REPORT NI 43-101

TECHNICAL REPORT ON THE

MINERAL RESOURCES AND RESERVES OF THE

WOLFRAM CAMP MINE PROJECT,

AUSTRALIA

Prepared for

Almonty Industries

by

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1 SUMMARY

1.1 Introduction and Overview

This report was prepared to provide a Technical Report compliant with the provisions of National Instrument 43-101 - Standards of Disclosure for Mineral Projects, ("NI 43-101"), and comprises a review and summary of Resource and Reserve Estimations for the Wolfram Camp Mine project, as of the end of March 2017. The project is located in the state of Queensland in Australia. These current estimates were completed during October, 2015. The mine is an open pit operation, although it has not been in production since March 2015. The principal mine product is currently a tungsten concentrate. From 2012-2013 the mine also produced a molybdenum concentrate.

This report was prepared by Adam Wheeler, at the request of Almonty Industries ("Almonty"). Assistance and technical detail were supplied by the technical personnel at Wolfram Camp. Adam Wheeler visited the mine site from June 18th-21st, 2014 and from October 28th – November 1st, 2014.

The Wolfram Camp mineralisation was discovered in 1894 and previous mining operations have been based either on surface eluvial mining of residual wolframite grains or on the underground extraction of high-grade pipes of erratic shape and lateral dimensions. The hard rock mines of the Wolfram Camp mineral field have recorded combined production of at least 10,000t of wolframite, molybdenite, bismuth and mixed concentrates. Eluvial and early hard rock production is poorly recorded. The main periods of hard rock mining were 1908-1920, 1967-1972 and 1978-1982.

After a very brief period of production in 2008 under former owners, the mine restarted open pit ore production during the latter months of 2011, and the mill was commissioned during the beginning of 2012. Production continued until March 2015, after mine production was stopped, so as to allow a number of improvements to be made to the processing facilities. It is intended to restart the mine in 2018.

1.2 Ownership

Almonty Industries Inc ("Almonty"), is a corporation governed by the Canada Business Corporations Act (the "CBCA"). Almonty trades on the TSX Venture Exchange (TSX-V) under the symbol "AII". Almonty owns a 100% interest of each of Wolfram Camp Mining Pty and Tropical Metals Pty, who collectively own 100% of the Wolfram Camp tungsten and molybdenum mine.

1.3 Geology and Mineralisation

The Wolfram Camp Mineral Field is dominated by the Ootann Supersuite granite intrusives and related greisen alteration and mineralisation. Hodgkinson Formation sediments occur to the north-east of the mineralised contact with the Permian-Carboniferous granite. Minor sulphide mineralisation has been seen in veinlets with quartz and minor calcite up to a few hundred metres from the contact.

The granite which hosts the mineralisation at Wolfram Camp is the James Creek Granite. This granite has been extensively altered over approximately 3km of the contact with the sediments and volcanics. The contact appears to dip at 40^o - 60^o to the north around the arcuate northern edge of the granite, but there is significant evidence to suggest that the current surface of the granite to the south of the exposed contact is close to the original intrusive contact.

Alteration and mineralisation occur near the contact and are considered to be related to post-intrusion hydrothermal activity. The quartz pipes and sheets formed in cooling fractures parallel to the contact and in vertical to sub-vertical tension joints. These fractures and joints were best developed in the vicinity of rolls and flexures in the contact.

There are three principal types of mineralisation. The first, quartz pipes, comprise white to clear or smoky quartz, commonly containing vugs and with lumps of wolframite, molybdenite, native bismuth (often coated with bismuthinite), scheelite, pyrite, arsenopyrite, pyrrhotite and minor calcite, siderite, chalcopyrite, fluorite, sphalerite, galena and cassiterite. The lumps of wolframite can be over 1m in diameter and molybdenite lumps can reach 0.5m in diameter. Grades vary between pipes but grades in individual pipes tend to be consistent. Some pipes are wolframite rich, while others are molybdenite rich. Pipes can vary in shape from cylindrical to sheets or elongated veins.

The second type of mineralisation occurs within quartz greisen zones, and consists of vuggy crystalline quartz with variable, and sometimes rich, disseminated wolframite, molybdenite, bismuth, scheelite, pyrite, arsenopyrite and other minor minerals including mica. Mineral grains of wolframite and molybdenite vary commonly between 0.5mm to 1cm, although finer and coarser grains do occur. Mineralised greisen is generally present around most pipes, and in some areas forms more or less continuous zones between the pipes. The third type of mineralisation occurs within mica greisen zones, with increasing amounts of muscovite and decreasing quartz; with only minor disseminated wolframite and molybdenite and other sulphide minerals. No relict granitic texture is visible. Grain sizes of the target minerals are similar to those in the quartz greisen.

Adjacent properties where historically mining and production have occurred, and that are at present the subject of ongoing exploration programmes, include Bamford Hill and Mount Carbine, 25km to the south and 65km to the NNE of Wolfram Camp respectively.

1.4 Database and Resource Estimation

A limited amount of drilling may have been carried out before the 1970s and although data exists for surface and underground drilling completed in the 1970s, there are no detailed records of this work. The various drilling programmes completed at Wolfram Camp since the 1970s are summarised below; in terms of diamond drillholes (DD) and reverse circulation (RC) drillholes, stemming from exploration work done by previous owners between 1995-2010, as well as blasthole exploration samples (BEX) completed by Almonty since 2014:

- 1981-82, Tenneco Oil and Minerals (Tenneco) – 12 DD holes.
- 1994-96, Allegiance Mining NL (Allegiance) – 37 RC holes.
- 2005-06, Queensland Ore Limited (QOL) – 163 holes, mostly RC.
- 2010, Planet Metals Limited (PML) - 200 holes, mostly RC.
- 2014-15, Almonty – 1,417 BEX holes.

The final data from these drilling programmes, after rejection of suspect/abandoned holes, consists of data from 351 reverse circulation holes covering 14,586m of drilling, data from 68 diamond drillholes covering 3,916m of drilling and data from 1,417 BEX holes covering 36,092m. These data contain assays for W, Mo and As. There are also assays for Bi and Sn in the diamond drillhole data.

Since restart of the mine in 2012 by Wolfram Camp Mining (WCM), grade control (GC) drilling results have been accumulated from open pit blasthole samples. This database now consists of data from 55,195 GC holes, covering over 321km of drilling. These data contains assays for W, Mo, Bi, As and Fe. Both databases exist in Excel form.

On the mine site the combined sources of drilling are used for the creation of a short-term planning resource block model, using Datamine software, which is regularly updated with more GC data. This block model covers the main upper part of the Wolfram Camp orebody underlying the current pit, as well as the adjoining Parrotts orebody to the north-west. It contains parent blocks sized as 5m x 5m x5m, with sub-blocks down to a size of 1m x 1m x 2.5m, with W (and derived WO_3) grades, estimated using inverse-distance weighting. In this estimation the model has been divided into 4 four different zones, in which quite different search orientations have been defined. These orientations have been derived from geological interpretation as well as observation of old mined workings.

An updated resource estimation has been developed by Adam Wheeler, using the application of CAE Datamine software. All available GC, DD, RC and BEX data have been used. In this methodology, 2.5m composites have been generated, and the mineralised zones have been demarcated based on 0.09% and 0.3% WO_3 grade thresholds. These zones have then been extrapolated into the resource model. Grades of WO_3 and MoS_2 have ultimately been estimated using ordinary kriging, with parameters tested against reconciliation block models from previous production.

1.5 Mine Planning

The current open cut is approximately 800m along strike. In general the pit is advanced with benches extended out to the design pit shell on the north-south sides, and is deepened in 3-4 sectors along strike.

Drilling and blasting will be carried out by a specialist D&B contractor. Blasts are planned over 5m bench heights, with combined ore and waste partitions. The individual models determined from GC drilling are used to delineate different categories of material for mining, based on cut-off levels of 0.07, 0.12 and 0.3% WO_3. Separate models for each blast area also built up.

Blastholes, 89mm in diameter, are drilled on 2.7 x 2.4 m pattern. Plastic hoses are placed in high grade holes, which are not blasted. This helps against excessive fragmentation of wolframite, and the hoses provide an estimate of blast displacement. All blasting generally uses ANFO. Subsequent to blasting, the positions of the plastic hoses are re-surveyed, and the original ore/waste delineations are modified according to the measured displacements, as well as by visual assessment by geologists. Different colour ribbons are used to demarcate the different ore/waste categories.

Digging of material is done with a backhoe excavator, sitting on top of the broken muckpile, loading 40t trucks. Digging is done in 3 vertical passes: the first for the heave above the original bench floor, the second for the 0-2.5m depth cut and the third for the 2.5-5m depth cut. Ribbons are marked up individually for each cut prior to mining, based on the blast displacements at the top of each cut. Any additional high grade material spotted visually by geologists is also mined and stockpiled separately.

Clay and topsoil overburden from the mine is stockpiled separate from other waste dumps, for use on closure for rehabilitation. Waste and mineralised waste loads are hauled to stockpiles, and ore is trucked to the ROM pad adjacent the processing facilities. Mineralised waste is either stockpiled or sent to the ROM pad and crushed. The mineralised waste stockpile is then screened, with <15mm material being sent to the mill, 15-50mm material is sent to the ore sorter, and >50mm material is sent to the crusher as required. Mined tonnages are reconciled against monthly stockpile surveys and these in turn are used to reconcile against the short-term planning block model.

A current pit design has been based on an updated pit optimisation completed on the updated resource block model, and this is the physical limit applied to the current reserve estimate.

1.6 Environmental Studies

The Environmental Management Plan, produced in 2007, covered the tenements ML20486 and ML20534, and dealt with the potential environmental impacts from mining and associated activities, including:

- Pit excavation;
- Product and topsoil/overburden stockpiling;
- On-site processing;
- Sediment control works;
- Limited fuel, diesel and explosive storage;
- Access tracks;
- Air quality
- Water management
- Noise and vibration
- Waste management
- Land and management
- Community, social and cultural issues
- Monitoring

WCM produce a Plan of Operations biennially and is prepared consistent with the following:

- Schedule of Conditions***** of Environmental Authority No. MIN102648011 (EA), dated 7 August 2012.
- Section 234(3) of the Environmental Protection Act 1994
- Department of Environment and Heritage Protection (DEHP) guidelines:
- Calculating Financial Assurance for Mining Projects (DERM 2011)
- Preparing a plan of operations and audit statement for level 1 mining projects (DEHP 2012b).
- DEHP information sheet Plan of operations (DEHP 2012a).

Each Plan of Operations is accompanied by an Environmental Audit Statement produced by independent consultants which highlights any shortcomings and non-compliance.

WCM produce weekly, monthly and annual reports which monitor all aspects of the mining operation, including environmental matters.

1.7 Mineral Processing

The process plant is primarily based on gravimetric separation, aimed at recovering a high grade wolframite concentrate. During 2013, it was able to crush 369kt of material and (after ore-sorting) process 259kt of ore, with an average feed grade of 0.25% WO_3. During 2014 the plant processed 345 kt of ore, with an average fed grade of 0.22% WO_3. The currently planned processing plant recovery is 71%; with upgrades to the crushing, ore sorting, spiral separators and shaking tables. The overall mill capacity has also been increased to 518 ktpa. The planned milling improvements are going to be implemented during 2017.

The primary crushing circuit employs a 90mm jaw crusher, with a nominal 51tph capacity, followed by a 25mm cone crushers. Crushed ore is passed through two double-deck dry screens, from which +30mm coarse material is fed to XRF ore sorters. Ore sorter rejects are sent for waste disposal. Material selected by the ore sorted is then passed onto fine cavity cone crushers. Finally accepted -2mm material will then be passed onto the spirals.

The fine and coarse fractions pass onto two parallel banks of triple start spiral classifiers and from there onto Wilfley shaking tables. Recoveries from the tables have been recently further improved with the use of flotation frames, with Xanthate to assist in sulphide removal.

The concentrate from the shaking tables is subjected to batch flotation to reduce the fine sulfide content. The sulfide reduced concentrate is dried and cooled. The accepted material is then transferred to the dressing plant. Here the material goes through a rotary diesel dryer, and from there onto a rare earth roll (RER) magnetic separator. The material is passed through the RER three times. The rejects from the RER, containing scheelite, are currently stored, but will be processed in the future with regrinding and flotation. The RER accepts are split into 3 streams. One stream with relatively high iron is passed through an electromagnetic (EM) unit at low magnetic settings. Low Fe material from the EM is blended back with the accepts from the RER. The high Fe material is retained and blended back when possible. The other 2 streams from the RER are bagged and assayed. Any material with high uranium and thorium (U+Th) is separated, and blended to allow the sale of acceptable concentrates.

Concentrate grades are typically 63% WO_3. The final saleable concentrate is bagged (weighed and sampled) and transported by semi-trailer to Brisbane.

1.8 Mineral Resource and Reserve Estimates

The evaluation work was carried out and prepared in compliance with Canadian National Instrument 43-101, and the mineral resources in this estimate were calculated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council in May, 2014. The current resource estimation is shown in Table 1-1 and Table 1-2. The resources shown are pit-constrained resources, based on an updated pit optimisation. There are no measured resources.

Table 1-1. Wolfram Camp – Indicated Mineral Resources
Pit-constrained resource, as of 31st August, 2015

Resource Category	Tonnes kt	WO_3 %	MoS_2 %
Indicated	514	0.23	0.07

Notes:

. Cut-off = 0.10% WO3
. Historic underground mined material removed
. Prices used in optimisation:

US \$/mtu WO_3	**400**
US \$/t MoS_2	**25,000**

. Minimum width = 1m
. Resources shown are inclusive of reserves

Table 1-2. Wolfram Camp – Inferred Mineral Resources
Pit-constrained resource, as of 31st August, 2015

Resource Category	Tonnes kt	WO_3 %	MoS_2 %
Inferred	1,879	0.31	0.08

Notes:

. Same cut-off and controls as above

The current reserve estimation, for an open pit mine plan developed from this resource base, is shown in Table 1-3.

Table 1-3. Wolfram Camp – Probable Mineral Reserves
At 31st August, 2015

Reserve	**Tonnes**	**WO_3**
Category	**kt**	**%**
Probable Reserves	375	0.22

Notes

- **Cut-off = 0.08% WO_3**
- **Mining factors of applied of**
  - **Dilution = 10%**
  - **Losses = 10%**
- **Pit design also contain 187kt of inferred resources at economic grades**

The pit design containing this reserve stems from an updated pit optimisation. The principal operating costs for future operation have been updated to US$14.08/t ore for processing and administration and US$3.69/t rock for open pit mining. The open pit design also contains 1,556 Kt of waste, which gives a strip ratio (waste:ore) of 4.2. The cut-off grade of 0.08% WO_3 stems from the breakeven cut-off grade calculated with an APT WO_3 price of US$364/mtu. Corresponding with this pit reserve and assumed metal price, a total operating margin of US$5.1M has been determined.

1.9 Conclusions

1. The Wolfram Camp open pit mine was producing for over 4 years, from 2012 - 2015. The open pit mining practices were progressively improved during this period, along with the planning and grade control systems.

2. Wolfram Camp has all permits and licenses to operate and remain in compliance with appropriate regulations. It has no restrictions with respect to waste dumping or tailings capacity.

3. Grade control (GC) samples from blasthole drilling in the open pit mining operations have in general corresponded fairly well with previous exploration diamond drilling (DD) and reverse circulation (RC) drilling results for the mined areas. This has supported the use of GC samples in resource estimation, and together with reconciliation information, has provided a very important assistance in the development of parameters for updated resource modelling.

4. In the author's opinion, the current resource and reserves estimates for Wolfram Camp are conservative, because of reasons which include:
   a) Areas within only relatively widely spaced exploration data, where some mineralised intersections will have been missed.
   b) The currently orebody model has been limited to a depth of 490m, which represents the approximate base of drilling information, not the geological base of the deposit.
   c) There are known mineralised extensions, both along-strike in both directions as well as at depth, where historical underground workings demonstrate mineralisation. At current metal price levels, these areas offer potential for future underground reserves.
   d) The very erratic distribution quartz pipes and mineralised greisens is unique to the Wolfram Camp area, and means that even with BEX drilling on a 10m x 10m grid, there will still be a high proportion of inferred resources as the pit deepens and advances.

5. Owing to the very erratic nature of mineralisation, and the relatively wide spacing of available exploration drilling, compared to the scale of mineralised structures, the proportion of *Inferred* to *Indicated* resources is high. As the pit advances with more blasthole sampling, progressively more reserves can be determined, approximately 25m beneath the base of the open pit at any time. Based on the optimisation results, where *Inferred* resources have been enabled, an open pit life of 4 years is suggested, before the additional contribution of potential extension zones.

6. Significant improvements are being made to the plant during the shutdown. These changes have affected the crushing, ore sorting, spiral separators and shaking tables, and should enable improved metallurgical recoveries, reduced processing costs, an increased mill capacity. There are also improvements to assist tailings disposal.

2 INTRODUCTION

2.1 Introduction

This Technical report was prepared in compliance with the provisions of National Instrument 43-101 - Standards of Disclosure for Mineral Projects, ("NI 43-101"), and comprises a review and summary of Resource and Reserve estimates for the Wolfram Camp Mine project, as of the end of August 2015. The current estimates were completed during October, 2015. The mine, an open pit operation, is located in the state of Queensland in Australia and at present produces a tungsten concentrate. From 2012-2013 the mine also produced a molybdenum concentrate.

This report was prepared by Adam Wheeler, at the request of Mr. N. Alves, of Almonty Industries. Assistance and technical detail were supplied by the technical personnel at Wolfram Camp. Adam Wheeler visited the site from June 18th-21st, 2014 and from October 28th – November 1st, 2014.

After a very brief period of production in 2008 under former owners, the mine restarted open pit ore production during the latter months of 2011, and the mill was commissioned during the beginning of 2012 and continued until mid-2015. Since that time the mine has been shut down, while mill improvements are being implemented.

2.2 Terms of Reference

Adam Wheeler was commissioned by Almonty Industries, to provide an updated resource and reserve estimation, which can be presented as an independent Technical Report on the Mineral Resources and Mineral Reserves at Wolfram Camp. This Technical Report has been prepared to be compliant with the provisions of National Instrument 43-101 - Standards of Disclosure for Mineral Projects ("NI 43-101"). The report is considered current as of October 31st, 2015.

The Qualified Person responsible for the preparation of this report is Adam Wheeler (C.Eng, Eur.Ing), an independent mining consultant. In addition to a site visit, Wheeler has carried out studies of all relevant parts of the available literature and documented results concerning the project and held discussions with technical personnel at Wolfram Camp regarding all pertinent aspects of the project.

The estimate of mineral resources contained in this report conforms to the CIM Mineral Resource and Mineral Reserve definitions (May, 2014) referred to in NI 43-101.

2.3 Sources of Information

In conducting this study, Adam Wheeler has relied on reports and information prepared by Wolfram Camp. The information on which this report is based includes the references shown in Section 27. Adam Wheeler has made all reasonable enquiries to establish the completeness and authenticity of the information provided, and a final draft of this report was provided to Almonty and Wolfram Camp, along with a written request to identify any material errors or omissions prior to finalisation.

2.4 Units and Currency

All measurement units used in this report are metric, and currency is expressed in US Dollars unless stated otherwise. The exchange rate used in the study described in this report is US$0.755 to 1.00 AUD, unless otherwise stated.

3 RELIANCE ON OTHER EXPERTS

Adam Wheeler has reviewed and analysed data provided by Wolfram Camp and has drawn his own conclusions there from. Adam Wheeler has not performed any independent exploration work, drilled any holes or carried out any sampling and assaying. While exercising all reasonable diligence in checking and confirmation, Adam Wheeler has relied upon the data presented by Wolfram Camp, and previous reports on the property in formulating his opinions.

Title to the mineral lands for the Wolfram Camp property has not been confirmed by Adam Wheeler and Adam Wheeler offers no opinion as to the validity of the exploration or mineral title claimed.

4 PROPERTY DESCRIPTION AND LOCATION

The Wolfram Camp tungsten-molybdenum-bismuth project is located 90km west of Cairns and approximately 18km outside the township of Dimbulah in northern Queensland (Figure 4-1 and Figure 4-2)). Wolfram Camp Mining Pty Ltd (WCM) is a wholly-owned subsidiary of Almonty Industries and holds 85% of the project; Tropical Metals Pty Ltd ("TMPL") (which is also a wholly owned subsidiary of Almonty) holds the remaining 15%. The project is located on the Chillagoe 1:250,000 Geological Sheet 7863, and on the Chillagoe 1:100,000 Topographic Sheet 7863, centred at AMG 84 coordinates 835000E and 811000N.

The Wolfram Camp Mining (WCM) joint venture partners currently hold four (4) Mining Leases, as shown in Table 4-1 and Figure 4-3. These leases are sufficient to cover the project's infrastructure requirements and resource areas as well as buffer zones. A large proportion of the surface of the mining licenses has been extensively disturbed by previous mining activities. The Mining licenses entitle WCM to machine-mine material for tungsten and molybdenum production, with full surface rights and access. A compensation agreement associated with these Mining Leases is with the Mareeba Shire Council, which requires a payment and ongoing maintenance at an agreed standard. A Native Title Agreement for these mining leases is also in place which requires an Annual payment to the Djungan people and certain conditions to be met, such as cultural heritage protection and employment. There are no other agreements associated with these Mining Leases.

WCM and TMPL collectively hold five (5) Exploration Permits, as also shown in Figure 4-3, with details shown in Table 4-2. This table also shows the forward commitments required to retain these Exploration Permits. These Exploration Permits, under Queensland's Mineral Resources Act 1989, allows the holder to take action to determine the existence, quality and quantity of minerals on, in or under land by methods which include prospecting, geophysical surveys, drilling, and sampling and testing of materials to determine mineral bearing capacity or properties of mineralisation.

Queensland State royalties are calculated and paid annually, based on 2.7% of the value received of shipment invoices minus shipping costs. The first $AUD100,000 of metal value is royalty-free each year.

As well as having a mining leases granted, the Department of Environment and Heritage Protection issues an Environmental Authority (EA) to Operate. The WCM EA is EPML00831213. A requirement of the EA is to lodge and have approved a Plan of Operation (PoO) for fixed periods of time. This is described in more detail in Section 20.2. The total calculated rehabilitation liability presented in the PoO for January 2017 was AUD2,528,500.

There are no other known factors or risks that may affect the rights or ability to work on the property.

Table 4-1. Mining Lease Details

Tenement	Area (ha)	Original Grant Date	Most Recent Term Sought Status	Status	Expiration Date
ML 20486	160.0	01-Dec-07	20 years	Granted	30-Nov-27
ML 20534	35.7	01-Dec-07	20 years	Granted	30-Nov-27
ML 5117	2.02	26-Sep-85	21 years	Granted	30-Sep-27
ML 4935	2.023	04-Mar-76	20 years	Granted	30-Sep-27

Table 4-2. Exploration Permit Details

Tenement	Area (km^2)	Original Grant Date	Most Recent Term Sought Status	Status	Expiration Date
EPM 8884	12.6	29-Sep-92	4 years	Granted	28-Sep-17
EPM 19109	22	29-May-14	2 years	Granted	28-May-18
EPM 16050	94.2	19-Jun-08	5 years	Granted	18-Jun-18
EPM 14028	188.4	10-Jun-04	5 years	Granted	09-Jun-18
EPM 25773	170	09-Jul-15	5 years	Granted	08-Jul-20

Figure 4-1. Regional Map Showing Wolfram Camp Project Location

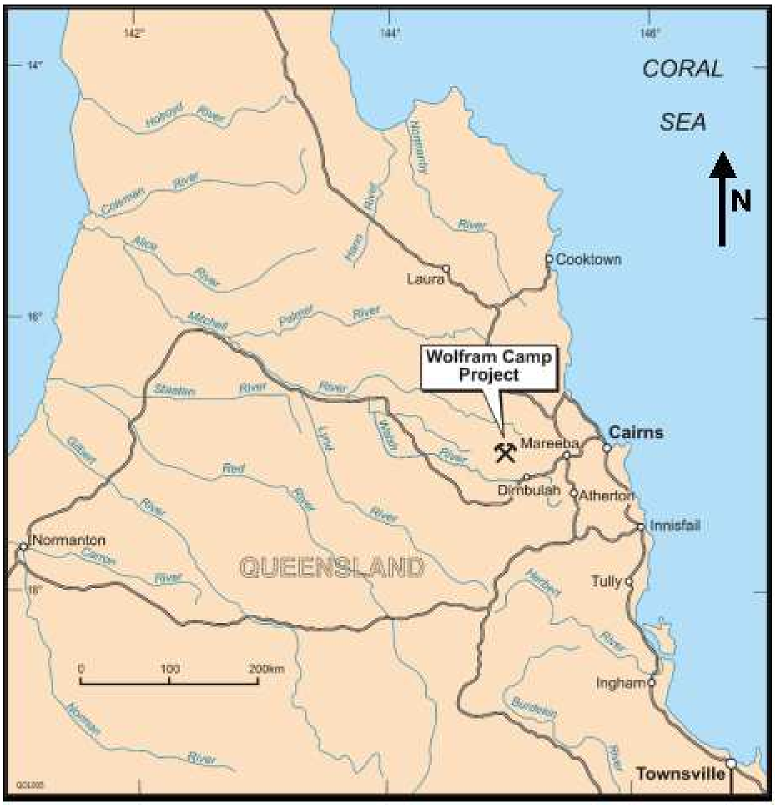


Figure 4-2. Wolfram Camp Location Plan

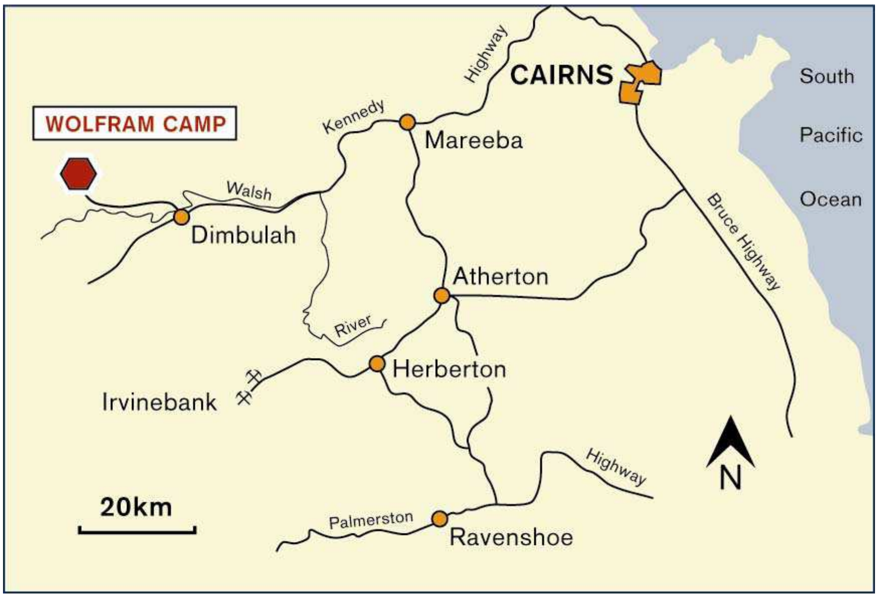
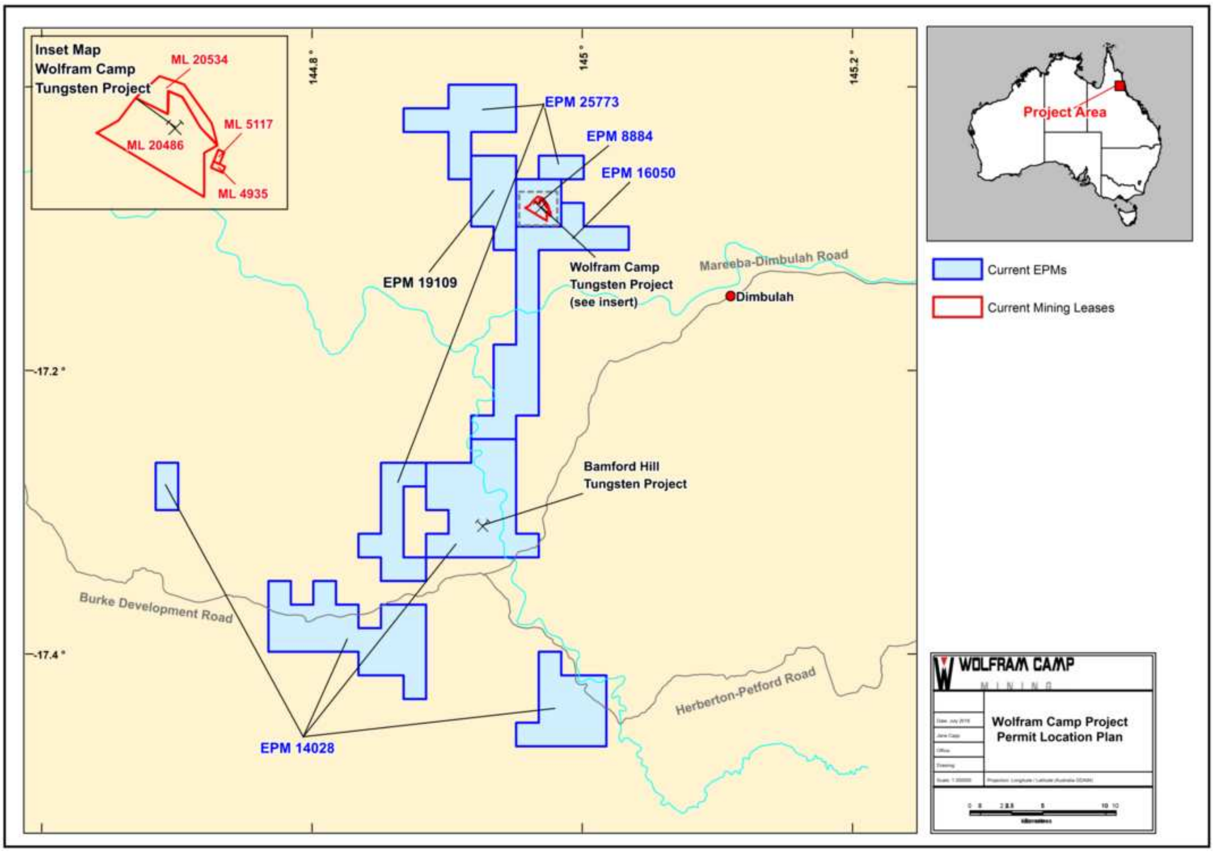


Figure 4-3. Wolfram Camp Project – Mining Leases and Exploration Permits



5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, PHYSIOGRAPHY

The Wolfram Camp tungsten-molybdenum-bismuth deposit occurs to the west of the Great Dividing Range, in the headwaters of the Walsh River which flows westwards, eventually reaching the Gulf of Carpentaria. The project lies in an area of moderate topography at an elevation of about 700m. The western portion of the deposit is cut by Bullaburrah Creek which flows south-westwards across the line of mineralisation and then turns south before turning south-eastwards to flow into the Walsh River, some 18km to the west of Dimbulah.

The undulating hills support ironbark and bloodwood dominated open woodland with a low native grass ground cover. A large proportion of the area is significantly degraded by previous mining and these areas are characterised by large populations of exotic weed species and relatively short lived coloniser species such as acacias. Significant areas of remnant vegetation cover are confined to the margins of the project area. The remnant flora is, however, quite diverse and variable with some areas displaying a well-developed understory. A large number of eucalypt species are present including species which are commonly found in the Mareeba-Dimbulah area, as well as several species which are normally encountered in areas much further to the west and north

The nearest town is Dimbulah, some 18km from the project, which has a population of around 1,000. The town supports two schools and has modern social and sporting facilities and lies on the main Burke Development Road connecting Mareeba to Chillagoe and on the Mareeba-Almaden-Forsayth railway line. The sections of highway and road between Mareeba-Dimbulah-Wolfram Camp are allweather, except for high flood levels during "the wet", when access to Wolfram Camp can be cut by the Walsh River and Bulluburrah Creek.

From Dimbulah a good surfaced road, "Wolfram Camp Road", reaches to within about 10km of the project site from which a well maintained dirt road continues to the site of the old Wolfram Camp township. From there access is *via* a gazetted track and then mine tracks, both of which were upgraded by Queensland Ores Ltd ("QOL") to allow all-year access to site.

Average rainfall is generally 75-100cms per year, with most falling during the annual wet season from December to March. Average annual evaporation rates are approximately 1600mm, therefore the site has a negative annual water balance.

The mean daily temperatures range from about 20^{o}C in winter to 30^{o}C in summer. Figures reported by the Bureau of Meteorology covering the period 1931 to 2004 for Dimbulah are shown in Table 5-1.

Table 5-1. Mean Daily Temperatures

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Mean Max Temp (°C)	34.0	32.9	32.5	31.6	29.5	27.8	27.4	28.8	30.9	33.3	34.2	35.0
Mean Min Temp (°C)	21,7	21.4	19.9	18.0	14.5	12.6	11.1	10.5	13.3	16.7	19.8	20.9

The prevailing wind direction is from the southeast with an average speed of 25km/hr.

The Walsh River valley supports intensive farming operations which have suffered severely since the banning of tobacco growing. Current crops include sugar cane, specialist fruits and nuts, ti-tree for oil, and other produce. Cattle stations surround the river plains and dominate the better quality high country. The nearest cropping and grazing activity to Wolfram Camp lies 3km to the south.

Power was established to site during the operations in the 1980s with a 22 kV line run from the nearest existing line which runs along the access road past the old Wolfram Camp town site, approximately 1.5km from the site of the more recent drilling.

6 PROJECT HISTORY

6.1 Introduction

The Wolfram Camp mineralisation was discovered in 1894 and previous mining operations have been based either on surface eluvial mining of residual wolframite grains or on the underground extraction of high-grade pipes of erratic shape and lateral dimensions. These pipes have ranged from less than 1m in diameter to 15m by 10m in plan, and have down-plunge lengths often exceeding 100m. The pipes comprise predominantly glassy white quartz with shoots containing coarse bungs of wolframite and molybdenite and occur within greisen-style alteration zones within a Carboniferous granodiorite near the intrusive contact with Devonian sediments and Carboniferous volcanics.

The hard rock mines of the Wolfram Camp mineral field have recorded combined production of at least 10,000t of wolframite, molybdenite, bismuth and mixed concentrates. Eluvial and early hard rock production is poorly recorded. The main periods of hard rock mining were 1908-1920, 1967-1972 and 1978-1982. A summary of historical activities at WCM are summarised in Table 6-1.

Table 6-1. Summary of WCM History

Period	Description
1894 - 1903	First operated from small separate mines
1903 - 1917	The Irvinebank Company - plant constructed for toll treatment. Many more mines developed, for both wolframite and molybdenite.
1917 - 1920	The Thermo Electric Ore Reduction Corporation Limited. Much large mines equipped.
1921 - 1967	Limited operations with adverse market conditions
1967 - 1972	Metals Exploration Limited. Leisner levels develped, plant re-established at Whiskey Creek. Production from some high grade pipes. Diamond drilling.
1972 - 1991	Mount Arthur Molybdenum Limited, further development and production. 8,000t mined from 1975-1981. Underground face sampling.
1992 - 1994	Great Northern Mining Corporation, limited work on site.
1994 - 1996	Allegiance Mining used option to carry out drilling programmes
	TMPL
2005	QOL diamond drilling
2008	PML diamond drilling
2011 -2013	Deutsche Rohstoff AG start open pit mining operations with refrubished plant.
2014-2015	Almonty take over WCM and continue open pit production
2016 - present	Mill enhancements

6.2 QOL1992-Present

6.2.1 Overview

Great Northern Mining Corporation NL (GNMC) acquired the project in 1992 but carried out only minimal work on the site. This included analysing the 138 fired-face samples taken previously by TOMA.

During 1994 and 1995 Allegiance Mining NL entered into an option agreement over the project and drilled 37 reverse circulation holes (1,726m), mainly to test between the former Lanski and Leisner mines (Alistair Barton and Associates, 1996). Due to the topography in this area and the limitations of the rig available at that time, the majority of these holes were not located in the optimum sites. As a consequence, potential targets still remain to be tested in this area. The programme also suffered from poor drilling recoveries. Some interesting results were achieved however, with one hole returning 9m at 0.61%WO_3 and 0.05%Mo.

Additional holes were drilled to the east of the Lane Decline development in the vicinity of the former Harp of Erin workings and returned encouraging intervals in a number of holes including 11m at 1.44%WO_3 and 0.74%Mo, and 5m at 2.28%WO_3. The Harp of Erin workings were based on a 5m diameter pipe recorded as being relatively rich.

Allegiance also undertook a bulk sampling programme on the tailings (Alistair Barton and Associates, 1996), with a particular emphasis on the potential to recover molybdenite which had not generally been recovered in the processing circuit. A total of around 1,000t of tailings was screened but the proposed treatment through a mobile process plant was never completed.

Allegiance's internal report on this work quotes tailings resources of around 57,000t but at grades which can only be considered to be unrealistic. Allegiance could not raise further funds and withdrew from the project.

GNMC sold the project to TMPL in 1998, since which time TMPL has researched and collated historical data. In 2002, a privately-owned company, Eclectic Investments Pty Limited (Eclectic) entered into an option to purchase the project from TMPL. Eclectic completed surface surveying, gridding in the Lane Decline to Brunjes Mine area, and underground mapping along the Lane to German Bill decline. Eclectic withdrew from the project due to depressed metal prices. All work, following the departure of TOMA up to Queensland Ores Ltd's (QOL) involvement, was undertaken using the TOMA survey grid.

All work, following the departure of TOMA up to QOL's involvement, was undertaken using the TOMA survey grid. Wolfram Camp Mining Pty Limited (WCM), then a wholly owned subsidiary of Queensland Ores Ltd (QOL), entered into a Farm-In Agreement with TMPL on June 2004 to earn an 85% interest in the project.

During the period June to December 2005, QOL drilled 36 diamond drillholes in the Wolfram Camp Exploration Permit for Minerals (EPM) 8884 for a total of 2437.8m. The drilling was carried out by Zen Drilling International Pty Limited, a subsidiary of Radial Drilling, using a Longyear 38 rig. Holes were collared in PQ (providing a nominal 85mm core) and reduced to HQ (providing a nominal 63.5mm core) once ground conditions were considered suitable. The majority of the diamond holes had at least one internal survey taken. All core has been photographed.

QOL's diamond drilling provided continuous core samples with very little core loss. In the main, QOL sampled its diamond holes on geological or mineral boundaries such that most intersections sent for assay were less than 1m, e.g.:-

D5 returned 0.6 m at 9.58% WO_3,
D8 returned 0.72 m at 10.01% WO_3 and 0.57 m at 28.18% WO_3,
D20 returned 0.6 m at 2.79% WO_3 and 2.63% MoS_2,
D23 returned 0.65 m at 1.21% WO_3 and 4.02% MoS_2,
D24 returned 0.4 m at 6.68% WO_3,
D29 returned 0.36 m at 6.08% WO_3, 0.24m at 8.33% MoS_2, 0.35 m at 1.30% WO_3 and 10.48% MoS_2, and 0.4 m at 5.76% WO_3,
D30 returned 0.35 m at 10.53% MoS_2,
D33 returned 0.61 m at 11.15% WO_3 and 2.95% MoS_2,
D34 returned 0.61 m at 9.21% WO_3,
D36 returned 0.63 m at 7.32% WO_3, and 0.52m at 4.98% WO_3

A number of holes intersected former underground workings. In order to continue these holes below these former workings, the hole were reamed down in PQ and then drilled at HQ through the PQ rods. This process was time-consuming and expensive but allowed access to the footwall of some of these former workings and therefore provided useful information.

In December 2005, 15 reverse circulation (RC) holes were drilled for a total of 939m using a 4.5" (114mm) bit. Between April and December 2006 QOL used the same Drill North rig to complete a further 112 RC holes totalling 5,357m, with 110 of these in the area of immediate interest and two in the Mulligan-McIntyre area. The RC drilling provided excellent recovery due to the high quality of the equipment and the competence of the drilling crews used in the operation.

Where former workings were intersected, efforts were made to extend the hole but in some cases this was not possible and the hole was abandoned. Three RC holes were abandoned when old workings were intersected, in order to minimise potential drilling problems, and a further three failed to penetrate thick mine fill dumped by previous operators in the vicinity of the Victory shaft.

All holes drilled by QOL were surveyed by Charles O'Neill Pty Limited, licensed consulting surveyors based in Cairns. To provide an accurate base for its work QOL commissioned the flying of aerial photography and orthophoto based topographical mapping in November 2005. In order to take account of track development since that date QOL commissioned a new set of data with the flying being undertaken in August 2007.

In both cases flying was undertaken by United Photos and Graphic Services Pty Limited of Blackburn, Victoria, a Member Firm of the Association of Aerial Surveyors Australia Inc. Ground control was established by Charles O'Neill Pty Limited, surveyors from Cairns. Topographical maps were produced by Survey Graphics, mapping consultants of Perth, WA.

The topography was recorded at 1m intervals and provided an excellent base for all requirements. All QOL boreholes were picked up by consulting surveyors Charles O'Neill Pty Limited.

