growth path will include developing existing resources, discovering new ore bodies, acquiring additional projects and is being led by an experienced exploration-to-production team with a proven track record.

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The information in this release that relates to Exploration Targets and Exploration Results is based on information compiled by John Hicks. Mr Hicks is a member of the Australasian Institute of Mining and Metallurgy (AusIMM) and is a full-time employee and shareholder of Panoramic Resources Limited. Mr Hicks also holds performance rights in relation to Panoramic Resources Limited.

The information in this report that relates Mineral Resources (specifically Lower Schmitz) is based on information compiled by Mr Paul Payne. Mr Payne is a Fellow of the Australasian Institute of Mining and Metallurgy (AusIMM) and consultant working for Payne Geological Services Pty Ltd (PayneGeo).

Both the aforementioned have sufficient experience that is relevant to the style of mineralisation and type of target/deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Both Mr Hicks and Mr Payne consent to the inclusion in this report of the matters based on the information in the form and context in which it appears.

No New Information or Data

This announcement contains references to exploration results and Mineral Resource estimates, all of which have been cross referenced to previous market announcements made by the Company. The Company confirms that it is not aware of any new information or data that materially affects the information included in the relevant market announcements and, in the case of estimates of Mineral Resources, that all material assumptions and technical parameters underpinning the estimates in the relevant market announcement continue to apply and have not materially changed.

Forward Looking Statements

This announcement may contain certain "forward-looking statements" which may not have been based solely on historical facts, but rather may be based on the Company's current expectations about future events and results. Where the Company expresses or implies an expectation or belief as to future events or results, such expectation or belief is expressed in good faith and believed to have a reasonable basis. However, forward looking statements are subject to risks, uncertainties, assumptions and other factors, which could cause actual results to differ materially from future results expressed, projected or implied by such forward-looking statements. Such risks include, but are not limited to metals price volatility, currency fluctuations, increased production costs and variances in ore grade or recovery rates from those assumed in mining plans, as well as political and operational risks in the Countries and States in which we operate or sell product to, and governmental regulation and judicial outcomes. For a more detailed discussion of such risks and other factors, see the Company's Annual Reports, as well as the Company's other filings. The Company does not undertake any obligation to release publicly any revisions to any "forward-looking statement" to reflect events or circumstances after the date of this announcement, or to reflect the occurrence of unanticipated events, except as may be required under applicable securities laws.

Appendix 1

Lanfranchi Project – Tabulation of Lower Schmitz Drill Hole Assay Results and 2012 JORC Compliance Tables

Hole	East (m)	North (m)	RL (m)	Dip (°)	Azi (°)	EOH (m)	From (m)	To (m)	Intercept	Cu (%)	Co (%)	SG g/cm³
SMT373A	391916.4	6513685.0	-800.0	-2.3	230.5	626.46	449.72	452.46	2.74m @ 1.19 %	0.09	0.02	2.96
							482.90	489.94	7.04m @ 5.29 %	0.39	0.11	3.53
							497.00	498.62	1.62m @ 1.06 %	0.11	0.03	2.99
							525.30	532.10	6.80m @ 5.53 %	0.44	0.10	3.53
							550.54	557.04	6.50m @ 6.63 %	0.36	0.12	3.71
SMT377	391470.6	6513874.2	-442.3	-33.5	174.4	490.80			*Abandoned*			
SMT377A	391470.6	6513874.2	-442.3	-33.5	174.4	821.40	700.80	715.40	14.60m @ 3.19 %	0.10	0.05	3.18
						including	701.80	706.41	4.61m @ 6.67 %	0.13	0.10	3.41
SMT377B	391470.6	6513874.2	-442.3	-33.5	174.4	703.80	675.64	680.85	5.21m @ 2.96 %	0.26	0.06	3.22
SMT377C	391470.6	6513874.2	-442.3	-33.5	174.4	680.30	599.15	600.25	1.10m @ 1.31 %	0.06	0.02	2.94
							651.40	661.20	9.80m @ 6.12 %	0.33	0.11	3.52
SMT378	391451.4	6514040.6	-503.2	-29.4	170.6	715.89	678.98	689.70	10.72m @ 6.15 %	0.46	0.10	3.62
						including	679.17	687.53	8.36m @ 7.24 %	0.54	0.12	3.79
SMT378A	391451.4	6514040.6	-503.2	-29.4	170.6	649.80			*NSR*			
SMT378B	391451.4	6514040.6	-503.2	-29.4	170.6	827.70	668.95	675.89	6.94m @ 7.64 %	0.50	0.13	3.85
							733.80	734.80	1.00m @ 1.00 %	0.07	0.02	3.04
							754.84	766.07	11.23m @ 7.58 %	0.56	0.12	3.59
SMT378C	391451.4	6514040.6	-503.2	-29.4	170.6	654.05	641.12	644.70	3.58m @ 4.44 %	0.20	0.08	3.39
SMT378D	391451.4	6514040.6	-503.2	-29.4	170.6	722.70	684.69	701.20	16.51m @ 4.87 %	0.43	0.10	3.40
						including	684.69	689.77	5.08m @ 10.51 %	0.95	0.22	4.25
SMT378E	391451.4	6514040.6	-503.2	-29.4	170.6	719.70	685.42	693.62	8.20m @ 6.69 %	0.51	0.11	3.70
SMT379	391504.4	6514044.1	-467.1	-43.1	171.9	671.43	651.47	652.72	1.25m @ 1.74 %	0.13	0.03	3.02
SMT381									*No SGs available*			
SMT382	391539.3	6513475.2	-867.1	15.7	203.3	127.80	73.20	79.55	6.35m @ 3.85 %	0.26	0.06	3.37
SMT383	391539.9	6513475.7	-867.3	15.7	188.8	101.10	73.94	77.66	3.72m @ 7.10 %	0.55	0.12	3.75
SMT384	391540.4	6513475.9	-867.3	17.6	174.2	98.10	76.16	81.79	5.63m @ 7.60 %	0.49	0.15	3.87
							87.00	88.38	1.38m @ 1.11 %	0.08	0.02	3.02
SMT385	391533.8	6513473.0	-866.1	24.7	204.6	111.00	62.33	64.99	2.66m @ 2.88 %	0.27	0.04	3.17
SMT386	391539.9	6513475.6	-866.7	26.6	189.0	88.92	65.33	69.83	4.50m @ 5.65 %	0.65	0.09	3.54
SMT387	391534.2	6513472.9	-867.5	5.4	196.3	149.00	122.39	123.58	1.19m @ 12.00 %	1.22	0.20	4.59
							125.70	127.54	1.84m @ 2.21 %	0.22	0.04	3.14
SMT388	391539.9	6513475.5	-867.7	5.4	189.0	133.95	93.11	94.78	1.67m @ 5.87 %	0.37	0.12	3.60
							117.00	119.00	2.00m @ 1.08 %	0.10	0.02	2.98
SMT391	391532.5	6513475.0	-867.1	13.2	248.4	197.03	183.35	184.84	1.49m @ 2.53 %	0.15	0.05	3.10
SMT392A	391562.0	6513485.1	-869.1	-23.4	184.5	347.79	96.70	97.87	1.17m @ 1.20 %	0.11	0.02	2.94
							122.14	126.37	4.23m @ 2.25 %	0.08	0.03	3.08
							153.20	156.22	3.02m @ 1.72 %	0.13	0.02	2.64
							167.13	168.24	1.11m @ 1.23 %	0.08	0.02	2.98
SMT393	391535.7	6513473.2	-868.1	-5.1	169.3	162.75			NSR			
SMT394	391535.0	6513472.9	-868.2	-7.2	184.0	164.16	126.84	133.59	6.75m @ 5.96 %	0.52	0.12	3.73
SMT395	391534.6	6513472.8	-868.1	-5.7	190.2	205.10	167.05	172.39	5.34m @ 6.56 %	0.41	0.15	3.73
SMT397	391533.1	6513473.5	-865.1	36.6	218.8	91.00	60.28	65.32	5.04m @ 4.56 %	0.34	0.08	3.47
SMT401	391535.6	6513473.6	-867.4	9.4	169.4	103.95	87.85	94.48	6.63m @ 5.38 %	0.32	0.11	3.58
							98.30	100.09	1.79m @ 1.37 %	0.10	0.03	2.96
SMT402	391534.6	6513472.8	-867.6	4.3	190.6	154.56	88.70	102.37	13.67m @ 6.26 %	0.36	0.10	3.57
							115.52	118.93	3.41m @ 6.61 %	0.67	0.13	3.74
							132.24	139.12	6.88m @ 6.66 %	0.30	0.12	3.49
SMT403	391534.2	6513472.9	-867.7	5.4	197.8	164.00			*NSR*			
SMT404	391535.0	6513472.9	-868.4	-13.9	184.0	222.87			*NSR*			
SMT405	391626.6	6513534.9	-868.4	28.3	230.0	166.72			*NSR*			
SMT406	391626.3	6513534.9	-868.5	26.3	243.7	181.98	161.57	162.70	1.13m @ 5.09 %	0.12	0.09	3.47
SMT407	391626.6	6513534.3	-867.8	36.3	226.1	160.82	138.34	143.01	4.67m @ 2.47 %	0.22	0.06	3.17
SMT408	391626.5	6513534.7	-868.0	33.7	236.3	170.00			*NSR*			
SMT410	391626.5	6513535.0	-867.5	42.7	246.9	172.97			*NSR*			
SMT411	391534.4	6513472.9	-867.9	-1.4	195.4	187.23	143.81	148.91	5.10m @ 3.99 %	0.30	0.07	3.37

Hole	East (m)	North (m)	RL (m)	Dip (°)	Azi (°)	EOH (m)	From (m)	To (m)	Intercept	Cu (%)	Co (%)	SG g/cm³
SMT413	391534.4	6513472.9	-868.3	-11.1	194.9	300.90			*NSR*			
SMT416	391535.2	6513473.0	-868.3	-11.4	180.4	167.18			*NSR*			
SMT420A	391535.5	6513473.2	-868.4	-15.8	173.5	195.60			*NSR*			
SMT425	391564.3	6513486.2	-868.5	-7.5	127.7	311.43			*NSR*			
SMT426	391536.1	6513473.6	-868.5	-19.2	160.1	62.34			*NSR*			
SMT427	391533.2	6513474.1	-867.8	0.7	225.0	246.30	218.84	224.47	5.63m @ 1.04 %	0.08	0.03	3.07

Notes:

1. *Intervals are down-hole lengths, not true-widths*
2. *Parameters: 1.0% Ni lower cut-off*
3. *Intercepts < 1.5 % m not included*
4. *Intercepts grades based on length weighting incorporating sample SG values*
5. *NSR – no significant result*

Lanfranchi Project – Table 1, Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	Virtually all sampling for exploration and resource estimation purposes at the Lanfranchi Nickel Mine (LNM) is based on diamond drill core. Sample selection is based on geological core logging. Individual samples typically vary between 0.2m and 1.2m in length.
Drilling techniques	Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	Diamond drilling at LNM is typically NQ2 or LTK60 size. Occasionally BQ and HQ core size holes have been drilled.
Drill sample recovery	Method of recording and assessing core and chip sample recoveries and results assessed. Measures taken to maximise sample recovery and ensure representative nature of the samples. Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	All recovered diamond core is metre marked by on site geologists; any core loss is determined and recorded as part of the geological logging process. Core recovery is typically 100 percent. No relationship exists between core recovery and grade.
Logging	Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. The total length and percentage of the relevant intersections logged.	All core is geologically and geotechnically logged to a standard appropriate for exploration and mineral resource estimation purposes. Core is logged from start to end of hole without gaps. Core photography is not undertaken. Drill holes are logged using Excel™ templates that are code restricted to ensure that only approved data can be entered. The Excel™ templates are then uploaded to the Lanfranchi SQL Server drill hole database via Datashed™.

Criteria	JORC Code explanation	Commentary
Sub-sampling techniques and sample preparation	If core, whether cut or sawn and whether quarter, half or all core taken. If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. For all sample types, the nature, quality and appropriateness of the sample preparation technique. Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. Whether sample sizes are appropriate to the grain size of the material being sampled.	All diamond core is cut using electric core saw and half core sampled for assay. Quarter core samples are sent as part of the LNM QAQC process for check assaying. Sample intervals typically vary between 0.2m and 1.2m and are positioned as to not cross geological boundaries.
Quality of assay data and laboratory tests	The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.	All LNM drill hole samples are analysed by Kalassay Group. The Laboratory process for LNM samples involves: Crush sample to <3mm, pulverise to 90% passing 75um (lab blanks introduced and pulverised at this point). From the pulverised sample, a 0.2g assay aliquot is taken and weighed then digested by 4-Acid digest and analysed by ICP-OES instrument. Laboratory QA/QC is performed on standards, blanks and duplicates. The LNM policy is to scrutinize the results for QA/QC standards and blanks when assay jobs are reported and to request re-runs if result are ± 1SD from the expected value.
	For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	No other geophysical or analytical tools have been used to estimate grade.
	Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	Certified Reference Material (QAQC) samples are routinely inserted during all sampling at LNM. In addition samples are routinely sent for check analysis at a different Laboratory. The QAQC results indicate that the diamond core assays being used for resource estimation at LNM are a fair representation of the material that has been sampled.
Verification of sampling and assaying	The verification of significant intersections by either independent or alternative company personnel.	Significant intersections are calculated by mine geologists and verified/reported on a monthly basis by the Geology Manager.
	The use of twinned holes.	Twinning of drillholes is not performed at LNM
	Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	Assay data are imported directly from the Kalassay assay files and QA/QC validated via Datashed™ to the LNM SQL drillhole database.
	Discuss any adjustment to assay data.	No adjustment to assay data is made.
Location of data points	Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	Drill hole collars are accurately surveyed for X,Y,Z and azimuth and dip by site Surveyors using "Total Station" control. Older holes may/may not have collar azimuth/dip measurements. Down-hole surveys are generally conducted using single shot or reflex multishot tools at 15m, 30m and every 30m thereafter.
	Specification of the grid system used. Quality and adequacy of topographic control.	The LNM drill hole database contains both MGA94 and local mine grid (KNO) coordinates. All site geological and mine planning work is performed in the local KNO grid system. Conversion from KNO grid to MGA GDA94 Zone 51 is based on a two point transformation: 389084.61E, 513790.88N = 389351.47E, 6513980.38N 389044.77E, 513543.54N = 389313.70E, 6513732.77N
Data spacing and distribution	Data spacing for reporting of Exploration Results.	LNM resource estimation drill holes are typically drilled on a regular grid spacing that varies according to the size and consistency of the resource being drilled. Due to the consistent grade and low Coefficient of Variation of nickel mineralisation generally, resource definition drilling at LNM is more for volume estimation purposes than grade estimation.