QOL, through its wholly owned subsidiary WCM, carried out nearly 10,000m of diamond and reverse circulation drilling and identified the potential for the relatively lower grade halo mineralisation around the previously mined high grade pipes to host economically viable material. Sufficient funds were raised to build a 150,000tpa processing plant which comprised a combination of flotation and gravity techniques. On-site construction work commenced in November 2007 when the mining leases were granted and the plant was handed over by the contractor in July 2008.

The process plant operated intermittently for less than three months but unfortunately, a combination of technical difficulties and a shortage of working capital, compounded by the GFC, resulted in a suspension of operations in November 2008. At this time QOL received financial support from Metallica Minerals Ltd (Metallica), with Metallica ending up with a 75% stake in QOL. QOL was subsequently renamed Planet Metals.

PML drilled 200 holes comprising 45 DD holes (WCD-037 TO WCD-081) totalling 2,269m and 155 RC holes (BP-001 to BP-104 and WCRC-139 TO WCRC-148) totalling 2,571m at the Wolfram Camp minesite between September 2009 and February 2010. The aim of the programme was to infill areas in the existing resource model of QOL, where there was a paucity of drill data and to provide additional geological information for an updated resource model. The holes were also drilled to provide additional geological and bulk density data for the Wolfram Camp, WO_3 + Mo +Bi deposit.

Assay results confirmed the company's previous understanding of the geology whereby mineralisation is mainly confined to quartz greisens with high grade zones occurring in quartz pipes. This style of mineralisation is very difficult to quantify, hence much of the RC drilling within the proposed pit was completed on a 10m by 10m spacing. The majority of the holes were drilled easterly at 50° to 60°.

As a result of the mineralisation often occurring as blebs, some very high grade zones were identified by the drilling. Best results included:

2m @ 26.3% WO_3 and 5.5% Mo from 2m (Hole BP-059

1m @ 16.7% WO_3 and 2.2% Mo from 27m (Hole WCD-044)

2m @ 3.4% WO_3 and 0.1% Mo from 13m (Hole WCD-058)

3m @ 2.9% WO_3 and 0.04% Mo from 5m (Hole BP-018)

A total of 10 RC holes (WCRC-139 to WCRC-148) were drilled outside the current pit boundary with eight of these holes drilled south of the current pit near the old Mulligan and MacIntyre mines. Two holes, RC-141 and RC-143 intersected significant widths of molybdenum mineralisation associated with quartz greisens adjacent to the old mine workings with RC-141 intersecting 7m @ 0.39% Mo from 66m and WCRC-143 intersecting 3m @ 0.69% Mo from 33m. These results indicate that there is the potential to identify additional resources outside the current pit boundaries.

Collars of holes drilled by PML were surveyed, using a differential GPS, by Charles O'Neil Pty Ltd, who were also responsible for surveying the historical holes drilled by QOL.

Due to the prevailing economic climate and poor ore reconciliation between the feasibility resource estimate and mine production the mine was placed on care and maintenance by Planet Metals in 2008. In 2009 Metallica Minerals acquired a majority share in Planet Metals (then QOL) and provided ongoing capital for evaluating the potential of the mine. Following the substantial infill drilling programme in 2009 – 2010, Golder Associates Pty Ltd. ("Golder") was requested by Planet Metals to provide an updated resource estimate for the Wolfram Camp W-Mo mine suitable for public reporting.

In May 2011, Deutsche Rohstoff AG acquired Wolfram Camp Mining Pty Ltd from Planet Metals and subsequently commenced geotechnical investigations and mine planning together with plant refurbishment. In May 2011, Deutsche Rohstoff acquired 100 percent of the Wolfram Camp Mining Pty Ltd (WCM), which held 85% of the decommissioned Wolfram Camp mine. In September and December 2011, DRAG acquired the outstanding 15% through the purchase of the Tropical Metals Pty Ltd, who owned a large exploration tenement holding which included the nearby tungsten deposit at Bamford Hill (as shown in Figure 9-7).

After the takeover by DRAG in May 2011, the target was to commence mining production as rapidly as possible. By autumn 2011, the processing plant and tailings storage facility were refurbished and repaired in preparation for the start of operations. In October 2011, DRAG entered into an offtake agreement with Global Tungsten & Powders (GTP), an American company belonging to the Austrian Plansee Group. Commissioning of the plant commenced in December the same year with production ramped up throughout 2012.

6.2.2 QOL – Diamond Drill Core Sampling

All core was transferred directly from the core barrel to correctly sized aluminium core trays at the rig site. Wooden core blocks were placed in the trays to record downhole depths at the end of each drill run. At intervals the core trays were carefully transported to a centralised core handling area.

Here the core was geologically logged by either company geological staff or experienced geological consultants. Alpha angles were measured throughout of any contacts or major discontinuities in the core, and where successful core orientation was achieved, beta angles were also measured. Basic geotechnical logging was carried out with Rock Quality Designation (RQD) factors calculated for all core.

Logging of the core enabled mineralised portions of the holes to be selected for assay. These selected samples were sawn such that one quarter core was sent to the laboratory. Sample intervals were selected on geological criteria, with the maximum sample length (other than two samples) of one metre. A sample collection method was introduced whereby the same progressive quarter core was selected for all intervals, irrespective of the distribution of mineralisation within the whole core,to eradicate any sampling bias.

The selected quarter cores were collected in calico bags over the designated interval, with sample number tags inserted with the sample and the sample number written on the bag. The calico bags were collected in larger polywoven bags on which the contained sample numbers were written. These polyweave bags were addressed to the laboratory and were sent to Mareeba Transport in batches for transport to the laboratory.

All core was stored in trays stacked under cover in a shed at QOL's house in Dimbulah.

Samples from QOL's diamond holes were transported to ALS Chemex's laboratory in Townsville where sample preparation was carried out. All samples were weighed, dried and crushed (two passes) to a nominal 6mm.

Samples containing coarse molybdenite which had been identified by QOL were spread on to a plastic mat and the coarse molybdenite was hand picked, weighed and bagged (Figure 6-1). The remainder of these samples, and the whole of the other samples, were individually pulverised to 85% passing 75microns. A 300gm extract from each sample was sent to ALS Chemex's Brisbane laboratory and analysed using XRF for Mo, W, Bi, As and Sn as requested by QOL.

With the coarse molybdenite samples, the weight of the hand-picked molybdenite was converted to Mo (multiplied by 0.5994), and this weight of Mo was divided by the original weight of the sample times 100 to establish the percentage Mo, which when added to the XRF result provided the total Mo content of the sample.

Figure 6-1. High Grade Molybdenite Intersection
(typical of those requiring hand picking during analysis)

ALS Chemex was selected as the laboratory to undertake the analyses of all samples produced by QOL from the Wolfram Camp project. ALS Chemex has had a long involvement in the project having worked with TOMA in the 1980s during which time it devised a systematic analytical process to handle the unusual mineralisation distribution present at Wolfram Camp. In addition, QOL personnel had a long term and positive relationship with that company.

6.2.3 QOL – Reverse Circulation Sampling

The RC samples were collected in plastic sacks at 1m intervals *via* a cyclone. All samples other than sediments were split using a Jones Riffle Splitter to produce a +/-2kg sample for analysis. This sample was collected directly in pre-numbered calico bags. A matching sample number tag was inserted in each bag which was then tied. Chips were logged for each metre at the drill rig with the logs recorded manually and later transferred to computer format.

Estimations of mineral content were made using small panned concentrates and an in-house classification built up early in the programme based on experience in the project. When assay results were obtained they were checked against estimates to ensure accuracy. Visual estimation of wolframite proved very efficient whereas visual estimation of molybdenite tended to exaggerate the expected grade. However, the order of magnitude of the estimate proved to be an excellent tool. A sub-sample of each metre was collected and placed in numbered plastic chip trays which were stored at QOL's house in Dimbulah.

The assay samples were put into polyweave sacks on which the contained sample numbers were written. These polyweave bags were sealed by tape and packed to ensure that samples could not be damaged in transit. The bags were addressed to the laboratory and sent to Mareeba Transport by QOL personnel in batches for transport to the laboratory.

The remainder of each sample was stored in numbered plastic sacks pending the receipt of assay results. If no unexpected or anomalous results were received, the samples were subsequently destroyed.

RC samples were sent directly to ALS-Chemex's laboratory in Brisbane for preparation and XRF analysis for Mo, W, Bi, As and Sn using the same preparation as per the diamond core samples.

As with all analyses, ALS Chemex carried out routine internal checks on the assays from QOL's Wolfram Camp samples.

6.2.4 QOL –Check Analyses Diamond Drill Samples

Historical evidence and a visit to the mineralised pillar in the 1.1 Stope in the Lane Decline workings clearly indicated the high nugget nature of the mineralisation at Wolfram Camp. It was for this reason that the close spaced drilling pattern of roughly 20m by 20m was selected as the best way to provide sufficient coverage such that, when the controls on mineralisation were better understood, estimates of resources compliant with JORC guidelines would be achievable.

The extent of the high nugget effect, and the need for the establishment of a systematic sample collection methodology for the diamond drill core, was highlighted by a programme of re-sampling initiated as part of early metallurgical testwork undertaken by Lycopodium Engineering Pty Limited and Ammtec Limited.

Assay results from a set of original samples and composites (ALS, Brisbane) were significantly different to those returned by the adjacent quarter cores over the same intervals (Ultratrace, Perth).

When pulps from the samples originally assayed at Ultratrace were subsequently tested at ALS, an excellent correlation in assay values was returned, indicating that the difference occurred in the samples rather than the assay laboratory or analytical technique used.

6.2.5 QOL –Check Analyses Reverse Circulation Samples

RC samples were sent directly to ALS-Chemex's laboratory in Brisbane for preparation and XRF analysis for Mo, W, Bi, As and Sn using the same preparation as per the diamond core samples. ALS Chemex carried out routine internal checks on the assays from QOL‟s Wolfram Camp samples.

Duplicates

Following the receipt of assay results from the first 49 RC holes, some discrepancies were noted between Mo assay results and visual estimates of molybdenite content. As a result, eighteen RC samples carrying high visible molybdenite content were manually re-split through a riffle-splitter. The resplit samples were sent to Townsville to undergo the same hand-picking process used with selected intervals of diamond core.

Results showed less variation in the Mo values than expected, although of the 18 samples, 16 did show a minor increase in Mo grade. However, repeatability of the W grade proved far more erratic with 15 samples showing increased W grade (including one from 0.095% to 2.92%) whilst 3 showed minor decreases in grade.

Blanks and Standard Samples

A programme involving the insertion of blanks and assay standard samples was initiated as a check on the laboratory analyses. Blanks were inserted in sample batches as samples with numbers ending in 00 and 50. Standards were inserted as samples ending in 25 and 75. Standards for this programme were acquired from CANMET Mining and Mineral Science Laboratories, Ottawa, Canada, and from CDN Resources Laboratories Limited, British Columbia, Canada.

Analyses of these samples showed excellent quality control in the laboratory, again indicating all variations are due to the nugget effect in the mineralisation.

At the completion of the RC drilling programme, 95 intervals were randomly selected for re-splitting. Again, this was achieved manually through a riffle splitter, with the split collected in a numbered calico bag and a sample number added to each bag. Of the 95 samples 42 (44%) showed an increase in Mo content in the second sample (with 12 showing variation of less than 5%) and 35 (37%) showed an increase in W content in the second sample (with 18 showing variation of less than 5%).

L Davis of Veronica Webster Pty Ltd ("VWPL"), who prepared the due diligence report, has had sight of copies of laboratory returns. All the laboratories used are National Association of Testing Authorities Australia (NATA) registered and have internal checking procedures.

There is no evidence to suggest that sample and assay data have not been acquired in accordance with acceptable industry standards.

Pulp Checks

QOL re-assayed a number of pulps from their original samples (which were assayed at Ultratrace) at ALS in Townsville and an excellent correlation between the two sets of assays was returned.

6.2.6 QOL – Bulk Density Measurements

During the core drilling programme QOL carried out 108 measurements of bulk density. These measurements were taken on air-dried samples. Of these, 61 samples were from material classified as waste and 47 were from material classified as ore.

The length-weighted average bulk density for the waste samples was 2.68, and for the ore the average was 2.81. However, none of the ore samples was representative of the more vuggy variety of host rock and neither were any highly mineralised samples tested. The actual range of bulk density values within the orebody is wide, ranging from, for example, 1.5 in extremely vuggy quartz pipes to 5 or 6 in massive mineralisation. As a large proportion of the current resources occur within the massive quartz greisen, and to maintain conservatism, QOL incorporated a figure of 2.7 in its evaluation.

6.2.7 Planet Metals Ltd – Diamond Drill Core Sampling

PML assigned sample numbers on the basis of the hole number and depths e.g. a sample from hole WCD-052 taken between 23 and 24m was given the sample number WCD052_23_24.

The HQ core samples were put into core trays and transported from the drill site to the sample preparation shed where they were marked up and logged. A geological and geotechnical log was completed for each of the holes. The geological logging system used by PML was similar to that used by the previous owners of the Wolfram Mine tenements.

In addition to logging the core, one sample was collected from each tray for bulk density analysis. Bulk density determinations were made by cutting the core into cylinders, measuring the length and diameter of the core with callipers then weighing the core cylinder. Core sampling was based on ½ core sampling, with limited selective sampling; as a consequence of the very spotty nature of the wolfram and molybdenum mineralisation the core was cut in such a way as to bisect the mineralisation with ideally equal portions being present on each half of the core

After geological logging, selected sections of core were cut in half and sent to ALS for analysis. Where large blebs of molybdenum or wolfram were evident in the core an attempt was made to cut through the bleb to ensure that the sample accurately reflected the mineralisation within the core. As a rule all intercepts of quartz greisen and quartz pipe material were cut as these two rock types contain the bulk of the high grade mineralisation, and in some circumstances sticks of core comprising quartz greisen appeared to be barren but when cut in half revealed blebs of wolfram in the centre.

All samples were assayed for W, Mo, Bi, As and Sn using the ME-XRF05 method; where samples exceeded the detection limits for that method, they were re-assayed using the ME-XRF15c method.

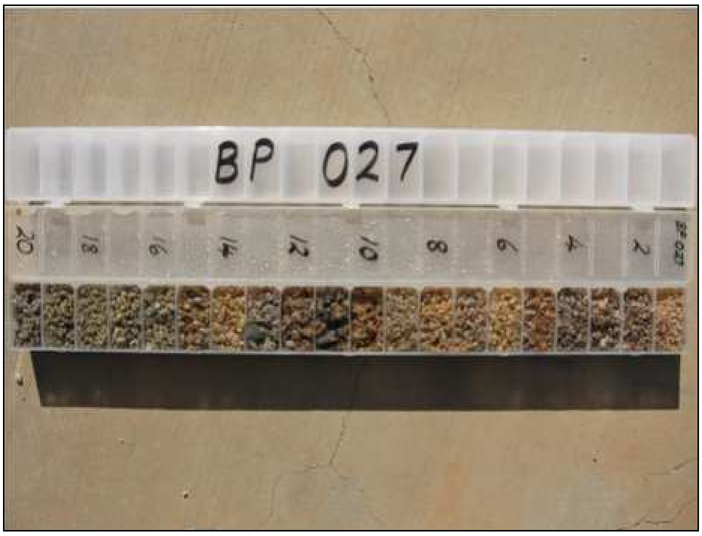
6.2.8 Planet Metals Ltd – Reverse Circulation Sampling

PML used an 87.5:12.5 riffle splitter attached to the base of the cyclone and a 2-3kg sample was collected in a calico bag beneath the 12.5 chute; the remainder of the sample was collected in a plastic bag and left on site pending receipt of analytical results.

Sample recovery in the mineralised zone is believed to be high for PML drill holes. Drill holes were sampled predominantly over 1m intervals. A sample from each one metre interval was put in a numbered chip tray, photographed and logged.

The RC samples collected by QOL were taken via a cyclone into plastic sacks at 1m intervals. All samples other than sediments were split using a Jones Riffle Splitter to produce a +/-2kg sample for analysis. This sample was collected directly in pre-numbered calico bags. A matching sample number tag was inserted in each bag which was then tied up. Planet Metals had an 87½ : 12½ riffle splitter attached to the base of the cyclone and a 2- 3kg sample was collected in a calico bag beneath the 12½ chute, the remainder of the sample was collected in a plastic bag and left on site pending assay results. A sample from each 1m interval was put in a numbered chip tray which was then photographed, (Figure 6-2), and each 1m sample was logged.

Figure 6-2. Numbered Chip Tray with 1m Samples



PML discontinued the practice of handpicking coarse grained molybdenite devised by QOL. W and Mo assaying was by ALS method ME-XRF05, with higher grade samples analysed by ALS method ME-XRF15c, which uses a lithium borate flux to produce a fused glass disc. ALS considered method ME-XRF15c to be more accurate than ME-XRF07; however, it has only been available since early 2009. ALS stated that due to the hardness of common tungsten minerals, in most cases higher concentrations of tungsten may cause bias in the order of 10-15% on the low side by method ME-XRF05. The fusion method ME-XRF15c does not suffer from these mineralogical effects.

6.2.9 Planet Metals Ltd – Check Analysis

The QAQC results for drilling indicated that the assays for the PML drilling programme were satisfactory for resource estimation purposes.

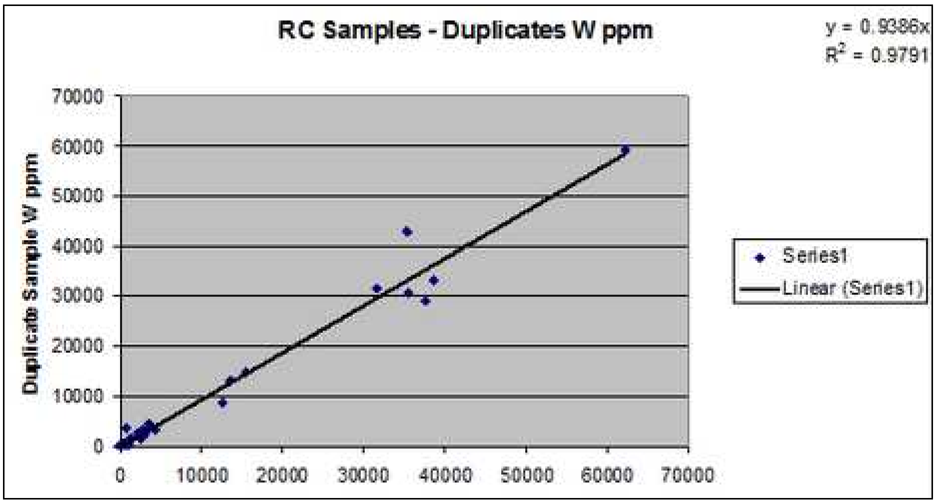
6.2.9.1 Duplicate Sampling

The duplicate assay results were analysed by Golder (A. Richmond) in 2010. A duplicate sampling programme was completed on RC samples and half core samples collected by PML and also on the quarter core samples collected by QOL. With respect to the RC samples, PML on receiving the assay results, selected samples which contained varying grades of wolfram and molybdenum mineralisation and re-sampled the same interval by re-splitting the portion of the original sample which was collected in plastic bags and left next to the drill hole. These samples were given the same sample number and submitted to the laboratory to be assayed by the same method as the original sample.

With the diamond core samples the other half of the core from the original sample was assayed (after being photographed) and re-assayed. As QOL had not undertaken any systematic duplicate sampling PML collected the other quarter core of samples taken by QOL and submitted them for assay.

Once the results were received, a regression analysis was completed on each data set for Tungsten (W), Molybdenum (Mo) and Bismuth (Bi) and a series of graphs were plotted for each element. The graphs for tungsten duplicates are shown in Figure 6-3 to Figure 6-5.

Figure 6-3. Comparison of RC Duplicate Samples



Results from the duplicate samples indicated that the quarter core samples collected by QOL showed large variations between each quarter of core, and the half core samples collected by PML show some variation, but not as great as the quarter core samples. QOL became aware of this issue when they checked a series of quarter core samples stating in their resource document that:- "*Assay results from a set of original samples and composites (ALS, Brisbane) were significantly different to those returned by the adjacent quarter cores over the same intervals (Ultratrace, Perth)*".

Figure 6-4. Comparison of ½ Core Diamond Drill Samples

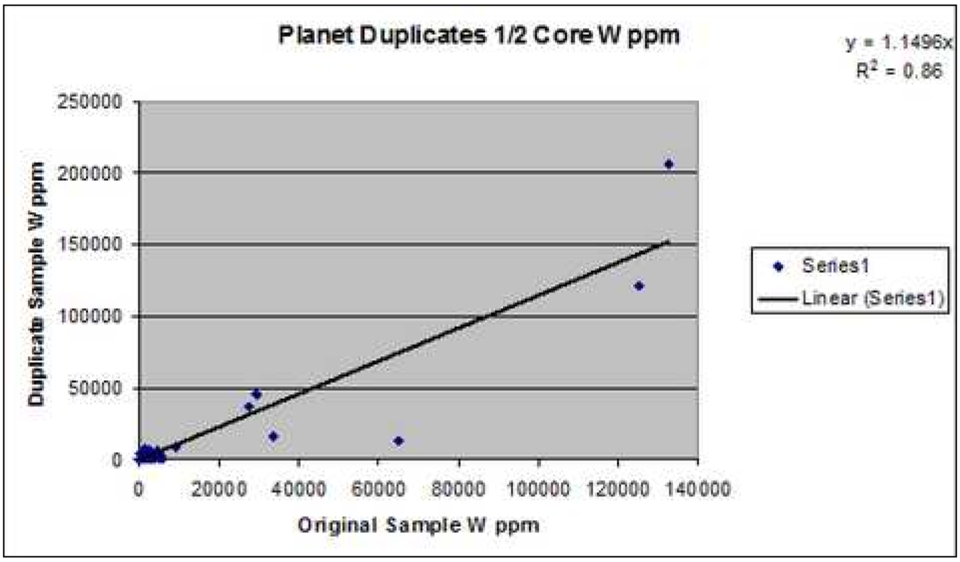
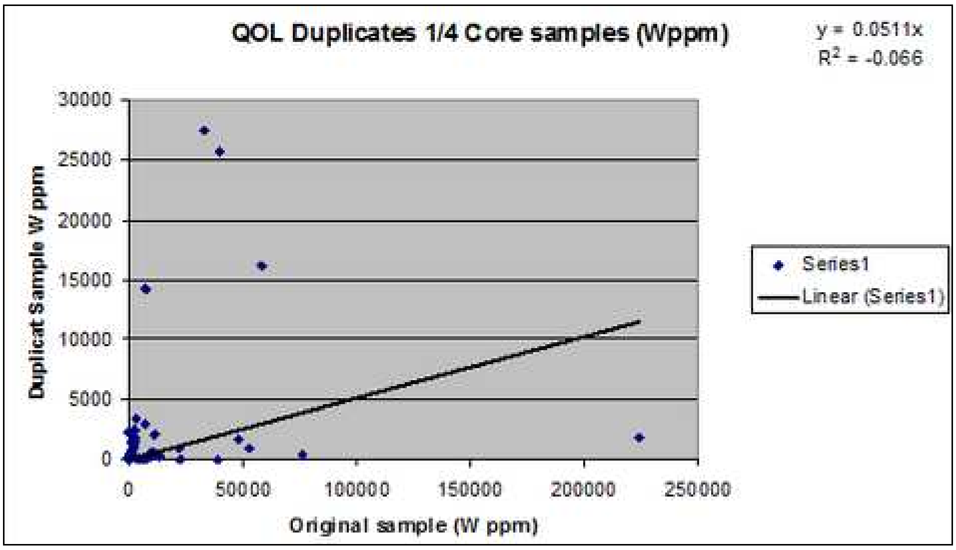


Figure 6-5. Comparison of ¼ Core Diamond Drill Samples



The data also revealed that the RC duplicates showed the greatest degree of correlation indicating that RC samples better reflected the actual grade in the ground as they have been homogenised and no sampling bias was introduced. Based on these results RC drilling would appear to provide the most representative samples for the mineralisation at Wolfram Camp. If diamond drilling is used then whole core samples should be taken to provide the best sample, thus avoiding any sample bias which is evident with using half or quarter core samples

The majority of the available sample data in the database are from RC drilling and it was concluded that they can be used with confidence; the half core and quarter core samples should be used with a lower degree of confidence. Much of the lack of precision in core arises from errors in cutting core, whilst preparation of a smaller initial sample size increases the nugget effect.

6.2.9.2 Standards

Planet Metals also inserted a series of standards obtained from Geostats Ltd in Perth into sample batches for both the RC and diamond drilling. Three standards comprising two molybdenum standards, GMO_01 and GMO_03 and one tungsten standard GW_01 were used.

In general the tungsten assays and the tungsten standard showed very strong correlation with a low percentage variance of between 0.23 and 2.37%, the variance for molybdenum was higher i.e. between 7.36 and 17.98%; in all cases the ALS sample was consistently lower than the standard. It is possible that molybdenum values have been underestimated for the samples submitted by Planet Metals Ltd.

6.2.10 Planet Metals Ltd – Density Measurements

After logging the core, PML collected one sample from each tray for dry bulk density measurements on air dried core samples. The methodology used by Planet Metals involved selecting a piece of core from each core tray, cutting it into a cylinder of at least 10cm in length, and using callipers to measure the length and diameter of the core (Figure 6-6). From these measurements the volume of each cylinder could be calculated. The core cylinders were weighed and a simple mass divided by volume calculation was completed to obtain bulk density information. Samples were taken from the different rock types and an average bulk density obtained for each rock type was estimated, as summarised in Table 6-2.

Figure 6-6. Measurement of Sample for Bulk Density Determination

Table 6-2. Bulk Density Measurements

Rock Type	Number of Samples	Average Dry g/cc
Decomposed Granite	4	2.65
Unaltered Granite	113	2.71
Altered Granite	182	2.74
Mica Greisen	176	2.85
Quartz Greisen	48	2.87
Quartz Pipe	3	2.52
Sediment	21	3.08

Dry bulk densities were assigned to blocks based on IK estimates of the proportion of each block belonging to one of four main lithology groups (granites, mica greisens, quartz greisens, quartz lode).
Bulk density values applied for each lithology group represented the average of an appropriate number of samples.

6.3 Historical Resource Estimates

6.3.1 Queensland Ores Ltd (QOL) (2007)

QOL believed that the density of drilling and the (assumed) good continuity of the interpretation allowed resources to be estimated into the *Measured* and *Indicated* categories under the guidelines established in the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the JORC Code, 2004 Edition)

A total of 160 drill holes completed by QOL (QOLWCD01-36 and QOLWCRC01-125 excluding hole 87), 3 holes drilled by TOMA (WDDH03-05) and 13 holes drilled by Allegiance (WC01 and 07-18) were accessed in the interpretation. Of the 176 drillholes used in the interpretation, 39 were diamond holes (36 by QOL and 3 by TOMA), and the remaining 137 were RC holes (124 by QOL and 13 by Allegiance). In reporting resources, a bottom cut of 0.1%WO_3-equivalent (WO_{3Eq}) was used to take account of likely operating costs.

Resource polygons were constructed on a local QOL grid on 10m E-W sections over 640m and subsequent interpretation established 37 lenses. The lenses were defined as polygons that were 'snapped' on to drillholes to ensure accuracy and end plates were placed approximately 5m from the last drillhole. Volumes for each of the lens polygons were calculated using QOL's Vulcan computer software. Tonnages were then calculated by incorporating a bulk density factor of 2.7 for mineralisation and all lithologies, as determined by QOL testwork on samples from a comprehensive programme.

A frequency distribution curve was plotted and indicated a case for a top-cut at 5.75%WO_{3Eq}. Incorporating a top-cut at this level would have required the cutting of 1.9% of the total assay results to the 5.75%WO_{3Eq} value. Rather than use this top-cut value it was decided that a more conservative option would be to utilise a 97.5 percentile top-cut. The 97.5 percentile for %WO_{3Eq} was therefore calculated from the 1016 length weighted samples available. The twenty fourth highest grade was 4.845%WO_{3Eq} and this was incorporated as the top-cut for %WO_{3Eq}.

During the grade estimation process, the distance between the block and the closest sample was recorded. This value was used to calculate a field in the block which flagged each block with a resource category – *Measured, Indicated, or Inferred*. The categories were defined as follows:

- *Measured* Resources – 0-15m to the closest sample point
- *Indicated* Resources – 15-25m to the closest sample point
- *Inferred* Resources – >25m to closest sample point

To take account of material within the mineralised lenses which had been removed by previous mining operations, the proportion of the total downhole intersections within mineralised lenses represented by voids in the drilling was calculated. Within the *Measured* and *Indicated* Resources, of a total of 834.38m of mineralised lens intersected in drilling, 82.26m or 9.26% represented voids. When the total *Measured* and *Indicated* Resource figures were estimated an amount of 9.26% was therefore removed to represent former workings in these zones and similarly 2.14% was removed from the *Inferred* Resource figure.

- ***Measured plus Indicated Resources*** ***709,706t at 0.42% WO_3 and 0.17% MoS_2***
- ***Inferred Resources*** ***238,324t at 0.4% WO_3 and 0.2% MoS_2.***

Davis (2011 Due Diligence) considered that the method used by QOL could not accurately define the actual position of mineralisation, rather it supplied a global figure for tonnes and grade within the hard zones (here, the mineralised lenses) of the block model. The primary nugget effect introduced extreme variability throughout the overall deposit.

Mine planning and scheduling was progressed by Coffey Mining. By December 2007 there had been significant movement in the commodity prices such that the WO_{3Eq} factor of MoS_2 had increased to 2.84. More hard data were available regarding operating costs and a break-even figure of 0.36% WO_{3Eq} was established as the operational bottom cut-off grade. An in-house re-evaluation of the resources quoted above using the new WO_3-Equivalence and the new bottom cut-off grade resulted in the following figures (rounded):-

- ***Measured*** ***351,900 t at 0.79% WO_3 and 0.26% MoS_2***
- ***Indicated*** ***65,200t at 0.67% WO_3 and 0.26% MoS_2***
- ***Inferred*** ***149,200t at 0.5% WO_3 and 0.3% MoS_2***

In February 2008 QOL undertook an in-house evaluation of the resources based on the removal of the top grade cut, using the 0.1%WO_{3Eq} bottom cut-off. This estimation returned:-

- ***Measured*** ***619,535t at 0.51% WO_3 and 0.17% MoS_2***
- ***Indicated*** ***104,814t at 0.47% WO_3 and 0.18% MoS_2***
- ***Inferred*** ***242,699t at 0.4% WO_3 and 0.2% MoS_2***

The results of this estimation represented a considerable increase in contained WO_3 and MoS_2 compared to the published resources.

Coffey Mining Pty Limited ("Coffey") was asked to develop a pit design which would maximise the economically viable recovery of minerals incorporating the following constraints:-

- The pit design was to be based on the resource estimate which did not incorporate a top cut, i.e. it was designed on the in-house estimate which incorporated a WO_{3Eq} factor of 2.84, no top cut, and a bottom cut-off grade, before dilution, of 0.36% WO_{3Eq}.

- The target mine production rate was to be 150,000t per annum of ore.
- Minimum pre-dilution thickness of 1m.
- 15% dilution at zero grade was to be incorporated whatever the thickness of the mineralised lens.
- 100% recovery of the resources with no mining losses.
- The pit base was set at 549mRL. At this depth the pit would start to impinge on the significant underground workings developed by Mount Arthur Molybdenum in the 1980s.
- No account was taken of the loss of ore in former workings.

After numerous iterations, Coffey‟s pit design and mine schedule (V12a) proposed the mining of 562,984t of ore at diluted grades of 0.52% WO_3 and 0.14% MoS_2, requiring the extraction of 3.9 Mt of waste, at an average stripping ratio of 6.9:1.

6.3.2 Resource Estimate for Planet Metals Ltd 2010 (Golder Associates Pty Ltd)

Golder used a broad envelope to model the mineralisation so there were only three geological domains: sediment outside the contact in the north-east, mineralised zone and unaltered granite.

Tungsten equivalence (W_{Eq}) was W_{Eq} = W plus 2.33 times Mo, based on the prevailing metal prices.

Conservative values of 10ppm W and 2ppm Mo were assigned to non-assayed intervals. High values were trimmed to W = 80,000, Mo = 20,000 and W_{Eq} = 100,000 (after estimation with uncut data). Semi variograms were analysed for W, Mo and W_{Eq}; downhole semi-variograms were used to determine the nugget value of 40%. A weak north-west trending anisotropy was apparent and there were definite ranges for the W assays below 1000ppm or 0.1%. The spatial relationship of higher values was uncertain but probably they were independent of each other and part of the nugget effect.

After selecting appropriate search criteria and data acceptance, Golder applied a median indicator (MIK) technique and estimated the block values. Golder finally classified resources into *Indicated* Resources (minimum of five holes with an average distance of less than 40m from the block) and *Inferred* Resources.

***Indicated* Resources, W_{Eq} cut-off grade of 0.25%, 0.78Mt grading 0.44% W and 0.13% Mo.**
***Inferred* Resources, W_{Eq} cut-off grade of 0.25%, 0.64Mt grading 0.52% W and 0.11% Mo.**

6.3.3 Resource Estimate for Hazelwood Resources Ltd. February 2011 (Golder Associates Pty Ltd)

In February 2011, Hazelwood Resources Limited of Perth requested Golder to update the resource estimate:

- Creating an MIK block model with an SMU of 3 by 3 by 1.5m.
- W_{Eq} would not be used for the block modelling but independent WO_3 and Mo estimates to allow reporting on both.
- Search ellipsoids would be oriented to allow for vertical continuity of mineralisation.
- A Whittle open pit optimisation was to be carried out.
- The update was to be constrained similarly to the 2010 estimate to allow direct comparison.
- Reporting was done for a number of cut-offs required by Hazelwood.

The unconstrained estimate (WO_3 MIK with *carried* Mo) at a cut-off of 0.25% WO_3 comprised:

***Indicated* Resources, WO_3 cut-off grade of 0.25%, 0.53Mt grading 0.78% WO_3 and 0.09% Mo.**
***Inferred* Resources, WO_3 cut-off grade of 0.25%, 0.44Mt grading 0.86% WO_3 and 0.07% Mo.**

The Mineral Resources at 0.05% WO_3 contained in the Wolfram Camp geological model, developed by Golder for Hazelwood are summarised in Table 6-3.

Table 6-3. Mineral Resources – Golders, February 2011

Category	Tonnes Kt	WO3 %	Mo %
Indicated	2,873	0.23	0.048
Inferred	2,213	0.25	0.044
Total	5,086	0.24	0.046

Notes

. Cut-off = 0.05% WO3

The semi-variography for all grade categories was modelled similarly with a nugget effect of 0.4 of the total variance. However, the total contained metal did not change greatly because of the lens interpretation.

6.3.4 Resource Estimate for Deutsche Rohstoff AG (DRAG) April 2011 (Martlet Consultants Pty Ltd)

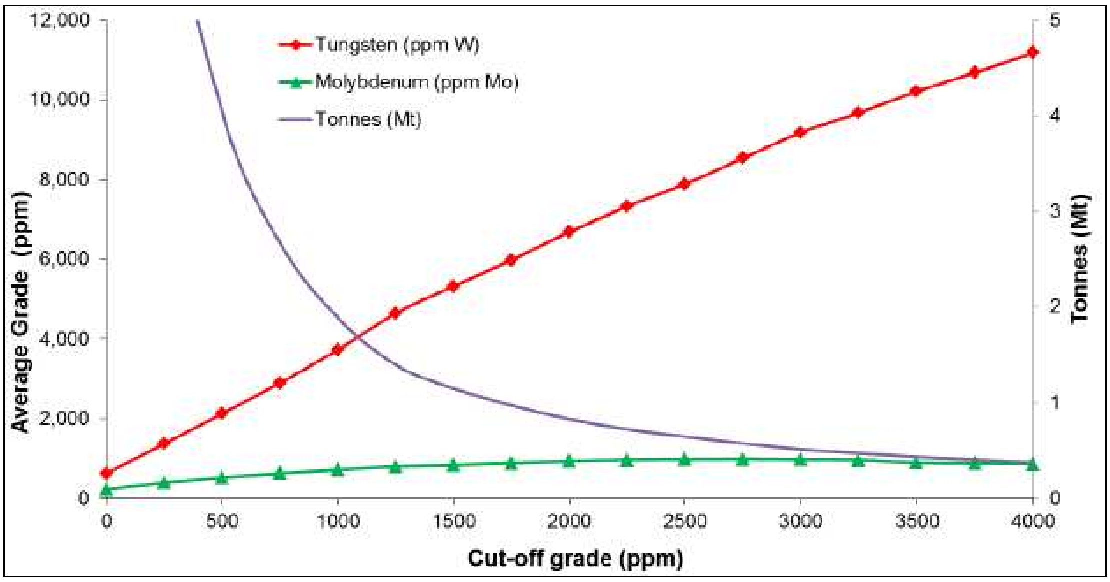
Revision of the Golder 2010 IK was undertaken by Martlet Consultants Pty Ltd. ("Martlet"), and amongst other conditions they were requested to report W tonnes and grade (per 6m bench) using a 0.1%W cut-off and no top cutting.

The 2010 IK model constructed for Planet by Martlet Consultants assumed that Mo was a direct contributor to the economics of a potential mining operation. Consequently, the 2010 IK model was based *on tungsten equivalent grade* and *cut-off* values. The 2011 IK model in the study was based *on W grade* and *cut-off values*. Mo was reported as a secondary element that could contribute economically to a mining operation. Grade/tonnage data and curves are shown in Table 6-4 and Figure 6-7**.**

Table 6-4. Martlet April 2011 Grade-Tonnage Table

Cut-off (%)	Tonnes (Mt)	W (%)	Mo (%)
0.025	7.10	0.137	0.040
0.050	4.05	0.212	0.052
0.075	2.67	0.290	0.063
0.100	1.89	0.373	0.072
0.125	1.40	0.464	0.080
0.150	1.15	0.533	0.084
0.175	0.97	0.599	0.089
0.200	0.83	0.669	0.094
0.225	0.72	0.733	0.097
0.250	0.65	0.789	0.098
0.275	0.58	0.854	0.099
0.300	0.51	0.919	0.097
0.325	0.47	0.968	0.096
0.350	0.44	1.021	0.090
0.375	0.40	1.068	0.088
0.400	0.38	1.118	0.087

Figure 6-7. Martlet April 2011 - Resource Grade/Tonnage Curves



Davis (2011) considered that the tonnes and grade differences between the February 2011 (Golder for Planet) and April 2011 (Martlet) estimates are as shown in Table 6-5; a reduction of 10% contained metal for the Martlet estimate, even though there was no grade cutting in that estimate was mainly a consequence of lower tonnage above cut-off which might be anticipated from the modelling of pipes rather than lenses; several parameters were involved, search distances, etc.

Table 6-5. Comparison of Golder (2011) and Martlet (2011) Resource Estimates

Estimate	Cut-off W%	Grade above Cut-off W%	Resource above Cut-off (Mt)	Metal Contained W(t)
Feb 2011 Golder	0.1	0.39	1.95	7,605
April 2011 Martlet	0.1	0.37	1.89	6,993
Feb 2011 Golder	0.2	0.65	0.97	6,305
April 2011 Martlet	0.2	0.67	0.83	5,561

- **It should be noted that all of the above historical estimates precede reopening of open pit mining operations in 2012.**
- **Subsequent reconciliation of mill and mine production results indicate that major changes in resource estimation methodology have been required.**
- **Therefore the QP is not treating these historical estimates as being particularly relevant to the current updated resource or reserve estimation work.**

7 GEOLOGICAL SETTING AND MINERALISATION

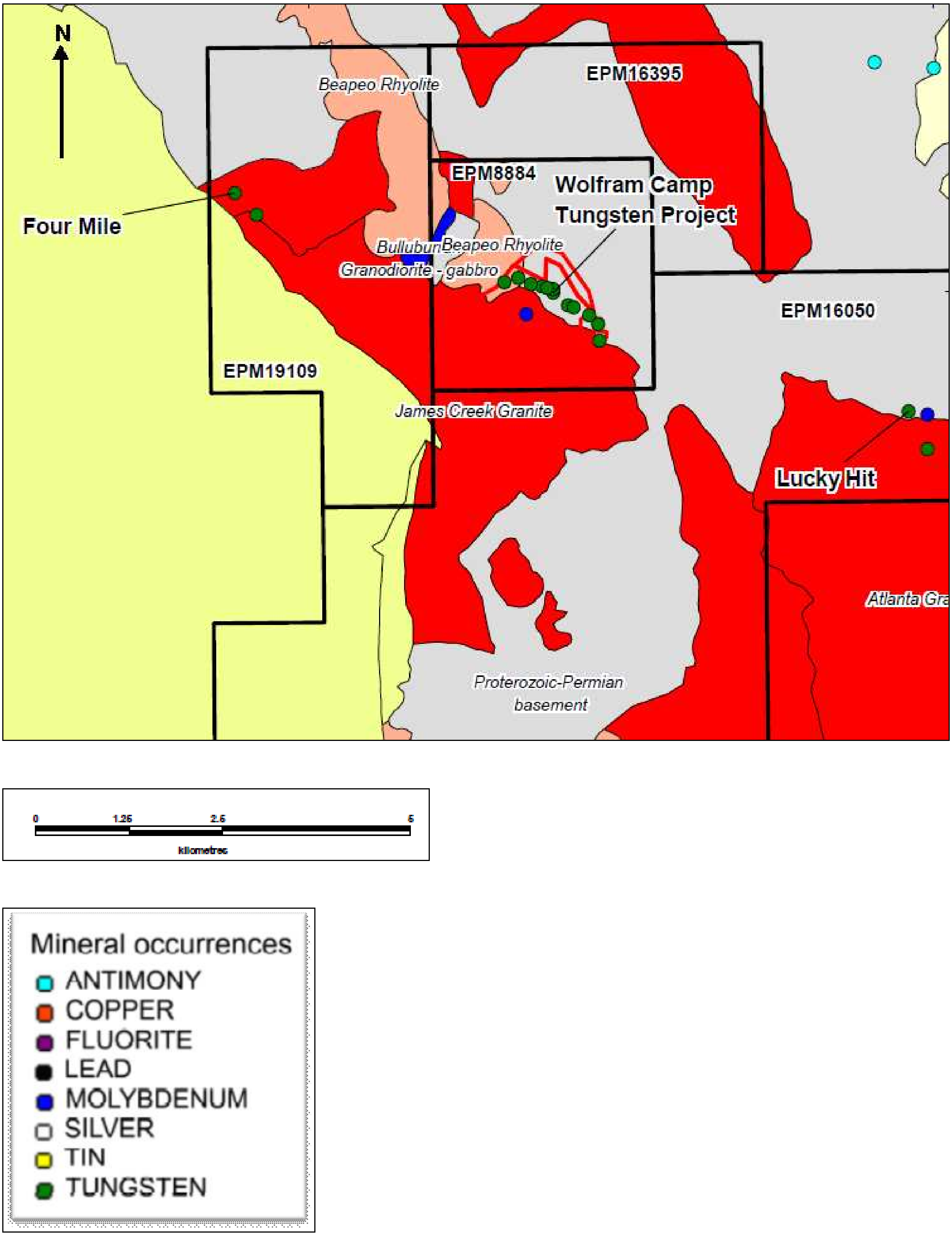
7.1 Regional Geology

The Wolfram Camp Mining Field is situated in the Hodgkinson Basin, which forms part of the Palaeozoic Tasman Geosyncline and comprises Middle to Upper Devonian flysch sequences intruded by a series of Late Carboniferous to Permian granitic rocks and overlain by the Carboniferous Featherbed Volcanics (de Keyser and Wolff, 1964).

The Wolfram Camp deposits (and others in the region) are usually associated with the Late Carboniferous-Early Permian Ootann Supersuite granites (Champion et al, 1991, and Dash et al, 1991) which are generally composed of biotite granite, hornblende-biotite granite and granodiorite. The Ootann Supersuite has a distinct W, Mo and Bi metallogenic association and the late stage siliceous (greisen) alteration at Wolfram Camp reflects this association (Figure 7-1.**)**.

A number of authors have noted the apparent linear configuration of the Bamford Hill – Eight Mile – Wolfram Camp – Mount Carbine tungsten workings.

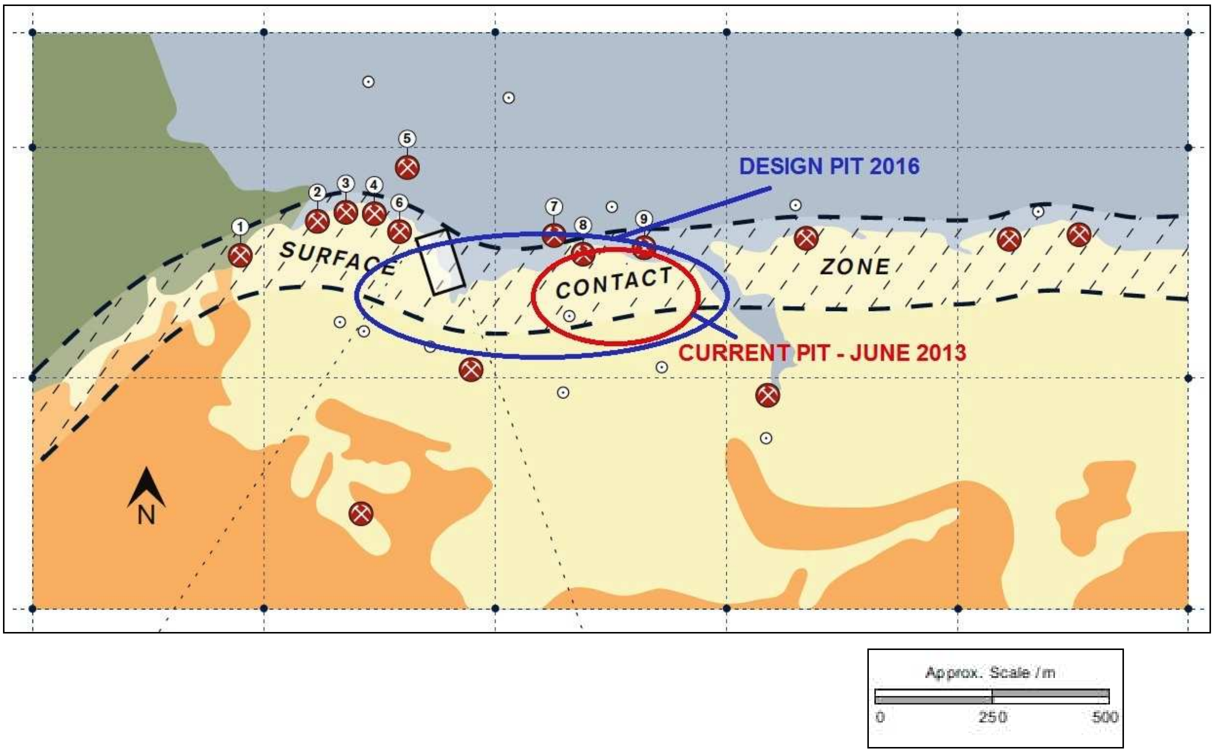
Figure 7-1. Igneous Geology and Mineral Occurrences of the Wolfram Camp Region



7.2 Project Geology

The Wolfram Camp Mineral Field is dominated by the Ootann Supersuite granite intrusives and related greisen alteration and mineralisation. Greisens are apparently developed at the upper contacts of intrusives usually capping apophyses, where late stage (post intrusive) gases and volatiles naturally accumulated, and are in contact with overlying hosts, in this case the sediments and volcanics of the Hodgkinson Formation (Figure 7-2, Figure 7-3 and Figure 7-4).

Figure 7-2. Wolfram Camp – Contact Zone with Adjacent Workings



Legend:

Altered Granite – very pale brown

Unaltered Granite – mid brown

Sediments – blue

Volcanics – green

TOMA Boreholes –circles with central dots

Mine Workings: 1. Forget-Me-Not; 2.Lanski; 3. Larkin; 4. Murphy-Geaney; 5. Leisner Shaft; 6. Hope; 7. Lane Decline; 8. Harp of Erin; 9. German Bill

Figure 7-3. Wolfram Camp – Detail showing Old Workings and Quartz Pipes

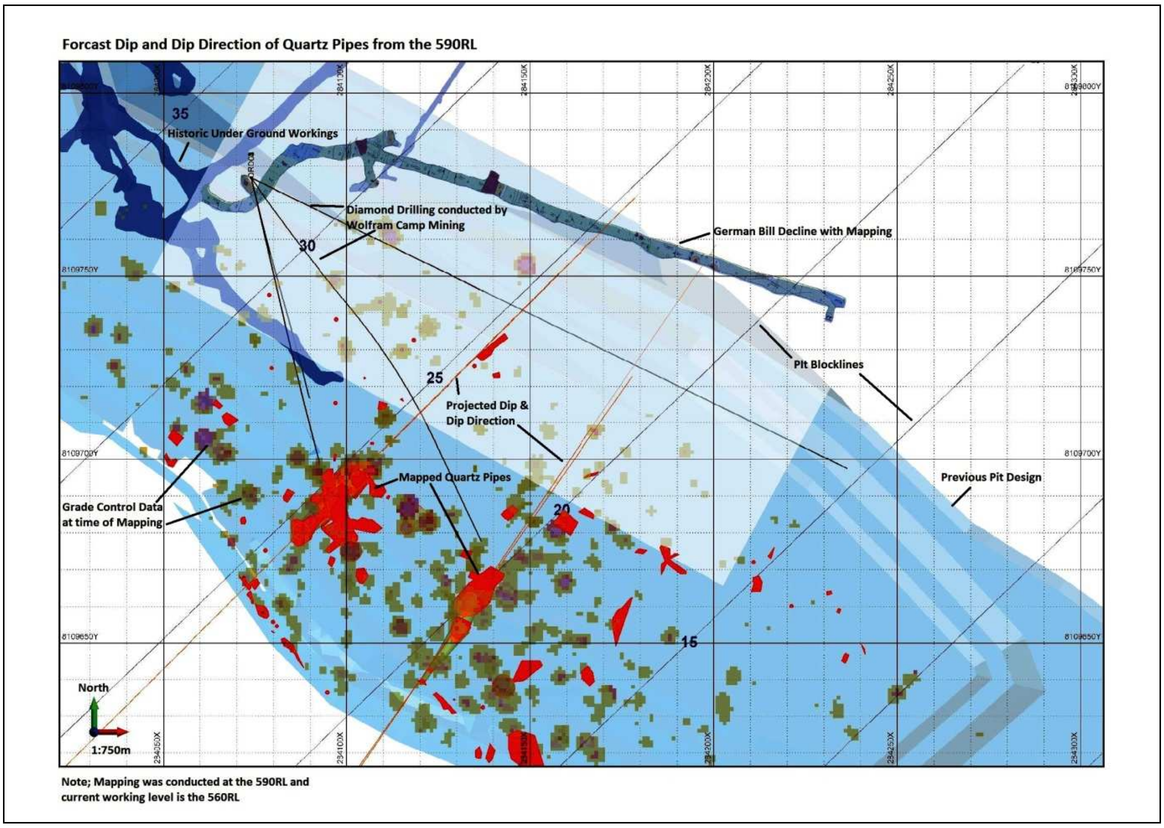
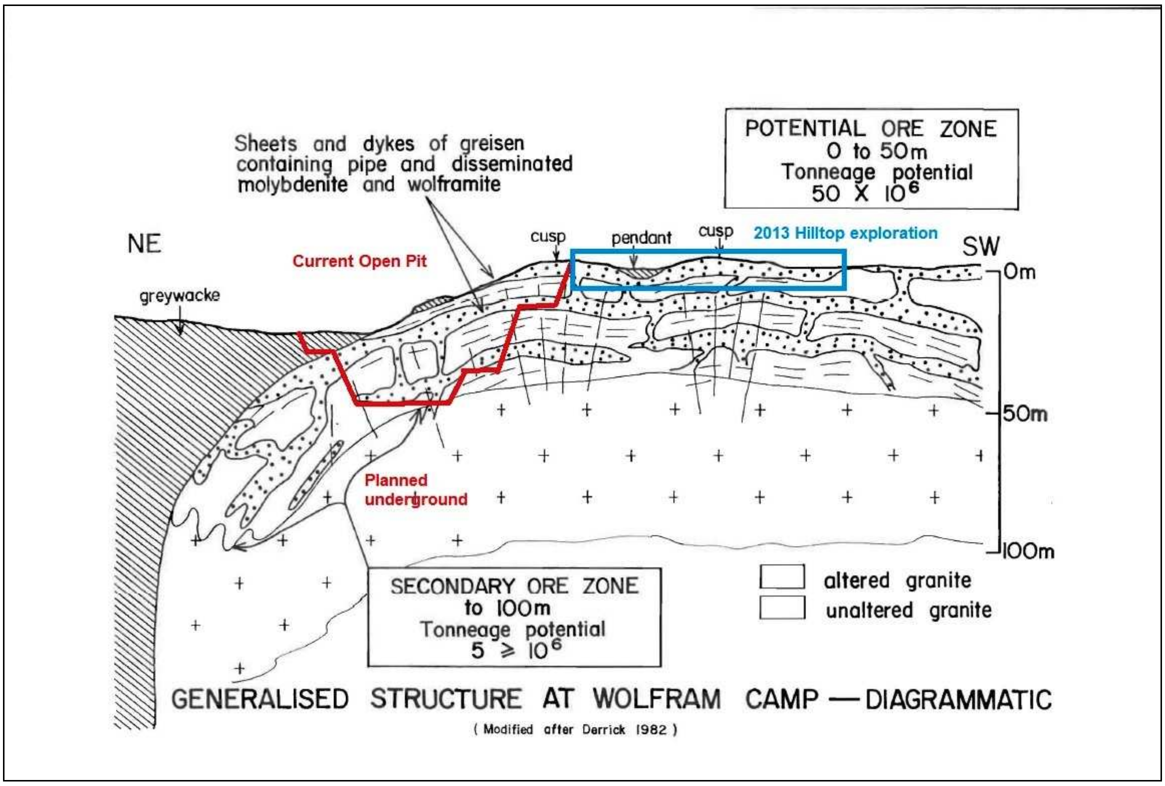


Figure 7-4. Wolfram Camp – Section showing Structure and Open Pit Outline



The granite which hosts the mineralisation at Wolfram Camp is the James Creek Granite. It is described as a pale to medium grey, pinkish grey or pink, fine to coarse grained biotite granite and leucogranite which has been dated at 291 +/-6 Ma.

Ashley (2006) provided a petrological report on a suite of samples for QOL and described the host granite as a moderately to strongly altered coarse grained muscovite-biotite monzogranite comprising dominant quartz with intergrown K-feldspar (probably microcline), sodic plagioclase and biotite.

The granite has been extensively altered over approximately 3km of the contact with the sediments and volcanics in a zone up to 500 m wide on surface, as shown in Figure 7-4. This contact appears to dip at 40^{o}- 60^{o} to the north around the arcuate northern edge of the granite, but there is significant evidence to suggest that the current surface of the granite to the south of the exposed contact is close to the original intrusive contact. Remnant outliers of sediment and the extension of the near-contact alteration for some 800m to the south of the exposed contact, in the nature of a blanket (based on sparse drilling in this area), provide some evidence for this.

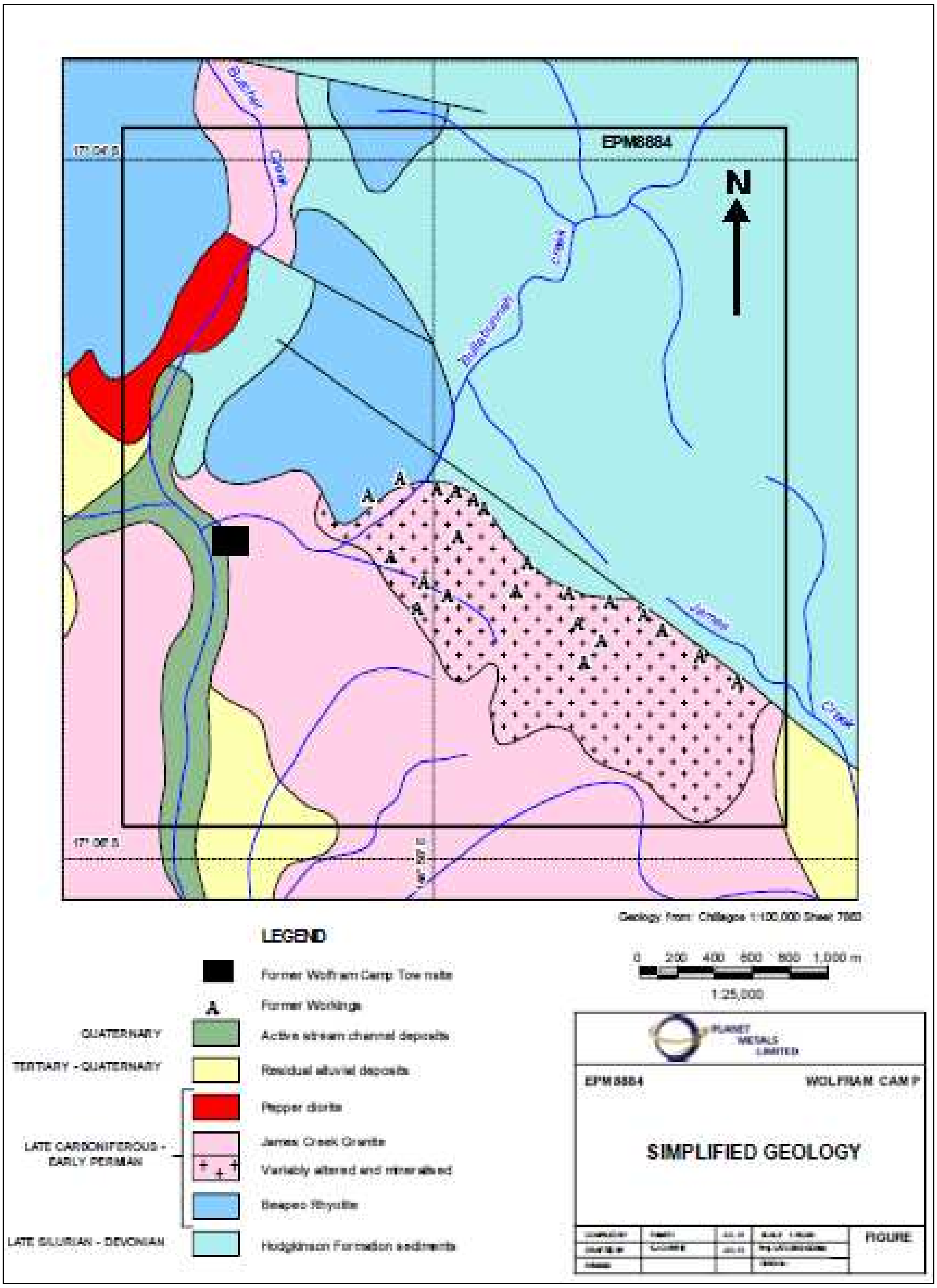
Both sub-horizontal and steep joints are seen in the high wall at the Wolfram Camp mine pit. These are common in granite batholiths giving rise to characteristic boulder strewn topography often with tors and are thought to develop when the batholith cools and crystallises. At Panasqueira in Portugal, sub-horizontal fractures are the important loci for mineralisation; however at Wolfram Camp the sub-horizontal fractures are not usually mineralised. Weathered layers which have caused problems in the treatment plant because of a high clay content are associated with sub-horizontal fractures in the higher part of the pit and are probably the result of ancient water tables developed in wet tropical to arid environments. These were not noticed on surface in the scree ridden slopes at Wolfram Camp but they may be recorded in some drill logs.

Hodgkinson Formation sediments occur to the north-east of the mineralised contact with the Permian-Carboniferous granite (Figure 7-5). These sediments have undergone penetrative deformation and are low-grade regional metamorphic rocks which have been folded and uplifted, and subsequently eroded to form a region of low relief (de Keyser and Wolff,1964). The contact metamorphism is low grade.

Minor sulphide mineralisation has been seen in veinlets with quartz and minor calcite up to a few hundred metres from the contact. Only very minor greisen alteration with associated wolframite, molybdenite and bismuth has been noted within the sediments and volcanics, and mostly occurs within a few metres of the contact.

The Featherbed Volcanics within the area of interest comprise mostly acid ignimbrites of a similar composition to the granite.

Figure 7-5. Simplified Geology of EPM8884, Wolfram Camp



7.3 Mineralisation and Alteration

7.3.1 Mineralisation

Alteration and mineralisation are considered to be related to a post intrusion (or very late stage) aqueous mineralising phase or phases, which produces a greisen. The mineralisation consists of erratic pods, pipes and veins, in which the majority of sulphide, wolframite and molybdenite is contained, scattered throughout the greisen zone, over a width of ~50 m. The greisen extends up to the contact but not into the host rocks. A comprehensive table identifying all minerals present in the quartz pipes and in vugs was produced by Ball (1920) and is shown in Table 7-1.

Table 7-1. Minerals Identified in Wolfram Camp Deposit Mineralisation

Minerals Occurring in Lode Quartz	Minerals Occurring in Vugs
Arsenopyrite	**Arsenopyrite**
Bismuth (native)	**Aragonite**
Bismuthinite	*Bismuthinite*
Bismuth Ochre	*Bismuth Ochre*
Bismutite	**Calcite**
Cassiterite	*Cassiterite*
Chalcopyrite	*Chalcopyrite*
Fluorspar	*Fluorspar*
Galena	*Galena*
Haematite	*Haematite*
Limonite	**Kaolinite**
Molybdenite	*Limonite*
Molybdite	*Molybdenite*
Powellite	*Pyrolusite*
Pyrite	**Pyrite**
Pyrrhotite	*Pyrrhotite*
Scheelite	**Quartz Crystals**
Scorodite	*Scheelite*
Siderite	*Scorodite*
Sphalerite	**Sericite**
Tungstite	*Siderite*
Wolframite	*Sphalerite*
	Turgite
	Wolframite

Italics - indicates minerals occurring in small to very small quantities

Morton and Ridgway (1944) noted that in most of the pipes mined to that date wolframite predominated whilst in some instances (Mulligan, McIntyre, Nil Desperandum) molybdenite was the main mineral. In all cases, however, wolframite, molybdenite and bismuth were all present. In most cases it had been noted that the rare metals crystallised separately but intergrowths involving all three
were fairly common. They also reported the characteristic development of vugs in the pipes, ranging from "*inches to a few feet across*". Occasionally these vugs occupied the full section of the pipe for many feet. The largest vug encountered to that date was in the Enterprise (German Bill) mine with reported dimensions of 20m by 10m by 7m. These vugs were more or less filled with quartz crystals and loose clayey and sericitic material containing a considerable number of minerals including the rare metals.

Underground sampling, mining records and drill results indicate an overall metal ratio of 10 W: 3:5 Mo: 1 Bi. The other sulphide minerals present in the mineralised zone are predominantly arsenopyrite, pyrrhotite and pyrite with trace chalcopyrite, sphalerite and galena. In total these sulphides occur less than the Mo content. The wolframite from Wolfram Camp tends towards the more iron rich variety ferberite ($FeWO_4$) as opposed to the more manganese-rich variety, huebnerite ($MnWO_4$). Mineralogical work undertaken by JKTech Pty Limited (2006 and 2007), on behalf of QOL, identified the following additional minerals in samples carrying low grade mineralisation within quartz greisen and granite samples:-albite; apatite; chlorite: euxenite; fluorite; MnFe oxides; monazite; orthoclase; rutile; sericite; thorite; Ti-magnetite; xenotime; and zircon.

Pipes have been historically the more important economically; they characteristically dip towards the contact and a few were reported to reach the contact and follow it. The course, size and shape of pipes changes abruptly. Pipes with maximum dimensions of as much as 14m x 9m (Murphy-Geaney) are mentioned but any with a diameter of >1.5m were considered good working size. Mined pipes have ranged from less than 1m in diameter to 15m by 10m in plan, and have down-plunge lengths often exceeding 100m. The pipes comprise predominantly glassy white quartz with shoots containing coarse patches of wolframite and molybdenite. The overall known extent of the mineralised pipes at Wolfram Camp covers a strike length of approximately 800m and a depth of approximately 170m. The average width of the zone containing the mineralised pipes is approximately 85m.

7.3.2 Alteration Zones

The complex alteration developed around the quartz pipes can be used to indicate proximity to mineralisation. The most recent classification was established by Tenneco for use in its borehole logging and underground mapping. Tenneco's classification was based on decreasing alteration away from the central quartz pipes, and defined the following:-

7.3.2.1 Type 1 – Quartz Pipe

Quartz pipes comprise white to clear or smoky quartz, commonly containing vugs and with lumps of wolframite, molybdenite, native bismuth (often coated with bismuthinite), scheelite, pyrite, arsenopyrite, pyrrhotite and minor calcite, siderite, chalcopyrite, fluorite, sphalerite, galena and cassiterite. The lumps of wolframite can be over 1m in diameter and molybdenite lumps can reach 0.5m in diameter. Grades vary between pipes but grades in individual pipes tend to be consistent. Some pipes are wolframite rich, while others are molybdenite rich. Pipes can vary in shape from cylindrical to sheets or elongate veins. Mineralised greisen was noted by Morton and Ridgway (1944) as being present around most pipes and they noted that its development increased in importance where the pipes neared the contact. In the mines around the Larkin the greisen ore was so well developed that they found a more or less continuous zone between the pipes which made it possible to mine part of the contact zone in bulk. Michael J Noakes and Associates (1981) noted that both surface and underground mapping by Metals Exploration NL ("Metals Ex") indicate that the greisen zones are elongated approximately north-south. Mapping by Tenneco in the Lane and Forget-me-not declines shows that well-developed quartz greisens are not a widespread rock type and generally restricted to the margins of pipes.

7.3.2.2 Type 2 – Quartz Greisen

The quartz greisen zone consists of vuggy crystalline quartz with variable, and sometimes rich, disseminated wolframite, molybdenite, bismuth, scheelite, pyrite, arsenopyrite and other minor minerals including mica. Mineral grains of wolframite and molybdenite vary commonly between 0.5mm to 1cm although finer and coarser grains do occur. Quartz greisen with disseminated wolframite was termed "spotted dog" ore by the miners, and that with finely disseminated molybdenite was termed "spotted dog lig".

7.3.2.3 Type 3 – Mica Greisen

This zone consists of variable but increasing amounts of muscovite and decreasing quartz with only minor disseminated wolframite and molybdenite and other sulphide minerals. No relict granitic texture is visible. Grain sizes of the target minerals are similar to those in the quartz greisen.

7.3.2.4 Type 4 – "Green Spot"

The so-called "green spot" alteration zone is blotchy cream and green argillite-sericite altered granite. There is generally little to no mineralisation in this zone although increasing molybdenite and scheelite have been recorded at depth.

7.3.2.5 Type 5

The outer zone is defined by variable silicification, muscovite, sericite and argillic alteration of granite. The alteration can be very weak to pervasive and hosts little mineralisation.

7.3.2.6 Weathering

Cameron (1903) noted that molybdenite did not seem to come to surface in the early mines, with its first appearance from 6-10m below surface. Wolframite and bismuth showed little alteration and occurred as resistant minerals on surface. In fact, one 35t accumulation of wolframite on the

Great-I-Am lease towards the eastern end of the field had led to the discovery of the largest deposits on the field to that date. Based on QOL's drilling results, surface oxidation is confined to the top 5m of the deposit with partial oxidation, down structures such as faults to depths of 20-30m.

8 DEPOSIT TYPES

The Wolfram Camp deposit is a quartz-rich pipe-like type deposit, with major element zoning around the pipes (Plimer, 1974). Similar to other pipe-like Mo-W-Bi(+/-Sn) deposits in the Tasman Geosyncline of Eastern Australia, it is hosted in the greisen altered margin and roof zone of a granite mass. Quartz greisens commonly form a rim of several metres wide around quartz pipes, with variable and generally lower grade mineralisation.

A significant amount of detailed mineralisation geology has been gained via historical reports and from TOMA's mapping of the Forget-me-not and Lane declines. The pods, pipes and veins are oriented quite haphazardly but dominantly steeply dipping (Figure 8-1 and Figure 8-2). This is known from the descriptions from the Mines Department (Geology of Australian Ore Deposits, Volume I, Fifth Empire Mining and Metallurgical Congress, Australia and New Zealand, 1953, pp 828), underground exposures at Lane and Forget-me not declines and the drilling for pipe structures by Metals Exploration NL.

Pipes have been historically the more important economically; they characteristically dip towards the contact and a few were reported to reach the contact and follow it. The course, size and shape of pipes changes abruptly. Pipes with maximum dimensions of as much as 14m x 9m (Murphy-Geaney) are mentioned but any with a diameter of >1.5m were considered good working size. Mined pipes have ranged from less than 1m in diameter to 15m by 10m in plan, and have down-plunge lengths often exceeding 100m. The pipes comprise predominantly glassy white quartz with shoots containing coarse patches of wolframite and molybdenite.

This pipe-like model and the very unusual asymmetrical zoning of the pipes has meant planning of exploration holes has been extremely difficult. The previous RC drilling campaigns have generally attempted a systematic coverage of 20m x 20m. Diamond drilling, which has been used much less, has generally been far less systematic and has often been targeting overall depth and along-strike extents of mineralisation, rather than being the fundamental basis of resource estimation.

"Flat Lodes" (*Ibid*) are noted as occurring on high ground where it is interpreted that the contact of the greisen zone was also flat. Importantly the lateral extent of the pipes is restricted; they do not form lenses. Historical sections show clearly the erratic and vertical nature of the majority of pipes (Figure 8-1 and Figure 8-2).

Davis (2011) states that the discussions in QOL (2007) and Golder (2011) documents, *("The quartz pipes and sheets formed in cooling fractures parallel to the contact and in vertical to subvertical tension joints. These fractures and joints were best developed in the vicinity of rolls and flexures in the contact*.") are correct but do not emphasise that the majority of structures are steeply dipping.

The correlations and implied continuity of flat-lying layered lenses seen on drill sections drawn by QOL through the deposit are probably incorrect. For instance, the high grade intersections seen in individual holes are unlikely to correlate with those in adjacent holes; it is more reasonable to assume shorter podiform bodies associated with each hole that are probably steeply dipping.

Figure 8-1. Section at Wolfram Camp showing Form and Distribution of Pipes (Geology of Australian Ore Deposits, Volume 1 1953)

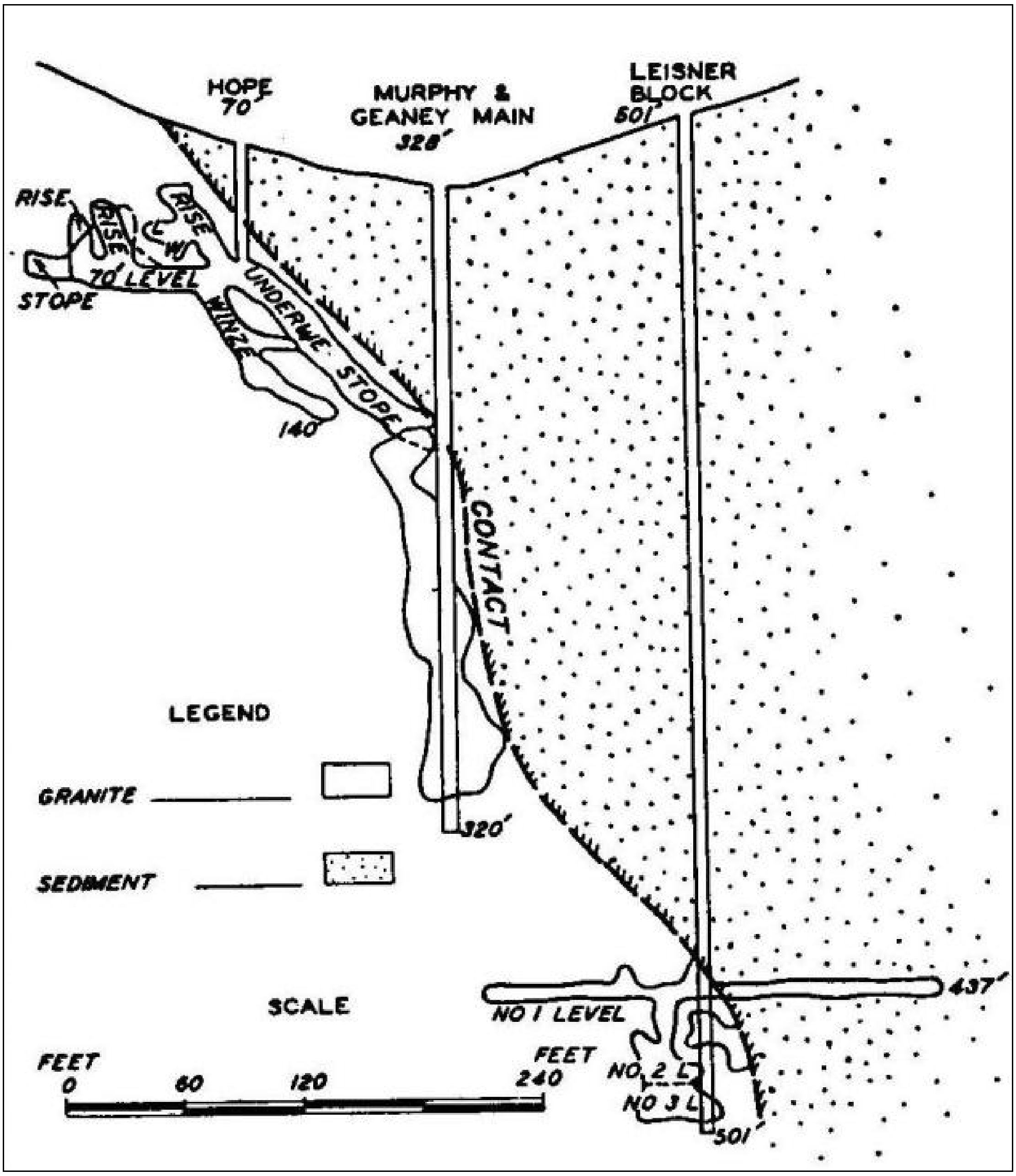
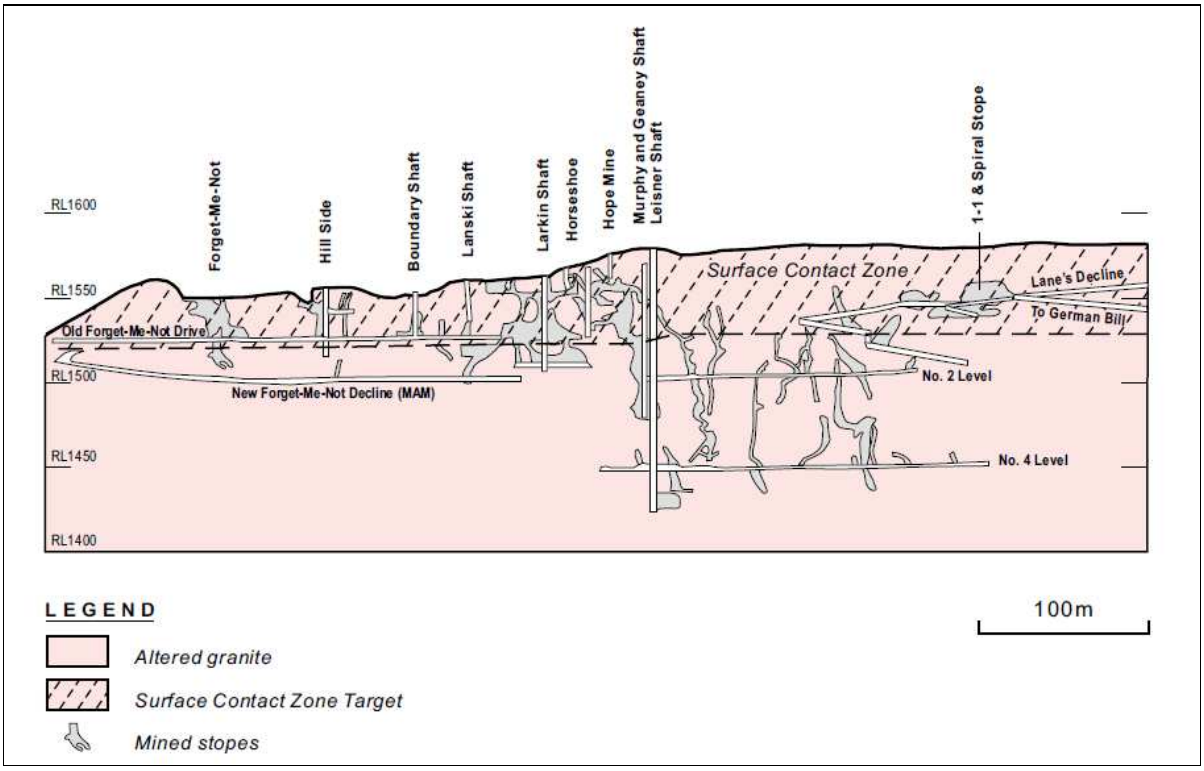


Figure 8-2. Longitudinal section - Wolfram Camp Greisen Zones and Stoped areas

(After Ball 1913 and others)



After site visits in March 2009 Metallica Minerals Limited ("Metallica") interpreted "*the mineralisation as being more sub-vertical and less horizontal–sub-horizontal. Overall throughout the deposit there is a general paucity of drilling (relative to mineralisation style) which complicates the geological interpretation and it is difficult to follow the "ore zones" between sections. However in areas where the drilling is more concentrated the mineralisation can be extrapolated between sections and does indicate a more vertical component especially in the all-important high grade zones*".

Gold Copper Exploration Limited when discussing the Bamford Hill mineralisation which is very similar in style to that of Wolfram Camp noted that "*The W-Mo-Bi assay results reflect both the random coarse grained nature of the mineralisation in the updip portion of the target zone and the more uniformly distributed fine to medium grained disseminated mineralisation downdip*". This is not observed at Wolfram Camp but the volume of pipe material may be expected to decrease farther away from the contact, as will the intensity of alteration/greisenation. The depth potential for pipes is not likely to be great; the mineralisation is more likely to be located in areas with lower hydrostatic pressure.

9 EXPLORATION

9.1 Summary

From the takeover by DRAG in May 2011 until 2014, no primary exploration work was done. Since 2014, the primary method of exploration has been blasthole exploration (BEX) drilling. These holes are generally 25m to 42m in length. To date, most of this BEX drilling has been in and adjacent to the immediate main pit and Parrotts areas.

Exploration Targets were prepared by WCM geologists, for proposed exploration programmes. They are based on regional and logical geology, geophysical surveys and in particular, historical data associated with previous underground production from mines in these areas. The Competent Person is satisfied that there are reasonable grounds for the assumptions employed in the generation of these targets. These Exploration Targets, along with the outlined exploration work connected with them, is summarised in Table 9-1.

Table 9-1. Exploration Targets

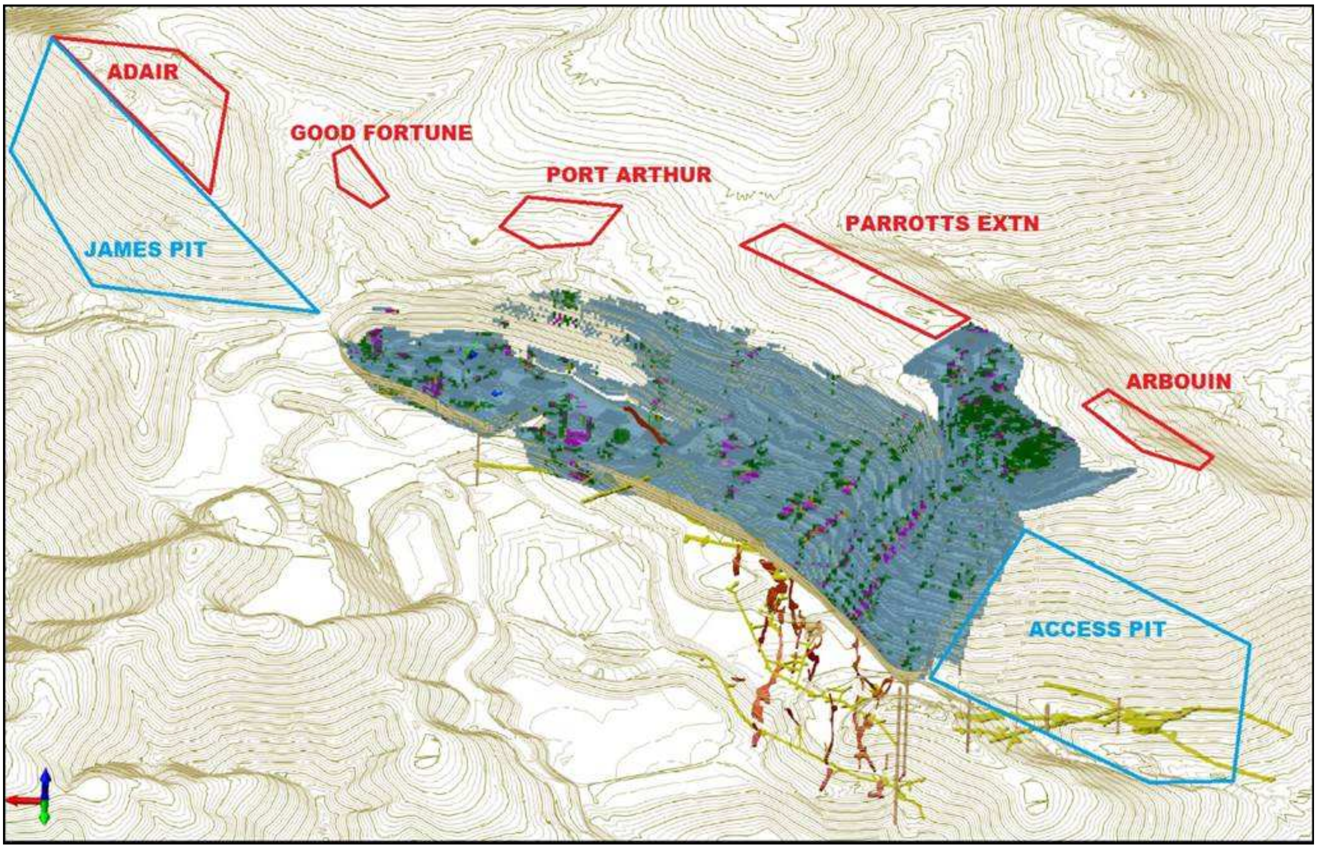
Exploration Area	Priority	Key Prospects	Budget	Exploration Target Approximations			Type Of Exploration	Geology	
			AUD x 1000	Mt	WO_3	Sn			
Wolfram Camp Area	1	Targeting extensions to the current resource model	250	3-5	0.15-0.25%		BEX Drilling, ~1000m/month 2017	Granite contact greisen-hosted W-Mo	
Bamford Hill	2	Bamford Hill Main Zone, Sunny Corner and Tiger's Tail high grade extensions.	25	2-3	0.15-0.25%		Integration of all previous mapping and data, channel sampling of exploration adit		
Four Mile	3	Four Mile + concealed cupola under base metal anomalies	15	0.5-1	0.15-0.25%		Data compilation, mapping geochemistry, target generation		
Eight Mile	3	Eight Mile, Captain Morgan, mapped alteration zones + concealed mineralised cupolas	15	1-2	0.15-0.25%				
Scardon's	4	Scardon's Top Camp	2.5	0.5-1	0.15-0.25%		Data compilation, mapping, target generation		
Sunnymount Group	4	Tommy Burns, Neville, Wolfram Line	7.5	0.1-0.5		0.3-0.5%		Structurally Controlled	Sn-W
Dover Castle Area	4	Extensions of known structures in EPM 14028	2.5	0.1-0.2		0.3-0.5%			Sn-Ag-In
Mistake Group	4	Mistake, Mystery, Spotted Dog, Hermit	2.5	0.1-0.2		0.3-0.5%			Sn-W-F
Koorboora Tinfield	4	Two Jacks	2.5	0.1-0.2		0.3-0.5%			Sn

9.2 Wolfram Camp Area – Priority 1

A number of potential resource extensions exist along-strike from the main Wolfram Camp pit, as well as off set and more to depth, as depicted in a 3D view in Figure 9-1. In Figure 9-2, a plan has been made of the chief quartz pipes associated with the old mines in the area, overlaid with the current pit design and drillhole data. These plans clearly show wolfram mineralisation over a 3 km strike length and explain the positions and potential sizes of these resource extensions to the current open pit. As these resources have been intersected by old historic workings, but do not have recent samples within them, they have been excluded from *Inferred* resources at the current time.