Criteria	JORC Code explanation	Commentary
	Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	Data spacing is deemed to be sufficient for Mineral Resource estimation and reporting. LNM exploration holes are not drilled on regular grid pattern.
	Whether sample compositing has been applied.	No sample compositing is undertaken; all core samples are logged and analysed in full.
Orientation of data in relation to geological structure	Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	Underground drill sites are not always ideally positioned for resource definition drilling however no sampling orientation bias is evident. The Ni grade is typically very consistent within individual resource domains and therefore drill orientation is not a determinant for reliable grade estimation
Sample security	The measures taken to ensure sample security.	All diamond core samples are taken directly from site to Kalassay for analysis via a local courier service. Sample security is considered adequate.
Audits or reviews	The results of any audits or reviews of sampling techniques and data.	No recent audit of the sampling techniques and procedures at LMN has been undertaken. All the LNM Mineral Resource estimates are audited by independent consultants BM Geological Services. Minor adjustments to model dimensions, geostatistical analysis and application of top-cuts (where required) and adjustments to search parameters have been made on occasions following this audit process.

Lanfranchi Project – Table 1, Section 2 – Reporting of Exploration Results

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings. The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	The Lanfranchi Nickel Mine (LNM) is an operating mine secured by a contiguous block of 35 Mineral Leases, 1 Mining Lease and 1 Prospecting Licence, covering the Tramways Dome 40km south of Kambalda in WA. All tenure is current and in good standing. Panoramic Resources Limited (Panoramic) has the right to explore for and mine all commodities within the tenements other than gold. The LNM is an operating mine with all statutory approvals and licences in place to operate. The mine operates under an off-take agreement to mine and deliver nickel ore to BHP-Billiton's Nickel West Kambalda concentrator.
Exploration done by other parties	Acknowledgment and appraisal of exploration by other parties.	The LNM tenements were purchased by Panoramic in 2004 from WMC Resources Ltd. WMC had held the Lanfranchi Tramways tenements and explored the region since 1967. WMC commenced mining at the LNM in 1976.
Geology	Deposit type, geological setting and style of mineralisation.	Panoramic mines nickel sulphide rich ore from several deposits at Lanfranchi. All deposits belong to the "classic' Kambalda style, komatiite hosted, nickel sulphide class of deposits.
Drill hole Information	A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes: o easting and northing of the drill hole collar o elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar o dip and azimuth of the hole o down hole length and interception depth o hole length. o If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	Panoramic routinely drills surface and/or underground exploration holes about the Tramways Dome in search of additional nickel sulphide mineralisation. Details of the LNM exploration holes mentioned in this accompanying document can be found in Table 1 of the document.
Data	In reporting Exploration Results, weighting averaging	Sample length weighted average grades are typically calculated using the

Criteria	JORC Code explanation	Commentary
aggregation methods	techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated. Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. The assumptions used for any reporting of metal equivalent values should be clearly stated.	Intercept Calculator within the DBMS DataShed™ for most publicly reported LNM exploration drill hole data. Parameters used are a1.0% Ni lower cut-off, minimum reporting intercept of 1m, and a maximum internal waste of 1.5 consecutive metres. For Lower Schmitz drill hole intercepts the process is essentially the same except the individual sample SG values are also incorporated in to the weighting calculation.
Relationship between mineralisation widths and intercept lengths	These relationships are particularly important in the reporting of Exploration Results. If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').	All LNM exploration drilling is conducted on the KNO local grid system. For public reporting purposes drill hole coordinates are expressed in MGA94 coordinates in accordance with JORC 2012 requirements. Where the geometry of the mineralisation is known the estimated true width of mineralisation will be reported. Where the mineralisation geometry is not sufficiently known the down-hole intersection length of mineralisation is reported, and clearly stated to be the case.
Diagrams	Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	Based on the material nature of the LNM exploration results being reported on, the diagram in the body of the accompanying report is considered sufficiently appropriate.
Balanced reporting	Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.	Based on the material nature of the LNM exploration results being reported on in the accompany document, the report is considered to be sufficiently balanced.
Other substantive exploration data	Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	No other exploration data is considered material to this report at this stage.
Further work	The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling). Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	Routine exploration drilling is ongoing at the LNM. The results reported herein will, at least in the short term, have a material effect on the planned exploration programs currently underway at the LNM. Immediate follow-up programs are being developed to undertake further work in the subject area of this release.

Lanfranchi Project – Table 1, Section 3 – Reporting of Mineral Resources

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. • Data validation procedures used.	• Data was captured electronically to prevent transcription errors. • Validation included comparison of assay results to logged geology to verify mineralised intervals.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• A site visit was not undertaken due to the project being on care and maintenance
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. • Nature of the data used and of any assumptions made. • The effect, if any, of alternative interpretations on Mineral Resource estimation. • The use of geology in guiding and controlling Mineral Resource estimation. • The factors affecting continuity both of grade and geology.	• The confidence in the geological interpretation is considered to be good due to the well understood and readily identifiable stratigraphy which is a key control on mineralisation position • The mineralisation is highly continuous and is defined by good quality drilling. • The deposit consists of a moderately plunging channel hosting matrix, massive and disseminated nickel sulphides

Criteria	JORC Code explanation	Commentary
		which have been interpreted based on logging and assay data from samples taken at regular intervals from underground diamond drill holes.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The Lower Schmitz Mineral Resource area extends over a strike length of 240m, a width of 70m and has a vertical extent of 170m commencing at 1100m below surface.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. • The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • The assumptions made regarding recovery of by-products. • Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation). • In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. • Any assumptions behind modelling of selective mining units. • Any assumptions about correlation between variables. • Description of how the geological interpretation was used to control the resource estimates. • Discussion of basis for using or not using grade cutting or capping. • The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	• A mineralisation envelope representing the massive/matrix nickel sulphides was used to define the deposit. • Vein composites were generated for each of the 30 holes passing through the envelope. • Inverse distance squared interpolation was used for estimation of Ni, Cu and Co with the ellipsoid oriented down the plunge of the mineralisation. • No high grade cuts were applied. • Surpac software was used for the estimation. • The parent block dimensions used were 10m NS by 8m EW by 8m vertical with sub-cells of 0.625m by 0.5m by 0.5m. The parent block size was selected on the basis of being approximately 50% of the average drill hole spacing in the better drilled part of the deposit. • There has been no mining at the deposit, although extensive production has occurred up-plunge of this mineralisation. • No previous resource estimates have been completed. • No assumptions have been made regarding recovery of by-products. • No estimation of deleterious elements was carried out. • An initial interpolation pass was used with a maximum range of 40m and extended to 80m for the unfilled blocks. • A minimum of 2 and a maximum of 5 composites were used. • Selective mining units were not modelled in the Mineral Resource model. The block size used in the model was based on drill sample spacing and lode orientation. • Strong correlation between Ni and Co was observed. • For validation, trend analysis was completed by comparing the interpolated blocks to the sample composite data within 20m northing intervals and by 20m vertical intervals.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• Tonnages and grades were estimated on a dry in situ basis. No moisture values were reviewed.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• The Mineral Resource has been reported at a 1% Ni cut-off based on local operating parameters.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• Extensive underground production has been carried out up-plunge of this deposit and it is assumed that this deposit will be mined using identical methods. • The deposit is considered to have sufficient grade and continuity to be considered for underground mining. • No mining parameters or modifying factors have been applied to the Mineral Resource.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• Metallurgical testing has not yet been compiled but the mineralogy is identical to deposits mined at the Lan Franchi operation prior to mine closure.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental	• Any development of the deposit will be part of the Lan Franchi project for which environmental permitting and waste management procedures are already in place.

Criteria	JORC Code explanation	Commentary
	impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• Bulk density determinations were made on samples from drill core using the weight in air/weight in water method. • Results were used to prepare a robust regression equation based on Ni grade. This was used to apply density in the block model.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• Mineral Resources were classified in accordance with the Australasian Code for the Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC, 2012). The Mineral Resource was classified as Indicated and Inferred Mineral Resource on the basis of data quality, sample spacing, and lode continuity. • The Indicated portion of the Mineral Resource was defined where good continuity of mineralisation was evident and where hole spacings were 20m by 20m or less. • The remaining portions of the deposit were classified as Inferred Mineral Resource. • The definition of mineralised zones is based on sound geological understanding producing a robust model of mineralised domains. This model is very similar to previously mined deposits at the project. • The Mineral Resource estimate appropriately reflects the view of the Competent Person.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• No audit of the estimate was carried out.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• The Lower Schmitz Mineral Resource estimate is considered to be reported with a high degree of confidence. The consistent lode geometry and continuity of mineralisation is reflected in the Mineral Resource classification. The data quality is good and the drill holes have detailed logs produced by qualified geologists. • The Mineral Resource statement relates to global estimates of tonnes and grade.

Appendix 2

Savannah Project – Tabulation of Drill Hole Assay Results and 2012 JORC Compliance Tables

Hole	East (m)	North (m)	RL (m)	Dip (°)	Azi (°)	EOH (m)	From (m)	To (m)	Intercept	Cu (%)	Co (%)
KUD1573	395885.5	8082589.8	1450.4	-37.1	111.9	304.30	267.18	290.15	22.97m @ 1.79 %	0.75	0.13
KUD1574	395885.1	8082589.5	1450.8	-29.3	124.5	350.00	265.50	280.70	15.20m @ 1.76 %	0.60	0.13
KUD1575	395885.4	8082588.9	1450.9	-18.2	129.6	294.70	270.30	286.70	16.40m @ 1.78 %	0.96	0.14
KUD1576	395885.4	8082589.8	1451.3	-10.8	111.5	385.90	333.90	356.95	23.05m @ 2.09 %	1.10	0.14
							362.95	367.90	4.95m @ 2.29 %	0.67	0.15
KUD1579	395885.4	8082588.7	1451.3	-8.6	132.3	327.30	302.80	312.60	9.80m @ 1.33 %	0.34	0.09
KUD1580	395885.6	8082590.9	1450.4	-45.7	83.9	381.30	321.90	325.28	3.38m @ 0.72 %	0.22	0.05
KUD1581	395885.5	8082589.5	1451.5	-4.7	118.1	386.40	358.55	370.90	12.35m @ 1.32 %	0.28	0.09

Notes:

- *Intervals are down-hole lengths, not true-width*
- *Parameters: 0.50% Ni lower-cut off, maximum internal waste 4.0m, minimum intercept 0.5m*
- NSR – no significant result; NS – no sample

Savannah Project – Table 1, Section 1 - Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The Savannah deposit and surrounding exploration areas are typically sampled by diamond drilling techniques. Over 1500 holes have been drilled for a total in excess of 220,000m. The majority of holes were drilled from underground drill platforms. • About the mine the drillhole spacing is a nominal 25x25m grid spacing over the extent of the mineralisation. • All drillhole collars were surveyed using Leica Total Station survey equipment by a registered surveyor. Downhole surveys were typically performed every 30 metres using either "Reflex EZ Shot" or "Flexit Smart Tools". • All diamond core is geologically logged with samples (typically between 0.2 metre to 1 metre long) defined by geological contacts. Analytical samples include a mix of full and sawn half core samples. Sample preparation typically involves pulverising the sample to 90% passing 75 µm followed by either a 3 or total 4 acid digest and analysis by either AAS or ICP OES.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• A mix of LTK60 and NQ2 sized diamond drilling has been used to obtain >90% of the data in the mine database. Exploration holes are typically NQ2 size. Some RC drilling has been used historically for the upper part of the mine.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• Diamond core recoveries are logged and recorded in the database. Overall recoveries are >99% and there are no apparent core loss issues or significant sample recovery problems. • Depths checked against core blocks, regular rod counts, driller breaks checked by fitting core together. • No relationship exists between sample recovery and grade
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.	• All holes have been geologically logged in full. Geotechnical logging was carried out on all diamond drill holes for recovery and RQD. Number of defects (per interval) and roughness was carried out around the ore zones. Structure type, alpha angle, infill, texture and healing is stored in the structure table of the database.

Criteria	JORC Code explanation	Commentary
	• Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography.	• Logging of diamond core RC samples recorded lithology, colour, mineralisation, structural (DDH only) and other features. Core was photographed wet.
	• The total length and percentage of the relevant intersections logged.	• All drill holes were logged in full.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken.	• Analytical core samples included a mix of full and sawn half core samples.
	• If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry.	• All samples from core
	• For all sample types, the nature, quality and appropriateness of the sample preparation technique.	• All core sampling and sample preparation followed industry best practice.
	• Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples.	• QC involved the addition of Savannah derived CRM assay standards, blanks, and duplicates. At least one form of QC was inserted in most sample batches.
	• Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling.	• Original versus duplicate assay results have always shown strong correlation due to massive sulphide rich nature of the orebody.
	• Whether sample sizes are appropriate to the grain size of the material being sampled.	• Sample sizes are considered appropriate to represent the Savannah style of mineralisation.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.	• The Savannah Nickel Mine (SNM) standard analytical technique is a 3-acid digest with an AAS finish. The method best approaches total dissolution for most minerals. Exploration samples sent off-site are analysed using a 4-acid digest with either ICP OES or AAS finish (AAS for ore grade samples).
	• For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	• No other analytical tools or techniques are employed.
	• Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• The onsite laboratory carries out sizing checks, uses internal standards, duplicates, replicates, blanks and repeats. A selection of roughly 10% of pulps was sent to external laboratories for repeat analysis and sizing checks. No bias has been identified.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel.	• Drilling and sampling procedures at the SNM have been inspected by many stakeholders since the project began.
	• The use of twinned holes.	• Throughout the life of the mine, there have been several instances where holes have been twinned, confirming intersections and continuity.
	• Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	• Holes are logged into Excel™ templates on laptops, data is then entered into MS Access™ database with user data entry front end built in. Data is ultimately transferred to SQL server from Perth office. Data periodically validated by site personnel.
	• Discuss any adjustment to assay data.	• No adjustments have been made to assay data.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	• All diamond drill hole collars were surveyed using Leica Total Station survey equipment by a registered surveyor. "Reflex EZ Shot" or "Flexit Smart Tool" was used for downhole surveys at approximately every 30m. Visual inspection in a 3D graphics environment using Surpac software failed to identify any obvious errors regarding the spatial position of drillhole collars or downhole surveys
	• Specification of the grid system used.	• The mine grid is a truncated 4 digit (MGA94) grid system. • Conversion from local grid to MGA GDA94 Zone 52 is calculated by applying truncated factor to local coords: E: +390000, N: +8080000N
	• Quality and adequacy of topographic control.	• Topographic control is of a high quality and is adequate for the resource estimation process
Data spacing and distribution	• Data spacing for reporting of Exploration Results.	• Nominal drill hole spacing of 25m (easting) by 25m (RL)
	• Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	• The mineralised domains delineated by the drill spacing show enough continuity to support the classification applied under the 2012 JORC Code.
	• Whether sample compositing has been applied.	• No sample compositing has been undertaken.

Criteria	JORC Code explanation	Commentary
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• Drill hole orientation was largely perpendicular to the orebody with the exception of the western extent where drill platform positions allowed only for oblique intersections. • No orientation sampling bias has been identified.
Sample security	• The measures taken to ensure sample security.	• Samples transported to onsite lab by SNM staff. Samples sent off site are road freighted (Nexus transport) and tracked using spreadsheets onsite.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits/reviews of the sampling techniques have been undertaken in recent time. The procedures used are considered to be industry standard. Mine to mill reconciliation records throughout the life of the Savannah Project provide confidence in the sampling procedures.

Savannah Project - Table 1, Section 2 - Reporting of Exploration Results

Criteria	JORC Code explanation	Commentary
Mineral tenement and land tenure status	• Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings. • The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	• The Savannah Nickel Mine (SNM) is an operating mine secured by 5 contiguous Mining Licences. All tenure is current and in good standing. SNM has the right to explore for and mine all commodities within the mine tenements. • The SNM is an operating mine with all statutory approvals and licences in place to operate. The mine has a long standing off-take agreement to mine and deliver nickel sulphide concentrate to the Jinchuan Group in China.
Exploration done by other parties	• Acknowledgment and appraisal of exploration by other parties.	• Since commissioning in 2004, SNM has conducted all recent exploration on the mine tenements.
Geology	• Deposit type, geological setting and style of mineralisation.	• The SNM is based on mining ore associated with the Savannah Intrusion; a palaeo-proterozoic mafic/ultramafic magma conduit. The Ni-Cu-Co rich massive sulphide mineralisation occurs as "classic" magmatic breccias developed about the more primitive, MgO rich ores basal parts of the conduit.
Drill hole Information	• A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes: o easting and northing of the drill hole collar o elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar o dip and azimuth of the hole o down hole length and interception depth o hole length. • If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	• All exploration at SNM is conducted on the Savannah mine grid, which is a "4 digit" truncated MGA grid. Conversion from local to MGA GDA94 Zone 52 is calculated by applying truncated factor to local coords: E: +390000, N: +8080000. RL equals AHD + 2,000m • Savannah underground diamond drill holes are typically NQ2 size, though some deep holes are commenced HQ size and then reduced. Deep surface holes are commenced PQ size, then reduced to HQ and eventually NQ2 size • All core is orientated and photographed prior to cutting and sampling • All intersection intervals are reported as down-hole lengths and not true widths • All assays are typically performed on the Savannah onsite laboratory, otherwise by SGS Laboratories in Perth
Data aggregation methods	• In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated. • Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. • The assumptions used for any reporting of metal equivalent values should be clearly stated.	• Weighted averages were calculated using parameters of 0.5% Ni lower cut-off, minimum reporting length of 1m and maximum internal waste of 7m. • Cu and Co grades were determined by the defined Ni grade interval, ie they are not calculated independently. • For all Savannah North Maiden Resource drill hole intercepts the process is essentially the same except the individual sample SG values are also incorporated in to the weighting calculation.
Relationship between mineralisation widths and intercept lengths	• These relationships are particularly important in the reporting of Exploration Results. • If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. • If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').	• The geometry of the mineralisation reported herein with respect to the drill holes being reported has not been established. • All intersection lengths reported in this accompanying release are down-hole lengths and not true widths.

Criteria	JORC Code explanation	Commentary
Diagrams	• Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	• Based on the limited level of data currently available for this area at Savannah it was deemed that a simplified plan and section view showing the location of the exploration drill results in relation to the main areas of the SNM operation was appropriate.
Balanced reporting	• Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.	• Based on the fact that exploration results reported herein are from several drill holes, located well away from other mine drill holes, the report is considered to be sufficiently balanced.
Other substantive exploration data	• Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.	• No other exploration data is considered material to this release at this stage.
Further work	• The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling). • Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	• The exploration results reported herein are for the Savannah North Project. Work is ongoing and further results will be reported if and when they become available.

News Release

PJD-54

30 September 2016

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PANORAMIC RESOURCES LTD



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ASX:PAN

MINERAL RESOURCES AND ORE RESERVES AT 30 JUNE 2016

Summary

Panoramic Resources Limited (**ASX Code: PAN**) is pleased to announce the Group's Mineral Resources and Ore Reserves Statement at 30 June 2016. Key points:

Resource / Reserve	FY2016	FY2015	% Change y-o-y
Total Nickel Resources	256,300 tonnes	218,600 tonnes	+ 17%
Total Nickel Reserves	22,300 tonnes	45,700 tonnes	- 51%
Total Copper Resources	83,200 tonnes	68,300 tonnes	+ 22%
Total Copper Reserves	11,500 tonnes	24,600 tonnes	- 53%
Total Cobalt Resources	10,400 tonnes	7,700 tonnes	+ 35%
Total Cobalt Reserves	900 tonnes	2,200 tonnes	- 59%
Total Gold Resources	1.27 million ounces	2.43 million ounces	- 48%
Total Platinum Resources	1.36 million ounces	1.36 million ounces	No change
Total Palladium Resources	1.44 million ounces	1.44 million ounces	No change

Nickel in Mineral Resources increased by 37,700 tonnes between FY2015 and FY2016 due to:

- additions at Savannah North (+54,400 tonnes) and Lower Schmitz (+6,700 tonnes);
- mining depletion at Savannah, Copernicus and Lanfranchi (-12,400 tonnes); and
- sterilisation / other changes (-11,000 tonnes).

The Group nickel Resources at 30 June 2016 do not include 65,500 tonnes of nickel contained added to the Savannah North Resource from the August 2016 Resource upgrade (*refer ASX announcement dated 24 August 2016*). **With the inclusion of the Resource upgrade at Savannah North, Group nickel Resources at 30 September 2016 are 321,800 tonnes contained nickel.**

Nickel in Ore Reserves declined by 23,400 tonnes between FY2015 and FY2016 due to:

- mining depletion at Savannah, Copernicus and Lanfranchi (-12,400 tonnes);
- exclusion of Savannah Sub-900 Fault material due to economic factors (-10,800 tonnes);
- an increase in Reserves at Metcalfe (+1,100 tonnes); and
- other changes (-1,300 tonnes).

Gold in Mineral Resources decreased by 1.17 million ounces between FY2015 and FY2016 due to the sale of the Company's 70% interest in the Mt Henry Project. There was no change to platinum and palladium Mineral Resources between FY2015 and FY2016.

Commodity price and US$:A$ exchange rate assumptions used for the FY2016 Ore Reserves, based on medium term forecasts compiled from a range of external parties, were:

- Nickel US$6.00/lb
- Copper US$2.50/lb
- Cobalt US$11.83/lb
- US$:A$ 0.74

Savannah Nickel Project (including Copernicus and Savannah North)

At Savannah, Ore Reserves as at 30 June 2016 are 1.56 million tonnes at 1.16% Ni, 0.74% Cu and 0.06% Co for 18,100 tonnes contained nickel, 11,500 tonnes contained copper and 900 tonnes contained cobalt. Savannah Ore Reserves declined by 23,500 tonnes nickel between FY2015 and FY2016 due to mining depletion at Savannah and Copernicus (-11,400 tonnes), exclusion of Savannah Sub-900 Fault Resources from the Ore Reserve due to economic factors (-10,800 tonnes), and other changes (-1,300 tonnes).

Mineral Resources at Savannah (including Copernicus and Savannah North) at 30 June 2016 are 10.5 million tonnes at 1.53% Ni, 0.79% Cu and 0.10% Co for 160,900 tonnes contained nickel, 83,200 tonnes contained copper and 10,400 tonnes contained cobalt. This is an increase of contained nickel in Mineral Resources of 32,000 tonnes between FY2015 and FY2016 due to additions at Savannah North (+54,400 tonnes), partly offset by mining depletion at Savannah and Copernicus (-11,400 tonnes), sterilisation and other changes (-10,900 tonnes).

Drilling at Savannah North during 2015 resulted in a Resource estimate of 6.88 million tonnes at 1.59% Ni for 109,600 tonnes contained nickel (*refer ASX announcement dated 1 October 2015*), which is included in the total Resource reported above. There was no resource drilling undertaken at Savannah or Copernicus during FY2016.

In March 2016, the Company commenced an infill and extensional drilling program on Savannah North, which led to a major increase of the Savannah North Resource to 175,100 tonnes nickel contained (*refer ASX announcement dated 24 August 2016*). As this work was completed after the 30 June 2016 cut-off date for the annual Group Resources and Reserves report, this Resource upgrade is not included in the 30 June 2016 Resource estimates. **Nickel in Resources at Savannah (including Copernicus and Savannah North) at 30 September 2016 is 226,400 tonnes.**

Lanfranchi Nickel Project

At Lanfranchi, Ore Reserves at 30 June 2016 are 0.21 million tonnes at 2.03% Ni for 4,200 tonnes contained nickel. Lanfranchi Ore Reserves increased by 100 tonnes of nickel between FY2015 and FY2016 due to inclusion of the additional levels of Metcalfe in the Reserve (+1,100 tonnes) offset by mining depletion (-1,000 tonnes).

Mineral Resources at Lanfranchi at 30 June 2016 are 5.65 million tonnes at 1.69% Ni for 95,500 tonnes contained nickel. This is an increase of nickel in Mineral Resources of 5,600 tonnes between FY2015 and FY2016 due to additions at Lower Schmitz (+6,700 tonnes), partly offset by mining depletion (-1,000 tonnes).

A maiden Resource estimate for Lower Schmitz of 131,000 tonnes at 5.1% Ni for 6,700 tonnes contained nickel was reported in the March 2016 quarter (*refer ASX announcement dated 28 April 2016*). No other resource drilling was completed at Lanfranchi in FY2016.

Gum Creek Gold Project

Mineral Resources at Gum Creek at 30 June 2016 are 17.4 million tonnes at 2.28g/t Au for 1.27 million ounces contained gold. There were no changes to Gum Creek Resources during FY2016. There are no Ore Reserves at Gum Creek.

Mt Henry Gold Project

In September 2015, the Company completed the sale of its 70% interest in the Mt Henry Project to Metals X Limited (*refer ASX announcement dated 16 September 2015*). The Company has no retained interest in this project.

Panton Platinum-Palladium Project

Mineral Resources at Panton at 30 June 2016 are 14.3 million tonnes at 2.19g/t Pt and 2.39g/t Pd for 984,000 ounces contained platinum and 1,081,000 ounces contained palladium. There were no changes to Panton Resources during FY2016.

Thunder Bay North Platinum-Palladium Project

Mineral Resources at Thunder Bay North at 30 June 2016 are 10.4 million tonnes at 1.13g/t Pt and 1.07g/t Pd for 377,000 ounces contained platinum and 355,000 ounces contained palladium. There were no changes to Thunder Bay North Resources during FY2016. Rio Tinto Exploration Canada Inc. (RTEC) has the right to earn up to 70% in the Thunder Bay North Project by spending C$20 million over five years, with a minimum spend of C$5 million before RTEC can withdraw.

Material Information Summary

In accordance with the ASX Listing Rules, a fair and balanced representation of the information provided in Appendix 1 must be presented in the body of the ASX announcement. That representation follows below. This information applies only to the Company's Savannah and Lanfranchi Nickel Projects where material changes to the Mineral Resource and Ore Reserve position occurred during the year due to additions, model updates, mining depletion and sterilisation.