The potential open pit extensions include the Parrotts, Hilltop, James Hilltop, Access and James Hill Pit. It is estimated that these exploration targets contain a potential 3-5 Mt of additional resources. It is anticipated that the exploration work required for these targets will take approximately 1 year.

Figure 9-1. 3D View Looking SE – Wolfram Camp Resource Extensions



Geophysics maps for the Wolfram Camp Area are shown in Figure 9-3 to Figure 9-6. These also support the strike extensions of the geology in this area.

Figure 9-2. Plan of Main Quartz Pipes Associated with Old Mines

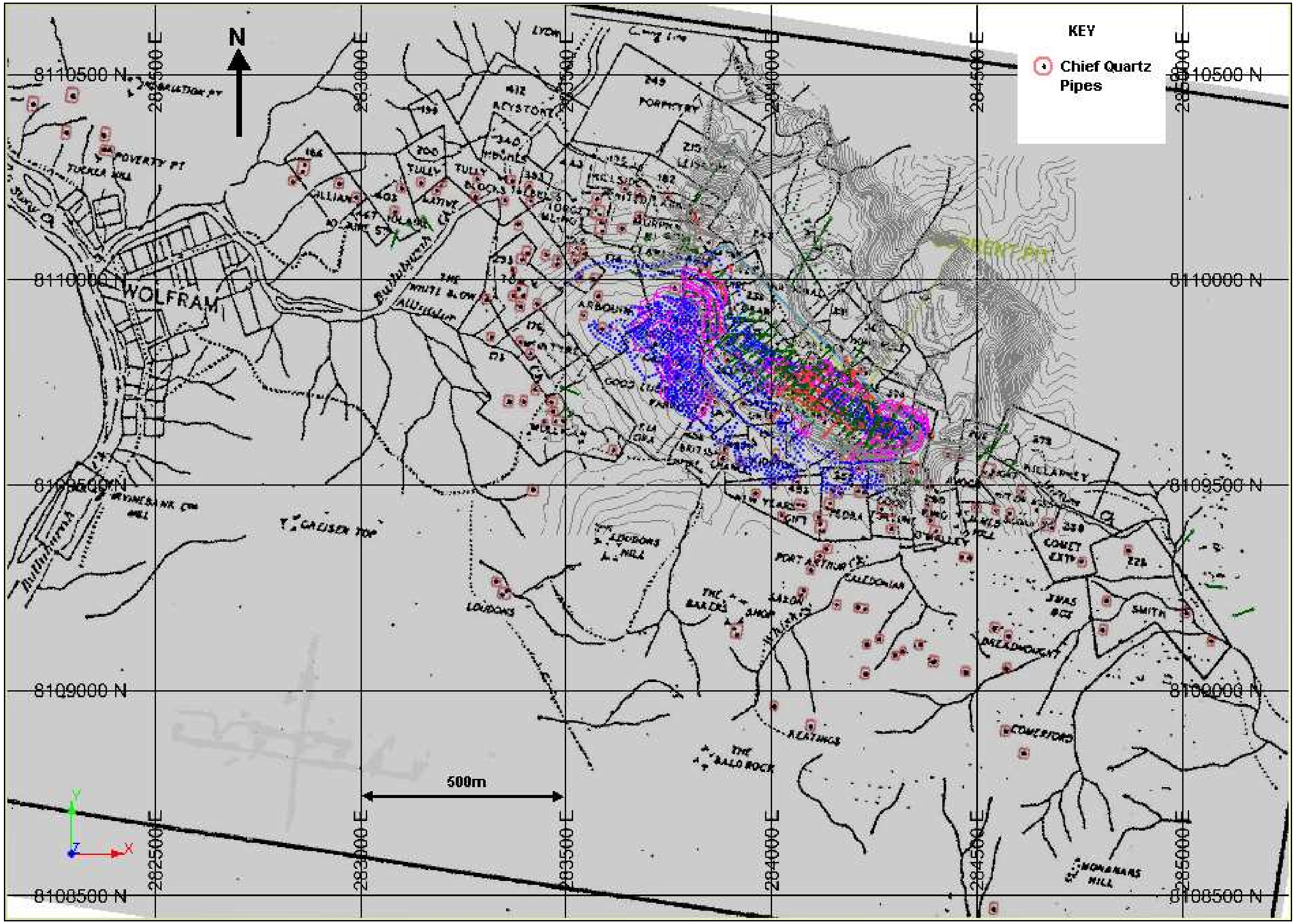


Figure 9-3. WCM Aeromagnetics

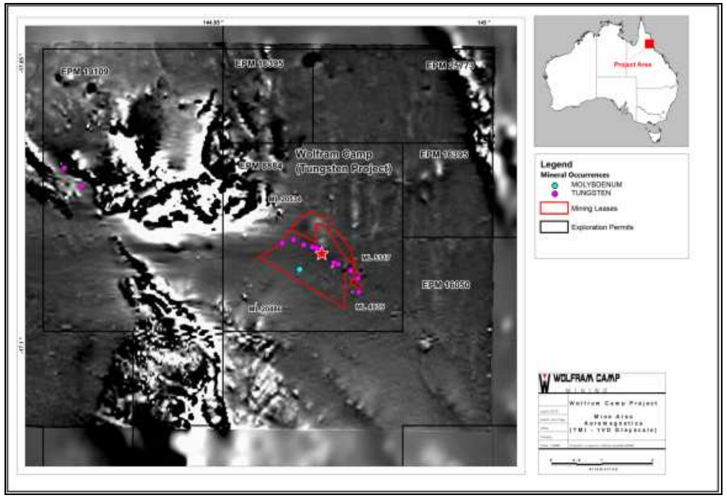


Figure 9-5. WCM - Thorium

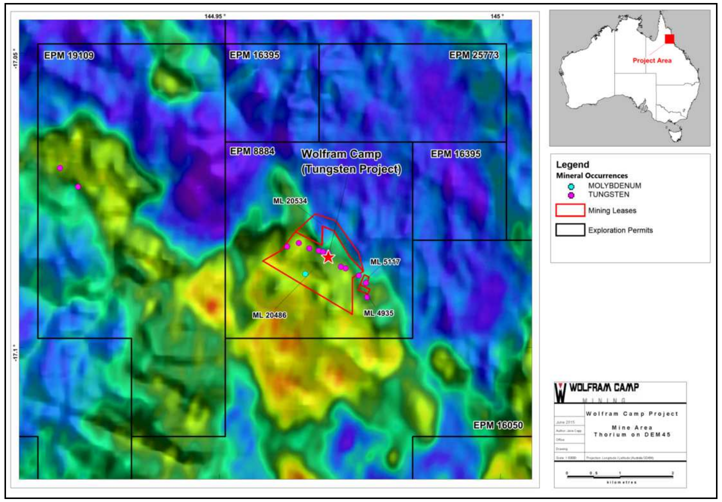


Figure 9-4. WCM - Potassium

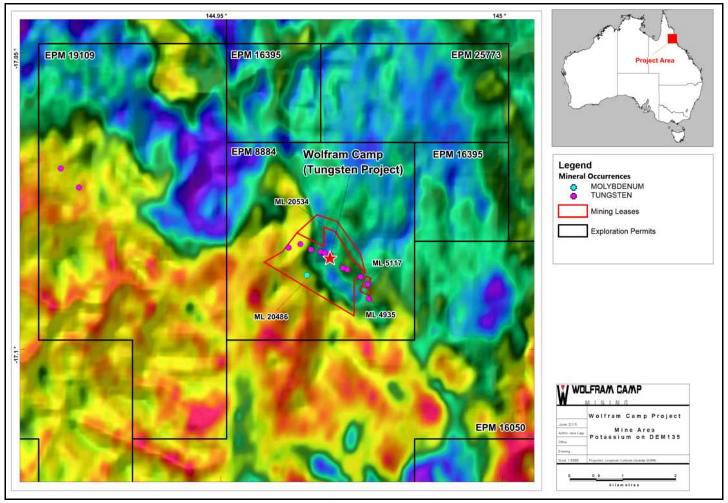
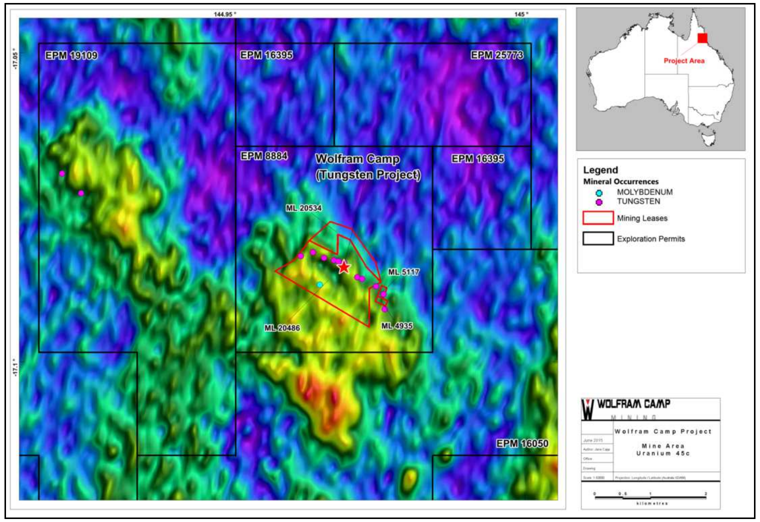


Figure 9-6. WCM - Uranium



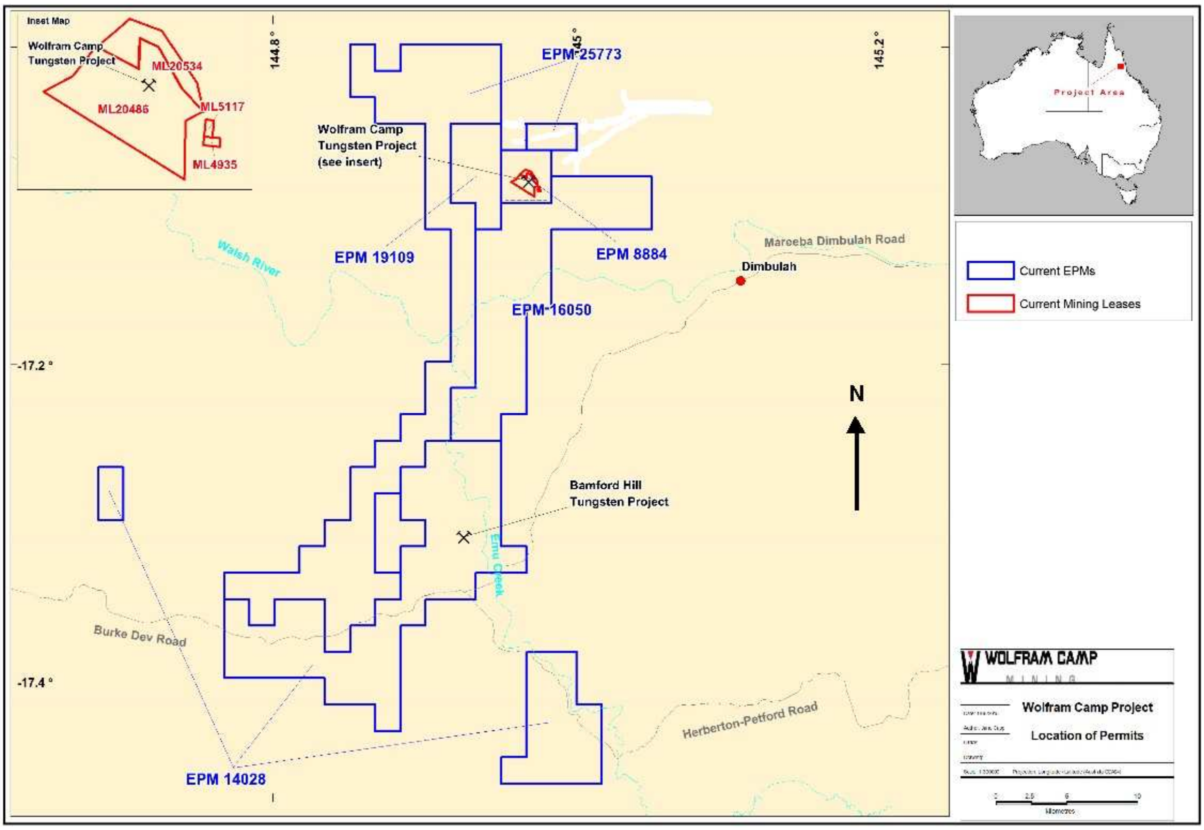
9.3 Bamford Hill – Priority 2

A second priority is exploration in the Bamford Hill area. The key targets are the Bamford Hill Main Zone, Sunny Corner and Tiger's Tail extensions. The exploration program is to include integration of all available mapping, drilling and digitized historical workings, along with survey pick-up of workings, including channel sampling in the exploration adit. The estimated budget for this work is AUD15,000, to commence evaluation of the exploration target of 2-3Mt, with a grade range of 0.15-0.25%WO_3, with higher grade underground extensions. It is anticipated that the exploration work required for these targets will take approximately 1 year.

The Bamford Hill tungsten-molybdenum deposit is located 25 km to the south of the Wolfram Camp Mine, in the southern part of the Bamford Hill - Wolfram Camp Corridor, as shown in Figure 9-7.

Figure 9-7. Location of Bamford Hill

In Relation to Wolfram Camp and the Exploration/Minerals licence Areas.



The Bamford Hill W-Mo-Bi deposits are geologically similar to Wolfram Camp with coarse-grained wolframite (+minor scheelite), molybdenite and bismuth contained within branching, quartz-rich, pipe-like orebodies within the greisenised flank of the high-level fractionated Bamford granite stock.

Wolfram was discovered here in 1893 (a year before Wolfram Camp), with the most extensive production from numerous underground workings during the period 1906-1920 with limited subsequent activity during periods of higher tungsten demand, including significant eluvial mining of remnant surfical deposits from 1979 to 1981, as shown in Figure 9-8.

The scale of the alteration system (over 2.5 km strike extending to depths of at least 250m) led to the most recent systematic exploration at Bamford Hill in the early 1980's which evaluated the bulk-tonnage / low grade potential of the central section of the mineralised contact zone which hosts the highest density of historical workings.

A program of diamond core and percussion drilling (~3,600m), exploratory underground development (Figure 9-9) and analysis of historical production records identified resource potential of 20-30Mt with a low (<0.1%) combined WO3, Mo + Bi grade, and also highlighted significant untested potential for higher-grade pipes.

Current exploration has focussed on the compilation and digitising of historical data, and geophysical trials to assist targeting more intensely mineralised zones within the greisen envelope. WCM plans to define resources to be exploited by future underground mining methods to supplement currently identified resources at Wolfram Camp.

The regional geology around Bamford Hill is shown in Figure 9-10. Geophysics maps for the Bamford Hill Area are shown in Figure 9-11 to Figure 9-14.

Figure 9-8. Bamford Hill – Geological Map Showing Mineral Occurrences

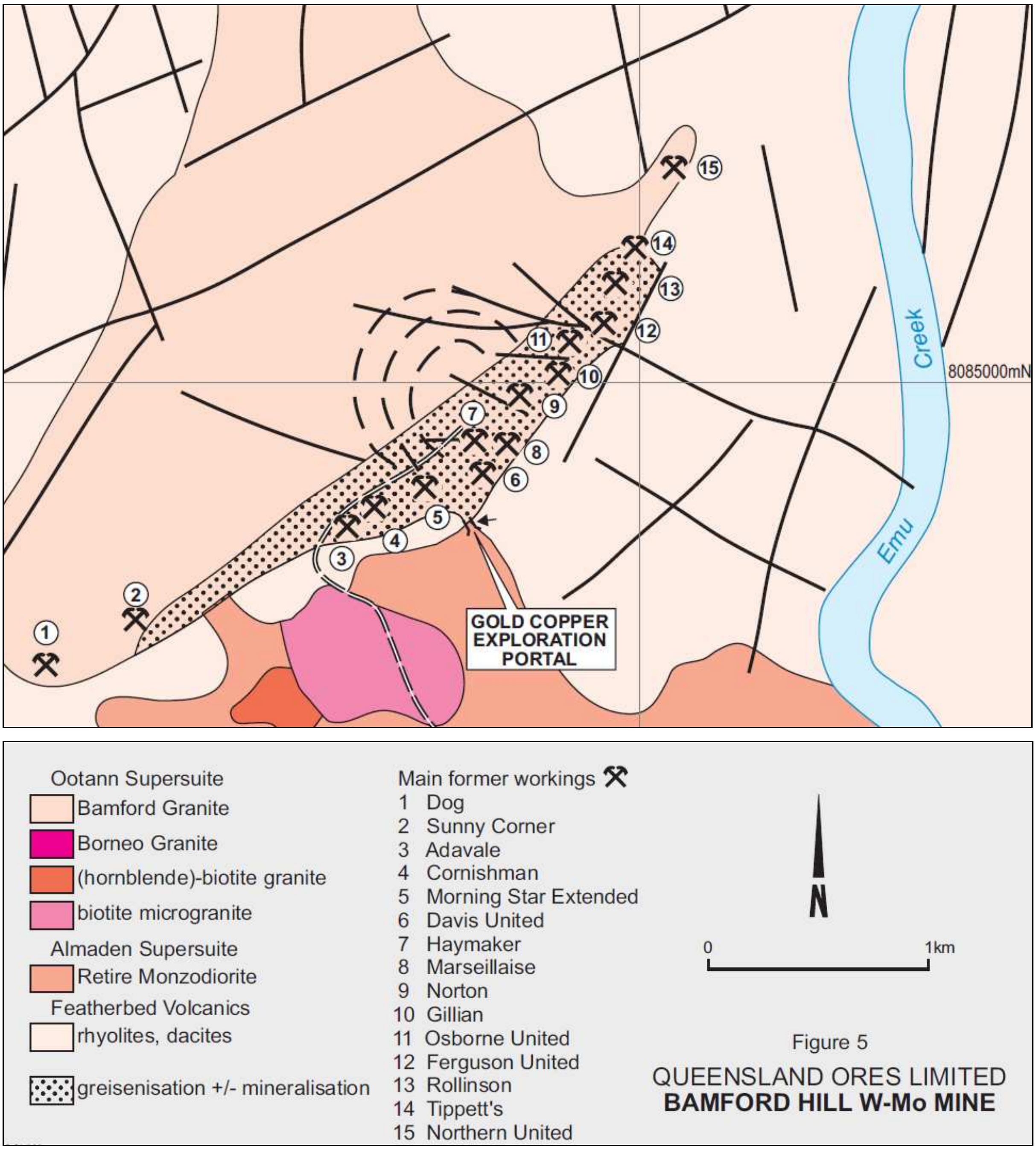


Figure 9-9. Bamford Hill – 450m Exploration Adit

Figure 9-10. Bamford Hill Regional Geology and Mineral Occurrences

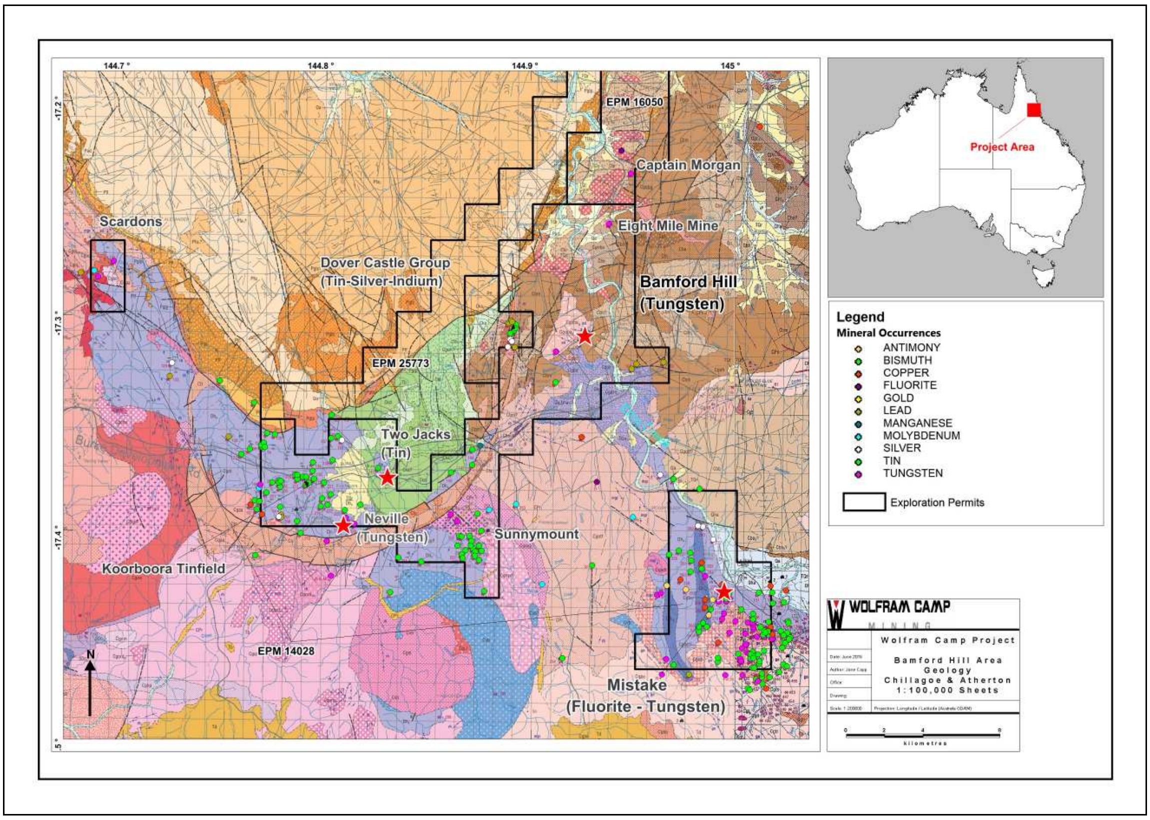


Figure 9-11. Bamford Hill – Iron Ratios

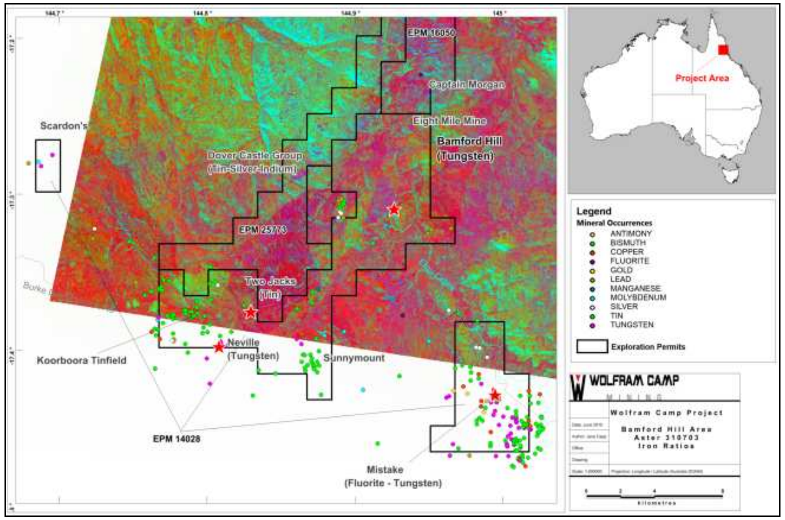


Figure 9-12. Bamford Hill – Magnetics

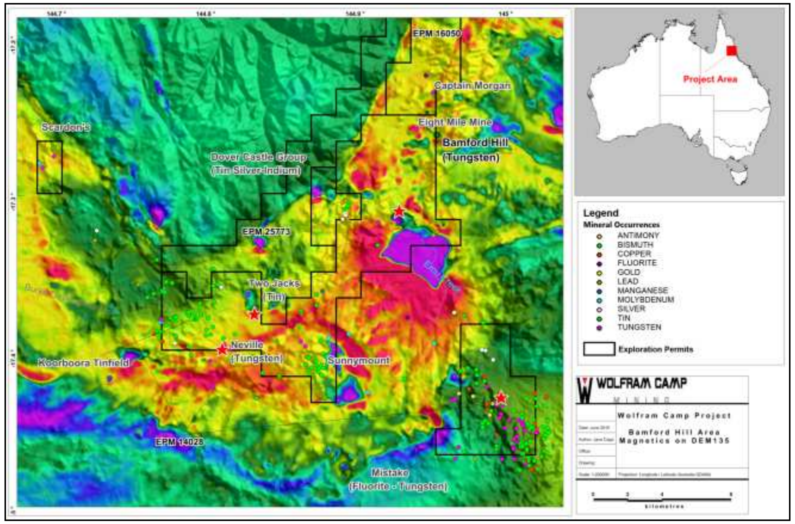


Figure 9-13. Bamford Hill – Potassium

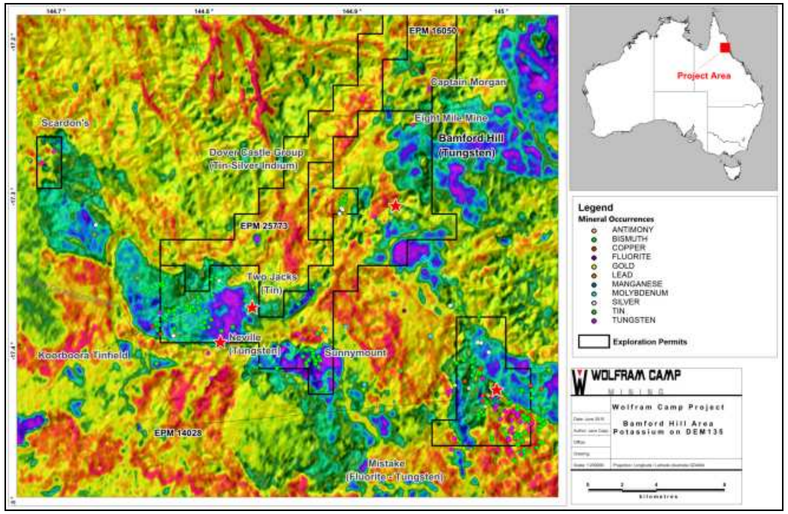
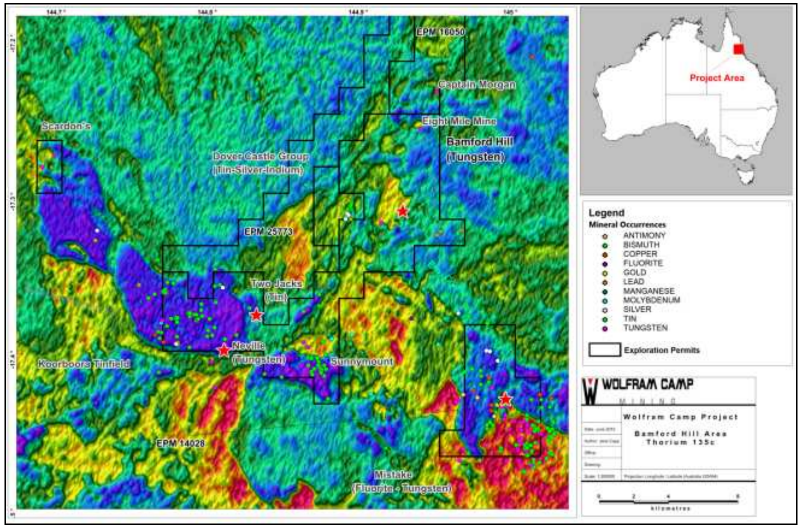


Figure 9-14. Bamford Hill – Thorium



9.4 Four Mile and Eight Mile – Priority 3

Four Mile - Granite contact greisen-hosted W-Mo

Exploration Target 0.5-1 Mt @ 0.15-0.25% WO_3

Key Prospects – Four Mile + concealed cupola under base metal anomalies.

1 year of exploration work - Data compilation, mapping + geochemistry, target generation + drilling.

Planned Program - Mapping / geochemistry AUD10,000 + trial RAB drilling (500m / AUD12,500)

Eight Mile - Granite contact greisen-hosted W-Mo

Exploration Target 1-2 Mt @ 0.15-0.25% WO_3

Key Prospects – Eight Mile, Captain Morgan, mapped alteration zones + concealed mineralised cupolas

1 year of exploration work - Data compilation, mapping + geochemistry, target generation + drilling

Planned Program - Mapping / geochemistry AUD10,000 + target generation AUD5,000

9.5 Scardon's – Priority 4

Scardon's - Granite contact greisen-hosted W-Mo

Exploration Target 0.5-1 Mt @ 0.15-0.25% WO_3.

Key Prospects – Scardon's Top Camp

Program - Data compilation, mapping, target generation AUD2,500

9.6 Other Surrounding Areas – Priority 4

These locations of these other areas are shown in Figure 9-16. It is anticipated that the exploration work required for these target areas will take approximately 1 year.

Sunnymount Group - Structurally controlled Sn-W

Exploration Target 0.1-0.5 Mt @ 0.3-0.5% Sn

Key Prospects – Tommy Burns, Neville, Wolfram Line

Program - Data compilation, mapping, target generation AUD7,500

The mines in the Sunnymount area, separated by the Tennyson Ring Dyke from the Koorboora area, were discovered 20 years after the latter area.

Dover Castle Area - Structurally controlled Sn-Ag-In

Exploration Target 0.1-0.2 Mt @ 0.3-0.5% Sn

Key Prospects – Extensions of known structures into EPM 14028

Program - Data compilation, mapping, target generation AUD2,500

Mistake Group - Structurally controlled Sn-W-F

Exploration Target 0.1-0.2 Mt @ 0.3-0.5% Sn

Key Prospects – Mistake, Mystery, Spotted Dog, Hermit

Program - Data compilation, mapping, target generation AUD2,500

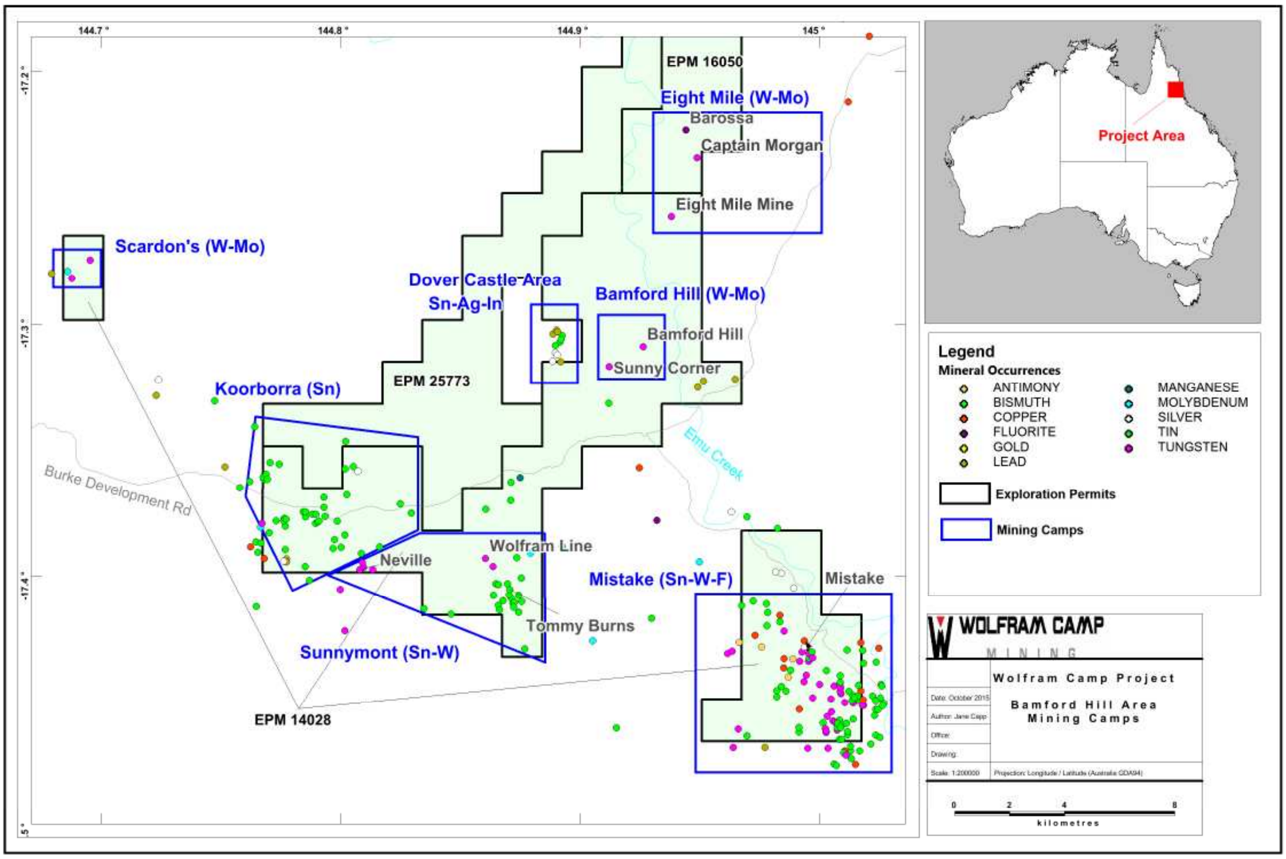
Koorboora Tinfield - Structurally controlled Sn

Exploration Target 0.1-0.2 Mt @ 0.3-0.5% Sn

Key Prospect – Two Jacks

Program - Data compilation, mapping, target generation AUD2,500

Figure 9-15. Other Exploration Areas



10 DRILLING

A limited amount of drilling may have been carried out before the 1970s and although data exist for surface and underground drilling completed in the 1970s there are no detailed records of this work.

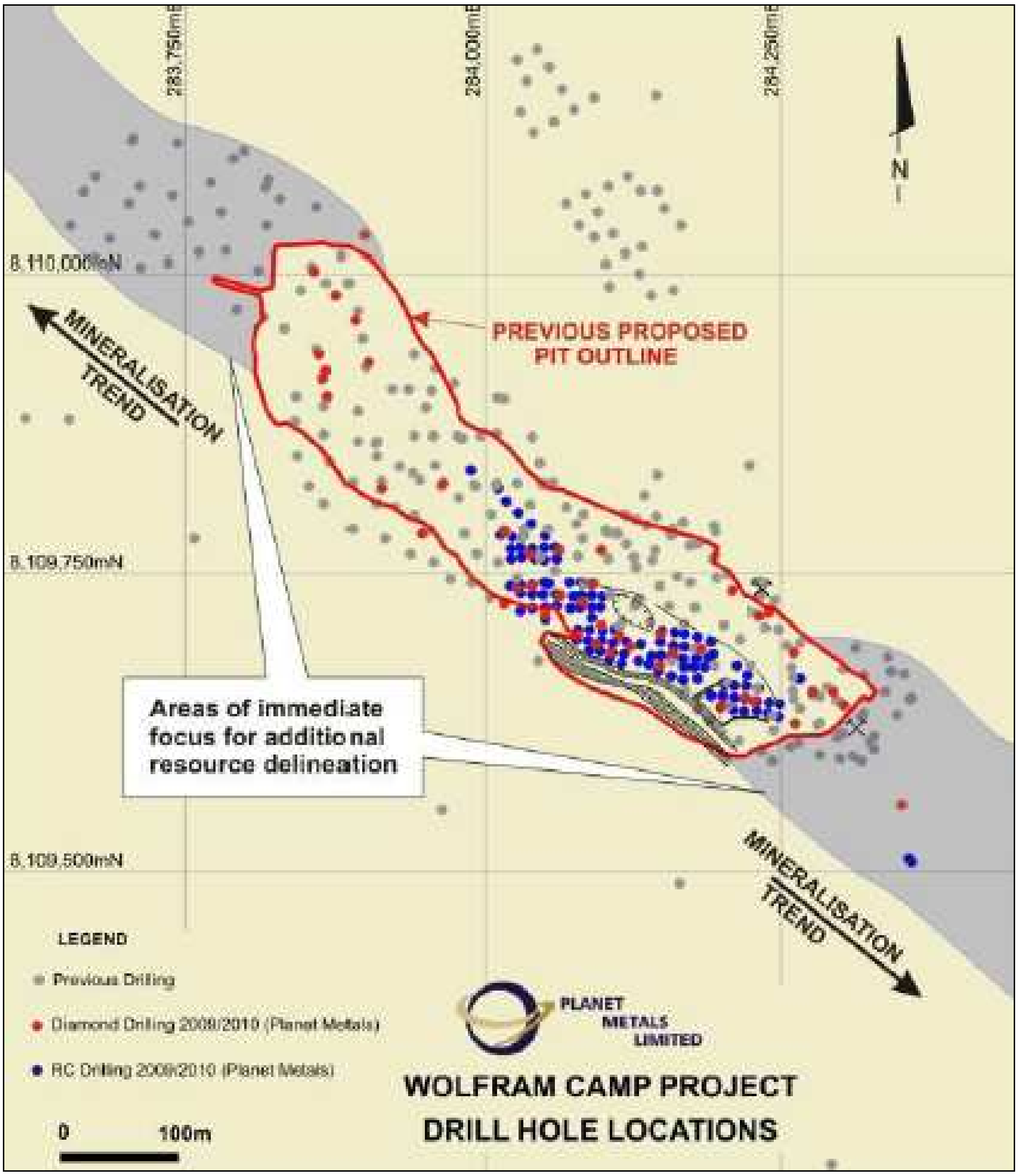
The various drilling programmes completed at Wolfram Camp since the 1970s are summarised below in Table 10-1; however only data from holes drilled since 2000 have been included in the resource estimates for Wolfram Camp.

Table 10-1. Drilling Summary – Historical Exploration Drilling

Company/Year	DD Holes		RC Holes		Underground Holes	
	No.	(m)	No.	(m)	No.	(m)
Metals Ex 1970s	16	1,388		798		10,161
Tenneco (TOMA) 1981/82	12	1,275				
Allegiance Mining NL 1994/95			37	1,726		
Queensland Ores Ltd						
2005	36	2,438	15	939		
2006			112	5,357		
Planet Metals Ltd 2009/2010	45	2,269	155	2,571		
Totals	**109**	**7,370**	**419**	**11,391**		**10,161**

The locations of historical exploration drillholes at Wolfram Camp are shown in Figure 10-1. Since 2014, up to August 2015, WCM have drilled 1,417 blasthole exploration (BEX) drillholes have also been drilled. These holes are generally 25m in length.

Figure 10-1. Wolfram Camp Borehole Locations
(from Golder Associates 2010)



11 SAMPLE PREPARATION, ANALYSES AND SECURITY

All aspects of sample preparation and analyses associated with former 2011 exploration drilling is described in Section 6.