Savannah Nickel Project

Drilling and Supporting Data

The Savannah Resource estimate is based almost entirely on data gathered from NQ2 or LTK60 underground diamond drill core. Holes are drilled on a nominal 25m x 25m grid spacing over the extent of mineralisation. Face sampling and pre-production drill data is also used to refine resource and reserve stope shape outlines. All drill core is photographed, geologically logged, and then halved for sampling. All drill core is spatially orientated to the mine grid by survey control. Down-hole surveys are typically performed every 30m by using either Reflex EZ Shot™ or Flexit Smart Tool™.

Sampling and Assaying

Core sample lengths are typically between 0.2m to 1m long based on logged geological boundaries. Whilst Savannah was in operation, sample analysis was via an on-site laboratory. For core and mining related samples, the standard analytical technique is a 3-acid digest with an AAS finish. This method best approaches total dissolution for most minerals. For exploration samples or other samples (QAQC) sent off-site that are included in the geological database, the analytical technique is 4-acid digest with either ICP OES or AAS finish (typically AAS for high grade ore samples).

Geology and Geological Interpretation

Nickel mineralisation at Savannah is associated with the Savannah Intrusion; a Palaeoproterozoic mafic/ultramafic intrusion.The Ni-Cu-Co rich massive sulphide mineralisation at Savannah occurs as "classic", readily recognisable magmatic breccia-textured ores developed about the more primitive MgO rich basal parts of the intrusion. Panoramic has been mining and exploring the Savannah orebody for over 10 years and has a sound knowledge and understanding of the geology and orientation of the orebody and a high level of confidence in the geological interpretation.

Database

The Savannah geological database is administered on a SQL Server by Panoramic's Database Manager in Perth. Data is captured on-site into Excel™ software templates using laptop computers and uploaded via "Datashed" to the site database, which in turn is automatically replicated to the SQL server in Perth. Regular exports of the database enable the Company's site and Perth based personnel to access the data. Validation of the database is undertaken regularly by Company geologists by plotting the data on plan and cross-sections and through visual 3D inspection using Surpac™ software.

Cut-off Grade

A 0.50% nickel cut-off grade with no minimum mining width is used to define mineralised shapes for resource modelling. This enables the entire mineralised part of the Savannah Intrusion to be encapsulated and available for conversion to Ore Reserve status once the appropriate mining and economic factors are applied. The 0.50% cut-off grade is a natural grade boundary between the magmatic breccia-textured ores and weaker disseminated mineralisation at Savannah.

An initial cut-off grade of 0.8% nickel is used for mine planning. An economic analysis is carried out for each planned stope and only stopes with a positive return are included in the Ore Reserve estimate. Commodity price and US$:A$ exchange rate assumptions are based on medium term forecasts compiled from a range of external parties.

Metallurgical and Mining Assumptions

The metallurgical features of the Savannah ore are well understood and have not materially changed since mining began in 2004. The sulphide ore is processed via a conventional crush, grind and flotation process to make a bulk Ni-Cu-Co concentrate free of any significant deleterious elements. Metallurgical recoveries used for Ore Reserve estimations are 86% for nickel, 95% for copper and 88% for cobalt. The concentrate is transported by road to Wyndham, then shipped to the Jinchuan Group's smelter/refinery in the Gansu province of north-west China.

Due to the favourable geometry and availability of paste backfill, ore is extracted by sublevel stoping methods. Ore development is conducted under geological control with face and sludge sampling routinely performed to refine the geological interpretations and stope design. Blast hole drilling is designed to minimise over-break and is set out by survey control. Mining dilution between 7.5-20% at zero grade is applied to stope and ore development depending on location in the mine and stope type (primary, secondary or sill pillar). The minimum underground mining width for development is 4.8m and 3.0m for stopes.

Estimation methodology

Ordinary Kriging techniques using Surpac™ software were used to estimate Ni, Co, Cu and density into the Savannah 3D Resource Block Model. Top-cut analysis was undertaken for each Resource domain using grade histograms, but generally no extreme values were detected. Variography was calculated for the domain with the largest sample population and the resultant variogram models adapted for the remaining domains. Check estimates by Panoramic staff using Inverse Distance Squared method yielded similar results to the Ordinary Kriged model. The Savannah Resource Model has been updated periodically since mining began in 2004, with differences in tonnage for successive updates accounted for by new drilling, mining depletion, sterilisation and new resource areas. Grade correlation between updated estimates has remained high.

Classification

The Resource classification system adopted at Savannah is based on the level of confidence as set out in the 2012 JORC Code guidelines. Measured Resources are defined by areas supported by strong drilling and confined up and down dip by mine development such that confidence in lode volume and continuity of grade is very high. Indicated Resources are defined by areas where geological confidence is high and drilling support is strong (equal to or less than 25m x 25m grid spacing). Inferred Resources are typically in areas where geological confidence is lower.

Proven Ore Reserves are based on Measured Resources subject to economic viability. Probable Ore Reserves are based on Indicated Resources subject to the economic viability.

Lanfranchi Nickel Project

Drilling and Supporting Data

The Lanfranchi Resource estimates are based almost entirely on data gathered from NQ2 or LTK60 underground diamond drill core. Holes are typically drilled on a regular grid pattern that varies according to the size and consistency of the resource being drilled. Due to the low coefficient of variation of the Lanfranchi Resources nickel grades, Resource definition drilling is more about defining shapes and volume estimation purposes than grade estimation. All drill core was geologically logged and then halved for sampling. All drill core was spatially orientated to the mine grid by survey control. Down-hole surveys are typically performed every 30m by either Reflex Multi-Shot or single shot tools.

Sampling and Assaying

All sampling for Resource estimation purposes at Lanfranchi was based on underground diamond drill core. Sample selection was based on geological core logging with individual samples typically between 0.2m and 1.2m in length.

All Resource drill-hole samples were analysed by the Kalassay Group in their Perth laboratory. The laboratory process for Lanfranchi samples involved reducing each sample by crushing and pulverising to 90% passing 75um. A 0.2g assay aliquot was taken from the pulverised sample and digested by 4-Acid digest and analysed by an ICP-OES instrument. Lanfranchi Certified Reference Material (QAQC) samples are routinely inserted in all sample batches submitted to Kalassay.

Geology and Geological Interpretation

All Lanfranchi Resources belong to the "classic Kambalda style" komatiite hosted nickel sulphide deposit. This deposit type has been extensively studied and mined in the Kambalda area since the late 1960s. Due to this knowledge and history, there is a high level of confidence in the geological interpretation of the Lanfranchi Resources. The strongly contrasting character between mineralised and un-mineralised lithologies is readily apparent and easy to identify.

Database

The Lanfranchi geological database is administered on a SQL Server by Panoramic's Database Manager in Perth. All Lanfranchi drill hole and resource samples are logged and recorded using code restricted Excel™ software templates to ensure that only approved data can be entered. The templates are uploaded to the SQL drill-hole database via the "Datashed" software. Uploads to the database on site are automatically replicated to the SQL server in Perth. Once Laboratory assay files have been scrutinised and finalised for QAQC, they are imported directly into the database.

Cut-off Grade

All Lanfranchi Resource models were constructed to a nominal 1.0% Ni cut-off grade. No minimum mining width assumptions were made during the resource wireframing or estimation process, but in some areas minor internal dilution was included to avoid over-complication of the wireframe shape and when it was obvious selective mining was not a realistic option. The 1.0% Ni cut-off grade is a natural grade boundary between Lanfranchi's low grade and high grade mineralisation.

For mine planning, an initial cut-off grade of 1% nickel is used, except for airleg mining areas, which is 2%. An economic analysis is carried out for each planned stope and only stopes with a positive return are included in the Ore Reserve estimate. Commodity price and US$:A$ exchange rate assumptions are based on medium term forecasts compiled from a range of external parties.

Metallurgical and Mining Assumptions

The metallurgical features of the Lanfranchi ore types are well understood as the ores have been processed at the Kambalda Nickel Concentrator since the1970s. As such, no new metallurgical studies were required. The Lanfranchi sulphide rich ores are suitable for processing via flotation to make a nickel concentrate. The Lanfranchi ore is processed under an Ore Tolling and Concentrate Purchase Agreement (OTCPA) with BHP Billiton Nickel West (BHPB) at the Kambalda Concentrator located about 40 kilometres to the north-west of Lanfranchi. Under the OTCPA, metallurgical recovery is determined by BHPB and is related to the average grade delivered on a monthly basis.

Due to favourable geometry and availability of paste backfill, sublevel stoping methods are employed to extract the thicker orebodies, while air-leg mining methods are used in narrower, high-grade ore zones. The hanging wall ultramafic rocks at Lanfranchi are typically weak and therefore all open stopes are extensively cable bolted and all ore development is shotcreted. Ore development is conducted under geological control and is routinely mapped and sampled. Blast-hole drilling is designed to minimise over-break and is set out by survey control.

For all stopes a mining dilution factor of 10% at zero grade was applied. The minimum stoping width is 3.0m and 1.8m for air-leg stopes. A 95% metal recovery factor was applied to all stopes and no Inferred Resources were included in the Ore Reserve.

Estimation methodology

The computer software package Surpac™ was used to develop all Lanfranchi Resource models. All resource models have been estimated using Inverse Distance Squared (ID2) methodology for simplicity and the amount of data available. Models that were run between 2007 and 2010 in parallel using ID2 and Ordinary Kriging methodology, produced very similar estimates, typically within ±4% on tonnes, ±0.3% in grade and <5% difference on a contained nickel basis.

Classification

The Resource classification system adopted at Lanfranchi is based on the level of confidence as set out in the 2012 JORC Code guidelines. The classification relies largely on drill density but with increased confidence in areas of ore development. Measured Resources are defined by areas of the resource with adjacent mining or development. Indicated Resources are assigned to areas of high geological confidence supported by a regular, systematic pattern of drilling.

Proven Ore Reserves are based on Measured Resources subject to economic viability. Probable Ore Reserves are based on Indicated Resources subject to the economic viability.

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NICKEL - MINERAL RESOURCES AS AT 30 JUNE 2016

Resource	Equity	Metal	Date of Resource	JORC Compliance	Measured		Indicated		Inferred		Total		Metal Tonnes
					Tonnes	(%)	Tonnes	(%)	Tonnes	(%)	Tonnes	(%)	
Savannah Project													
Savannah (above 900)	100%	Nickel	Jun-16	2012	1,275,000	1.51	759,000	1.20			2,034,000	1.39	28,300
		Copper				0.87		0.90				0.88	17,900
		Cobalt				0.07		0.07				0.07	1,400
Savannah (below 900)	100%	Nickel	Jun-16	2012	780,000	1.64	125,000	1.72			905,000	1.65	14,900
		Copper				0.76		0.75				0.76	6,900
		Cobalt				0.10		0.09				0.10	900
Savannah North	100%	Nickel	Jun-16	2012			4,780,000	1.51	2,103,000	1.77	6,883,000	1.59	109,600
		Copper						0.72		0.88		0.77	52,900
		Cobalt						0.11		0.12		0.11	7,800
Copernicus (O/P)	100%	Nickel	Jun-15	2012	132,000	0.97					132,000	0.97	1,300
		Copper				0.52						0.52	700
		Cobalt				0.03						0.03	0
Copernicus (U/G)	100%	Nickel	Jul-10	2004			508,000	1.30	25,000	0.98	532,000	1.29	6,800
		Copper						0.91		0.69		0.90	4,800
		Cobalt						0.05		0.02		0.05	300
Lanfranchi Project	100%	Nickel											
Cruikshank			Apr-11	2004			2,018,000	1.42	611,000	0.79	2,629,000	1.28	33,600
Deacon			Mar-14	2012	89,000	2.99			134,000	1.70	224,000	2.22	5,000
Gigantus			Jul-07	2004					652,000	1.63	652,000	1.63	10,600
Helmut South			May-14	2012									
Helmut South Ext			Apr-14	2012	21,000	4.54	29,000	2.87			50,000	3.59	1,800
John			Jul-07	2004					291,000	1.42	291,000	1.42	4,100
Lanfranchi			Apr-14	2012	40,000	4.12	55,000	4.40	63,000	3.49	158,000	3.97	6,300
Martin			Feb-12	2012			47,000	3.58	7,000	4.16	54,000	3.66	2,000
McComish			Jul-07	2004					992,000	1.49	992,000	1.49	14,800
Metcalfe			Jan-14	2012			280,000	1.99	111,000	1.35	391,000	1.81	7,100
Schmitz			Jul-13	2012	30,000	4.92	23,000	3.93	16,000	2.95	69,000	4.14	2,900
Lower Schmitz			Mar-16	2012			51,000	5.60	79,000	4.80	131,000	5.11	6,700
Winner			Jul-11	2004			14,000	4.40			14,000	4.40	600
Total (Equity)		Nickel											256,300
		Copper											83,200
		Cobalt											10,400

Notes:

- *Figures have been rounded and therefore may not add up exactly to the reported totals*
- *All resources are inclusive of reserves*
- *Savannah Project Resource cutoff grade is 0.50% Ni*
- *Copernicus Project Resource cutoff grade is 0.50% Ni*
- *Lanfranchi Project Resource cutoff grade is 1.00% Ni*

Competent Person Statement

The information in this report that relates to Mineral Resources is based on information compiled by or reviewed by Paul Hetherington (MAusIMM) for the Savannah Project Resource and Copernicus Project Resource and Bradley Robinson (MAusIMM) for the Lanfranchi Project Resources. The aforementioned were formerly full-time employees of Panoramic Resources Limited. The aforementioned have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. The aforementioned consent to the inclusion in the release of the matters based on their information in the form and context in which it appears.

The information in this report that relates Mineral Resources at Lower Schmitz is based on information compiled by Mr Paul Payne. Mr Payne is a Fellow of the Australasian Institute of Mining and Metallurgy (AusIMM) and consultant working for Payne Geological Services Pty Ltd (PayneGeo). Mr Payne has sufficient experience that is relevant to the style of mineralisation and type of target/deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Mr Payne consents to the inclusion in this report of the matters based on the information in the form and context in which it appears.