11.1 Grade Control Sampling, Analyses and Security

Blasthole drilling is completed over 5m benches, so these holes, with sub-drilling, are usually 5-5.5m in length. Over each hole, 2 samples are taken. The first from 0-2.5m, the second from 2.5m to final hole depth. All blasthole samples since 2014 have been taken using the rig-mounted Sandvik sample splitter, as shown in Figure 11-1, which also allows the taking of field duplicates, at a frequency of 1 in 10.

Figure 11-1. Collection of Blasthole Sample Material

A flowsheet depicting the current on-site sample preparation and assaying procedure is shown in Figure 11 2. A photograph of the on-site facilities is shown in Figure 11-3.

Figure 11-2. Sample Preparation Facility

Sample 1.5 - 3.5 kg

Drier 105^{o}C, minimum 12 hours

Weighed

Disc pulverizer - 85% passing 600µm

Rotary Splitter > Rejects

Accepts 300g

3 Ring Mill (LM-1 Pulverizer) - 85% passing 75µm

Niton XRF

W grade

Figure 11-3. On-Site Sample Preparation and Laboratory Facility

There are two ED-XRF spectrometers used in the laboratory, a Niton XL3T 700 and a Panalytical Epsilon 3XL. Generally the Niton instrument is used for grade control/geological samples, and the Epsilon instrument is used for plant and concentrate samples.

The Epsilon 3XL XRF programmes have been calibrated against prepared WCM samples that were assayed externally by ALS. The concentrations of most of the major elements of interest, including tungsten, were determined by ALS using an oxidising fusion method with XRF finish (XRF-15c). The Epsilon 3XL XRF provides greater instrument control, has helium purging of the optical path and the deconvolution algorithms are more powerful and easily manipulated, hence the Epsilon 3XL XRF has been used as the master analytical unit.

The Niton XL3t XRF provides a direct tungsten reading using a factory programmed calibration and deconvolution algorithm. To improve reading accuracy, a secondary calibration has been introduced by an algorithm used to generate a pseudo element, which is reported as WO_3. This algorithm has been developed by comparison with both Epsilon and ALS assays, using reference CRM materials. This has been an on-going analysis, leading to better assaying accuracy, focussed on tungsten. The Niton instrument (originally handheld) has also been mounted in a shielded test stand, to improve safety and to further reduce measurement variables related to sample presentation.

11.2 Quality Control

To improve the quality control of samples taken from blasthole exploration and regular grade control (GC) sampling, a Sandvik rig-mounted splitter was used since December 2014, as shown in Figure 11-4. A summary of the QA-QC results associated with this BEX drilling since 2014 is shown in Table 11-1 to Table 11-4.

The Epsilon XRF equipment was used for BEX data from November 2014 up to April 2015. After this the Niton XRF equipment has been used for BEX data analysis.

Figure 11-4. Rig-Mounted Sample Splitter

Table 11-1. Summary of BEX QAQC Sample Frequencies

		All	Field Dup	Lab Dup	CRM	Si02	SQC	Blanks	Primary samples
Nov 14 - Apr 15	Number	10,845	638	1,357	542	110	111	385	8,041
BEX Epsilon E3	Frequency		8%	17%	7%	1%	1%	5%	
April 15 - Aug 15	Number	4,628	330	625	176	39	36	163	3,226
BEX Niton	Frequency		10%	19%	5%	1%	1%	5%	

Notes
. CRM = Certified reference material
. SQC = Internal standard calibration

Table 11-2. Summary of BEX Duplicates' Results

Data Source	Type of duplicate	Number	Mean Original	Mean Duplicate	HARD @90%	Correlation Coeff	Slope	Miss-Classification @ Cut-Offs 0.07% WO3	Miss-Classification @ Cut-Offs 0.12% WO3
FD E3 Splitter	Field Duplicate	637	0.056	0.055	29%	0.977	0.975	2.7%	1.9%
FD E3 Pre-Splitter	Field Duplicate	100	0.083	0.082	30%	0.999	0.956	1.0%	1.0%
FD Niton Splitter	Field Duplicate	505	0.145	0.141	31%	0.999	0.956	2.4%	0.4%
Deep Drilling	Field Duplicate	216	0.048	0.050	33%	0.984	1.126	1.9%	0.5%
CD WO3	Coarse Duplicate	168	0.117	0.118	20%	1.000	1.010	2.4%	0.6%
PD WO3	Pulp Duplicate	379	0.110	0.109	16%	0.999	0.968	1.1%	1.1%
CD WO3	Coarse Duplicate	260	0.091	0.089	16%	0.999	0.944	0.4%	0.0%
PD WO3	Pulp Duplicate	547	0.088	0.088	11%	1.000	0.996	0.2%	0.0%
Ext_Epsilon	External Duplicate	869	0.420	0.433	10%	0.988	0.956	2.0%	1.7%
Ext_Niton	External Duplicate	864	0.433	0.537	16%	0.948	1.200	2.5%	2.1%

Notes
. For external duplicates, indices were determined with a filter of >0.05 <3%WO3
. Results above summarised for WO3 only

Table 11-3. Summary of BEX Blanks' Results

	Niton Blanks		Epsilon Blanks	
Count	188		348	
Max	0.038		0.027	
Min	-0.001		-0.002	
>0	171		130	
>0.005	56	29.8%	22	6.3%
>0.01	25	13.3%	10	2.9%
>0.02	5	2.7%	2	0.6%

Table 11-4. Summary of Certified Standards' Assays

	CRM MP-2 0.820 (WO3)		% Out of Range	CRM BH-1 0.532 (WO3)		% Out of Range	SiO2		% Out of Range
Epsilon Standards	Number	152		Number	152		Number	152	
	Average	0.830		Average	0.537		Average	0.001	
	St dev.	0.024		St dev.	0.010		St dev.	0.001	
	2σ	0.048	0.7%	2σ	0.021	6.6%	2σ	0.002	1.3%
	3σ	0.073	0.0%	3σ	0.031	0.0%	3σ	0.002	0.7%
	CRM MP-2 0.820 (WO3)		% Out of Range	CRM BH-1 0.532 (WO3)		% Out of Range	SiO2		% Out of Range
Niton Standards	Number	95		Number	95		Number	95	
	Average	0.789		Average	0.538		Average	0.001	
	St dev.	0.017		St dev.	0.009		St dev.	0.001	
	2σ	0.035	2.1%	2σ	0.019	5.3%	2σ	0.002	2.1%
	3σ	0.052	1.1%	3σ	0.028	0.0%	3σ	0.003	0.0%

12 DATA VERIFICATION

Data verification procedures that have been applied by the qualified person include:

- Inspection of all active mining, milling, sampling and laboratory facilities on-site.
- Import of supplied drillhole database and reprocessing of these data to check for any sequence, overlap or out-of-range errors.
- Import of grade-control sample database and reprocessing of these data to check for any sequence or out-of-range errors.
- Check analysis of a study completed by the geological department, to compare grades between fine and coarse material from blasthole samples grades.
- Use of all imported sample data for analysis against the on-site resource model and grade-control model.
- Import of the 2014 on-site resource block model, and check calculations on this to ensure it corresponds with the mine’s own current resource figures.
- Import of the current grade control model, and analysis of its contents to test that its contents correspond with mine production figures.
- Generation of a retrospective resource block model, dating back to March 13, to enable comparison of resource modelling parameters with respect to actual mine production.
- Contiguity analysis, to test the degree of smoothing that may occur with reverse circulation sampling.
- Analysis of mine production reports, to look at planned operating cost levels and applied cut-offs grades.

12.1 On-Site Laboratory – Quality Control Procedures

Current quality control procedures include:

- Granulometric tests – disc and ring mill. These tests have started relatively recently.
- Monthly mass checks of rings and rollers. Rings rejected after a 30% loss.
- XRF analysers – daily checks:
    - Controlled reference material (CRM) checks.
    - Silica – blank.
    - CRM diluted with silica.
    - Standard quality control (SQC) sample (0.24% WO_3)

The CRM sample used is a Canmet MP-2 sample, which has a grade of 0.65%W+/-0.02%.

12.2 Drillhole Database

Data related to the drill holes were supplied in .csv format from export of the geological department's Micromine system. For diamond drillhole and reverse circulation data, these included separate files for collars, lithologies, assays and survey data. For grade control (GC) data, separate files were imported from collar and assay data. Verification checks on these data included:

- Importation of data into Datamine, and logical combination of the files, through the desurveying process. No sequence, overlap or out-of-range errors were encountered.

- Checking of all drillhole data against supplied actual and historical topographies - no errors were encountered.

- Checking of drillhole data against historical and current geological sections.

- Checking of drillhole and GC data against on-site resource and grade-control models, in terms of general WO_3 grade distributions. In general a logical correspondence was observed.

12.3 Blasthole Sample Data

The hydrocyclone used on the blasthole rigs removes fines (back dust) from the cuttings (approximately 20%), leaving the coarser material to be deposited on the bench floor from a drop box. Samples have been and are being taken from the drop-box only. A study was completed by the mine in Dec 2013, to ensure that sample grades are not being biased by non-sampling of back-dust. These data, consisting of 48 sets of paired samples, were also analysed by the qualified person. These results are displayed diagrammatically in Figure 12-1, for low grade ranges. A summary of the average % difference of the drop-box sample grade and the total combined WO_3 sample grade is shown in Figure 12-2. A summary of key results derived from these data is shown in Table 12-1. These results support the absence of bias by taking only drop-box samples.

Figure 12-1. Coarse v Fine WO_3 Blasthole Grades

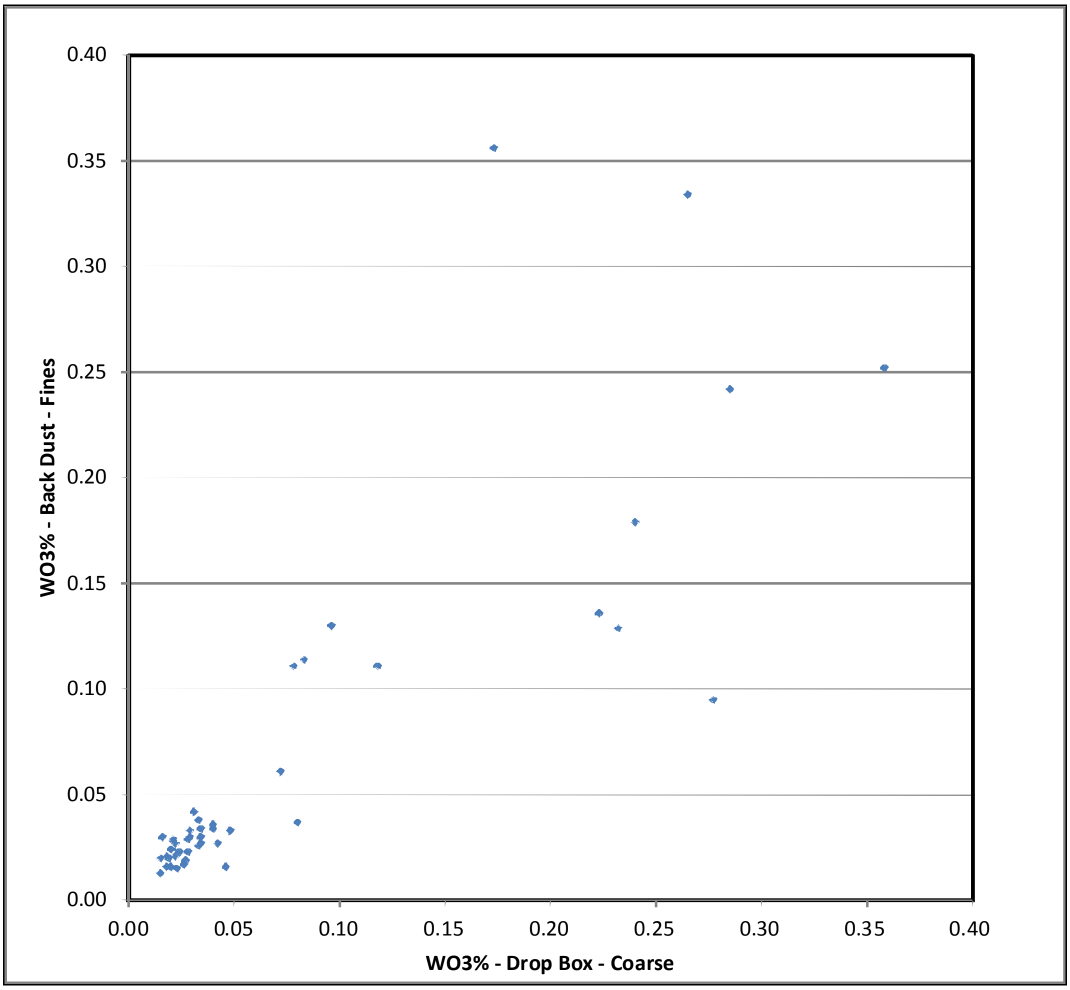


Figure 12-2. % Difference in Coarse versus Combined WO3 Grades

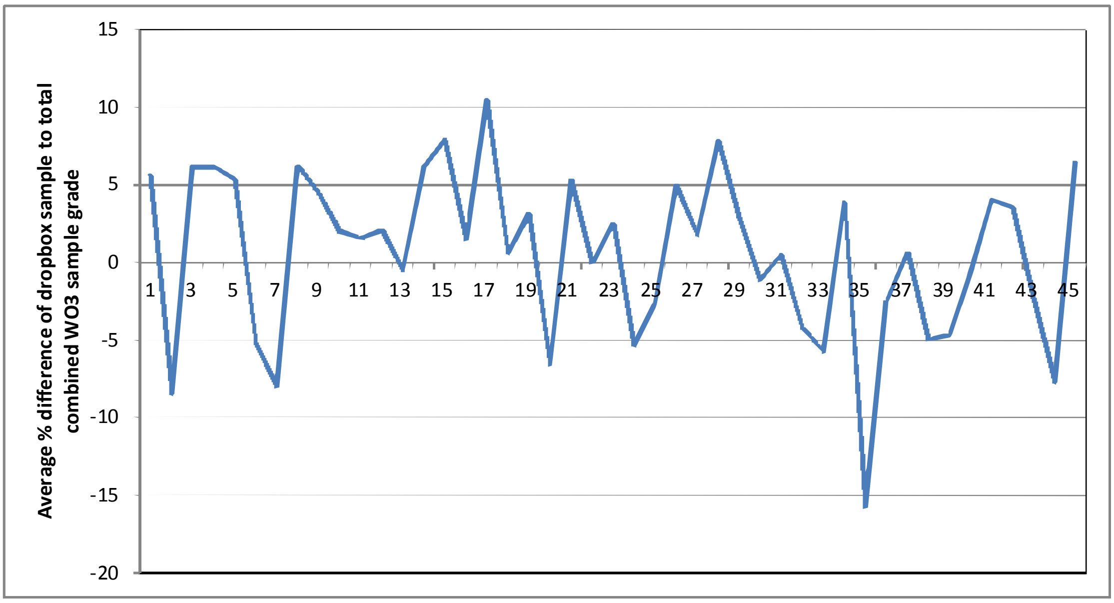


Table 12-1. Summary Results – Blasthole Split Grade Analysis

Average % difference of dropbox sample to total combined WO3 sample grade	0.61%
Correlation Coefficient - Coarse: Fine WO3 Grades	98%
% Mis-Match of Ore:Waste, based on a 0.12%WO3 cut-off	4.2%

For the first 1.5 years of production (since 2012), samples were taken by spear samples sampled from collar cones, or taken from cuttings dumped onto a rubber mat, for each 2.5 m. All blasthole samples since 2014 have been taken using the rig-mounted Sandvik sample splitter, which also allows the taking of field duplicates, at a frequency of 1 in 10.

12.4 Mine Block Model Analysis

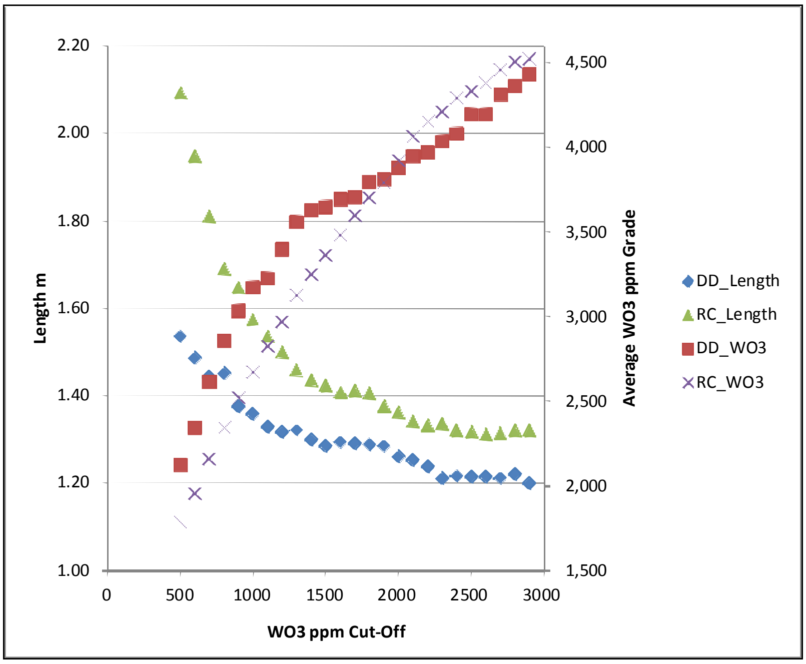
During the update of the 2014 resource block model, the procedure was also used so as to build up resources extracted back to June 2013. Evaluation results of this part of the model were used in the verification process, as summarised in Table 14-14.

12.5 Grade Smoothing Analysis

Comments have been made in previous due diligence studies related to smearing of grades with RC data. A contiguity analysis has been completed in the current review, where for all RC samples inside the defined mineralised zone, the average length and grade are determined for all intersections above a certain cut-off. This process is repeated through a series of cut-offs and the results collated. The same procedure is then repeated for DD samples.

A summary graph of the results is shown in Figure 12-3. For the average grade calculations in this 2014 analysis, a top-cut of 0.5% WO_3 was applied (the current top-cut level has been revised to 1.1% WO_3). At a 0.12% WO_3 cut-off, the average intersection length goes from approximately 1.3m to 1.5m in going from DD to RC samples. However, bearing in mind that that there is also almost 5x as much RC drilling as DD drilling, it is considered this degree of smoothing is not excessive, and the use of RC samples is still acceptable for resource modelling purposes. Another important factor in favour of the RC sampling is the much larger sample volume, as RC involves whole samples, while the DD samples only represent ½ or ¼ core samples.

Figure 12-3. Contiguity Analysis on WO_3 Grades Within DD and RC Samples



12.6 Summary

Since 2014, regular QA/QC procedures have been implemented for all GC and BEX drilling. For samples derived earlier than this, sample data were verified as far as possible by the QP, as described above.

In the opinion of the QP, the verification results obtained in the current study support the resource estimation results that have been derived.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Queensland Ores Limited (QOL), who started the development of the Wolfam Camp deposit in Queensland, commissioned Lycopodium Engineering Pty Ltd (Lycopodium) to assist in developing a metallurgical flowsheet to produce both molybdenum and wolfram concentrates from the deposit. A preliminary metallurgical testwork programme was conducted under the guidance of Lycopodium Engineering (Lycopodium) on samples representing the three main ore types; granite, pipe and greisen ore. These ore types make up the majority of the resource base. Testwork was conducted by a number of laboratories:

- *Australian Metallurgical and Mineral Testing Consultants (AMMTEC)* Perth, Australia -sample preparation, sizing, grind size, gravity, magnetic, comminution characterisation and flotation.
- *Julius Krutschnitt Minerals Research Centre (JK Tech),* Brisbane, Australia - drop weight test evaluation and mineral liberation analysis.
- *Roger Townend and Associates*, Perth, Australia - mineralogy.
- *CSIRO Minerals* Bentley Western Australia - WHIMS Testwork. (Report provided in AMMTEC report).
- *Outotec Pty Ltd (Outokumpu)* - Thickening Testwork. (Report provided in AMMTEC report Appendix V1).

In the reports, reference was made to wolfram recovery, which is generally reported based on the assay WO_3 and is calculated from tungsten assays. Reporting WO_3 assays and recovery is standard industry practice and was adopted for the report. In some cases, AMMTEC reported tungsten (W) assays and where appropriate these were converted to WO_3 by applying a multiplier of 1.26. The aim of the metallurgical testwork programme was to:

- Generate preliminary metallurgical data on the recovery potential of producing a concentrate of both molybdenum and wolfram of a saleable grade.
- Produce key metallurgical data for the design of a full scale production operation.

The study basis for the operation was:

- Feed grade of 0.4% W (or 0.5% WO_3) and 0.3% Mo.
- Target tungsten concentrate grade of 52% W (or 65% WO_3).
- Target molybdenum concentrate grade of 50% Mo.
- Target recoveries of 80% W (or WO_3) and 80% Mo.

13.2 Testwork

13.2.1 Introduction

The testwork programme was conducted in two stages:

Stage 1: Preliminary testwork to establish basic parameters for molybdenum flotation and wolfram gravity recovery. The first stage of testwork consisted of bench scale testwork which was completed at the Ammtec Laboratory in Perth. This included some comminution parameter tests, gravity and magnetic separation tests, and flotation tests. The testwork was performed on composites made up from the three ore samples by equal weight (master composite) and subsequently, on the higher grade greisen ore composite.

The economic minerals in the ore contain molybdenum and tungsten. Tungsten minerals are wolframite ($(Mn,Fe)WO_4$) and scheelite ($CaWO_4$). The Molybdenum mineral is molybdenite (MoS_2).

Stage 2: Develop a flowsheet and design basis for the wolfram gravity circuit and provide further design data for the molybdenum flotation circuit.

13.2.2 Stage 1 Testwork Summary

The key findings of the testwork are summarised below:

13.2.2.1 Mineralogy Analysis

• The tungsten is present as both wolframite and scheelite with wolframite crystals having 71 - 72% WO_3 composition (2 analyses). If this is a reflection of the average tungsten grade of the tungsten minerals, then concentrate would have to contain more than 90% of the tungsten minerals to meet the targeted grade.
• Wolframite and scheelite are present as discrete crystals up to 1mm.
• The main ore mineral is wolframite occurring as discrete prismatic particles of up to 1mm (commonly 0.8 mm in the high grade Greisen composite sample).
• The minor ore mineral; scheelite, occurs as discrete prismatic particles, commonly occurring as composites with wolframite (hence potential to recover scheelite *via* magnetic wolframite).
• The ore mineral molybdenite occurs as discrete flakes.
• Major high SG (5.0) gangue minerals are pyrite and marcasite (FeS_2) which are hydrophobic and only magnetic when heated. These minerals are a significant issue for the flotation process and the gravity separation process but unlikely to be an issue in the magnetic separation process.
• Minor high SG (7.0) gangue minerals are bismuthinite (Bi_2S_3) and bismuth which are hydrophobic and non-magnetic. These minerals may be a significant issue for the flotation and the gravity separation process.
• Non-sulphide gangue minerals include quartz which has a low SG of 2.7 and is non magnetic. If liberated from the valuable minerals, the quartz should pose no major metallurgical recovery issue.

• Another minor gangue mineral of interest is siderite (SG of 4.0), non-sulphide but magnetic.

13.2.2.2 Head Assay Analysis

• The overall grade of the master composite was 1,990ppm tungsten and 720ppm molybdenum.
• The greisen is significantly higher grade than the pipe or granite material.
• The pipe and granite samples are half the expected mine resource grade.
• The tungsten calculated values for the pipe sample of 795ppm, 970ppm and 1,314ppm, rather than the sole assay grade of 390ppm, are likely to be the most accurate representation of its head assay grade.
• Bismuth levels are significant, if they are concentrated into the molybdenum concentrate.

13.2.2.3 Size by Size Assay Analysis

• For the coarse size by size analysis where the three variability composites were crushed to 100% passing 3.35mm, all had around 80% of the tungsten, molybdenum and bismuth in 70% of the mass in the -2mm fraction.
• Similarly for all three composites, the -1mm fraction contained some 50% of the tungsten, molybdenum and bismuth in 35% of the mass.
• There were no significant opportunities for scalping of either a high grade concentrate or a low grade waste stream based on the size data available.

13.2.2.4 Comminution

For the master composite:

• The abrasion index of 0.3768 is classified as average. The indications are that moderate liner and steel ball wear can be expected.
• The rod mill work indices are low at 12.0kW/t.
• The ball mill work indices are above average at 18.7kW/t.

13.2.2.5 Gravity (tabling) Testwork - Wolfram Recovery

- Highest grade concentrate obtained from the -1.00 mm grind in a single pass was 4% W at a recovery of 59%.
- Highest grade concentrate obtained from the -0.25 mm grind in a single pass was 13% W at a recovery of 34%.
- Mass recovery was 3% and 0.5% for the -1.00 mm and the -0.25 mm grind respectively, at the above grades.
- A grind size of -1.00 mm has a significant negative effect on the liberation of the tungsten minerals.
- From these tests, the grind size of -0.25 mm was selected as the grind size for all other tabling testwork.

13.2.2.6 Tabling and Magnetic Separation Testwork - Wolfram Recovery

• Sequential tabling and magnetic separation testwork on a master composite sample produced a concentrate with a tungsten grade of 29.6% W (or 37.3% WO_3) at a recovery of 42%.
• Similarly on a greisen composite sample the same separation produced a concentrate with a tungsten grade of 26.9% W (or 33.9% WO_3) at a recovery of 25.6%.
• Mineralogical investigation of the master composite concentrate attributed the low grade to contamination by the gangue mineral siderite. Siderite is strongly magnetic.
• There were high wolfram losses (approximately 20%) to the gravity/magnetic tails. Size analysis on the master composite gravity tail indicates that the wolfram is located in the slimes fraction (minus 25 microns) and hence lost to table tails.

13.2.2.7 Knelson Concentration and Magnetic Separation - Wolfram Recovery

• Knelson concentration and subsequent magnetic separation testwork on a greisen composite sample produced a concentrate with a tungsten grade of 14.6% W (or 18.4% WO_3) at a recovery of 42%. These results are poorer than the previous gravity/magnetic testwork results.

13.2.2.8 Flotation Testwork - Molybdenum Recovery

• Flotation of master composite samples at two different grind sizes indicated that there is no benefit to rougher performance in floating at a finer grind size than P80 150 um. Hence, further rougher flotation testwork was conducted at a P80 150um during Stage 1.
• Cleaning of the master composite rougher concentrate achieved the target grade (50% Mo) at the target recovery (80.9%).
• Flotation of the gravity/magnetic tails samples for the master composite and the greisen composite achieved the targeted molybdenum grade (49% and 52% respectively) but at lower than target recoveries of 68% and 75%, respectively.
• There was evidence from the sulphur assays that the samples underwent oxidation in the gravity/magnetic tails samples prior to the flotation testwork as the samples were initially oven dried then re-pulped.

13.2.3 Stage 2 Testwork Summary

13.2.3.1 Review

The objectives of the Stage 2 testwork and whether they were met are reviewed below.

- *Establish that a representative sample of ore has been tested through the proposed flowsheet to confirm the design*

Sample selection was handled entirely by QOL and no documentary evidence of sample location or representativity was supplied. A number of different samples have been supplied to feed the testwork programme. However this has been based on the need to supply enough mass to feed the unit operations. Representivity of samples has not been established at this point.

- *Complete variability testing of the unit operations based on available samples*

Variability testing on greisen and quartz pipe samples was completed and showed similar behaviour to the master composite although with a slightly higher ball work index. No implications are apparent for the comminution circuit design which will comprise closed circuit crushing and closed circuit ball milling.

While some limited variability testing has been undertaken on both flotation (master composite, quartz pipe, granite and greisen) it is unknown whether this forms a representative cross section of the ore body types which will be encountered.

- *Confirm recoveries and grades for the molybdenum flotation circuit*

Flotation of whole ore on both quartz pipe and greisen/quartz pipe samples achieved the target grade of above 50% molybdenum. The float reagents have been established as reported herein but dosages for full scale treatment need to be confirmed. However, batch flotation testing in Stage 2 gave significantly lower recoveries than expected from Stage 1 with the quartz pipe sample reporting 67% molybdenum recovery and the greisen/quartz sample reporting 45 to 55% molybdenum recovery. Depression of bismuth in both cases was effective with 86 - 96% of bismuth reporting to cleaner tail.

These samples were obtained as bulk 10t samples and stored on the surface for up to 10 years. Subsequent MLA analysis of cleaner tails showed the minerals to be highly oxidised with copper and lead sulphates, copper oxide and iron arsenate. As a result, a fresh sample was obtained from QOL to investigate the lower molybdenum recovery further. The fresh sample was used to complete locked cycle testing of the molybdenum flotation regime and gave very good results as shown below (Table 13-1):

Table 13-1. Locked Cycle Flotation Tests - Cycles 4, 5 and 6 Metallurgical Balance

Product	Wt %	Mo %	Mo Dist %	S %	S Dist %	W ppm	W Dist ppm	Bi %	Bi Dist %
Mo Cleaner Con	0.91	53.5	94.17	37.1	75.61	2433	0.06	1.32	6.64
Mo Rougher Tail	1.14	0.93	2.05	8.70	22.20	28001	0.84	13.1	82.55
Scavenger Tail	97.95	0.02	3.78	0.01	2.19	38405	99.10	0.02	10.81
Calculated Head	100.00	0.52	100.00	0.45	100.0 0	37959	100.00	0.18	100.00
Assay Head		0.508		0.46		45800		0.192	

Batch flotation testing on this locked cycle sample gave molybdenum recoveries between 60 - 80% depending on the collector used. Whilst the locked cycle test has given good results, the poorer batch test results are cause for some concern and further batch and variability work is recommended to establish the cause of this poorer performance and to ensure that target grades for molybdenum can be reached for all ore types expected to form mill feed.

- *Develop a gravity based flowsheet for production of a wolfram concentrate*

Stage 1 testwork was focused on a tabling and gravity separation flowsheet for recovery of wolfram. This flowsheet gave poor recovery of tungsten and failed to make a saleable grade of concentrate (>65% WO_3). Poor recovery was attributed to loss of wolfram to slime tail on the table. To counter this, stage 2 testwork focused on the use of a centrifugal concentrator (the Kelsey jig) to maximise fine wolfram recovery in a roughing stage.

This was followed by investigations into magnetic separation, tabling and cleaner jigging. Rougher results indicated that tungsten recoveries of up to 96% to a concentrate mass of 8% could be achieved. These results are summarised in Table 13-2.

Table 13-2. GR/QP composite KCJ Rougher - Run 2

Test	Spin (Hz)	Tails Water (lpm)	Solids			W Assay (%)			Con Wt (%)	Con W Dist'n (%)
			Feed kg/h	Con kg	Tail kg	Con	Tail	Calc'd Feed		
1	40	12	43.9	0.061	0.670	0.663	0.004	0.059	8.39	93.82
2	40	12	44.6	0.060	0.683	0.699	0.005	0.061	8.11	92.51
3	40	12	43.9	0.063	0.669	0.681	0.002	0.060	8.57	96.96
4	40	12	39.9	0.060	0.605	0.672	0.004	0.064	8.97	94.30
5	40	12	34.9	0.058	0.520	0.642	0.004	0.068	9.98	94.68
6	40	12	30.7	0.060	0.452	0.482	0.006	0.062	11.76	91.46

Tabling of the rougher jig concentrate achieved grades of 45% WO_3 but at very low recovery. Selective magnetic separation on hutch concentrate was able to achieve very high grades (68% WO_3) but also at very low recoveries. At this point a separate approach to treatment of coarse and fine concentrate was adopted. The rougher concentrate was screened at 150 micron with the coarse fraction treated on a table and the fine fraction subjected to further cleaner jig processing (Table 13-3).

Table 13-3. KCJ Run 5 and 6 Summary

Sample Identity	Wt % Ret	Assay Data %					
		WO_3	Fe_2O_3	Mo	S	Bi	SiO_2
KCJ Run 5 Combined KCJ Concentrate + Hutch Concentrate 150 µm Screening							
+150µm KCJ Con	6.83	27.90	12.40	0.086	0.34	0.172	51.3
-150µm KCJ Con	93.17	2.72	2.46	0.017	0.04	0.015	88.1
Total	**100.00**						
KCJ Run 5 Concentrate +150 µm Wet Table Separation							
Cleaner Table Con	***39.40***	***68.0***	***20.3***	***0.155***	***0.52***	***0.321***	***1.31***
Cleaner Table Mids	12.92	6.58	27.3	0.078	1.12	0.071	47.8
Cleaner Table Tails	17.95	0.732	1.38	0.008	0.06	0.006	93.0
Rougher Table Tails	29.73	0.720	0.88	0.013	0.06	0.005	93.8
Total	**100.00**						
KCJ Cleaner Run 6 - KCJ Run 5 Concentrate -150 µm							
Cleaner KCJ Con	***3.27***	***46.3***	***29.5***	***0.148***	***0.29***	***0.127***	***14.9***
Cleaner KCJ Hutch Conc	***0.92***	***39.0***	***36.1***	***0.111***	***0.54***	***0.138***	***15.3***
Cleaner KCJ Tail	95.81	0.48	1.81	0.006	0.03	0.006	91.1
Cleaner KCJ Con (Rpt)		0-09	2.2	0.005	0.03	<0.01	91.2
Total	**100.00**						

Source: Appendix 1 - AMMTEC Report No. A10020, Appendix XXII

The fine jig concentrate was subsequently upgraded to 52% WO_3 across a fine table at 85% stage recovery. The testwork established that a gravity-based flowsheet can be used to produce a wolfram concentrate; however maintaining saleable grade of >65% WO_3 from the fine stream is likely to prove problematic. The tests also showed that sliming perhaps as a consequence of over grinding, had fine wolfram short circuiting to tails in both jigging and wet tabling.

Further testing to establish the cause of this and the nature of the wolfram affected should be undertaken. Magnetic separation of the gravity concentrates did not substantially improve product quality and should not be included in the final plant design.

- *Establish that a saleable grade of wolfram concentrate can be produced*

The results above indicate that a saleable wolfram concentrate (>65% WO_3) can be produced based on the testwork completed on the samples supplied. However, the testwork shows that it will be substantially more difficult to produce a high grade concentrate from the finer fractions of the rougher jig concentrate stream, and grade control will need to rely on improving the concentrate with material from the coarser stream.

- *Confirm recovery for wolfram gravity circuit*

The gravity flowsheet involves a number of unit processes each of which contributes to losses in wolfram recovery. Much of the gravity testwork has been completed on a batch scale so predicting circuit recovery from this data is not possible. In addition, few of the tests have been repeated for reproducibility nor have significant variability tests been done on differing samples. However, using the batch data at its best and disregarding losses due to variable grade or gains due to recycling of intermediate streams, the following observations are made (Table 13-4**)**.

Table 13-4. Observations on Recovery for the Wolfram Gravity Circuit

Stage	Recovery (%)	Source
Wolfram recovery to flotation tail	99	Locked cycle test GS2798
Wolfram recovery to deslime underflow	99	Roche estimate
Wolfram recovery to rougher jig conc	94.5	Rougher jig run 5
Wolfram recovery to coarse conc table	95.7	Run 5 coarse table test (7% of rougher jig conc mass)
Wolfram recovery to deslime underflow	99	Roche estimate
Wolfram recovery to cleaner jig conc	80	Cleaner jig Run 6 (93% of rougher jig conc mass)
Wolfram recovery to fine conc table	85	Run 6 fine tabling test
Overall circuit recovery	**60**	**Product of stage recoveries**

- *Provide key metallurgical design data to allow the equipment sizing to be confirmed*

At this point, sufficient basic data existed for design of the comminution and flotation circuits although additional variability testing on the flotation circuit was recommended.

The complete flowsheet from grinding through to final tabling of wolfram concentrate had been tested once via jig run 5 and 6 combined. This consisted of a series of batch tests run in series. Locked cycle flotation was conducted once on a relatively higher grade sample of ore supplied by QOL. The impact of recirculating loads and intermediate products on the circuit mass balance was not established. To quantify this for the purpose of gravity plant design, Roche completed a mass balance using in-house data.

In short, a substantial part of the Wolfram Camp design is based on batch testing of limited ore types together with vendor assumptions.

13.2.3.2 Sample Selection

Stage 2 testwork was completed sequentially on a series of different samples. These samples and the testwork completed are described in Table 13-5.

Table 13-5. Metallurgical Sample Summary

Test Series	Sample	Testing
A	Marketing Composite	• Gravity (Tabling) • Magnetic Separation • Rougher / Cleaner Flotation •Mineralogy - Mineral Liberation Analysis • Thickening (Outotec)
B	Greisen composite and quartz pipe composite	• Comminution (JK Tech) • Heavy Liquid Separation
C	Greisen / quartz pipe combined composite	• Bulk Flotation • KCJ Gravity Separation • Wet Tabling and Panning • Magnetic Separation
D	Quartz pipe (high grade) composite	• Bulk Flotation • KCJ Gravity Separation • Wet Tabling • Magnetic Separation
E	Locked cycle composite	• Flotation
F	Synthetic wolfram feed	• KCJ Gravity Separation • Magnetic Separation • WHIMS

		• Wet Tabling
G	Locked cycle tailings	• KCJ Gravity Separation • Wet Tabling • Magnetic Separation
H	Combined wolfram concentrates Molybdenite concentrates (greisen/quartz pipe composite)	• Electrostatic Separation • Bulk Density Determinations • Mineralogy – QEMSCAN

Some of this testwork was based on recombining products from other tests after assaying and also creating synthetic samples by adding high grade material to lower grade tails. This approach, while not ideal, was necessitated by:

- Lack of available material at the commencement of the project.
- The low grade of the feed which required a large mass of material to generate sufficient concentrate for further processing testwork.
- The requirement to obtain design data as quickly as possible to support the parallel plant design activity.

All sample selection was completed by QOL; however, no documentation on sample origin has been provided.

13.2.3.3 Mineralogy Analysis

Stage 1 testwork examined the mineralogy of three ore types after concentrating the value minerals by screening and heavy liquid separation. By contrast, Stage 2 mineralogical analysis was completed on specific process fractions to determine the minerals present and their liberation.

Mineralogical Liberation Analysis (MLA) was carried out on gravity and flotation samples from the marketing composite testwork. QEMSCAN Analysis looked at a molybdenum cleaner concentrate from the greisen/quartz pipe composite.

MLA Analysis

• For all samples, molybdenite (MoS_2) was the only molybdenum bearing mineral present.
• Remaining molybdenite in the flotation tails was predominantly liberated at the flotation grind size of 250 microns.
• Tungsten minerals present in the final flotation tails included wolframite, scheelite ($CaWO_4$) and raspite ($PbWO_4$). The dominant mineral in the rougher flotation tails was wolframite and raspite in the cleaner flotation tails.

• The major tungsten mineral in the gravity/magnetic products was wolframite with scheelite present to a lesser extent and no significant raspite.
• Other minerals present, largely as contaminants included siderite, pyrite, quartz and other silicates. Bismuth as bismuthinite was mostly liberated but locked as bursaite.

QEMSCAN Analysis

The concentrate sample was separated using a hand-magnet and the "mags" and non-mags analysed.

• The magnetic fraction was predominantly magnetite with accessory pyrrhotite, pyrite and ferromagnesian silicates.
• Molybdenum was dominant in the non-magnetics as liberated flakes mostly in the 20 -
30 μm size range.

13.2.3.4 Head Assay Analysis

• The marketing composite had an average overall grade of 2,425ppm W and 732ppm Mo which compares with 1,990ppm W and 720ppm Mo for the master composite in the Stage 1 Testwork.
• All composites were lower in head grade than the target value for feed grades of 0.4% W and 0.3% Mo. Head grades ranged from 20 - 50% of these values.
• Bismuth levels are significant, if concentrated into the molybdenum concentrate.

13.2.3.5 Size by Size Assay Analysis and Heavy Liquid Separation

• Samples of the greisen and quartz pipe composites were coarse crushed to -12.5 mm and screened at 6.3 mm, 2.0 mm and 0.5 mm for heavy liquid separation. The -0.5 mm fraction was assayed but not separated.
• Size by size analysis for the greisen composite showed 83% of the molybdenum in the coarse +6.3 mm fraction and a significant percentage of the wolfram (47%) in the 0.5 mm fines.
• Size by size analysis for the quartz pipe composite saw an even distribution of wolfram and molybdenum in the size fractions with no significant concentration.
• Heavy liquid separation of the greisen composite size fractions had the highest weight recovery to sinks of some 49% at an SG of 2.8 corresponding to the highest molybdenum recovery of 82%. The major wolfram deportment was in the -0.5mm fines at 47%. This fraction was not separated in TBE.
• For the quartz pipe composite the highest deportments were also in the 0.5mm fines at nearly 40% of the wolfram and 19% of the molybdenum. The next highest wolfram and molybdenum deportments were in the SG 3.0 sinks for the three size fractions.

13.2.3.6 Comminution

• JK SAG mill comminution tests determined the SMC Drop Weight Index (Dwi) at 2.6 for the greisen ore and 2.7 for the quartz pipe which indicates a low competency ore.
• The results for the greisen and quartz pipe sample were similar to the master composite from the Stage 1 Testwork.
• The respective abrasion indices at 0.4165g and 0.3218g were classified as average, indicating that moderate liner and steel ball wear can be expected.
• Similarly, the rod mill work indices at 11.9kWh/t and 12.9kWh/t were low and consistent with the values from the Stage 1 testwork.
• The ball mill work indices at 19.3kWh/t and 21.7kWh/t are higher than in the previous testwork. These results were based on a finer closing screen size of 70 micron. A previous test using a product screen size of 242 µm gave a work index of 14.2kWh/t.