NICKEL - ORE RESERVE AS AT 30 JUNE 2016

Reserve	Equity	Metal	Date of Reserve	JORC Compliance	Proven		Probable		Total		Metal Tonnes
					Tonnes	(%)	Tonnes	(%)	Tonnes	(%)	
Savannah Project											
Above 900 Fault	100%	Nickel	Jun-16	2012	1,365,000	1.15	194,000	1.24	1,558,000	1.16	18,100
		Copper				0.66		1.28		0.74	11,500
		Cobalt				0.06		0.07		0.06	900
Below 900 Fault	100%	Nickel	Jun-16	2012							
		Copper									
		Cobalt									
Copernicus Open Pit	100%	Nickel	Jun-16	2004							
		Copper									
		Cobalt									
Lanfranchi Project	100%	Nickel									
Deacon			Jun-16	2012			42,000	2.67	42,000	2.67	1,100
Metcalfe			Jun-16	2012			113,000	1.57	113,000	1.57	1,800
Lanfranchi			Jun-16	2012			11,000	2.56	11,000	2.56	300
Schmitz			Jun-16	2012			15,000	2.96	15,000	2.96	500
Helmut Sth Ext			Jun-16	2012			27,000	2.19	27,000	2.19	600
Total (Equity)		Nickel									22,300
		Copper									11,500
		Cobalt									900

Notes:

- *Figures have been rounded and therefore may not add up exactly to the reported totals*
- *All reserves are inclusive of resources*
- *Savannah Project Reserve cutoff grade is 0.80% Ni*
- *Lanfranchi Project Reserve cutoff grade is 1.00% Ni except for airleg mining which is 2.00% Ni*

Competent Person Statement

Information in this report relating to Ore Reserves has been compiled by or reviewed by Lilong Chen (MAusIMM). The aforementioned is a full-time employee of Panoramic Resources Limited. The aforementioned has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as a Competent Person as defined in the 2012 Edition of the Australian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. The aforementioned consents to the inclusion in the release of the matters based on his information in the form and context in which it appears.

GOLD - MINERAL RESOURCES AS AT 30 JUNE 2016

Resource	Equity	Metal	Date of Resource	JORC Compliance	Measured		Indicated		Inferred		Total		Metal (Au oz)
					Tonnes	Au (g/t)	Tonnes	Au (g/t)	Tonnes	Au (g/t)	Tonnes	Au (g/t)	
Gum Creek Project	100%	Gold											
Swan OC			Jun-15	2012			2,250,000	2.57	990,000	2.36	3,240,000	2.51	261,100
Heron South			Oct-12	2004			1,000,000	2.31	136,000	1.41	1,136,000	2.20	80,300
Howards			Jul-13	2012			5,255,000	1.07	716,000	1.01	5,971,000	1.06	204,000
Specimen Well			Mar-06	2004			289,000	2.06	72,000	1.79	361,000	2.00	23,200
Toedter			Mar-06	2004					661,000	1.62	661,000	1.62	34,400
Eagles Peak			Mar-06	2004			13,000	3.46			13,000	3.46	1,400
Orion			Mar-06	2004			22,000	3.04			22,000	3.04	2,200
Deep South			Mar-06	2004			20,000	3.02			20,000	3.02	1,900
Shiraz			Jul-13	2012			2,476,000	0.84	440,000	0.76	2,916,000	0.83	77,600
Swan UG			Jun-15	2012			207,000	8.71	77,000	11.25	284,000	9.40	85,800
Swift UG			Jun-15	2012					46,000	10.25	46,000	10.25	15,200
Omega UG			Mar-06	2004			31,000	9.20			31,000	9.20	9,200
Kingfisher UG			Mar-04	2004			390,000	6.80			390,000	6.80	85,300
Wilsons UG			Jul-13	2012			2,131,000	5.33	136,000	5.97	2,267,000	5.37	391,500
Total (Equity)		Gold			-	-	14,084,000	2.32	3,274,000	2.12	17,358,000	2.28	1,273,100

Notes:

- *Swan OC resource cutoff grade is 0.7 g/t. The resources have been partially diluted over a minimum mining width of 2.5m and confined to a A$2,000 Whittle pit shell*
- *Eagles Peak Resource cutoff grade is 1.2 g/t*
- *Orion Resource cutoff grade is 1.3 g/t*
- *Deep South Resource cutoff grade is 1.2 g/t*
- *Swan UG Resource cutoff grade is 4.0 g/t for resource wireframes near historic workings and 6.0 g/t for resource wireframes away from historic workings. The resource is based on an approximate 2.5m minimum vertical mining width.*
- *Swift UG Resource cutoff grade is 6.0 g/t.*
- *Omega UG Resource cutoff grade is 3.0 g/t*
- *Kingfisher UG Resource cutoff grade is 3.0 g/t*
- *Heron South resource cutoff grade is 0.5 g/t*
- *Howards resource cutoff grade is 0.4g/t*
- *Specimen Well resource cutoff grade is 0.5 g/t*
- *Toedter resource cutoff grade is 0.5 g/t*
- *Wilsons resource cutoff grade is 1.0 g/t for lower grade domains and 2.0g/t for high grade domains*

Cross references to previous market announcements:

- *Swan, Swift, Howards, Shiraz, Wilsons – refer ASX announcement dated 30 September 2015 titled "Mineral Resources and Ore Reserves at 30 June 2015"*
- *Heron South – refer ASX announcement dated 17 October 2012 titled "Gidgee Resource Upgrade – Howards and Heron South"*
- *Specimen Well, Toedter – refer ASX announcement dated 21 June 2012 titled "Significant Upgrade in Gold Resource at Gidgee"*
- *Eagles Peak, Orion, Deep South, Omega UG, Kingfisher UG – refer Legend Mining Limited (ASX:LEG) announcement dated 19 March 2007 titled "Legend Mining Limited Annual Report 31st December 2006"*

No New Information or Data

The Gold Mineral Resource estimates tabled above have been previously reported, and the relevant market announcements cross referenced. The Company confirms that it is not aware of any new information or data that materially affects the information included in the relevant market announcements and, in the case of estimates of Mineral Resources, that all material assumptions and technical parameters underpinning the estimates in the relevant market announcement continue to apply and have not materially changed.

PLATINUM GROUP METALS (PGM) - MINERAL RESOURCES AS AT 30 JUNE 2016

Thunder Bay North Project

Resource	Equity	Date of Resource	JORC Compliance	Tonnage	Grade									Metal (oz)	
					Pt (g/t)	Pd (g/t)	Rh (g/t)	Au (g/t)	Ag (g/t)	Cu (%)	Ni (%)	Co %	Pt-Eq (g/t)	Pt (oz ,000)	Pd (oz ,000)
Open Pit	**100%**	**Jan-11**	**2004**												
Indicated				8,460,000	1.04	0.98	0.04	0.07	1.50	0.25	0.18	0.014	2.13	283	267
Inferred				53,000	0.96	0.89	0.04	0.07	1.60	0.22	0.18	0.014	2.00	2	2
Underground	**100%**	**Feb-12**	**2004**												
Indicated				1,369,000	1.65	1.54	0.08	0.11	2.60	0.43	0.24	0.016	3.67	73	68
Inferred				472,000	1.32	1.25	0.06	0.09	2.10	0.36	0.19	0.011	2.97	20	19
Total (Equity)				10,354,000										377	355

Notes - Open Pit Resource:

The open pit Mineral Resource is reported at a cut-off grade of 0.59 g/t Pt-Eq within a Lerchs-Grossman resource pit shell optimized on Pt-Eq. The strip ratio (waste:ore) of this pit is 9.5:1. The platinum-equivalency formula is based on assumed metal prices and overall recoveries. The Pt-Eq formula is: Pt-Eq g/t = Pt g/t + Pd g/t x 0.3204 + Au g/t x 0.6379 + Ag g/t x 0.0062 + Cu g/t x 0.00011 + Total Ni g/t x 0.000195 + Total Co g/t x 0.000124 + Rh g/t x 2.1816. The conversion factor shown in the formula for each metal represents the conversion from each metal to platinum on a recovered value basis. The assumed metal prices used in the Pt-Eq formula are: Pt US$1,595/oz, Pd US$512/oz, Au US$1,015/oz, Ag US$15.74/oz, Cu US$2.20/lb, Ni US$7.71/lb, Co US$7.71/lb and Rh US$3,479/oz. The assumed combined flotation and PlatsolTM process recoveries used in the Pt-Eq formula are: Pt 76%, Pd 75%, Au 76%, Ag 55%, Cu 86%, Ni 44%, Co 28% and Rh 76%. The assumed refinery payables are: Pt 98%, Pd 98%, Au 97%, Ag 85%, Cu 100%, Ni 100%, Co 100% and Rh 98%.

Notes - Underground Resources:

The underground mineral resource is reported at a cut-off grade of 1.94g/t Pt-Eq. The Pt-Eq formula is: Pt-Eq g/t = Pt g/t + Pd g/t x 0.2721 + Au g/t x 0.3968 + Ag g/t x 0.0084 + Cu g/t x 0.000118 + Sulphide Ni g/t x 0.000433 + Sulphide Co g/t x 0.000428 + Rh g/t x 2.7211. The assumed metal prices used in the Pt-Eq formula are: Pt US$1,470/oz, Pd US$400/oz, Rh US$4,000/oz, Au US$875/oz, Ag US$14.30/oz, Cu US$2.10/lb, Ni US$7.30/lb and Co US$13.00/lb. The assumed process recoveries used in the Pt-Eq formula are: Pt 75%, Pd 75%, Rh 75%, Au 50%, Ag 50%, Cu 90%, and Ni and Co in sulphide 90%. The assumed smelter recoveries used in the Pt-Eq formula are Pt 85%, Pd 85%, Rh 85%, Au 85%, Ag 85%, Cu 85%, Ni 90% and Co 50%. Ni and Co in sulphide were estimated by linear regression of MgO to total Ni and total Co respectively. The regression formula for Ni in sulphide (NiSx) is: NiSx = Ni - (MgO% x 60.35 - 551.43). The regression formula for Co in sulphide (CoSx) is: CoSx = Co - (MgO% x 4.45 - 9.25).

Cross references to previous market announcements:

- *Open pit Resources – refer Magma Metals Limited (ASX:MMW) announcement dated 7 February 2011 titled "Positive Scoping Study for Thunder Bay North Project"*
- *Underground Resources – refer Magma Metals Limited (ASX:MMW) announcement dated 23 February 2012 titled "Magma Metals Increases Mineral Resources at TBN to 790,000 Platinum-Equivalent Ounces"*

No New Information or Data

The Thunder Bay North Mineral Resource estimates tabled above have been previously reported, and the relevant market announcements cross referenced. The Company confirms that it is not aware of any new information or data that materially affects the information included in the relevant market announcements and, in the case of estimates of Mineral Resources, that all material assumptions and technical parameters underpinning the estimates in the relevant market announcement continue to apply and have not materially changed.

PLATINUM GROUP METALS (PGM) - MINERAL RESOURCES AS AT 30 JUNE 2016

Panton Project

Resource	Equity	Date of Resource	JORC Compliance	Tonnage	Grade					Metal (oz)	
					Pt (g/t)	Pd (g/t)	Au (g/t)	Ni (%)	Cu (%)	Pt (oz ,000)	Pd (oz ,000)
Top Reef	**100%**	**Mar-12**	**2012**								
Measured				4,400,000	2.46	2.83	0.42	0.28	0.08	348	400
Indicated				4,130,000	2.73	3.21	0.38	0.31	0.09	363	426
Inferred				1,560,000	2.10	2.35	0.38	0.36	0.13	105	118
Middle Reef	**100%**	**Mar-12**	**2012**								
Measured				2,130,000	1.36	1.09	0.10	0.18	0.03	93	75
Indicated				1,500,000	1.56	1.28	0.10	0.19	0.04	75	62
Inferred				600,000	1.22	1.07	0.10	0.19	0.05	24	21
Total (Equity)				14,320,000	2.19	2.39	0.31	0.27	0.08	984	1,081

Cross references to previous market announcements:

- *refer ASX announcement dated 30 September 2015 titled "Mineral Resources and Ore Reserves at 30 June 2015"*

No New Information or Data

The Panton Mineral Resource estimates tabled above have been previously reported, and the relevant market announcements cross referenced. The Company confirms that it is not aware of any new information or data that materially affects the information included in the relevant market announcements and, in the case of estimates of Mineral Resources, that all material assumptions and technical parameters underpinning the estimates in the relevant market announcement continue to apply and have not materially changed.

Appendix 1
JORC Code 2012 Edition - Compliance Tables

Table 1 - Savannah Nickel Mine
Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• The deposit was sampled by diamond drilling techniques. Over 1500 holes have been drilled for a total in excess of 220,000m. The majority of holes were drilled from underground drill platforms. • The drillhole spacing is a nominal 25 x 25 metre grid spacing over the extent of the mineralization, except for Savannah North where the nominal spacing is a 50 x 50 metre grid spacing.. • All drillhole collars were surveyed using Leica Total Station survey equipment by a registered surveyor. Downhole surveys were typically performed every 30 metres using either "Reflex EZ Shot" or "Flexit Smart Tools". • All diamond core was geologically logged with samples (typically between 0.2 metre to 1 metre long) defined by geological contacts. Analytical samples included a mix of full and sawn half core samples. Sample preparation included pulverising to 90% passing 75 µm followed by total 4 acid digest and analysis by ICP OES.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• A mix of LTK60 and NQ2 sized diamond drilling has been used to obtain >90% of the data used in the estimate. Some RC drilling has been used historically for the upper part of the resource.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• Diamond core recoveries are logged and recorded in the database. Overall recoveries are >99% and there are no apparent core loss issues or significant sample recovery problems. • Depths checked against core blocks, regular rod counts, driller breaks checked by fitting core together. • No relationship exists between sample recovery and grade
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• All holes have been geologically logged in full. Geotechnical logging was carried out on all diamond drillholes for recovery and RQD. Number of defects (per interval) and roughness was carried out around the ore zones. Structure type, alpha angle, infill, texture and healing is stored in the structure table of the database. • Recorded core logging attributes include lithology, colour, mineralisation, structural and other features. • All core is photographed. • All drillholes were logged in full.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• Analytical core samples are dominantly sawn half NQ2 samples. • All resource definition samples are diamond core only.. • All core sampling and sample preparation followed industry best practice. • QC involved the addition of Savannah derived CRM assay standards, blanks, and duplicates. At least one form of QC was inserted in most sample batches. • Original versus duplicate assay results have always shown strong correlation due to massive sulphide rich nature of the orebody. • Sample sizes are considered appropriate to represent the Savannah style of mineralisation.
Quality of assay	• The nature, quality and appropriateness of the assaying	• The Savannah Nickel Mine (SNM) onsite laboratory

Criteria	JORC Code explanation	Commentary
data and laboratory tests	and laboratory procedures used and whether the technique is considered partial or total. • For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc. • Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	standard analytical technique is a 3-acid digest with an AAS finish. The method best approaches total dissolution for most minerals The onsite exploration sample analytical method for Ni,Cu,Co is AAS 22S. The onsite laboratory is run by SGS Laboratory Services. Exploration samples sent off-site are analysed using a 4-acid digest with either ICP OES or AAS finish (AAS for ore grade samples). • No other analytical tools or techniques are employed. • The onsite laboratory carries out sizing checks, uses internal standards, duplicates, replicates, blanks and repeats. A selection of roughly 10% of pulps was sent to external laboratories for repeat analysis and sizing checks. No bias has been identified.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel. • The use of twinned holes. • Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. • Discuss any adjustment to assay data.	• Drill hole intercepts are automatically reported from the SQL drill hole database. Significant intercepts are manually checked before reporting. • Throughout the life of the mine, there have been several instances where holes have been twinned, confirming intersections and continuity. • Holes are logged into Excel templates on laptops, data is then entered into MS Access database with user data entry front end built in. Data is ultimately transferred to SQL server from Perth office. Data periodically validated by site personnel. • No adjustments have been made to assay data.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. • Specification of the grid system used. • Quality and adequacy of topographic control.	• All diamond drillhole collars were surveyed using Leica Total Station survey equipment by a registered surveyor. "Reflex EZ Shot" or "Flexit Smart Tool" was used for downhole surveys at approximately every 30m. Visual inspection in a 3D graphics environment using "Surpac" software failed to identify any obvious errors regarding the spatial position of drillhole collars or downhole surveys • The mine grid is a truncated 4 digit (MGA94) grid system. Conversion from local grid to MGA GDA94 Zone 52 is calculated by applying truncated factor to local coords:E: +390000, N: +8080000N • Topographic control is well established, RL equals AHD + 2,000m.
Data spacing and distribution	• Data spacing for reporting of Exploration Results. • Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. • Whether sample compositing has been applied.	• Exploration drill holes are spaced on a geological basis as opposed to a nominal drill hole spacing. • For the most part, resource drilling is conducted on a regular spacing, sufficient to achieve the objectives of the drill program. For Savannah this is a nominal 25m x 25m grid pattern. For the Savannah North Resource definition programs the nominal spacing is 50m x 50m. • The mineralized domains delineated by the drill spacing show enough continuity to support the classification applied under the 2012 JORC Code. • No sample compositing has been undertaken.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. • If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	• Where possible drill hole orientation is largely perpendicular to the orebody. Underground drill access limitations prevents this occurring in some areas. • No orientation sampling bias has been identified.
Sample security	• The measures taken to ensure sample security.	• Samples transported to onsite lab by Panoramic staff. Samples sent off site are road freighted by a third party freight contractor and tracked using spreadsheets onsite.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits/reviews of the sampling techniques have been undertaken in recent time. The procedures used are considered to be industry standard. Mine to mill reconciliation records throughout the life of the Savannah Project provide confidence in the sampling procedures.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. • Data validation procedures used.	• An Excel™ software logging template with lookup tables and fixed formatting is used for logging and data collection. • Data validation checks are performed every time a drillhole is entered to the database using a checklist.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The competent person was formerly a site based, full time employee of Panoramic.
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. • Nature of the data used and of any assumptions made. • The effect, if any, of alternative interpretations on Mineral Resource estimation. • The use of geology in guiding and controlling Mineral Resource estimation. • The factors affecting continuity both of grade and geology.	• The confidence of the geological interpretation is high which has been confirmed by mapping and greater than 10 years of operational experience. • No other interpretations have been considered as the current model is demonstrably robust. • Geological controls were used to create the domains, namely, lithology, massive sulphide content, major structures • One of the main domains is affected by 2 major cross-cutting mafic dykes, the geometry and thickness of which are well understood.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.	• The resource is 350m along strike (east), varies in thickness from 1 to 50m and averages 8m thick, from the surface to 900m depth.
Estimation and modelling techniques	• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. • The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • The assumptions made regarding recovery of by-products. • Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation). • In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. • Any assumptions behind modelling of selective mining units. • Any assumptions about correlation between variables. • Description of how the geological interpretation was used to control the resource estimates. • Discussion of basis for using or not using grade cutting or capping.	• Ordinary Kriging was employed using Surpac™ software to estimate Ni, Co, Cu and Density into a 3D block model. Top cut analysis was undertaken for each domain using grade histograms, no extreme values were detected and therefore no top cuts applied. Variography was calculated for the domain with the largest sample population and the resultant variogram models were adapted for the remaining domains. • Check estimates by Panoramic staff using Inverse Distance squared method has yielded similar results. The estimate has been updated periodically since mining began in 2004, differences in tonnage for each successive update have been accounted for by new drilling, depletion for mining, and new resource areas. Grade correlation between updated estimates has always remained high. • By-product credits for copper and cobalt form part of the off-take agreement between Panoramic and Jinchuan. • No deleterious elements have been modeled in the resource estimate; the Savannah orebody has low MgO and negligible Arsenic levels. • All block estimates were based on interpolation into 4m N x 20m E x 10m RL parent cells, sub celling to 0.5m N x 2.5m E x 1.25m RL. Block discretisation points were set to 2(Y) x 5(X) x 4(Z) points. The block dimensions are over half the average drill spacing of 25m. A search radius of 150m was used with a minimum of 8 samples and a maximum of 50 samples for all domains. • No selective mining units were assumed in the estimate. • Nickel and cobalt show a very strong correlation. Nickel and copper are more variable. • The geological interpretation was used to derive the domains using massive sulphide content, lithology and structural boundaries. These were wireframed and used as hard boundaries to flag sample data for estimation. • Statistical analysis of the grade populations indicated no

Criteria	JORC Code explanation	Commentary
	• The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	extreme values and a low coefficient of variation. • Validation included comparing the raw data statistics to block estimates, volumes of wireframes to block model volumes, drillholes and block model value plots were produced for a visual check of the grades. Good reconciliation data exists between mined and milled figures.
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• Tonnages estimated on a dry basis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• 0.5%Ni was used as a cut-off when defining the mineralised wireframes. Generally, this is the grade boundary between strongly disseminated sulphides and the ultramafic footwall unit.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• Mining at Savannah has been ongoing since 2004 and has a proven history of well reconciled Resource to Reserve to Production reconciliation therefore mining factors are typically only applied during Ore Reserve conversion.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• Savannah ore has been successfully treated through a 1MTPA SAG mill and flotation circuit since commissioning in 2004. The metallurgical nature of the mineral resource in this estimate has not changed. Metallurgical factors are addressed in Ore Reserve conversion.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• SNM operates under the conditions set out by an environmental license to operate.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• Bulk density determinations are routinely performed. Most determinations involved calculating the core volume and weighing the core in air. Regular checks using the water immersion technique are also carried out to derive a regression formula of measured density versus nickel grade. The regression formula is used to populate missing density values. • Voids within the mineralized zones are non-existent • The search parameters for density were the same as nickel for all domains. Waste material was assigned a value of 2.88, determined from the regression formula.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in	• The classification adopted is based on the level of confidence as set out in the JORC 2012 guidelines. • Measured Resources are defined by areas supported by strong drilling and confined up and down dip by mine development such that confidence in lode volume and

Criteria	JORC Code explanation	Commentary
	continuity of geology and metal values, quality, quantity and distribution of the data).	continuity and grade is very high. Indicated Resources are defined by areas where geological confidence is high and drilling support is strong (equal to or greater than 25m x 25m grid spacing).
	• Whether the result appropriately reflects the Competent Person's view of the deposit.	• The estimate appropriately reflects the view of the competent person.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• The resource estimate has been peer reviewed on site and by Panoramic's corporate technical team.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.	• The relative accuracy of the resource estimate is considered robust as it has been compiled as per the guidelines of the 2012 JORC Code, and knowledge gained from extensive operational history of the mine.
	• The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.	• The statement relates to global estimates of tonnes and grade.
	• These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• Mine to mill reconciliation records throughout the life of the Savannah Project provide confidence in the accuracy of the resource estimate.

Section 4 - Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve. • Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.	• The Savannah 2015 Resource models were used as the basis for conversion to an Ore Reserve. • These models were updated due to mining depletion, sterilisation and geological interpretations based on results from ore development, face sampling, drive mapping and pre-production drilling. • Mineral Resources are inclusive of Ore Reserves.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The competent person is a full time Panoramic Resources employee who conducts routine site visits as part of normal working duties.
Study status	• The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves. • The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.	• Underground mining commenced in January 2005 and has continued for over 10 years until being placed in care and maintenance in May 2016. • The current mine design, mining method, operating parameters, modifying factors, actual costs and knowledge gained from over 10 years of production are used in the Ore Reserve estimate.
Cut-off parameters	• The basis of the cut-off grade(s) or quality parameters applied.	• An initial cut-off grade of 0.8% nickel is used based on an economic assessment and current operating and market parameters. • Economic analysis is carried out for each planned stope and only stopes with a positive return are included in the Ore Reserve estimate.
Mining factors or assumptions	• The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design). • The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc. • The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling.	• The Savannah mine predominantly applies an open stoping with paste fill mining method. • Detailed stope designs are used where access has been developed otherwise preliminary stope designs are used. • A seismic monitoring and management system has been in place for over 10 years of mine production. • Stress and structural models have been developed and are used to identify appropriate mining sequences, stope spans and ground support requirements. • Routine site visits and inspections are conducted by consultant geotechnical engineers.

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td></td><td>• The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate).
• The mining dilution factors used.
• The mining recovery factors used.
• Any minimum mining widths used.
• The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion.
• The infrastructure requirements of the selected mining methods.</td><td>• Grade control assumptions are in line with practice developed over the previous 10 years of operation including ore development is routinely mapped and sampled, stope production is routinely sampled, and monthly mine production is reconciled to milled tonnes and grade.
• Mining dilution at zero grade is applied to stopes and ore development.
• Minimum underground development width is 4.8m and minimum stoping width is 3.0m.
• Mining dilution and recovery factors are applied to development and the detailed and preliminary stope designs as per below:
<table><tr><th>Type</th><th>Dilution</th><th>Mining recovery</th></tr><tr><td>Development</td><td>10%</td><td>95%</td></tr><tr><td>Above 500 fault
-Primary and Secondary stopes
-Sill pillar under paste
-Pit Pillar</td><td>
10%
20%
20%</td><td>
95%
95%
75%</td></tr><tr><td>Below 500 fault
-Primary and Secondary stopes
-Sill Pillars
-Sill Pillars & Stopes with no fill</td><td>
7.5 to 15%
15%
10%</td><td>
95%
90 to 95%
75%</td></tr><tr><td>Below 900 fault
-All stopes</td><td>
10%</td><td>
95%</td></tr></table>• No Inferred Resources are included in the Ore Reserve estimate.
• Infrastructure requirements (other than future capital development) for the selected mining method are established.</td></tr>
<tr><td>Metallurgical factors or assumptions</td><td>• The metallurgical process proposed and the appropriateness of that process to the style of mineralisation.
• Whether the metallurgical process is well-tested technology or novel in nature.
• The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied.
• Any assumptions or allowances made for deleterious elements.
• The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole.
• For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?</td><td>• The metallurgical process is a conventional nickel sulphide flotation technique involving crushing, grinding and flotation to produce a bulk nickel, copper and cobalt concentrate.
• Savannah ore has been successfully treated through a 1MTPA SAG mill and flotation circuit since commissioning in 2004.
• The metallurgical nature is consistent throughout the resource and as such no domaining has been applied.
• Metallurgical recoveries are calculated from plant feed grades in the LOM plan and are based on over 10 years of historical plant performance. Metallurgical recoveries approximate 86% for Nickel, 95% for Copper and 88% for Cobalt.
• Savannah produces a clean bulk nickel, copper and cobalt concentrate and since commissioning in 2004 there have been no deleterious material penalties. As such no allowance has been made for deleterious material.
• The Ore Reserve estimate has been based on appropriate mineralogy and metallurgical factors to meet the existing concentrate off-take specifications.</td></tr>
<tr><td>Environmental</td><td>• The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.</td><td>• Savannah operates under the conditions set out by an environmental license to operate.
• Waste is placed on approved waste dumps or used as backfill in mined voids.</td></tr>
<tr><td>Infrastructure</td><td>• The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour,</td><td>• The Savannah mine has substantial infrastructure in place including a paste fill plant, major electrical and pumping networks, a 1Mtpa throughput plant, a fully</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
	accommodation; or the ease with which the infrastructure can be provided, or accessed.	equipped laboratory, extensive workshop, administration facilities and a 215 person single person quarters camp and tailings storage facility.
Costs	• The derivation of, or assumptions made, regarding projected capital costs in the study. • The methodology used to estimate operating costs. • Allowances made for the content of deleterious elements. • The derivation of assumptions made of metal or commodity price(s), for the principal minerals and co-products. • The source of exchange rates used in the study. • Derivation of transportation charges. • The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc. • The allowances made for royalties payable, both Government and private.	• Costs are based on actual costs occurred in mining, processing and transportation over the FY2016 financial year to May 2016 when the mine was placed in care and maintenance. • Capital underground development costs are derived from the LOM plan and actual costs as per above. • Other capital costs are related to equipment and infrastructure costs and are based on quotes or historical actual costs. • Closure costs have not been included. • Metal prices and exchange rate assumptions are based on the median of a range of external market analysts medium term forecasts. • Flat rate metal prices for nickel of USD6.00/lb, copper of USD2.50/lb and cobalt of USD11.83/lb were used. • Flat rate USD/AUD exchange rate of 0.74 was used. • Net Smelter Return (NSR) factors were sourced from the existing concentrate off-take contract. • WA government and Traditional Owner royalties are included.
Revenue factors	• The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc. • The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.	• Revenue factors are based on metal production in concentrate from the LOM plan, flat rate metal prices for nickel, copper and cobalt (above), flat rate USD/AUD exchange rate (above) and the NSR factors in the existing concentrate off-take contract.
Market assessment	• The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future. • A customer and competitor analysis along with the identification of likely market windows for the product. • Price and volume forecasts and the basis for these forecasts. • For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.	• The concentrate is contracted for sale to Jinchuan Group of China until April 2020. The Savannah concentrate will be trucked to Wyndham Port and then shipped to Jinchuan's smelter/refinery in the Gansu province, northwest China upon a re-commencement of operations.
Economic	• The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc. • NPV ranges and sensitivity to variations in the significant assumptions and inputs.	• Internal cash flow estimates apply an 8% real discount rate for NPV analysis and only economically viable ores are considered for mining. • Sensitivity analysis of key financial and physical parameters is applied to the LOM plan.
Social	• The status of agreements with key stakeholders and matters leading to social licence to operate.	• The Savannah mine is fully permitted and has a co-existence agreement in place with Traditional Owners.
Other	• To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves: • Any identified material naturally occurring risks. • The status of material legal agreements and marketing arrangements. • The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	• No significant unresolved material matters relating to naturally occurring risks, third party agreements or governmental/statutory approvals currently exist.
Classification	• The basis for the classification of the Ore Reserves into varying confidence categories. • Whether the result appropriately reflects the Competent Person's view of the deposit.	• The classification adopted is based on the level of confidence as set out in the 2012 JORC guidelines • Proven Ore Reserves are based on Measured Resources subject to economic viability.