13.2.3.7 Flotation Testwork - Molybdenum Recovery

Marketing Composite

• Bulk flotation of the gravity middlings and tail for the marketing composite, after grinding to a P80 of 250µm, resulted in a grade of 45.6% Mo at a recovery of 73.3% after one stage of cleaning. Two cleaning stages improved the grade to 47.7% Mo but in open circuit, led to a recovery drop to 71.4%.

• Float conditions were 34% solids (w/w) using diesel (collector), NaHS (bismuth depressant) and IF50 (frother).

Greisen/Quartz Pipe Composite

• Bulk flotation of the greisen/quartz pipe composite was ahead of gravity separation as proposed for the final plant. The float tail was fed to the Kelsey Jig and wet tabled.
• Concentrates from a two-stage rougher float recovered 85% of the molybdenum but with a grade of less than 10% Mo. Bismuth floated with the molybdenum, recovering 84% at a grade just under 7% Bi.
• Subsequent three stage cleaner flotation saw the molybdenum grade improve to the required 50% Mo but with recovery dropping to 45% in the batched test. NaHS depressed the bismuth and the recovery to the cleaner tail was 86% albeit at only 7% bismuth.
• As with the marketing composite, float conditions were at 34% solids (w/w) and used diesel, NaHS and IF50 frother. Rougher flotation used air for bubble formation and nitrogen in the cleaners to suppress oxidation of the sulphur minerals.

Quartz Pipe Composite

• Flotation of the quartz pipe composite applied the same conditions as the greisen/quartz composite with diesel and PAX as collectors, NaHS as depressant for bismuth with lime used for pH adjustment and IF50 was the frother. Float density was 34% solids (w/w), the rougher float time totalled 50 minutes, with 10,7 and 4 minutes for the three cleaners. Conditioning time was 4 - 5 mins.
• The mass split to tails in the rougher float was 99%, the concentrate recovering 90% of the molybdenum at a grade of 4.3% molybdenum. The bismuth response to the rougher concentrate was only 57% recovery at a grade of nearly 5% Bi.
• Cleaner flotation resulted in the 3rd concentrate achieving the 50% molybdenum grade but with recovery decreasing to 66.5%. Bismuth recovery to the cleaner tails was 96% at a 5.7% Bi grade.

Locked Cycle Composite

• To better simulate the final plant conditions, locked cycle flotation of a molybdenum/wolfram ore (Ex-QOL) was undertaken.
• The circuit configuration included open circuit rougher/scavenging and two stages of molybdenum cleaning with the second cleaner tail returned to the first cleaner feed.
• Two sighter tests established the float conditions, namely 34% solids (w/w), diesel and PAX as the collectors, W55 as the frother and one bulk addition of NaHS in the molybdenum rougher.

• Results were excellent surpassing process targets with a 94% recovery of molybdenum and an 83% recovery for bismuth to the molybdenum rougher tail. Grades were 53.5% molybdenum and 13% bismuth.

13.2.3.8 Gravity and Magnetic Separation Testwork - Wolfram Recovery

Marketing Composite

• Sequential tabling and magnetic separation testwork on the marketing composite produced concentrate with tungsten grades for 3 size fractions as follows:

- +250 µm - 32.9% W (41.5% WO_3) at a recovery of 44.7%
- +106 µm - 50.4% W (63.6% WO_3) at a recovery of 86.0%
- -106 µm - 31.4% W (39.6% WO_3) at a recovery of 73.1%.

• Panning these concentrates improved grades to 47.1% W, 53.8% W and 52.0% W respectively but pan tails still contained significant wolfram.
• It was initially proposed to have flotation follow gravity separation in the flowsheet and this is how this section of testwork was done. It was subsequently decided that flotation would precede gravity separation as this improved molybdenum concentrate grade and recovery. In addition, by removing all sulphides *via* a bulk float, the gravity separation process for wolfram was improved.

Greisen/Quartz Pipe Composite

• Bulk float tails were cyclone deslimed at 10µm ahead of gravity concentration using a Kelsey Centrifugal Jig (KCJ). High SG minerals were concentrated in the cyclone underflow. The underflow accounted for a 97% mass split containing 96% of the wolfram. Cycloning was shown to be an effective concentration step and recommended for inclusion in the final plant design. Desliming also had the benefit of improving the efficiency of the jig.
• KCJ Run 1 tested a range of jig conditions and showed recovery and grade to improve with higher feedrate, higher wash water rate and higher centrifugal force.
• KCJ Run 2 under the optimum conditions from Run 1, achieved 90 - 96% recoveries at grades of 0.65 - 0.70% wolfram for all six runs.
• Size by size analysis of the concentrates and the jig tails showed the wolfram distribution in line with the weight splits.
• Tabling the jig concentrate and magnetically separating the first and second cleaner concentrates recovered 90% of the wolfram to the non-mags but at a grade of only 12% tungsten. Panning improved the grade to 44% at an 86% recovery. Further magnetic concentration saw the non-mags assay 49% tungsten and recovery increase to 96%.

Quartz Pipe Composite

• Bulk float tails were cyclone deslimed at 10µm ahead of gravity concentration. Cyclone desliming again concentrated wolfram to the underflow containing 96% of the wolfram with a similar mass split.
• The combined jig product concentrate and hutch concentrate from the rougher KCJ (Run 3) recovered 80% of the wolfram at a grade of 4.5% tungsten.
• Cleaning the combined rougher concentrate through the KCJ had the cleaner concentrate recovery at over 97% but the grade dropping to 2.7% wolfram.
• Wet screening the cleaner concentrate at 106µm and wet tabling the undersize had the table concentrate grade at 45% W and recovery at 52%. Wolfram slimes again reported to the table tails indicating over grinding.
• The +106µm was magnetically separated in two stages (low intensity to remove quartz followed by high intensity to separate siderite). Results indicated 80% by weight removed as the 500G magnetic fraction but only 50% of the wolfram was recovered in the final product as the 4500G magnetic fraction. Consequently 47% of the wolfram remained in the 7000G non-mags indicating contamination with magnetic siderite.

Synthetic Feed Composite

• This sample was constructed from a blend of high grade concentrate and previous jig run tails. No desliming was therefore carried out as the tails had been deslimed already.
• The bulk test results for KCJ run 4 show the overall recovery of wolfram to be 76.8% for the combined concentrate and hutch concentrate. However, the calculated grade of the total concentrate (concentrate+ hutch

con) was only 6% W. Whilst this gives a good upgrade ratio of 14:1, additional cleaning would be required to achieve a saleable grade of greater than 65% WO_3.

• Cleaning the combined concentrate by dry magnetic separation gave high grade (67% WO_3) at low recovery (61%). Cleaning the combined concentrate by wet magnetic separation (WHIMS) gave lower grade (30% WO_3) at higher recovery (78%).

• Wet tabling of the +125 µm mags fraction from WHIMS testing improved the concentrate grade to 73% WO_3 with only 5% loss in recovery. However the +125 µm mags fraction was less than 10% of the mass.

• Wet tabling of the 7000G non mags produced a grade of 16% WO_3 at a recovery of 67%. The upgrade ratio for wet tabling was 13:1 and the rejection of the gangue minerals of iron and silica was also significant. As this fraction forms 61% of the total WHIMS feed this would significantly reduce the overall concentrate grade and recovery.

• KCJ runs 5 and 6 were used to test a separate approach to treatment of coarse and fine concentrate. KCJ run 5 used recombined products from run 2 and 4 and was run as a bulk rougher jig test to produce a concentrate for further processing. Run 5 produced a wolfram recovery of 89% to a grade of 3.4% W. The rougher concentrate was screened at 150 µm with the coarse fraction treated on a table and the fine fraction subjected to further cleaner jig processing via jig run 6. Coarse concentrate achieved very high grades of 68% WO_3 at 95% stage recovery. The finer minus 150 µm stream was treated in the cleaner jig and reported a grade of 43% WO_3 at 80% stage recovery. This was subsequently upgraded to 52% WO_3 across a fine table at 85% stage recovery.

• KCJ Run 7 was carried out to try to improve the cleaner jig results achieved in KCJ Run 6. Small adjustments were made to stroke and bed depth and the feed sample was recombined from KCJ Run 6. Results of KCJ Run 7 indicate that whilst concentrate grade improved, recovery dropped significantly.

13.2.3.9 Electrostatic Separation

Combined Wolfram Concentrates Electrostatic separation of a combined tungsten concentrate was carried out to establish if uranium and thorium minerals could be concentrated or removed from the wolfram concentrate. Thorium minerals were concentrated in the non-conductor fraction with a 73% recovery. Uranium recovery was not as effective; while 46% reported to the non-conductors, some 40% remained in the conductor fraction. Wolfram remained in the conductors and mids. This work was commissioned by QOL and the target level of uranium and thorium in concentrate was not quantified.

13.2.3.10 Thickening

Marketing Composite

The unit thickener area or capacity requirement was between 1.0 and 1.5 t/m^2h although the latter gave poorer overflow clarities. Underflow density was above 60% solids (w/w) in the test and this would be expected to reach 65% at full scale.

13.2.4 JKTech Pty Ltd

The Julius Krutschnitt Minerals Research Centre (JK Tech), Brisbane, Australia undertook- drop weight test evaluation and mineral liberation analysis.

13.2.4.1 MLA analysis of Quartz Greisen and Quartz Pipe Samples – August 2006

- Queensland Ores Limited requested quantitative mineral analyses of a range of different size fractions from two W/Mo ores using the Mineral Liberation Analyser (MLA). The client requested occurrence and liberation data on tungsten and wolfram containing minerals (wolframite, scheelite and molybdenite). The ores, labelled quartz greisen and quartz pipe, assayed at ~0.4 - 3% W and 0.2-1.5% Mo. The client supplied 5 size fractions of each ore: -500/+250µm; -250/+125µm; -125/+75µm; - 75/+38µm and -38µm.

13.2.4.2 MLA analysis of Quartz Greisen and Quartz Granite Samples – August 2006

- Queensland Ores Limited requested quantitative mineral analyses of a range of different size fractions from two low grade W/Mo ores using the Mineral Liberation Analyser (MLA). The client requested occurrence and liberation data on tungsten and wolfram-containing minerals (wolframite, scheelite and molybdenite) and siderite. The ores, labelled quartz greisen and quartz granite, assayed at ~0.04 – 0.4% W and 0.03-0.1% Mo. The client supplied 5 size fractions of each ore: - 500/+250µm; -250/+125µm; -125/+75µm; -75/+38µm and -38µm.

13.2.4.3 SMC TEST - Samples from the Wolfram Camp Deposit – September 2006

- SAG Mill Comminution test (SMC) test data for two samples from the Wolfram Camp Deposit were received by JKTech from Ammtec Pty Limited on September, 2006, for data analysis. The samples were identified as GR sample and QP sample. The test results were forwarded to SMC Testing Pty Ltd for analysis. Analysis and reporting were completed on September 15, 2006.

13.2.4.4 MLA analysis - WC Gravity Separation Products – December 2006

- Queensland Ores Limited requested quantitative mineral analyses of a range of gravity separation products.

13.2.4.5 MLA analysis of WC Rougher and 1st Cleaner Tailings - February-2007

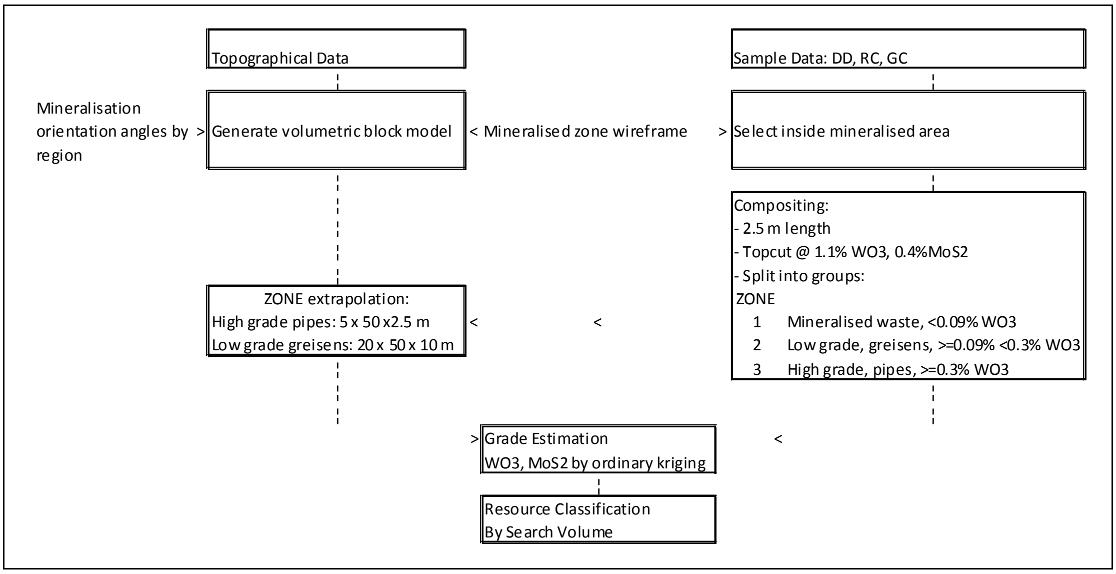
Queensland Ores Limited requested MLA quantitative mineralogical analyses of a Rougher Tailing and a 1st Cleaner Tailings. The client wished to obtain information on the overall mineralogy with specific emphasis on molybdenum mineralogy. The Rougher Tail contains ~0.04% Mo and the Cleaner Tailings ~1.1% Mo.

14 MINERAL RESOURCE ESTIMATES

14.1 General Methodology

An updated mineral resource estimation was completed, with an effective date of August 31st 2015, using a three-dimensional block modelling approach, with refinement of parameters by test work against a reconciliation model, built up on site with grade control data. The current updated model was generated using the application of CAE Datamine software. The on-site geological models are now also built up using Datamine software. The general methodology for the current update is described in the flowsheet in Figure 14-1.

Figure 14-1. Block Modelling Methodology



The modelling methodology is dominated by the extrapolation of three different types of zones within the principal overall mineralised zones, which correspond with the observed geology of pipes and greisens, as well as the grade ranges that have been encountered and applied during the last 2+years of mine production. These zones have been demarcated by the following grade ranges:

1. Mineralised waste, >=0.07 <0.09% WO_3
2. Low grade, greisens, >=0.09% <0.3% WO_3
3. High grade, pipes, >=0.3% WO_3

2.5m composites were generated from drillhole data, and then split into these zone groupings. The composites were then subsequently used for extrapolation of these zones within the framework of a volumetric block model. The orientation and scale parameters of this extrapolation have been derived from geological mapping at the mine, as well as observation of the historical mined workings. Grades of WO_3 and MoS2 have been estimated with ordinary kriging, using the extrapolated zones as hard boundaries.

14.2 Sample Database

A summary of the available sample data is shown in Table 14-1.

Table 14-1. Summary of Sample Database

					WO_3		MoS2	
Type of Sample	DTYPE	Holes	Length m	Average Length / Hole m	Samples	Holes With Samples	Samples	Holes With Samples
Diamond drillholes	DD	68	3,916	58	1,129	51	1,139	51
Reverse Circulation Holes	RC	351	14,586	42	9,984	348	9,980	348
Blasthole Grade Control Samples	GC	55,195	321,701	5.8	90,125	46,169	90,729	46,507
Blasthole exploration drilling	BEX	1,417	36,092	25	13,833	1,394	13,472	1,390

A plan of the exploration sample data, overlain on the current site contours, is shown in Figure 14-2. A long section showing the sample data is shown in Figure 14-3. As can be seen from these plots, the RC drilling is on approximately regular section lines spaced at approximately 20m apart. The section lines are oriented at approximately 135^{o} – 45^{o}, with most of the RC holes inclined at 60^{o} to the south-west.

The diamond drillholes are drilled more sporadically, mostly drilled along strike of the overall deposit, with holes generally inclined at approximately 50^{o} to the south-east. The grade control (GC) samples are generally taken as 2 x 2.5m samples over approximately 5m benches. The more recent blasthole exploration (BEX) samples are generally 25m long, on a 10m spaced drilling grid, and some in-fill of 42.5m.

Other elements assays available in the sample database include As, Bi, Sn and Fe. Lithologies logged in both the DD, RC and BEX data followed the coding system summarised in Table 14-2. Typical cross-sections showing lithology codes and WO_3 grades are shown in Figure 14-4 to Figure 14-8. The position of this section lines are depicted on the reference plan.

Table 14-2. Exploration Drillhole Lithology Codes

ICODE	GEOCODE	DESCRIPTION
1	GA	Altered Granite - 'green spot', argillite-sericite alteration.
2	GRm	Mica Greisen. Minor disseminated wolframite.
3	GRq	Quartz Greisen. Disseminated wolframite, commonly around quartz pipes.
4	Pq	Quartz Pipes. With lumps wolframite and molybenite.
5	FILL	Cavities/fill.
6	S	Hodkinson's Sediments
8	Ap, Flt	Other rocktypes, fault intersections

Figure 14-2. Plan of Exploration Sample Data

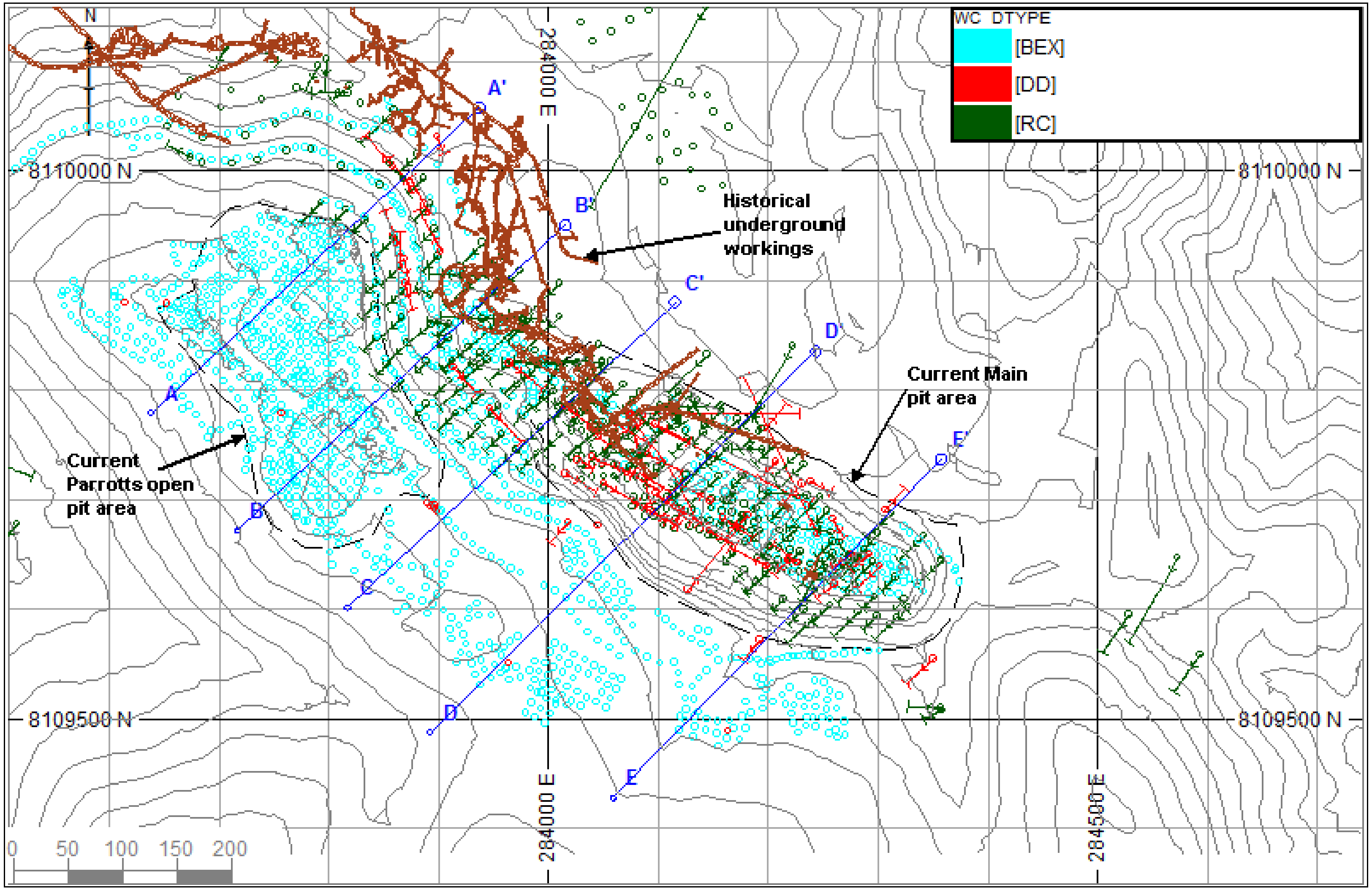


Figure 14-3. Long Section, Showing Exploration Sample Data

Azimuth 127°

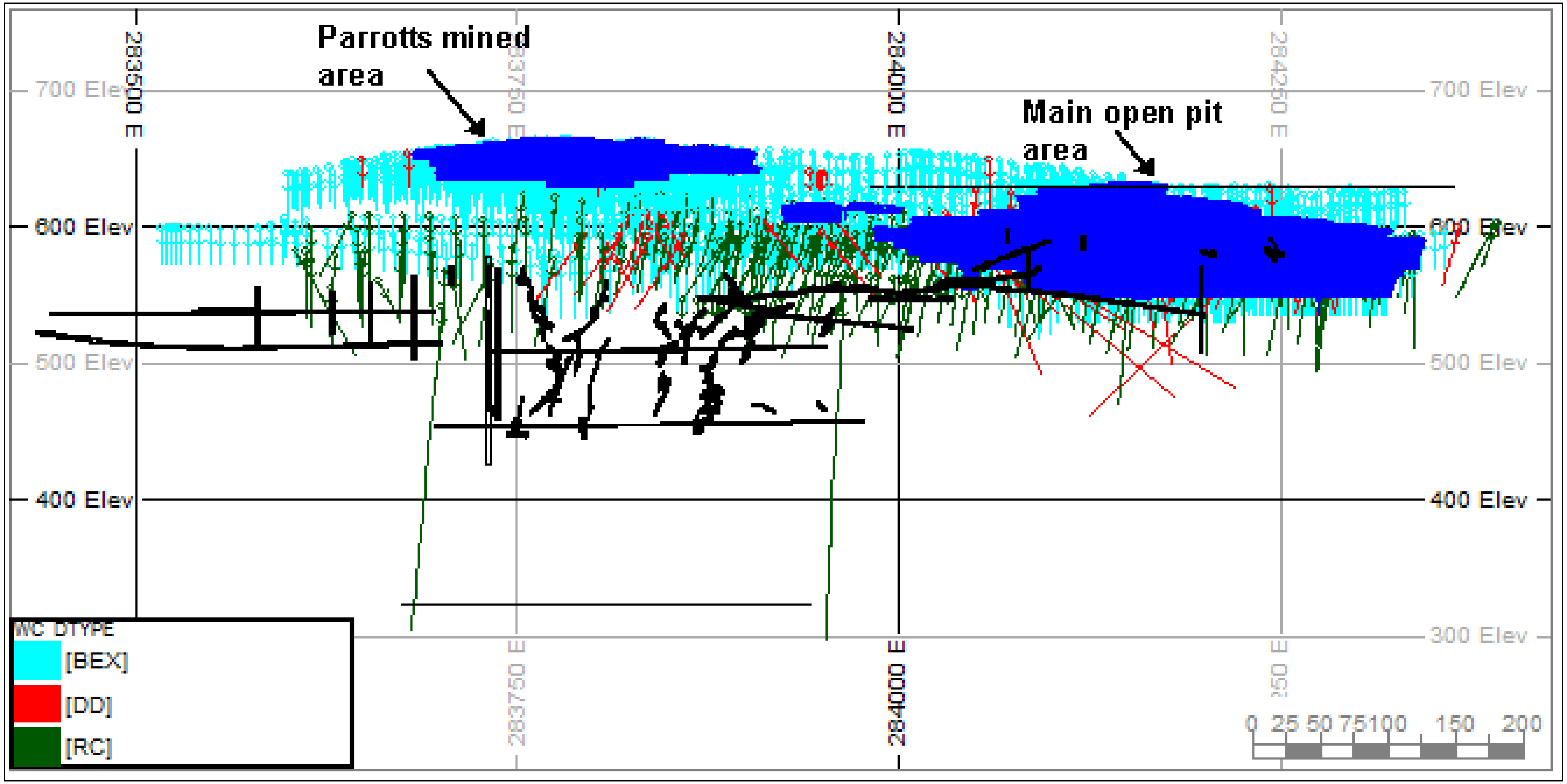


Figure 14-4. Section A-A' – Mineralised Zone Interpretation

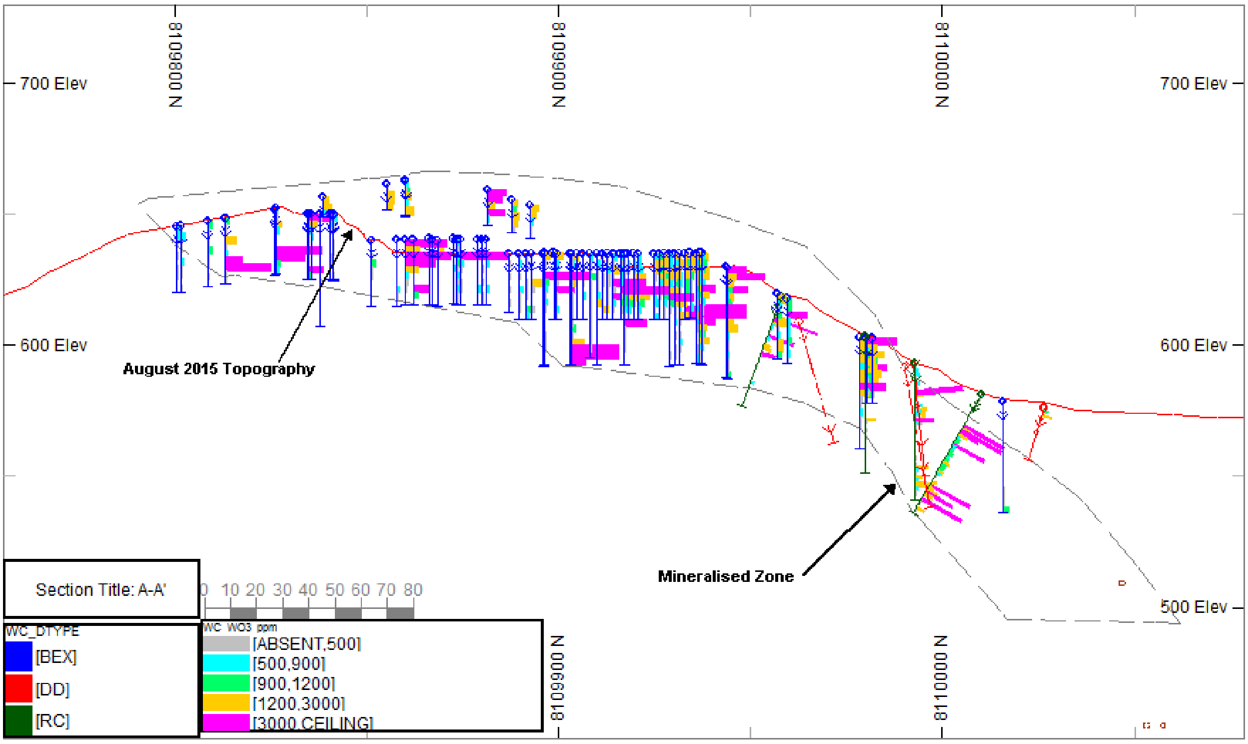


Figure 14-5. Section B-B' – Mineralised Zone Interpretation

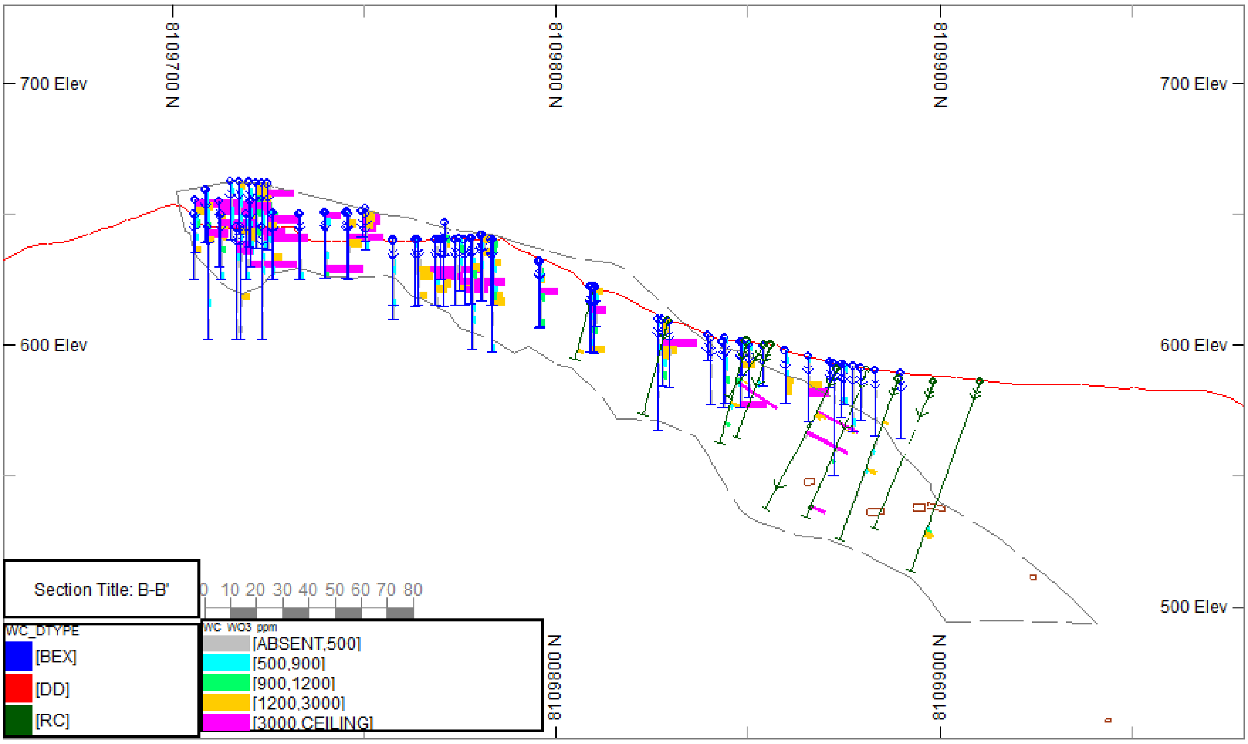


Figure 14-6. Section C-C' - Mineralised Zone Interpretation

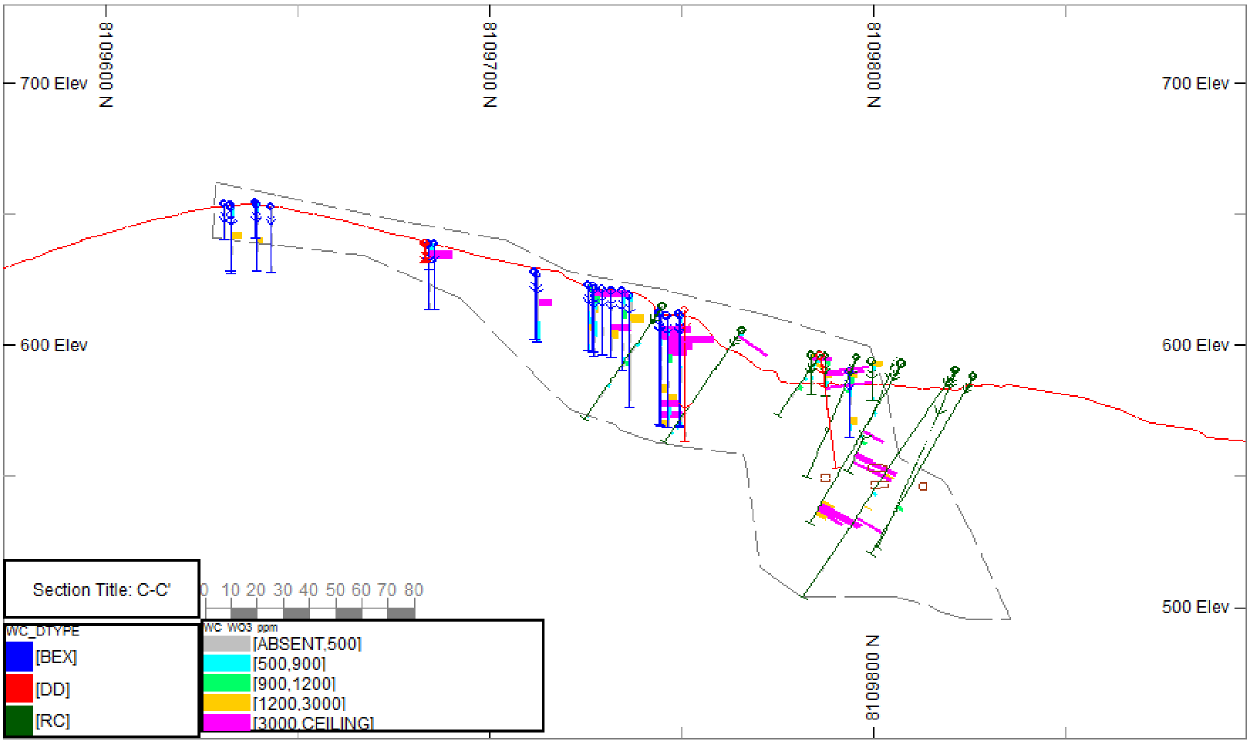


Figure 14-7. Section D-D' - Mineralised Zone Interpretation

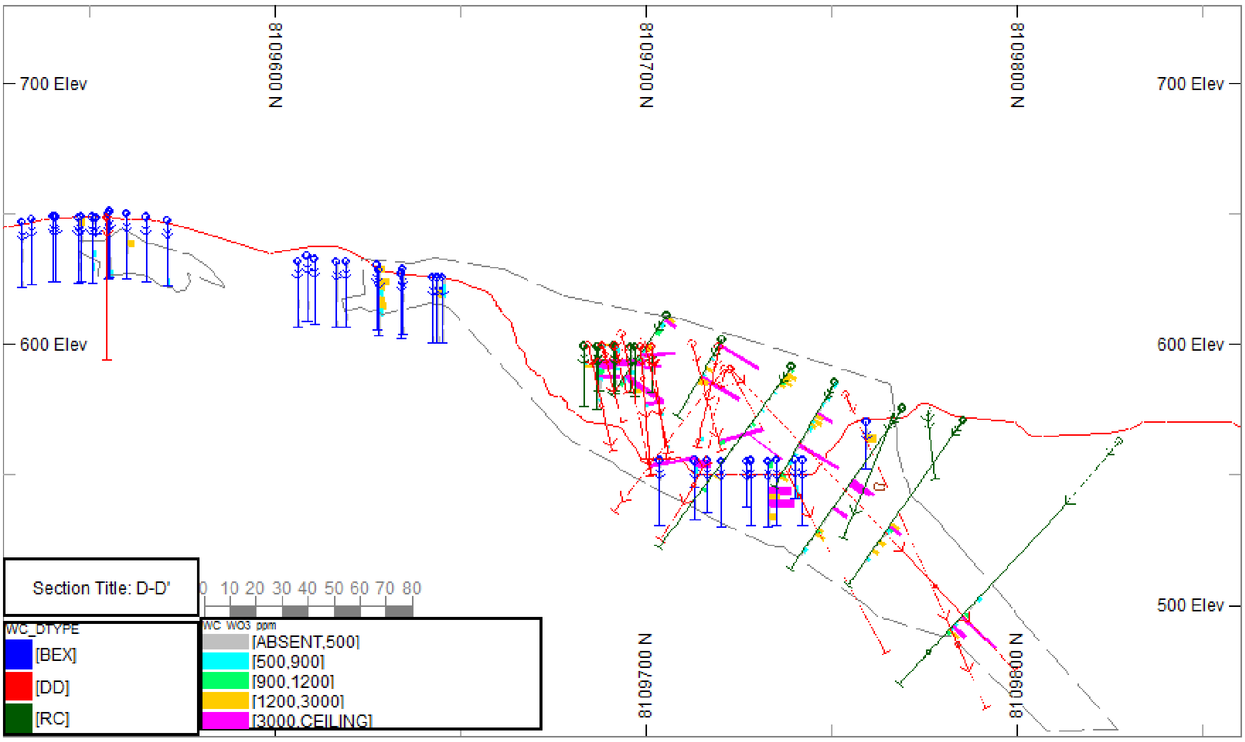
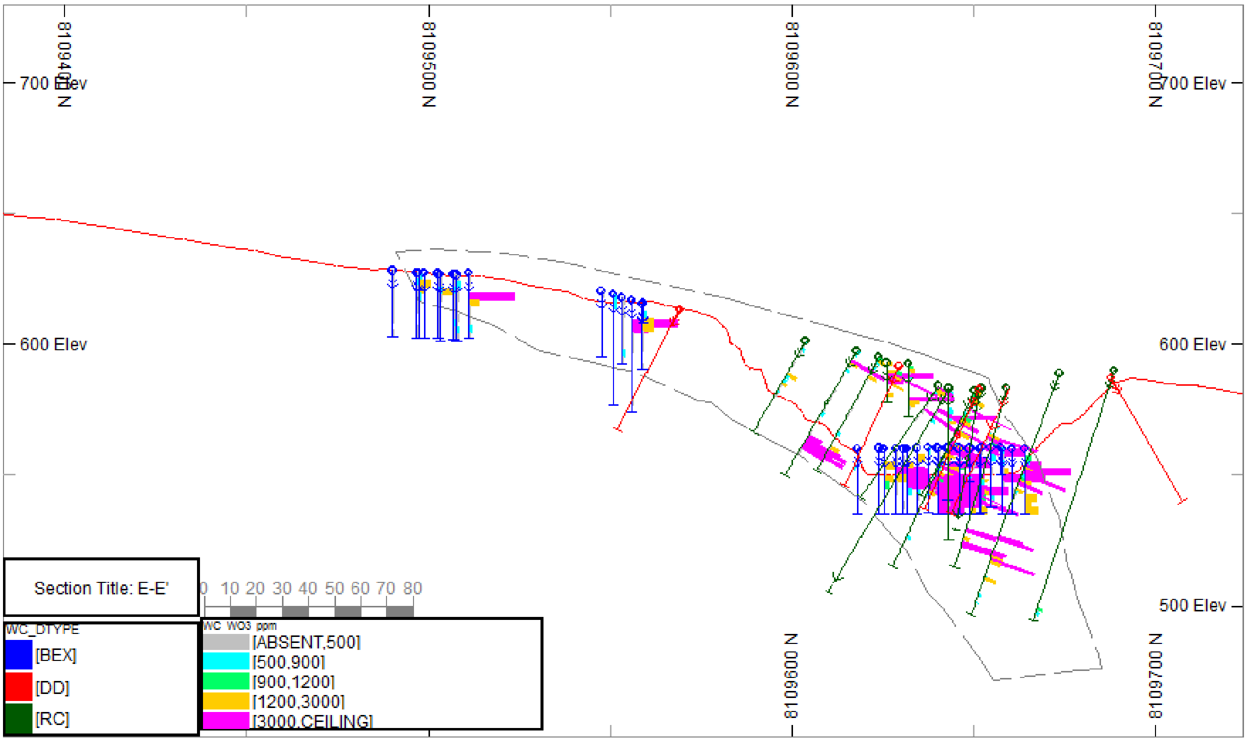


Figure 14-8. Section E-E' - Mineralised Zone Interpretation



In order to analyse the data available from blasthole GC samples, reconciled production results were compiled from Jun 2012 through till March 2014 (22 months), and compared with the data from the short-term planning block model, whose grades are derived predominantly from GC samples. These results are summarised in Table 14-3. In order to evaluate the GC model, the same sequential lower grade cut-offs were applied as used during the same chosen time period. These results, with a dilution and losses of 10% applied to the GC model, correspond very closely, with a difference in contained WO_3 metal content of less than 2% over the 22 months. These results support the use of the blasthole GC data in updated resource estimations.