Criteria	JORC Code explanation	Commentary
	• The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).	• Probable Ore Reserves are based on Indicated Resources subject to the economic viability. • The estimate appropriately reflects the view of the competent person. • No Inferred Resources are included in the Ore Reserve estimate.
Audits or reviews	• The results of any audits or reviews of Ore Reserve estimates.	• The Ore Reserve estimate, parent data and economic evaluation is reviewed by Panoramic Resources senior management.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage. • It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• The relative accuracy of the Ore Reserve estimate is considered robust as it is based on the knowledge gained from extensive operational history of the mine. • All currently reported Ore Reserve estimations are considered representative on a global scale. • Mine to mill reconciliation records throughout the life of the Savannah Mine provide confidence in the accuracy of the Ore Reserve estimate.

Table 1 – Lanfranchi Nickel Mine

Section 1 - Sampling Techniques and Data

(Criteria in this section apply to all succeeding sections.)

Criteria	JORC Code explanation	Commentary
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• All sampling for resource estimation purposes at Lanfranchi Nickel Mine (LNM) is based on diamond drill core. Sample selection is based on geological core logging. Individual samples typically vary between 0.2m and 1.2m in length.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc).	• Diamond drilling at LNM is typically NQ2 or LTK60 size. Occasionally BQ and HQ core size holes have been drilled.
Drill sample recovery	• Method of recording and assessing core and chip sample recoveries and results assessed. • Measures taken to maximise sample recovery and ensure representative nature of the samples. • Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	• All recovered diamond core is metre marked by on site geologists; any core loss is determined and recorded as part of the geological logging process. Core recovery is typically 100 percent. • No relationship exists between core recovery and grade.
Logging	• Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. • Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. • The total length and percentage of the relevant intersections logged.	• All core is geologically and geotechnically logged to a standard appropriate for mineral resource estimation purposes. Core is logged from start to end of hole without gaps. Core photography is not undertaken. Drillholes are logged using Excel templates that are code restricted to ensure that only approved data can be entered. The Excel templates are then uploaded to the Lanfranchi SQL Server drillhole database via Datashed.
Sub-sampling techniques and sample preparation	• If core, whether cut or sawn and whether quarter, half or all core taken. • If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. • For all sample types, the nature, quality and appropriateness of the sample preparation technique. • Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. • Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. • Whether sample sizes are appropriate to the grain size of the material being sampled.	• All diamond core is cut using a clipper brick saw and half core sampled for assay. Quarter core samples are sent as part of the LNM QAQC process for check assaying. Sample intervals typically vary between 0.2m and 1.2m and are positioned as to not cross geological boundaries.
Quality of assay data and laboratory tests	• The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.	• All LNM drillhole samples are analysed by Kalassay Group's Kalgoorlie laboratory. The Laboratory process for LNM samples involves: Crush sample to <3mm, pulverise to 90% passing 75um (lab blanks introduced and pulverised at this point). From the pulverised sample, a 0.2g assay aliquot is taken and weighed then digested by 4-Acid digest and

Criteria	JORC Code explanation	Commentary
		analysed by ICP-OES instrument. Laboratory QA/QC is performed on standards, blanks and duplicates. The LNM policy is to scrutinize the results for QA/QC standards and blanks when assay jobs are reported and to request re-runs if result are ± 1SD from the expected value.
	• For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	• No other geophysical or analytical tools have been used to estimate grade.
	• Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	• Certified Reference Material (QAQC) samples are routinely inserted during all sampling at LNM. In addition samples are routinely sent for check analysis at a different Laboratory. The QAQC results indicate that the diamond core assays being used for resource estimation at LNM are a fair representation of the material that has been sampled.
Verification of sampling and assaying	• The verification of significant intersections by either independent or alternative company personnel.	• Significant intersections are calculated by mine geologists and verified/reported on a monthly basis by the Geology Manager.
	• The use of twinned holes.	• Twinning of drillholes is not performed at LNM
	• Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	• Assay data are imported directly from the Kalassay assay files and QA/QC validated via Datashed to the LNM SQL drillhole database.
	• Discuss any adjustment to assay data.	• No adjustment to assay data is made.
Location of data points	• Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.	• Drillhole collars are accurately surveyed for X,Y,Z and azimuth & Dip by site Surveyors using "Total Station" control. Older holes may/may not have collar azimuth/dip measurements. Downhole surveys are generally conducted using single shot or reflex multishot tools at 15m, 30m and every 30m thereafter.
	• Specification of the grid system used.	• The LNM drillhole database contains both MGA94 and local mine grid (KNO) coordinates. All site geological and mine planning work is performed in the local KNO grid system.
		• Conversion from KNO grid to MGA GDA94 Zone 51 is based on a two point transformation: 389084.61E, 513790.88N = 389351.47E, 6513980.38N 389044.77E, 513543.54N = 389313.70E, 6513732.77N
	• Quality and adequacy of topographic control.	
Data spacing and distribution	• Data spacing for reporting of Exploration Results.	• LNM resource estimation drill holes are typically drilled on a regular grid spacing that varies according to the size and consistency of the resource being drilled. Due to the consistent grade and low Coefficient of Variation of nickel mineralisation generally, resource definition drilling at LNM is more for volume estimation purposes than grade estimation.
	• Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied.	• Data spacing is deemed to be sufficient for Mineral Resource estimation and reporting.
	• Whether sample compositing has been applied.	• No sample compositing is undertaken; all core samples are logged and analysed in full.
Orientation of data in relation to geological structure	• Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.	• Underground drill sites are not always ideally positioned for resource definition drilling however no sampling orientation bias is evident. The Ni grade is typically very consistent within individual resource domains and therefore drill orientation is not a determinant for reliable grade estimation
	• If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	
Sample security	• The measures taken to ensure sample security.	• All diamond core samples are taken directly from site to Kalassay for analysis via a local courier service. Sample security is considered adequate.
Audits or reviews	• The results of any audits or reviews of sampling techniques and data.	• No audits/reviews of the sampling techniques have been undertaken in recent time. The procedures used are considered to be industry standard. Mine to mill reconciliation records throughout the life of the Project provide confidence in the sampling procedures.

Section 3 - Estimation and Reporting of Mineral Resources

(Criteria listed in section 1, and where relevant in section 2, also apply to this section.)

<table>
<tr><th>Criteria</th><th>JORC Code explanation</th><th>Commentary</th></tr>
<tr><td>Database integrity</td><td>• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.
• Data validation procedures used.</td><td>• All LNM drillhole and resource samples are logged and recorded using MS Excel templates that are code restricted to ensure that only approved data can be entered. The Excel templates are uploaded to the LNM SQL drillhole database via Datashed software, this also ensures only approved data can be entered into the database. Once Laboratory assays files have been scrutinised and finalised for QAQC they are imported directly into database to ensure there are no transcription errors.</td></tr>
<tr><td>Site visits</td><td>• Comment on any site visits undertaken by the Competent Person and the outcome of those visits.
• If no site visits have been undertaken indicate why this is the case.</td><td>• The competent person (except Lower Schmitz) was formerly a site based Panoramic employee on a rostered, FIFO arrangement.
• The competent person for the 2016 Lower Schmitz Resource estimate is Mr Paul Payne of Payne Geological Services Pty Ltd. A site visit was not undertaken due to the project being on care and maintenance.</td></tr>
<tr><td>Geological interpretation</td><td>• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.
• Nature of the data used and of any assumptions made.
• The effect, if any, of alternative interpretations on Mineral Resource estimation.
• The use of geology in guiding and controlling Mineral Resource estimation.
• The factors affecting continuity both of grade and geology.</td><td>• There is a high level of confidence in the geological interpretation of all LNM resources due to; the extensive operating experience, and the readily recognizable, strongly contrasting mineralised and un-mineralised lithologies. Composites are individually selected for each drillhole based on logging and cut-off grade boundaries rather than using an intercept method. This method ensures where drillholes skim in and out of mineralisation along a resource edge the mineralised grades for the hole are used in the estimation process even though due to the complexity of the wireframe interpretation they may fall just outside the wireframe shape. The standard composite length is 1m. Wireframes are based on drillhole intercepts, survey pick-ups, face mapping and sludge sampling where available. Although rock chip, grab and sludge hole data is also available, they are not used in the estimation process to ensure clustering of lower quality does not bias the estimation process, as such only diamond drilling samples are used for estimation. Wireframes are used to constrain the estimation process to ensure rigid geological boundaries are adhered to. All wireframes are constructed to a 1.0% Ni cut-off grade</td></tr>
<tr><td>Dimensions</td><td>• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.</td><td>
<table>
<tr><th rowspan="2">Resource</th><th rowspan="2">Length</th><th rowspan="2">Width</th><th colspan="2">Depth below surface (limit)</th></tr>
<tr><th>Upper</th><th>Lower</th></tr>
<tr><td>Deacon</td><td>915</td><td>115</td><td>850</td><td>1315</td></tr>
<tr><td>Helmut South Extension</td><td>185</td><td>40</td><td>935</td><td>1160</td></tr>
<tr><td>Lanfranchi</td><td>185</td><td>80</td><td>630</td><td>820</td></tr>
<tr><td>Metcalfe</td><td>285</td><td>40</td><td>810</td><td>935</td></tr>
<tr><td>Martin</td><td>160</td><td>40</td><td>275</td><td>440</td></tr>
<tr><td>Lower Schmitz</td><td>200</td><td>75</td><td>1100</td><td></td></tr>
<tr><td>Cruickshank</td><td>800</td><td>100</td><td>0</td><td>450</td></tr>
<tr><td>Gigantus</td><td>300</td><td>250</td><td>200</td><td>450</td></tr>
</table>
</td></tr>
<tr><td>Estimation and modelling techniques</td><td>• The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of</td><td>• The computer software package "Surpac" was used for all resource models. All resource models have been estimated using ID2 methodology for simplicity and the amount of data available. Previous models run between 2007 and 2010 had run ID2 and OK models in parallel. This approach produced very similar estimates that were,</td></tr>
</table>

Criteria	JORC Code explanation	Commentary
	computer software and parameters used. • The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. • The assumptions made regarding recovery of by-products. • Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation). • In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. • Any assumptions behind modelling of selective mining units. • Any assumptions about correlation between variables. • Description of how the geological interpretation was used to control the resource estimates. • Discussion of basis for using or not using grade cutting or capping. • The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	typically within ±4% on tonnes, ±0.3% in grade and <5% difference on a contained nickel basis. Confidence in the modelling process is also high due to good reconciliation with production data over many years, especially within the Deacon resource. Block model parent cell sizes are matched to drillhole spacing for each resource model with search ellipses aligned parallel to mineralisation trends. Estimated elements in all resource models are as follows: Ni, As, Cu, Co, Fe, MgO, S. Domaining within the model is based upon primary lithology types and separate estimation passes are conducted to ensure both composites and search ellipses are realistic for each rock type. In the case of the Helmut South Extension (HSE) resource, geostatistical analysis of the ore population identified two distinct sample populations. Due to the complexity of the two populations within the main ore shape, they were unable to be domained separately. In this case a top-cut was applied to ensure that overestimation of the resource did not occur. The top-cut was established following an audit of the HSE resource model by BM Geological Services. Minimal stoping has been completed in the HSE resource to adequately study the estimation versus actual data. No other top-cuts were applied across the other resource models. Validation of the resource estimate was completed by onscreen visual validation of block grades vs. drillhole assays in sectional view and via Swanson plots of composites grades vs. block grades. In the case of the Deacon resource, the model composites and block model reports were compared in 100m increments along strike with very good correlation between "data in" and "data out".
Moisture	• Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	• Tonnages are estimated on a dry basis.
Cut-off parameters	• The basis of the adopted cut-off grade(s) or quality parameters applied.	• All LNM resource models are modelled to a nominal cut-off grade of 1% Ni. In some cases, minor internal dilution was included in the intercept to avoid over-complication of the wireframe shape.
Mining factors or assumptions	• Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	• No minimum mining width assumptions are made during the resource wireframing or estimation process. Mining parameters, including minimum mining width assumptions are applied during the conversion to Ore Reserves. Internal dilution is included during the resource estimation process when it is obvious selective mining is not a realistic option.
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• No metallurgical factors or assumptions are made during the Resource estimation process. These matters are addressed during conversion to Ore Reserve.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts,	• No environmental factors or assumptions are made during the Resource estimation process.