Table 14-3. Comparison – Production Data With Reconciliation GC Model

Month	Mine Production Data						
	Low Grade WO3 Cut-Off %	Low Grade Tonnes Kt	Low Grade WO3 %	High Grade Tonnes Kt	High Grade WO3 %	Total Ore Tonnes Kt	Total Ore WO3 %
Jun-12	0.08	29.88	0.11	5.59	0.41	35	0.15
Jul-12	0.08	20.86	0.12	4.29	0.30	25	0.15
Aug-12	0.08	16.45	0.14	1.34	0.70	18	0.13
Sep-12	0.08	7.68	0.18	0.13	1.07	8	0.18
Oct-12	0.08	26.22	0.15	2.58	0.91	29	0.22
Nov-12	0.08	13.22	0.15	1.62	0.64	15	0.20
Dec-12	0.08	12.66	0.15	2.27	0.51	15	0.20
Jan-13	0.08	25.82	0.11	1.49	0.48	27	0.13
Feb-13	0.08	30.54	0.14	2.78	0.54	33	0.17
Mar-13	0.08	45.99	0.13	2.26	0.49	48	0.15
Apr-13	0.08	29.49	0.17	3.31	0.55	33	0.21
May-13	0.08	27.33	0.14	0.99	0.37	28	0.15
Jun-13	0.08	47.13	0.13	3.00	0.35	50	0.14
Jul-13	0.10	27.28	0.17	1.24	0.37	29	0.18
Aug-13	0.10	35.83	0.15	2.70	0.37	39	0.17
Sep-13	0.10	34.38	0.15	2.46	0.40	37	0.17
Oct-13	0.10	40.61	0.19	3.80	0.38	44	0.21
Nov-13	0.10	30.30	0.16	1.66	0.40	32	0.17
Dec-13	0.10	44.66	0.15	6.59	0.27	51	0.17
Jan-14	0.12	40.47	0.16	0.79	0.36	41	0.16
Feb-14	0.12	30.25	0.19	0.81	0.44	31	0.20
Mar-14	0.12	27.67	0.18	0.67	0.46	28	0.19
Total		**645**	**0.15**	**52**	**0.44**	**697**	**0.17**

		Total Ore Tonnes Kt	Total Ore WO3 %
In-Situ GC Model		**660**	**0.20**
After Application of			
Dilution	**10%**		
Losses	**10%**		
		653	**0.19**

14.3 Interpretation

An overall mineralised envelope has been defined at the mine, based on key lithological boundaries, the northern part of which is the main contact between the granites and Hodgkinson sediments. The mineralised zone defines an approximately 50m wide zones of greisen altered granites which hosts the bulk of the mineralised pipe structures. This is shown schematically in Figure 14-9 and as a mineralised wireframe model in a 3D view in Figure 14-10.

Figure 14-9. Schematic of Altered Granite Zone

(After Plimer, 1974)

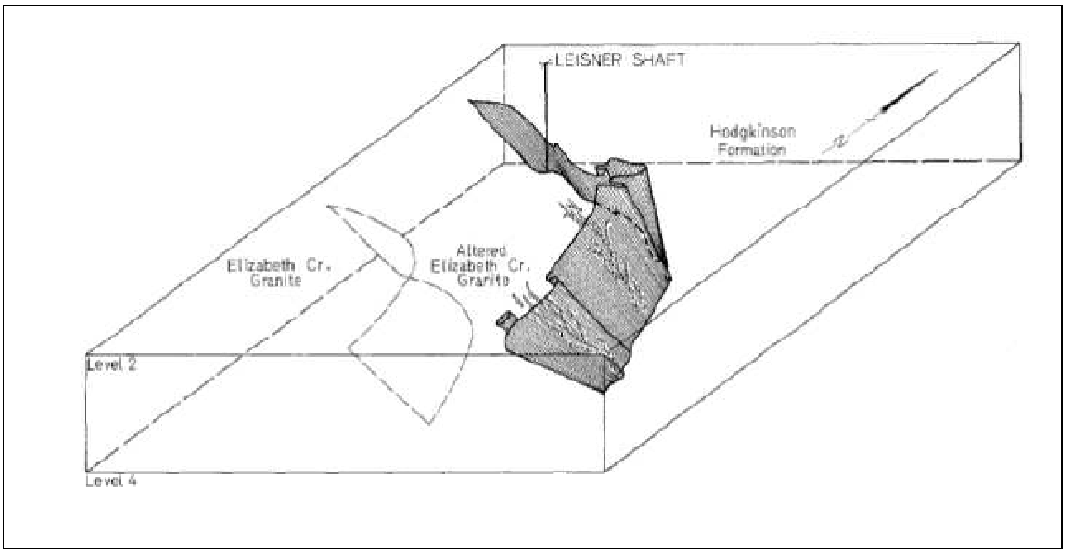
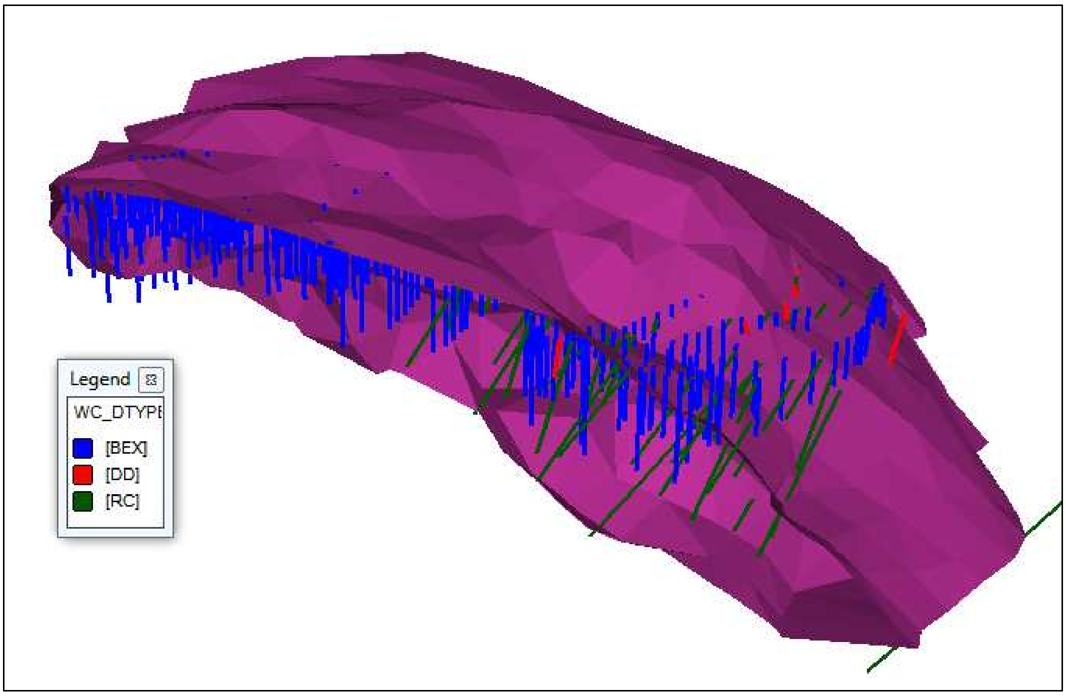


Figure 14-10. 3D View of Mineralised Zone Wireframe Model



This overall mineralised zone has generally been defined down to an elevation of approximately 500m RL. This does not represent the base of the deposit, more the approximate extent of available data. The historical underground workings extend down to an elevation of approximately 420m RL. Overall dimensions of the defined mineralised zone are summarised in Table 14-4.

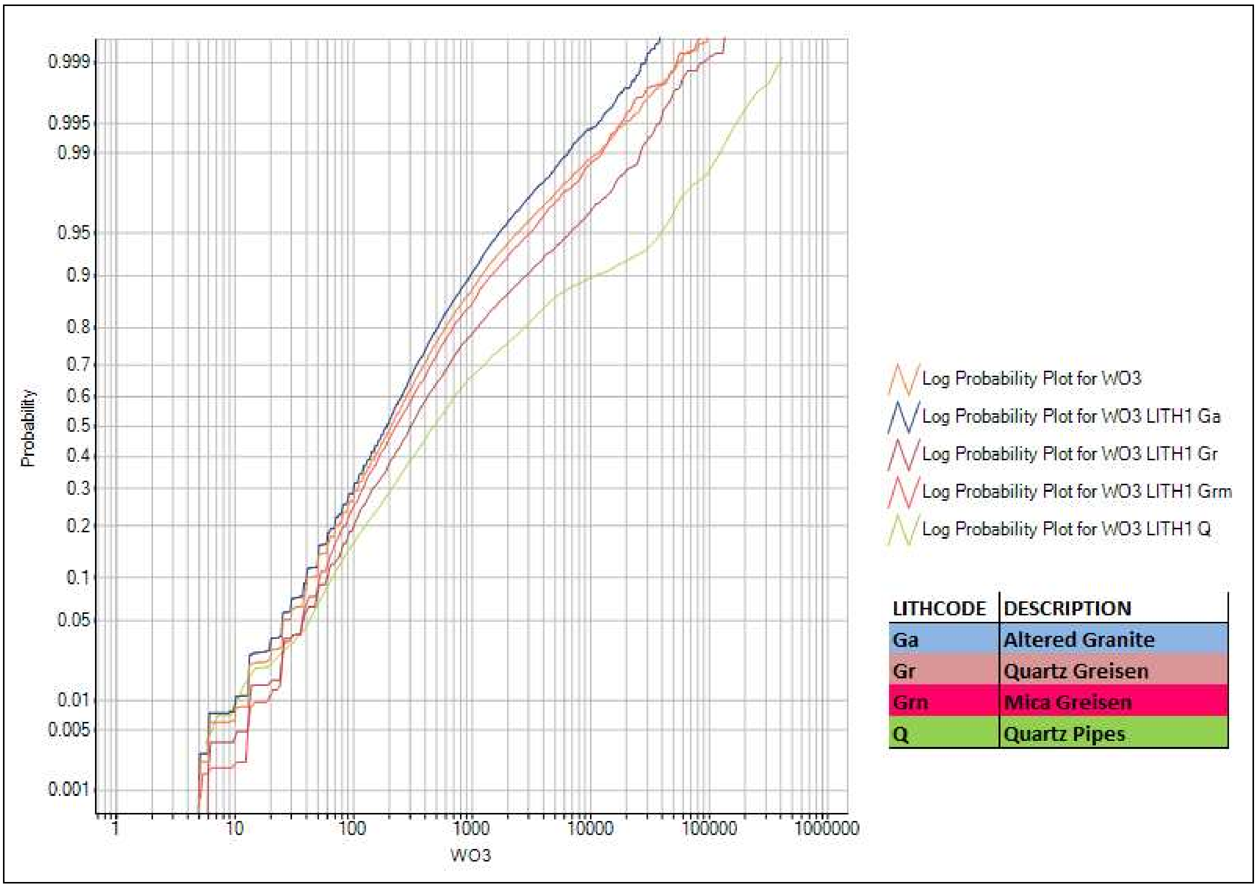
Table 14-4. Defined Mineralised Zone Dimensions

	Defined Base		Horizontal Width (m)		Dip Range (°)
Strike Length (m)	Elevation m RL	Max. depth m	Max	Average	
800	500	170	140	85	20 - 50

Most of the underground workings are located in, or very close to, the northern contact. Within the overall mineralised zone, there are very many pipe and greisen structures, which individually are far too complex to be individually interpreted purely from exploration drillhole (RC, DD or BEX) data. A log-probability of exploration drillhole WO_3 grades, split by principal logged lithology, is shown in Figure 14-11.

Figure 14-11. Log-Probability Plot – WO_3 By Lithology Coding

(WO_3 grades shown in ppm)



Observations of the grades depicted in Figure 14-11 include:

- Broadly individual approximately log-normal populations do occur within each of the main lithologies, increasing in grade from altered granite, to mica greisen, to quartz greisen and then quartz pipes, as would be expected.

- Although generally higher grades do occur within quartz pipes, clearly not all quartz pipes are significantly mineralised: 50% of the quartz pipe grades fall below 0.04% WO_3.

Taking into account the observations above, discussions with the geologists on site, and testwork with respect to their developed grade control reconciliation model, it was decided to model the mineralised greisen and pipe structures by the following approach:

a) Generation of 2.5m composites from the drillhole data.
b) Assignment of 3 types of mineralised zone codings, based on the composited WO_3 grade:
   1. Mineralised waste, >=0.07% <0.09% WO_3
   2. Low grade, greisens, >=0.09% <0.3% WO_3
   3. High grade, pipes, >=0.3% WO_3
c) Extrapolation of these ZONE codings, creating sub-cells reflecting these zone structures.

Test modelling was employed in the development of this methodology and parameters, using the following steps:

1. **Reconciliation Model, March 13 - March 14.** Blocks from the short-term planning block model, representing the volume mined between March 2013 and March 2014, were retrieved into a separate block model. Grades in this model predominantly stem from the grade control (GC) samples taken during blast hole drilling. This model was used as the reference for calibration of resource modelling parameters.

2. **Volumetric Block Model, March 13 - March 14.** An empty volumetric block model was set up for between the same topographical surfaces.

3. **Extrapolation and Grade Estimation Testing**. WO_3 grades were now estimated into this test volumetric model, using only composites derived from DD and RC exploration holes. Various zone extrapolation, compositing, search strategy, and grade estimation parameters were tested. The reconciliation GC model is depicted in cross-section in Figure 14-12, and the final best-fit test model is depicted on the same section line in Figure 14-13. The drillhole data on the section line is shown in Figure 14-14. Comparative bench plans are shown in Figure 14-15 and Figure 14-16.

Figure 14-12. Section B-B' – Reconciliation Model, March 13- March 14

(Based on GC data)

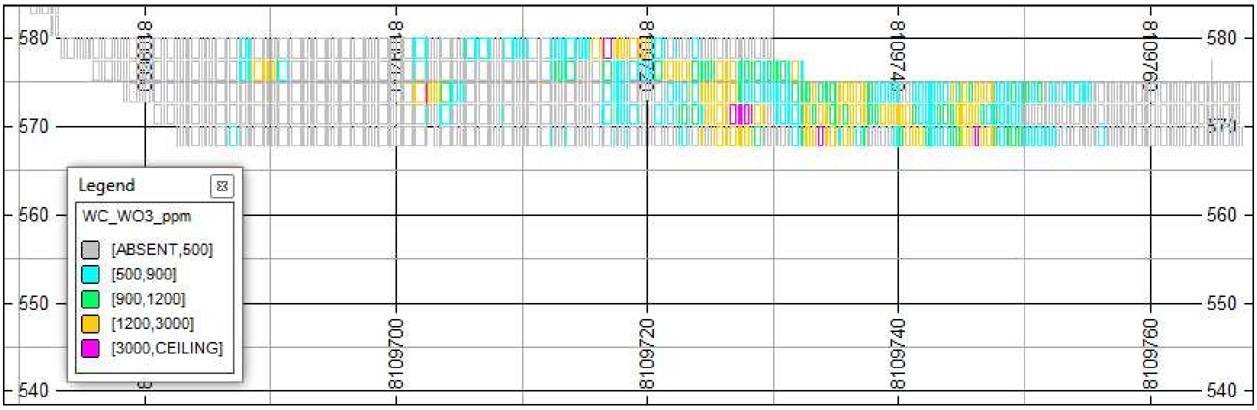


Figure 14-13. Section B-B' – Test Evaluation Model, March 13- March 14

(Based only on DD and RC data)

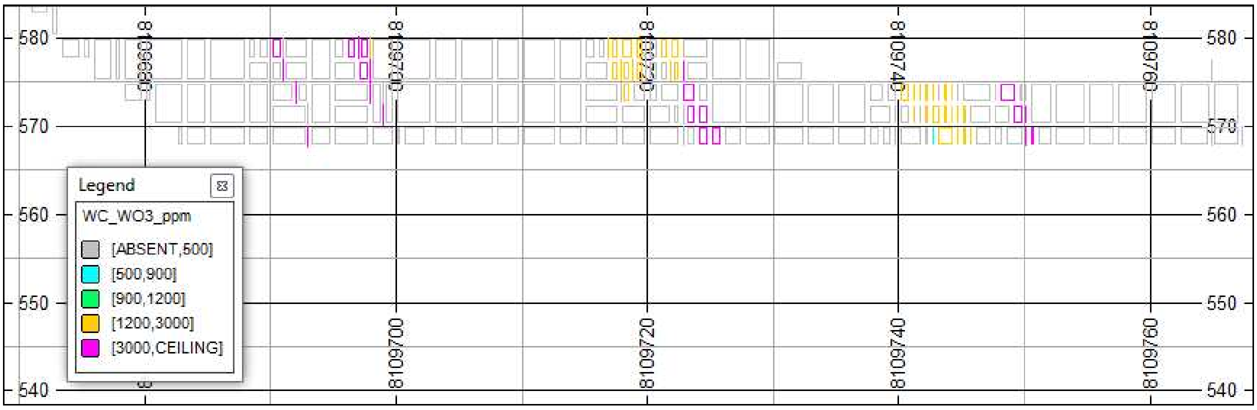


Figure 14-14. Section B-B' – Drillhole Data

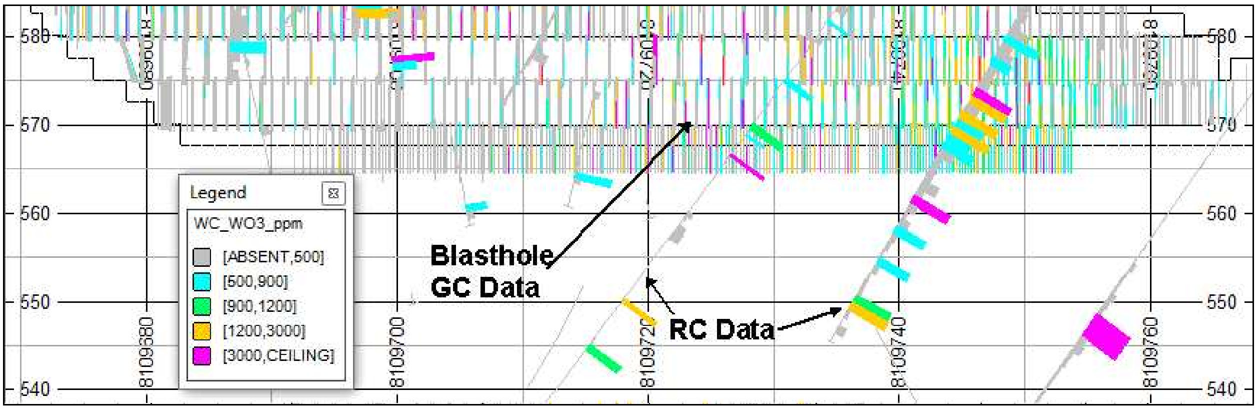


Figure 14-15. Plan at 573.25mRL - Reconciliation Model, March 13- March 14

(Based on GC data)

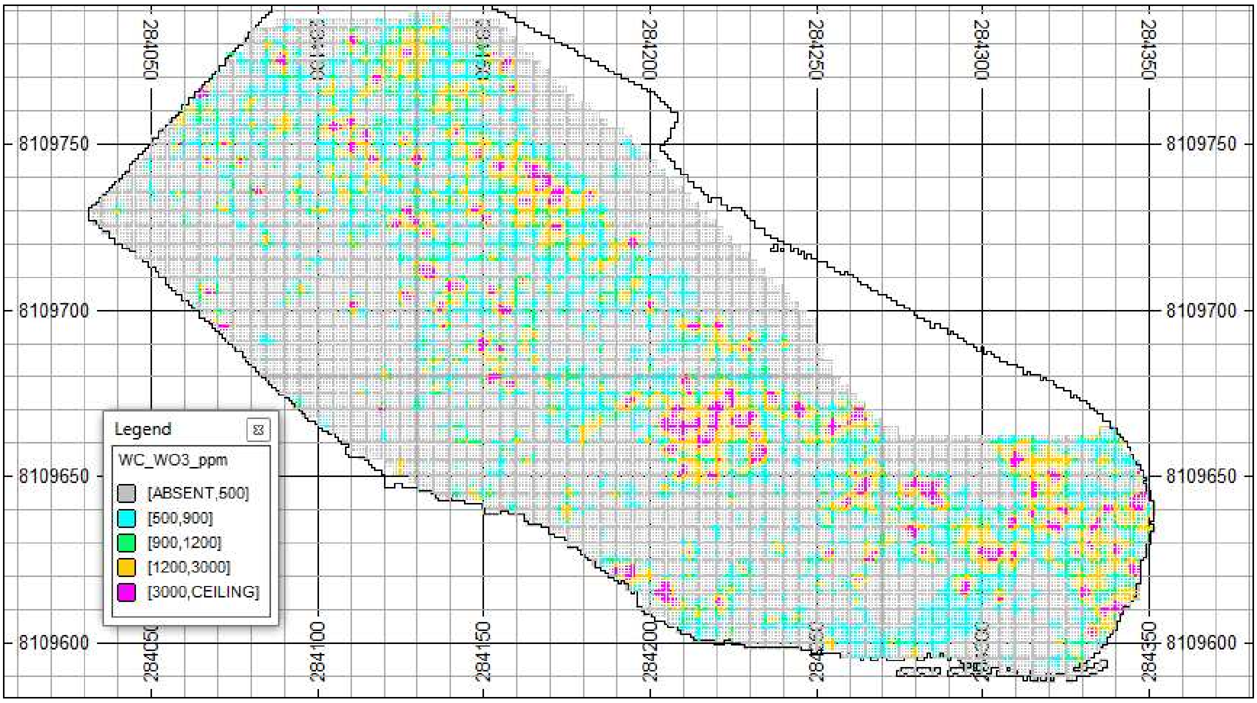
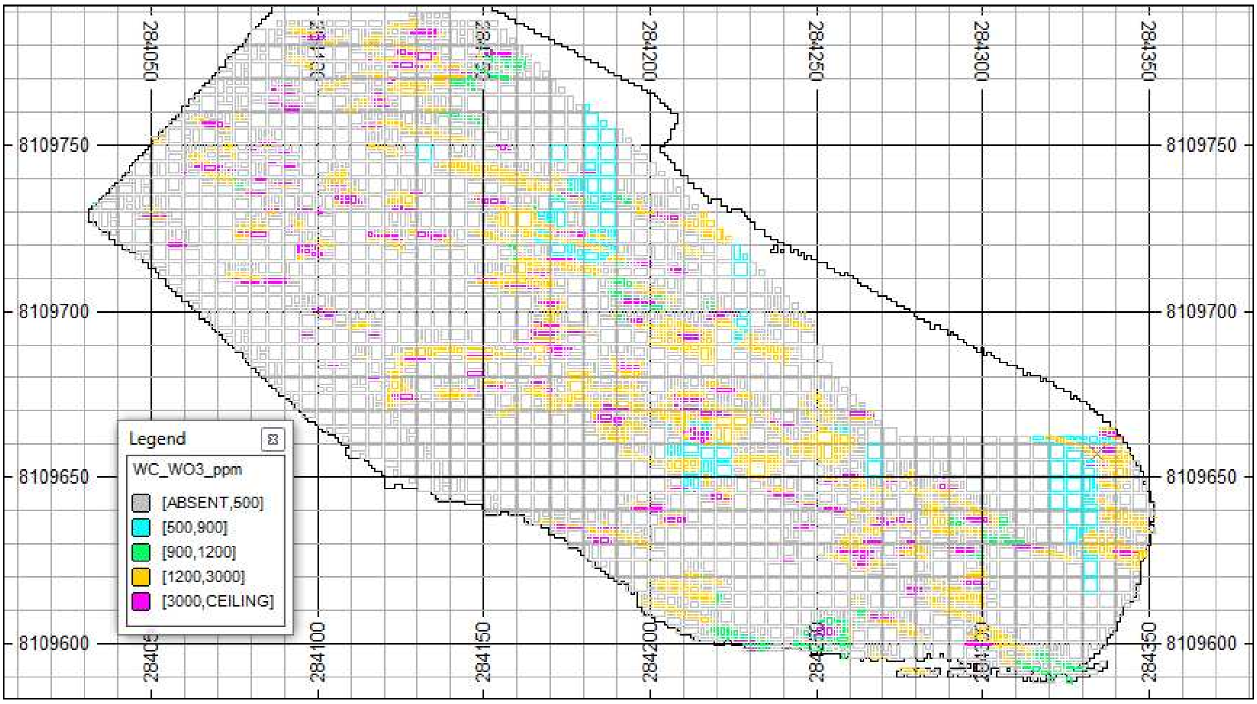


Figure 14-16. Plan at 573.25mRL - Test Evaluation Model, March 13- March 14

(Based only on DD and RC data)



From this test work a final set of grade estimation parameters were established, as summarised in Table 14-11.

14.4 Sample Selection and Compositing

All available DD, RC, BEX and GC samples were selected inside the supplied mineralised wireframe envelope. A summary of the selected data is shown in Table 14-5.

Table 14-5. Selected Sample Summary

					WO_3		MoS2	
Type of Sample	DTYPE	Holes	Length m	Average Length / Hole m	Samples	Holes With Samples	Samples	Holes With Samples
Diamond drillholes	DD	63	3,370	53	1,045	48	1,047	48
Reverse Circulation Holes	RC	305	10,208	33	8,361	304	8,357	304
Blasthole Grade Control Samples	GC	52,639	305,862	5.8	87,848	45,148	88,502	45,486
Blasthole exploration drilling	BEX	1,364	29,883	22	11,514	1,342	11,268	1,336

These samples were then composited, using the controls summarised below:

1. Composite length 2.5m. This compositing length was applied as slightly variable, such that an equal composite length of 2.5m was applied across each intersection. This length was chosen as it corresponds to the ore mining flitch height in the open pit.

2. Minimum composite length = 0.5m.

3. Minimum/maximum gap length = 0.125 / 0m.

4. ZONE assignment onto composites, such that:

    - ZONE=1, Mineralised waste, >=0.07% <0.09% WO_3
    - ZONE=2, Low grade, greisens, >=0.09% <0.3% WO_3
    - ZONE=3, High grade, pipes, >=0.3% WO_3

5. Top-cut values were assigned to WO_3 and MoS_2 composited grades, as follows:

    WO_3 - top-cut = 1.1%

    MoS_2 - top-cut = 0.4%

These top-cut values stem from:

- Decile analyses.
- Log-probability plots.
- Coefficient of variation (CV) analyses.
- Test modelling and comparison with reconciliation results.

Table 14-6 depicts the decile analysis results for RC grades of selected samples. From this it can be seen that the 5% of the BEX assays above a grade of approximately 1.1% WO_3 contain more than 39% of all sampled metal. Log-probability plots of selected samples are shown in Figure 14-17 and Figure 14-18, for WO_3 and MoS_2, respectively. Results for a CV analysis of WO3 for the selected BEX and GC samples are depicted graphically in Figure 14-19 and Figure 14-20. Decile analysis results for all data sets, for both WO_3 and MoS_2, as well as CV plots, are shown in Appendix A.

From all of these analyses, top-cut levels of 1.1% WO_3 and 0.4% MoS_2 were selected and applied.

Table 14-6. Decile Analyses – WO_3, Selected BEX Samples

Q%_FROM	Q%_TO	NUMBER	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
0	10	770	120	105	133	230,448	2.7
10	20	769	140	133	150	269,678	3.1
20	30	771	162	150	183	312,606	3.6
30	40	770	189	183	200	364,964	4.3
40	50	768	219	200	234	423,097	4.9
50	60	769	258	234	284	497,548	5.8
60	70	770	314	284	350	605,137	7.1
70	80	772	398	350	450	767,113	8.9
80	90	769	562	450	717	1,081,688	12.6
90	100	772	2,086	717	43,954	4,026,688	46.9
90	91	77	741	717	767	142,573	1.7
91	92	77	811	767	851	156,080	1.8
92	93	78	905	851	951	175,613	2.0
93	94	77	1,026	951	1,101	196,452	2.3
94	95	77	1,178	1,101	1,268	226,728	2.6
95	96	78	1,364	1,268	1,468	266,037	3.1
96	97	76	1,684	1,468	1,868	322,477	3.8
97	98	77	2,115	1,885	2,435	407,104	4.7
98	99	78	3,003	2,435	3,837	584,018	6.8
99	100	77	8,008	3,837	43,954	1,549,607	18.1
0	100	7700	445	105	43,954	8,578,967	100.0

Figure 14-17. Log-Probability Plot, WO_3, Selected Samples

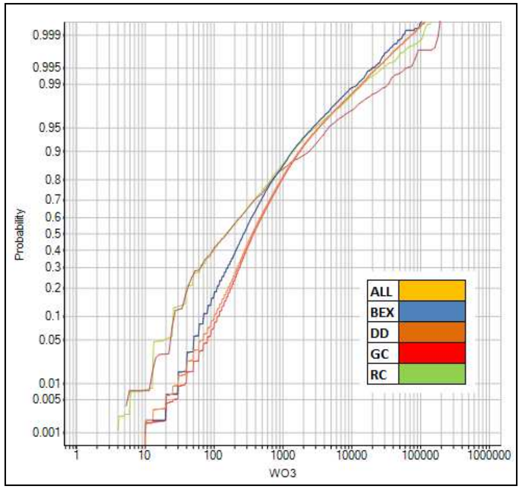


Figure 14-18. Log-Probability Plot, MoS_2, Selected Samples

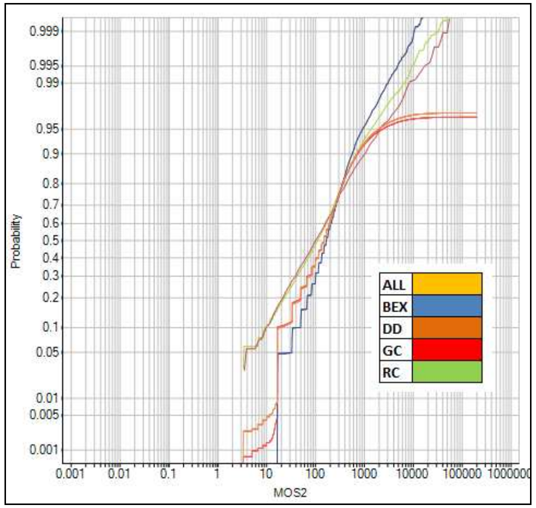


Figure 14-19. CV Analysis – BEX Samples, WO_3

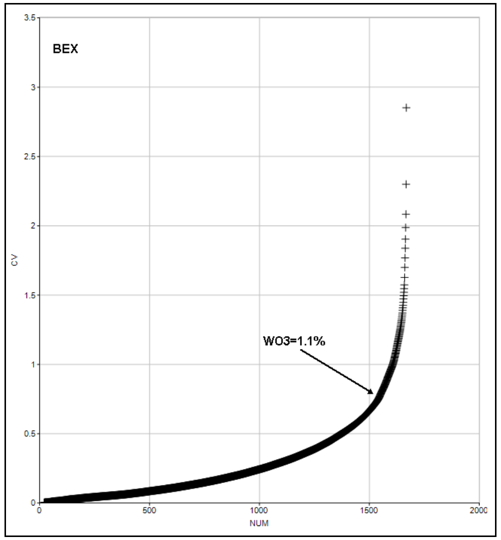
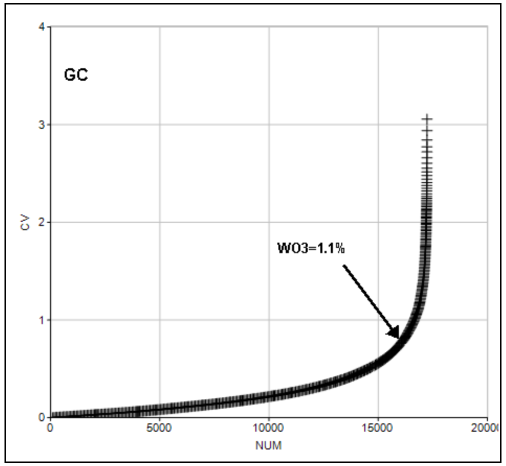


Figure 14-20. CV Analysis - GC Samples, WO_3



14.5 Geostatistics

A statistical summary of the selected samples is shown in Table 14-7. These statistics are divided by sample type, as well as shown overall. It can be seen that all of the coefficient of variation (CV) values are very high. The overall statistics also show statistics of As, Bi and Sn. Log-probability plots of WO_3 grades in these selected samples are shown in Figure 14-11 and Figure 14-17. Log-probability plots of MoS_2 samples grades are shown in Figure 14-21.

Table 14-7. Summary Statistics of Selected Samples

FIELD	Unit	TYPE	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	CV
WO_3	ppm	DD	1,475	5.00	189,796	1,668	95,432,418	9,769	848	5.9
WO_3	ppm	GC	88,793	0.00	563,210	1,098	37,505,267	6,124	824	5.6
WO_3	ppm	RC	8,732	4.00	361,936	1,059	55,074,795	7,421	632	7.0
WO_3	ppm	BEX	11,516	10.00	349,270	917	27,718,592	5,265	624	5.7
WO_3	ppm	ALL	110,516	0.00	563,210	1,081	37,226,760	6,101	808	5.6
MoS_2	ppm	DD	1,475	3.34	52,045	614	7,245,434	2,692	507	4.4
MoS_2	ppm	GC	89,441	0.00	201,356	315	2,306,016	1,519	274	4.8
MoS_2	ppm	RC	8,732	3.34	146,459	457	8,115,364	2,849	397	6.2
MoS_2	ppm	BEX	11,270	8.34	43,954	323	798,677	894	288	2.8
MoS_2	ppm	ALL	110,918	0.00	201,356	320	2,341,012	1,530	280	4.8
Bi	ppm	GC	1,823	0.00	25,168	228	2,883,318	1,698	144	7.5
Bi	ppm	ALL	1,823	0.00	25,168	228	2,883,318	1,698	144	7.5
As	ppm	DD	1,318	2.00	140,850	603	52,333,992	7,234	139	12.0
As	ppm	GC	82,398	0.00	156,870	183	1,861,567	1,364	204	7.5
As	ppm	RC	8,490	0.00	57,400	166	1,808,457	1,345	104	8.1
As	ppm	BEX	9,932	0.00	27,658	153	259,208	509	148	3.3
As	ppm	ALL	102,138	0.00	156,870	181	1,858,424	1,363	195	7.5

A statistical summary of the generated composites is shown in Table 14-8, with corresponding log-probability plots from Figure 14-21 to Figure 14-24.

Table 14-8. Summary Statistics of 2.5m Composites

FIELD	DTYPE	ZONE	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	CV
WO3	RC	1	281	700	895	792	3,037	55	792	0.1
	GC	1	6,156	700	900	790	3,263	57	790	0.1
	DD	1	37	706	895	789	3,451	59	789	0.1
	BEX	1	501	700	890	785	3,113	56	785	0.1
	ALL	1	6,975	700	900	789	3,250	57	789	0.1
	RC	2	645	908	2,976	1,544	213,544	462	1,542	0.3
	GC	2	14,725	900	3,000	1,509	277,932	527	1,506	0.3
	DD	2	78	933	2,951	1,754	381,336	618	1,755	0.4
	BEX	2	1,350	900	2,990	1,508	286,076	535	1,504	0.4
	ALL	2	16,798	900	3,000	1,510	277,648	527	1,507	0.3
	RC	3	325	3,001	11,000	7,514	8,971,282	2,995	7,566	0.4
	GC	3	4,976	3,000	11,000	6,716	9,071,406	3,012	6,727	0.4
	DD	3	75	3,090	11,000	7,461	10,474,251	3,236	7,515	0.4
	BEX	3	513	3,000	11,000	6,649	8,925,317	2,988	6,656	0.4
	ALL	3	5,889	3,000	11,000	6,735	9,082,538	3,014	6,747	0.4
MoS_2	RC	1	281	3	4,000	441	308,653	556	474	1.3
	GC	1	6,138	-	4,000	331	193,650	440	329	1.3
	DD	1	37	20	4,000	485	574,753	758	463	1.6
	BEX	1	501	33	4,000	373	197,555	444	358	1.2
	ALL	1	6,957	-	4,000	336	197,004	444	334	1.3
	RC	2	645	3	4,000	804	1,088,528	1,043	868	1.3
	GC	2	14,684	-	4,000	453	393,337	627	444	1.4
	DD	2	78	15	4,000	788	1,096,464	1,047	944	1.3
	BEX	2	1,343	17	4,000	491	387,615	623	473	1.3
	ALL	2	16,750	-	4,000	463	410,048	640	454	1.4
	RC	3	325	8	4,000	1,207	1,694,180	1,302	1,369	1.1
	GC	3	4,965	-	4,000	884	1,214,033	1,102	908	1.2
	DD	3	75	3	4,000	1,173	1,996,020	1,413	1,840	1.2
	BEX	3	511	17	4,000	966	1,310,065	1,145	988	1.2
	ALL	3	5,876	-	4,000	900	1,241,777	1,114	929	1.2

Notes

. CV = coefficient of variation

. All units above in ppm

. Top-cut levels applied:

WO_3	11,000 ppm
MoS_2	4000 ppm

. ZONE demarcation:

ZONE	WO_3 %
1	>=0.7 <0.09
2	>=0.09 <0.3
3	>=0.3

Figure 14-21. Log-Probability Plot, MoS_2 – Selected Samples

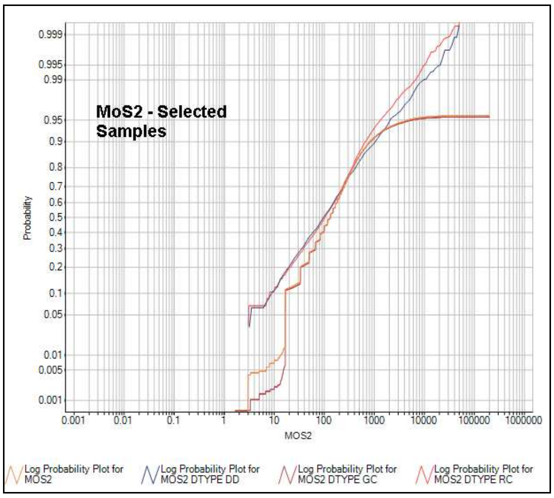


Figure 14-22. Log-Probability Plot, WO_3 – Composites by ZONE

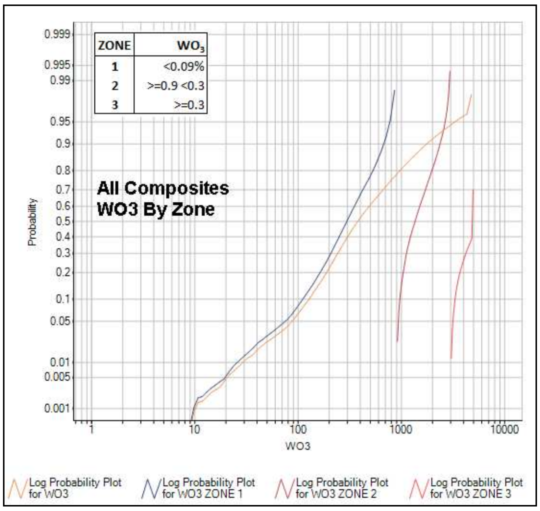


Figure 14-23. Log-Probability Plot, WO_3 – DD and RC Composites by ZONE

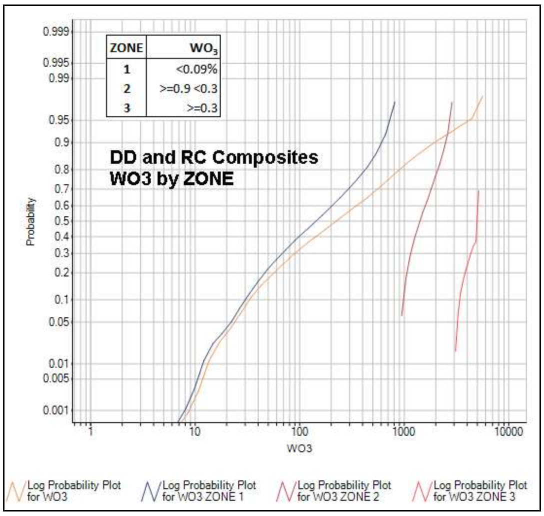
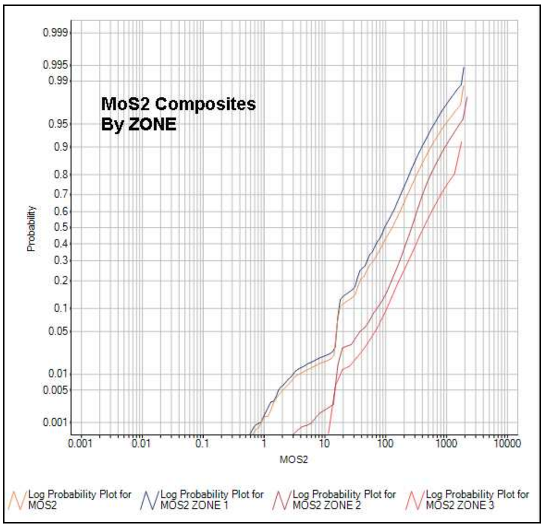


Figure 14-24. Log-Probability Plot, MoS_2 – Composites by ZONE



It can be seen from Table 14-8 that the coefficient of variation values have been reduced to near or below 1.0, by the effect of compositing and top-cut application. Table 14-8 also shows summary statistics for just DD and RC drillhole composites, as well as for all composites, which includes the vast number of GC composites. It can be seen the WO_3 averages are very similar with or without GC composites, which further supports the inclusion of GC composites for resource estimation.

Experimental variograms were generated for the generated composite data sets. Model variograms were fitted in each case, as depicted in Figure 14-25 and Figure 14-26. For the WO_3 variograms, the overall range is generally 30-40m, with over 2/3 the overall variability being reached by about 15m. The MoS_2 variograms generally have a longer range of approximately 50m. All of the model variogram parameters are summarised in Table 14-9.