Criteria	JORC Code explanation	Commentary
	particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.	• For all LNM Resources a nickel vs. SG correlation plot is generated using all available SG data. All samples without an SG measurement are assigned a calculated SG value based on the regression analysis. During the estimation process, actual SG measurements were given priority over calculated values. LNM has an extensive SG database, generated over many years of operating experience. Determinations have typically been performed using the water immersion technique. The technique is adequate due to low core porosity, fresh rock underground environment of the LNM operation.
Classification	• The basis for the classification of the Mineral Resources into varying confidence categories. • Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). • Whether the result appropriately reflects the Competent Person's view of the deposit.	• Classification of the LNM resource models are based primarily on drill density in conjunction with increased confidence from existing ore development. Significant ore development has been completed within the Deacon Resource which enables a Measured category to be applied to most parts of this Resource.
Audits or reviews	• The results of any audits or reviews of Mineral Resource estimates.	• In March 2013, BM Geological Services (BMGS) conducted an independent review of all LNM Resource estimates. The review covered the building and interpolation of grades in the relevant block models and their representation of grades based on the composite files. Overall BMGS concluded the resource models grades compared favourably with drill hole composite grades. Concern was expressed however about the representivity of the Helmut South Extension (HSE) Resource estimation, and the potential for overcalling of grades within the BM. To alleviate this issue, top-cuts were applied by BMGS and the model re-run.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• A high level of confidence exists for all LNM Resource estimates. The estimates are based on many years of operating experience. Recent mine production was derived from 4 of the 6 underground resources with good reconciled agreement between Resource/Reserve grades and mine production. • All estimates are global estimates of tonnes and grade.

Section 4 - Estimation and Reporting of Ore Reserves

(Criteria listed in section 1, and where relevant in sections 2 and 3, also apply to this section.)

Criteria	JORC Code explanation	Commentary
Mineral Resource estimate for conversion to Ore Reserves	• Description of the Mineral Resource estimate used as a basis for the conversion to an Ore Reserve • Clear statement as to whether the Mineral Resources are reported additional to, or inclusive of, the Ore Reserves.	• The Lanfranchi 2015 Resource models were used as the basis for conversion to an Ore Reserve. • These models were updated due to mining depletion, sterilisation and geological interpretations based on results from ore development, face sampling, drive mapping and pre-production drilling. • Mineral resources are inclusive of the Ore Reserves.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• The competent person is a full time Panoramic Resources employee who conducts routine site visits as part of normal working duties.
Study status	• The type and level of study undertaken to enable Mineral Resources to be converted to Ore Reserves. • The Code requires that a study to at least Pre-Feasibility Study level has been undertaken to convert Mineral Resources to Ore Reserves. Such studies will have been carried out and will have determined a mine plan that is technically achievable and economically viable, and that material Modifying Factors have been considered.	• Underground mining recommenced in 2005 and has continued for over 10 years until being placed in care and maintenance in November 2015. • The current mine design, mining method, operating parameters, modifying factors, actual costs and knowledge gained from over 10 years of production are used in the Ore Reserve estimate.
Cut-off parameters	• The basis of the cut-off grade(s) or quality parameters applied.	• For mine planning, an initial cut-off grade of 1% Ni is used, except for air-leg mining which is 2.0%Ni. • Economic analysis is carried out for each planned stope, and only stopes with a positive return are included in the Ore Reserve estimate.
Mining factors or assumptions	• The method and assumptions used as reported in the Pre-Feasibility or Feasibility Study to convert the Mineral Resource to an Ore Reserve (i.e. either by application of appropriate factors by optimisation or by preliminary or detailed design). • The choice, nature and appropriateness of the selected mining method(s) and other mining parameters including associated design issues such as pre-strip, access, etc. • The assumptions made regarding geotechnical parameters (eg pit slopes, stope sizes, etc), grade control and pre-production drilling. • The major assumptions made and Mineral Resource model used for pit and stope optimisation (if appropriate). • The mining dilution factors used. • The mining recovery factors used. • Any minimum mining widths used. • The manner in which Inferred Mineral Resources are utilised in mining studies and the sensitivity of the outcome to their inclusion. • The infrastructure requirements of the selected mining methods.	• Wide ore zones from Deacon, HSE and Metcalfe are extracted using an open stoping with paste fill mining method. • Narrow high grade ore zones from Lanfranchi and Schmitz are mined using air-leg mining methods. • Detailed stope designs are used where access has been developed otherwise preliminary stope designs are used. • A seismic monitoring and management system is in place. • Stress models are regularly updated and used to determine appropriate mining sequences, stope spans and ground support requirements. • Routine site visits and inspections are conducted by consultant geotechnical engineers. • Grade control assumptions are in line with practice developed over the previous 10 years of operation including ore development is routinely mapped and sampled, stope production is routinely sampled and monthly mine production is reconciled to milled tonnes and grade. • Mining dilution of 10% at zero grade is applied to stopes; • Mining dilution of 4% at zero grade is applied to jumbo ore development; • 95% metal recovery is applied to stopes. • 99% metal recovery is applied to jumbo ore development. • 70% extraction factor is used for air-leg mining where pillars are used to support the hanging wall. • Minimum stoping width is 3.0m. • Minimum air-leg stope width is 1.8m. • No Inferred Resources are included in the Ore Reserve estimate. • Infrastructure requirements (other than future capital development) for the selected mining method are established.
Metallurgical factors or assumptions	• The metallurgical process proposed and the appropriateness of that process to the style of mineralisation. • Whether the metallurgical process is well-tested technology or novel in nature.	• Lanfranchi ore is a nickel sulphide that can be treated using conventional nickel sulphide techniques involving crushing, grinding and flotation to produce a concentrate. The ore has been processed at the Kambalda Concentrator since the 1970's. • The Lanfranchi ore is processed under an Ore Tolling and

Criteria	JORC Code explanation	Commentary
	• The nature, amount and representativeness of metallurgical test work undertaken, the nature of the metallurgical domaining applied and the corresponding metallurgical recovery factors applied. • Any assumptions or allowances made for deleterious elements. • The existence of any bulk sample or pilot scale test work and the degree to which such samples are considered representative of the orebody as a whole. • For minerals that are defined by a specification, has the ore reserve estimation been based on the appropriate mineralogy to meet the specifications?	Concentrate Purchase Agreement (OTCPA) with BHP Billiton Nickel West (BHP) until February 2019. • Recovery is defined in the OTCPA and is based on the averaged nickel head grade delivered on a monthly basis. • There are insignificant amounts of deleterious elements in the current Resources.
Environmental	• The status of studies of potential environmental impacts of the mining and processing operation. Details of waste rock characterisation and the consideration of potential sites, status of design options considered and, where applicable, the status of approvals for process residue storage and waste dumps should be reported.	• Lanfranchi operates under the conditions set out by an environmental license to operate. • Waste rock is inert basalt and classified as NAF. • Waste is placed on approved waste dumps or used as backfill in mined voids.
Infrastructure	• The existence of appropriate infrastructure: availability of land for plant development, power, water, transportation (particularly for bulk commodities), labour, accommodation; or the ease with which the infrastructure can be provided, or accessed.	• The Lanfranchi mine has substantial mine infrastructure in place, including a paste fill plant, major electrical and pumping networks, administration facilities and a 150 persons accommodation village.
Costs	• The derivation of, or assumptions made, regarding projected capital costs in the study. • The methodology used to estimate operating costs. • Allowances made for the content of deleterious elements. • The derivation of assumptions made of metal or commodity price(s), for the principal minerals and co-products. • The source of exchange rates used in the study. • Derivation of transportation charges. • The basis for forecasting or source of treatment and refining charges, penalties for failure to meet specification, etc. • The allowances made for royalties payable, both Government and private.	• Capital costs are related to the renewal of existing mining equipment and are based on quotes from equipment suppliers or historical actual costs. • Operating costs are based on actual mining and transportation costs over the FY2015 financial year. • Processing costs are based on the OTCPA with BHP. • Closure costs have not been included. • Metal prices and exchange rate assumptions are based on the median of a range of external market analysts medium term forecasts. • Flat rate metal prices for nickel of USD6.00/lb and copper of USD2.50/lb were used. • Flat rate USD/AUD exchange rate of 0.74 was used. • Payability and penalties for delivering off-spec ore are defined in the OTCPA. Based on operating history and the extremely low-levels of deleterious elements any occurrence of such penalties is considered highly unlikely. • WA government royalties are included.
Revenue factors	• The derivation of, or assumptions made regarding revenue factors including head grade, metal or commodity price(s) exchange rates, transportation and treatment charges, penalties, net smelter returns, etc. • The derivation of assumptions made of metal or commodity price(s), for the principal metals, minerals and co-products.	• The head-grade is derived from the LOM plan and the geological model (with suitable modifying factors applied). • Revenue is calculated from the flat rate nickel and copper prices (above), flat rate USD/AUD exchange rate (above) and the recovery and NSR factors as defined in the OTCPA.
Market assessment	• The demand, supply and stock situation for the particular commodity, consumption trends and factors likely to affect supply and demand into the future. • A customer and competitor analysis along with the identification of likely market windows for the product. • Price and volume forecasts and the basis for these forecasts. • For industrial minerals the customer specification, testing and acceptance requirements prior to a supply contract.	• The OTCPA to purchase the ore is in place with BHP until February 2019.
Economic	• The inputs to the economic analysis to produce the net present value (NPV) in the study, the source and confidence of these economic inputs including estimated inflation, discount rate, etc. • NPV ranges and sensitivity to variations in the significant assumptions and inputs.	• Internal cash flow estimates apply an 8% real discount rate for NPV analysis and only economically viable ores are considered for mining. • Sensitivity analysis of key financial and physical parameters is undertaken as part of the Ore Reserve process
Social	• The status of agreements with key stakeholders and matters leading to social licence to operate.	• Licence to operate from WA State Government. • Pre native title mining tenements for current Ore Reserves. • Good relationships with local community and strong

Criteria	JORC Code explanation	Commentary
		cooperation with neighbouring mining operations.
Other	• To the extent relevant, the impact of the following on the project and/or on the estimation and classification of the Ore Reserves: • Any identified material naturally occurring risks. • The status of material legal agreements and marketing arrangements. • The status of governmental agreements and approvals critical to the viability of the project, such as mineral tenement status, and government and statutory approvals. There must be reasonable grounds to expect that all necessary Government approvals will be received within the timeframes anticipated in the Pre-Feasibility or Feasibility study. Highlight and discuss the materiality of any unresolved matter that is dependent on a third party on which extraction of the reserve is contingent.	• No significant unresolved material matters relating to either naturally occurring risks, third party agreements or governmental/statutory approvals, currently exist. • The OTCPA to purchase the ore is in place with BHP until February 2019.
Classification	• The basis for the classification of the Ore Reserves into varying confidence categories. • Whether the result appropriately reflects the Competent Person's view of the deposit. • The proportion of Probable Ore Reserves that have been derived from Measured Mineral Resources (if any).	• The classification adopted is based on the level of confidence as set out in the JORC 2012 guidelines • Proven Ore Reserves are based on Measured Resources subject to economic viability. • Probable Ore Reserves are based on Indicated Resources subject to economic viability. • The estimate appropriately reflects the view of the competent person. • No Inferred Resources are used in the Ore Reserve estimate.
Audits or reviews	• The results of any audits or reviews of Ore Reserve estimates.	• The Ore Reserve estimate, parent data and economic evaluation is reviewed by Panoramic Resources senior management.
Discussion of relative accuracy/ confidence	• Where appropriate a statement of the relative accuracy and confidence level in the Ore Reserve estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the reserve within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors which could affect the relative accuracy and confidence of the estimate. • The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. • Accuracy and confidence discussions should extend to specific discussions of any applied Modifying Factors that may have a material impact on Ore Reserve viability, or for which there are remaining areas of uncertainty at the current study stage. • It is recognised that this may not be possible or appropriate in all circumstances. These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	• The relative accuracy of the Ore Reserve estimate is considered robust as it is based on the knowledge gained from extensive operational history of the mine. • All currently reported Ore Reserve estimations are considered representative on a global scale. • Mine to mill reconciliation records throughout the life of Lanfranchi provide confidence in the accuracy of the Ore Reserve estimate.