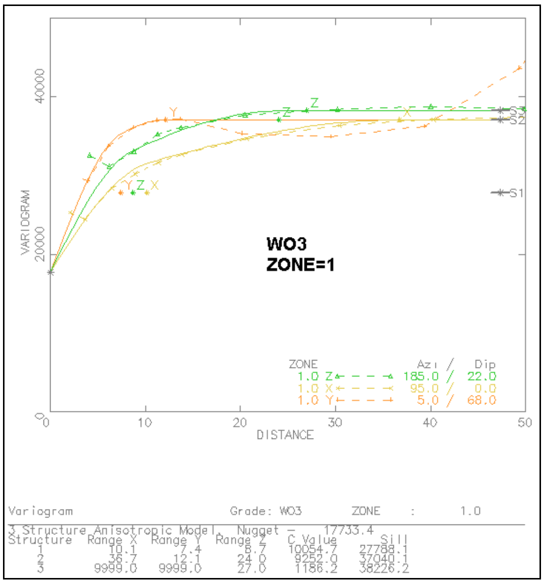
Table 14-9. Model Variogram Parameters

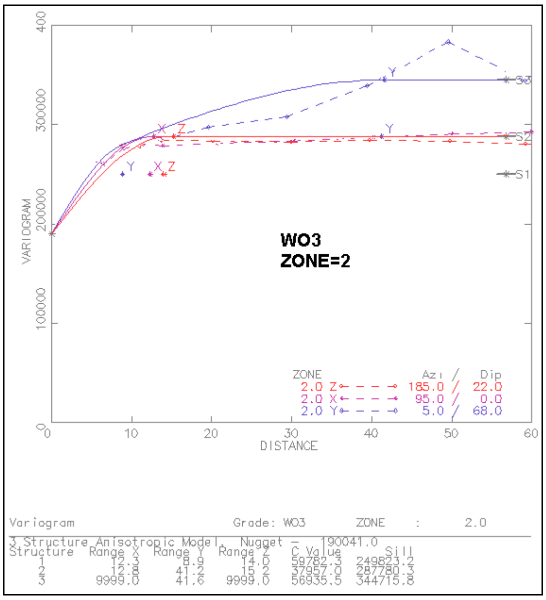
Element	Reference Number	ZONE	Nugget	Range 1 (m)			C1	Range 2 (m)			C2	Range 3 (m)			C2
				1	2	3		1	2	3		1	2	3	
WO3	1	1	17,733	10	7	9	10,055	37	12	24	9,252	9,999	9,999	27	1,186
WO3	2	2	190,041	12	9	14	59,782	13	41	15	37,957	9,999	42	9,999	56,936
WO3	3	3	194,371	3	5	5	173,499	28	27	28	39,135	-	-	-	-
MoS2	11	1	0.47	15	12	4	0.14	57	42	61	0.16	9,999	95	88	0.26
MoS2	12	2	0.19	13	5	5	0.16	95	34	66	0.30	9,999	73	88	0.19
MoS2	13	3	0.31	13	7	6	0.21	95	62	42	0.13	9,999	63	60	0.13

Notes:

Variogram Orientation		
Rotation (^{o}) About Each Axis		
3 (Z)	2 (Y)	1 (X)
5	0	68

Figure 14-25. Experimental and Model Variograms – WO_3





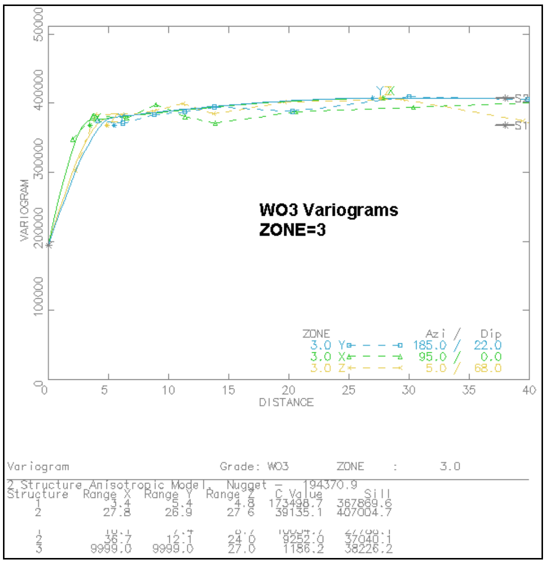
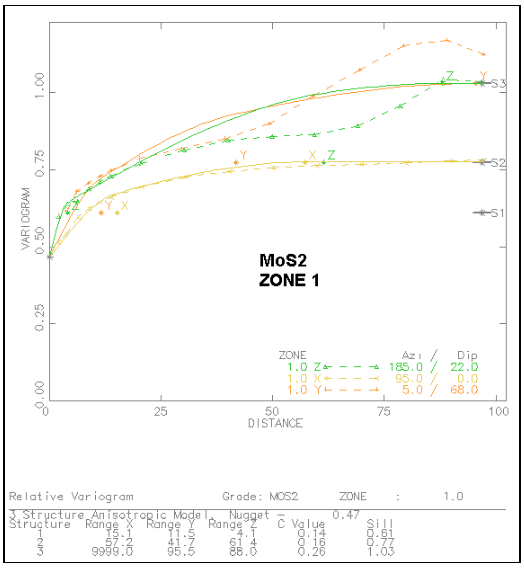
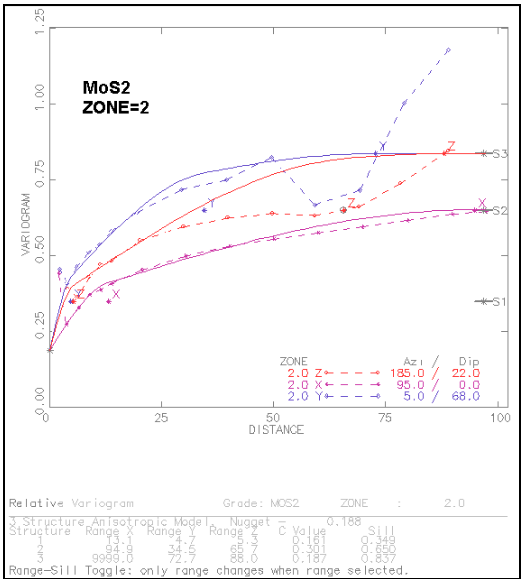
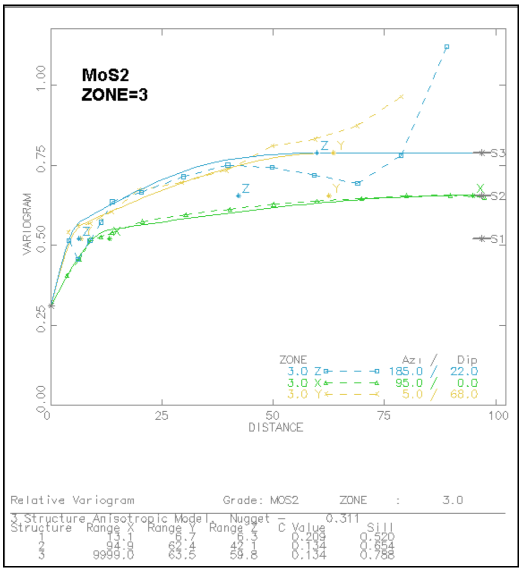


Figure 14-26. Experimental and Model Variograms – MoS_2







14.6 Volumetric Modelling

An overall block model prototype was set up using the parameters summarised in Table 14-10. A parent block size of 5m x 5m x 5m was selected. Laterally, 5m is approximately a quarter of the average RC sample spacing of approximately 20m. Vertically 5m is the principal mining bench height.

Table 14-10. Resource Model Prototype

	Min	Max	Range	Size	Number
	m	*m*	*m*	*m*	
X	283,500	284,500	1000	5	200
Y	8,109,400	8,110,250	850	5	170
Z	440	680	240	5	48

Physical controls used, in the form of wireframe models, during the generation of the volumetric block model include:

- Mined topography, as of end-August, 2015.
- Mineralised zone envelope.
- Overall pre-open-pit mining topography.
- Underground excavations.

Sub-cells were created at the edge of these structures, down to a resolution of 1m. A global density value of 2.7t/m^3 was also set into all blocks.

Strings were also defined in cross-sections, that were used for the generation of mineralisation orientation angles in the volumetric block model. These angles were subsequently used as dynamic anisotropy controls orient search ellipses during grade estimation.

Fields set into the volumetric block model include:

AZONE	Region identifier
TRDIPDIR	Dip direction of mineralisation
TRDIP	Dip of mineralisation
FILL	=0 in-situ; =1 filled e.g. areas now higher than original pre-mining topography
UG	=1 sub-blocks within historical underground workings.
MINZ	=1 within overall mineralised zone.

14.7 Densities

Consistent with previous resource estimation work and current practice at the mine, an average density value of 2.7t/m^3 has been assumed for both mineralised and unmineralised material. This value is also reasonably consistent with previous bulk density measurements, as summarised in Table 6-1.

14.8 Grade Estimation

As discussed earlier in Section 14.3, estimation parameters were derived by detailed test work on modelling of material that was mined between March 2013 and March 2014. This led to grade estimation being done as two stage process:

1. Extrapolation of greisen and pipe zones based on pre-defined grade ranges.
2. Estimation of WO_3 and MoS_2 grades within these zone structures.

A summary of the extrapolation and estimation parameters is shown in Table 14-11.

Table 14-11. Grade Estimation Parameters

	ZONE / Search	Search Distances (m)			OCTANT CONTROL	MIN No. of Composites	MIN No. of Drillholes
		X (1)	Y (2)	Z (3)			
ZONE extrapolation	**3: >=0.3% WO_3**	5	50	2.5		1	
	2: >=0.09 <0.3% WO_3	20	50	10		1	
WO_3/MoS_2 estimation	**1st**	5	12.5	2.5	Yes	5	3
	2nd	10	25	5	Yes	5	3
	3rd	20	50	10	No	1	1

Notes

. ZONE extrapolation based on nearest neighbour estimation

. Dynamic anisotropy set of per AZONE region, such that:

X : Strike direction

Y : Dip direction

Z : Cross-strike direction

. Main WO_3 and MoS_2 grades interpolated with ordinary kriging (OK)

. Max no. of composites = 24

. Octant controls:

Min. no. of octants = 3

Min. no. of composites per octant = 1

Max. no. of composites per octant = 3

. Grades also estimated with NN and ID^2 for validation purposes

Initially the ZONE identifiers on the composites were extrapolated. Much tighter distance limits were applied to the pipe (>=0.3% WO_3) structures, than the mineralised greisen (>=0.09 <0.3% WO_3) structures. In this zone extrapolation process, sub-blocks were generated down to minimum size of 1m x 1m x 2.5m.

For grade estimation, octant controls were used for the first two attempted searches, so as to provide an extra degree of declustering, to reduce the overall influence of GC samples, particularly for those upper parts of the model just underneath the current mining benches. For both ZONE and grade estimation, the search ellipses were oriented using the angles derived from dynamic anisotropy.

A final 3rd search was used without any minimum number of composite limits, so as to ensure that practically all extrapolated greisen/pipe blocks did receive some WO_3 / MoS_2 grades. The principal method of WO_3 grade interpolation used was ordinary kriging (OK). However, for subsequent testing and validation purposes, alternative WO_3 grade values were also interpolated using nearest-neighbour and inverse-distance weighting methods.

14.9 Mineral Resource Classification

A search volume field (SVOL) was generated in the block model during grade estimation, recording which of the progressive searches had been successful, along with another field, (NUM), which recorded how many composites had been used in each blocks' grade estimation. These field values were utilised in setting resource classification categories. The criteria used in setting these categories are summarised in Table 14-12.

The very erratic distribution quartz pipes and mineralised greisens is unique to the Wolfram Camp area, and means that even with BEX drilling on a 10m x 10m grid, there will still be a high proportion of inferred resources as the pit deepens and advances.

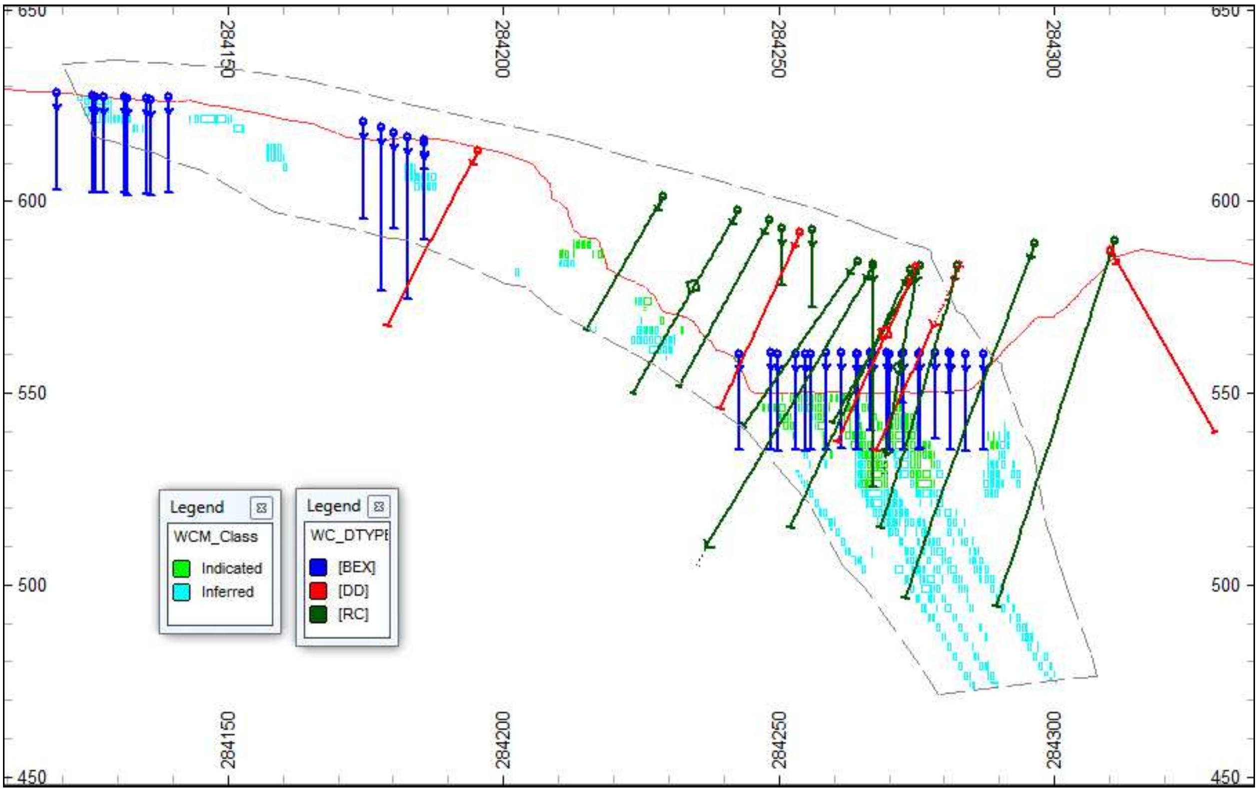
Table 14-12. Resource Classification Criteria

Category	Description
Measured	(No material currently classified as measured)
Indicated	At least 5 composites from at least 3 holes, within at least 3 octants, with a search of 10m (along-strike) x 25m (down-dip).
Inferred	Max extrapolation of 20m along-strike or 50m down-dip from individual composites.

Important aspects of these resource classification criteria include:

- There are no *Measured* resources. Despite some blocks, particularly those just underneath the current mined benches, having abundant samples nearby, it is considered that there is insufficient detailed interpretation control to justify the setting of any material with a *Measured* resource category.
- The maximum set of search distances used for *Indicated* resources were approximately 10m (along-strike) x 25m (down-dip). This corresponds approximately with the range of the WO_3 variograms for mineralised greisen material. Additional controls were also imposed for *Indicated* resources, such that *Indicated* blocks had to have grades stemming from samples from at least 3 different drill holes, and within at least 3 different octants.
- *Inferred* blocks have extrapolated a maximum distance of 50m (down-dip).

Figure 14-27. Example of Resource Classification – Section E-E'



14.10 Model Validation

14.10.1 Visual Comparisons

Sections were created through the resource block model, and compared with the drillhole composites used in for the grade estimation. A reference plan for these sections is shown in Figure 14-28. Sections showing grades of WO_3 are shown in Figure 14-29 through to Figure 14-34. These sections only show DD, RC and BEX holes, just to make plots clearer; however GC-derived composites were also used in grade estimation.

Figure 14-28. Reference Plan

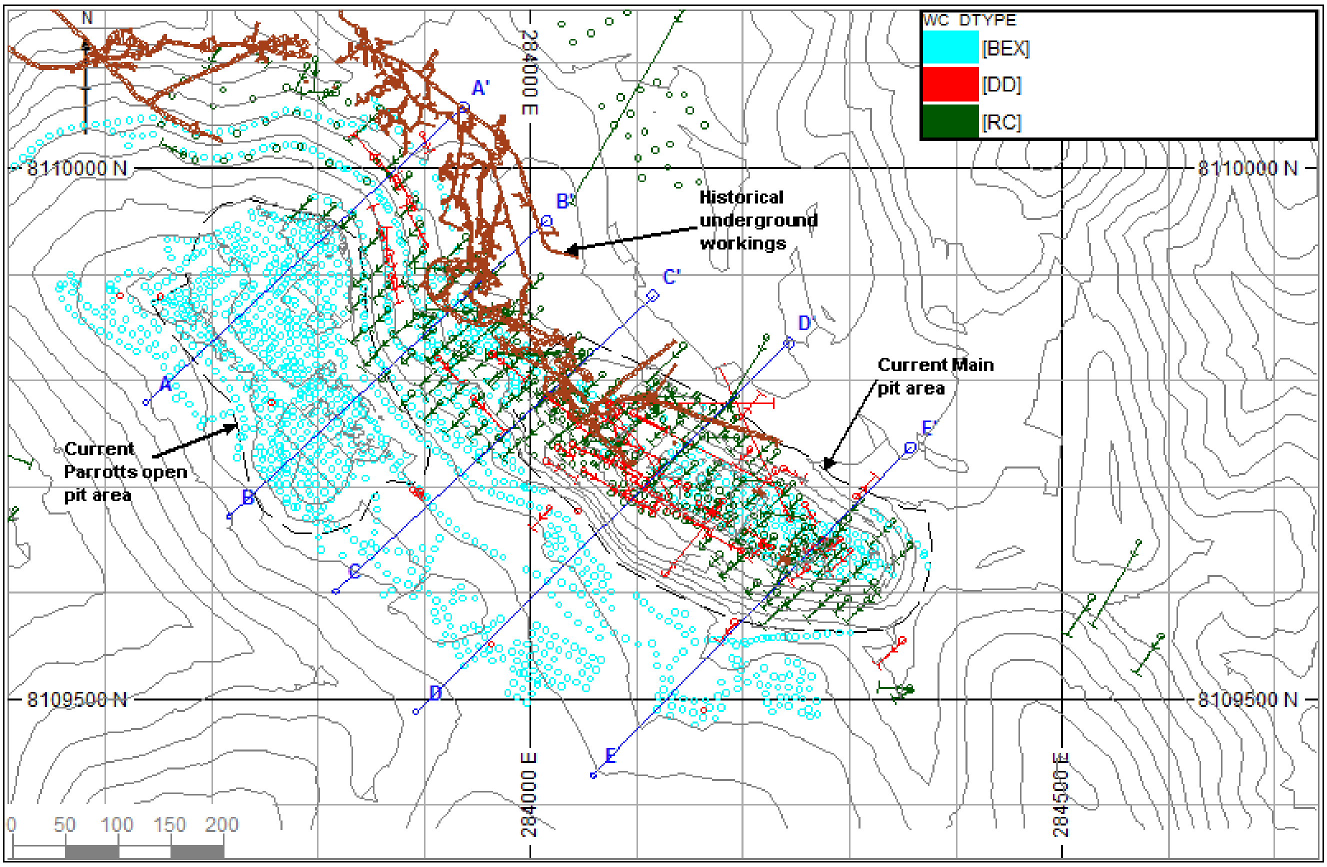


Figure 14-29. Section A-A' WO_3

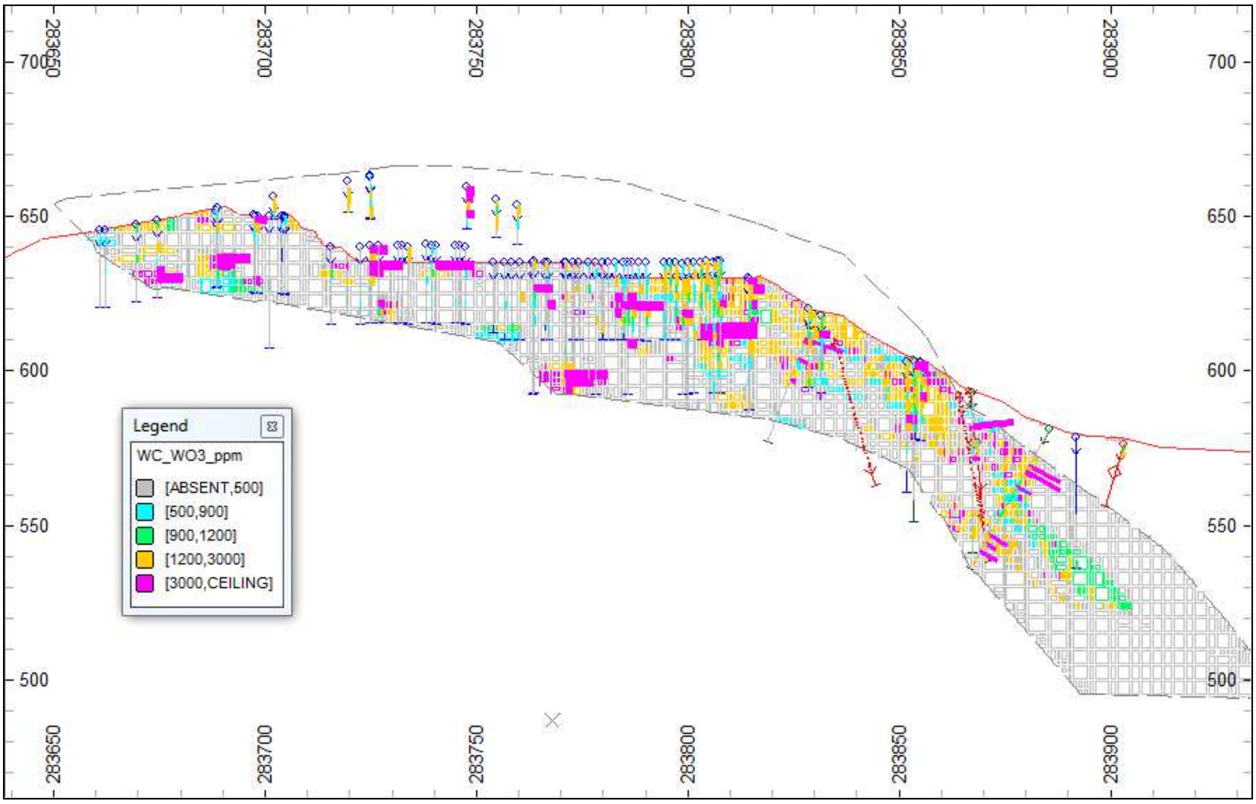


Figure 14-30. Section B-B' WO_3

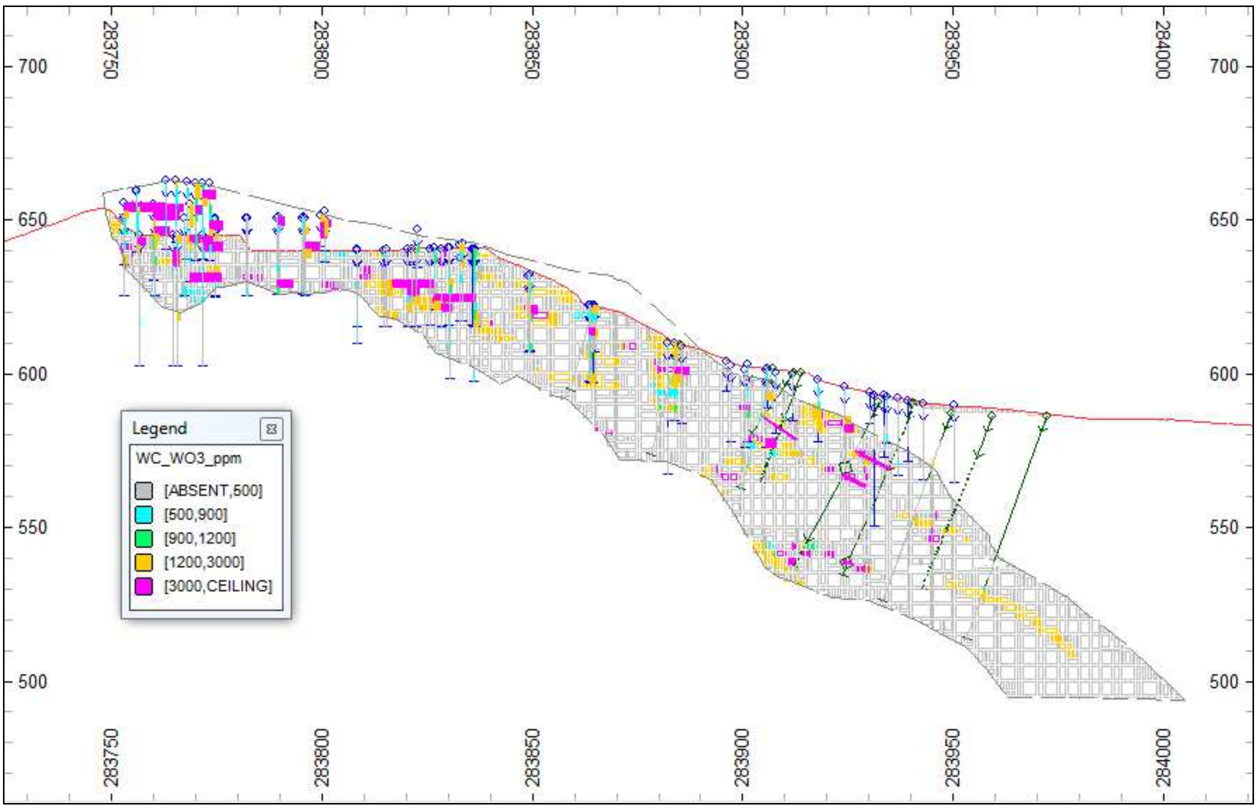


Figure 14-31. Section C-C' WO_3

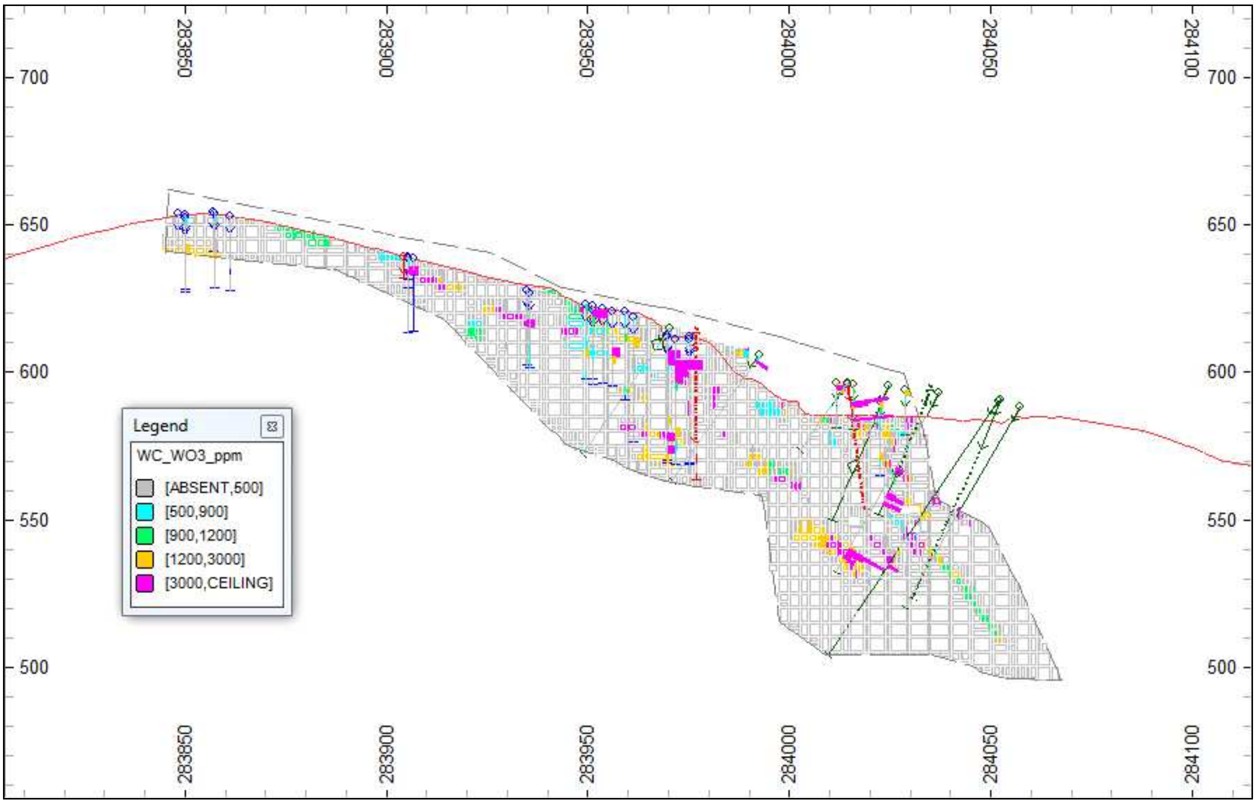


Figure 14-32. Section D-D'' WO_3

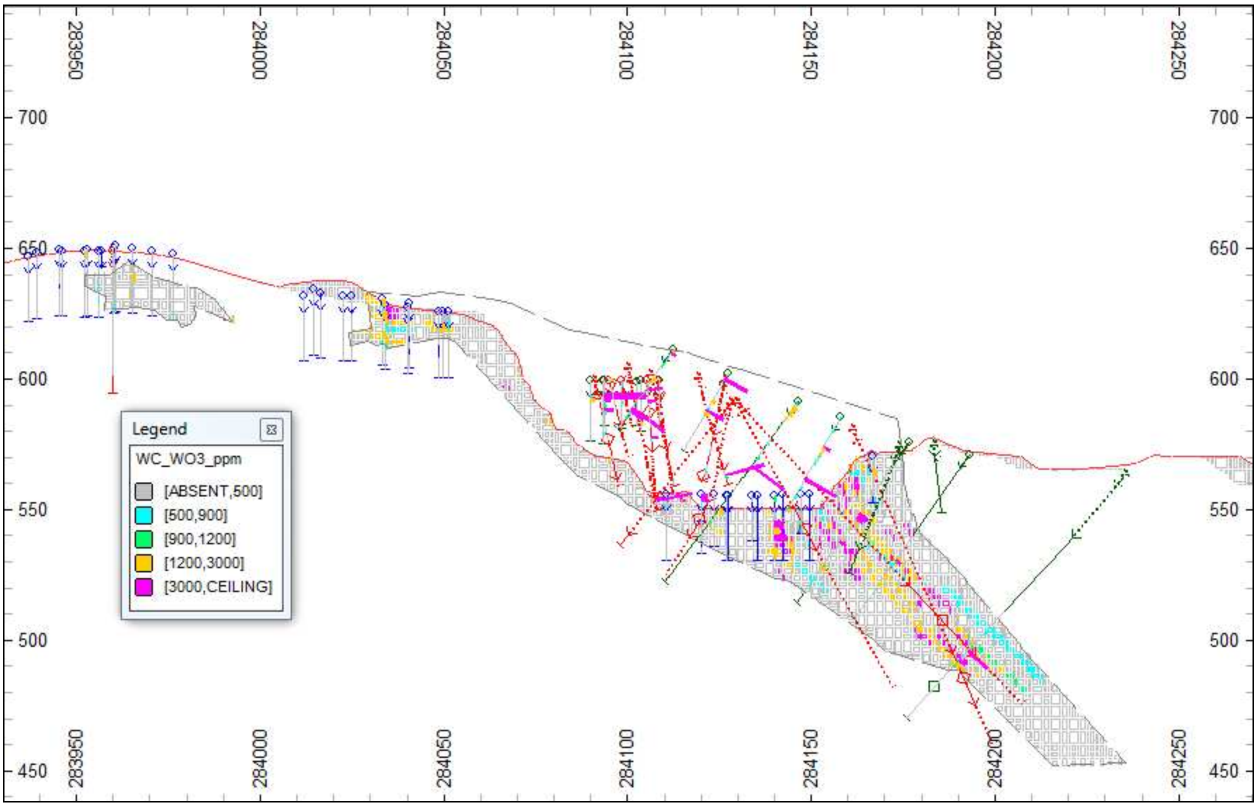
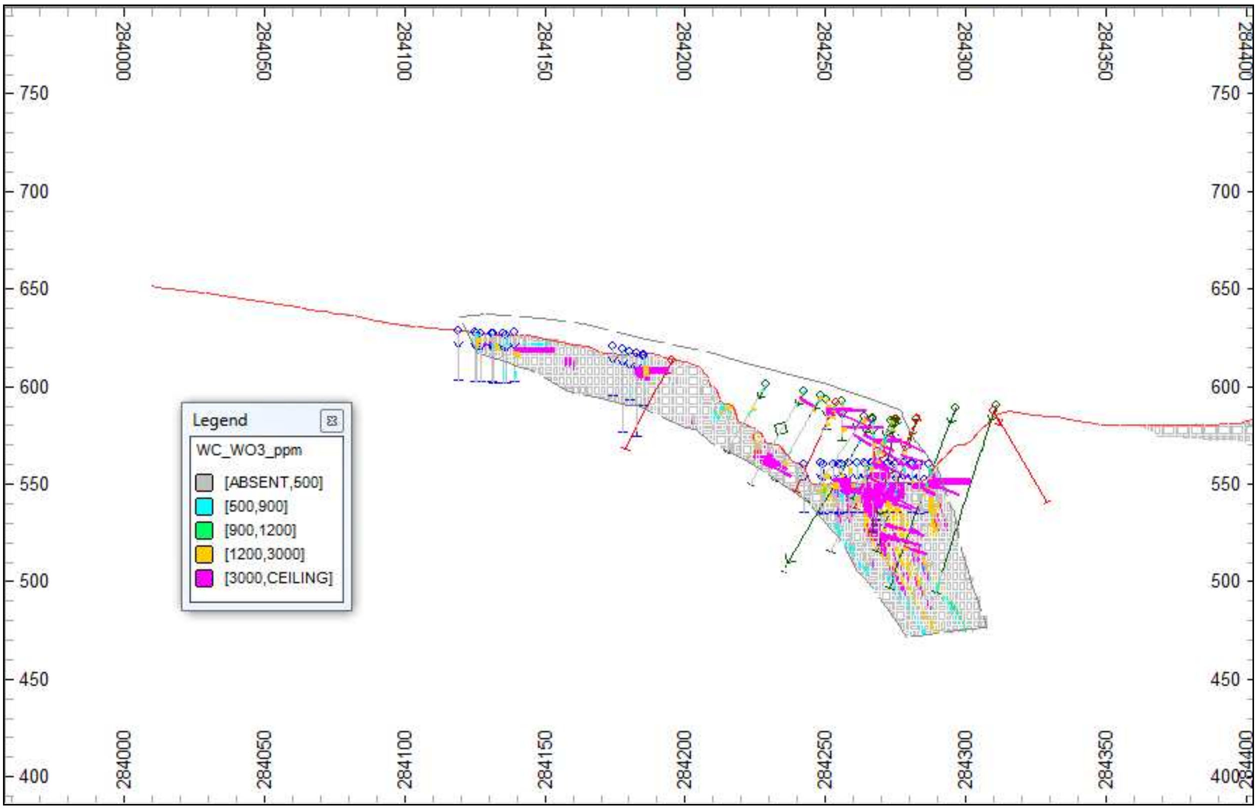


Figure 14-33. Section E-E' WO_3



14.10.2 Comparison of Global Average Grades

A comparison was made of the average WO_3 model grades, for all resource levels, with the corresponding average sample and composite grades for the different modelled beds. These results are summarised in Table 14-13.

Table 14-13. Comparison of Global Average Grades

	Zone 1 >0.09% WO3				Zone 2 >=0.09% <0.3% WO3				Zone 3 >=0.3% WO3			
Field	Composites	Model Grades			Composites	Model Grades			Composites	Model Grades		
	All	OK	ID	NN	All	OK	ID	NN	All	OK	ID	NN
WO_3	789	790	789	789	1,510	1,529	1,528	1,525	6,735	6,874	6,901	6,894
MoS_2	311	377	377	377	397	503	500	501	625	708	708	710

Notes

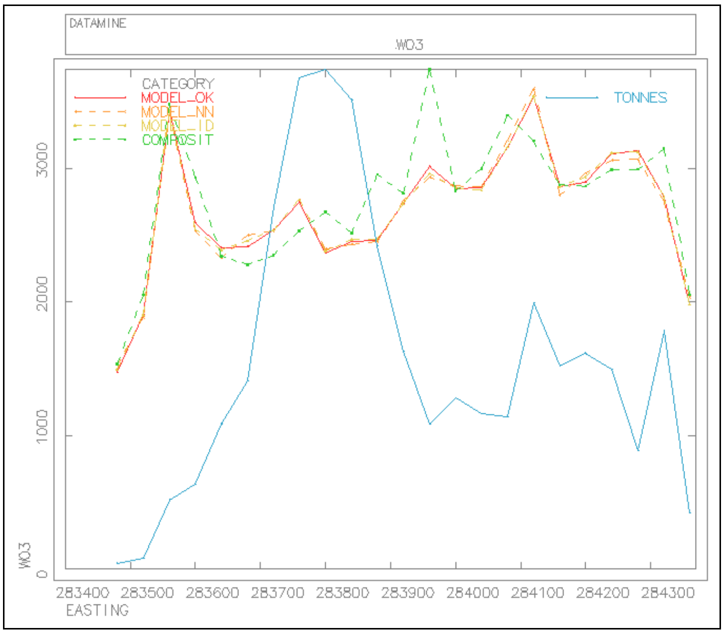
- **All grade units above shown in ppm**
- **OK - ordinary kriging**
- **ID - inverse distance weighting (^2)**
- **NN - nearest neighbour**
- **No cut-off grades applied**
- **Indicated and inferred resources used**

These results compare fairly well, within the principal zone types, broadly corresponding to medium grade greisen material and high grade pipe material.

14.10.3 Comparison of Local Average Grades

As part of the model validation process, grade profiles (swath plots) were also produced on 40m slices, and the average grades (derived from different estimation methods) per slice, compared with the composites on the same slices. An examples of a grade profile plot is shown in Figure 14-34, for WO_3 within ZONE=2 material. This shows a favourable comparison between composite grades and model grades, derived from kriging, inverse distance weighting and nearest neighbour estimation. All of the grade profiles produced are shown in Appendix A.

Figure 14-34. Example Grade Profile Plot – WO_3



14.10.4 Historical Comparison

A historical comparison of previous resource estimates is summarised in Table 14-15. This comparison is complicated by the range of previous methodologies and cut-off levels applied. The most recent evaluation, prior to the start of open pit operations in 2012, was Martlett 2011. At a cut-off of 0.1%W (approximately 0.13% WO_3) this gave a combined resource of 1.89 Mt with a grade of 0.47% WO_3. The current updated resource contains a combined resource of 2.7 Mt with a grade of 0.29% WO_3. Since 2012, a number of significant changes have been made in the current resource estimation methodology, which include:

- **Top-Cutting of Outlier Grades**. No top-cutting of outlier grades was applied in the Martlett 2011 estimation. Current top-cut levels were applied after a number of different types of analysis, as described in section 14.4.

- **Estimation Methodology**. The Martlett 2011 estimation used a multiple-indicator-kriging (MIK) process, whereas the current estimation uses zonal extrapolation of mineralised greisen and pipe structures. It should also be noted that the prev-2012 models will not have incorporated a high proportion of low grade greisen material, owing to the much sparser density of exploration-only data.

- **Inferred Extrapolation.** The current estimation has a maximum extrapolation distance of 50m, for *Inferred* material. The Martlett 2011 estimation used a maximum extrapolation distance of 250m. Given the sporadic frequency of mineralised pipe and greisen material that has been observed in the excavated pit ore, it is considered that the current maximum extrapolation distance of 50m (down-dip) is a reasonable limiting assumption.

- **Pit Production.** One of the key differences between the current estimation and those done previously is that now there is over 4 years of pit production. This has allowed reconciliation from production results, compared with the updated resource block model referenced to previous March 2014 topography.

- **Orientation of Mineralisation.** Previous estimations had all used one search orientation for the whole deposit. Although this orientation was modified between different studies, only one orientation was applied in each case. The current estimate uses dynamic anisotropy to use local orientations of mineralisation.

Table 14-14. Reconciliation March 2014 – August 2015

	Production Data		Block Model	
	Tonnes Kt	WO_3 ppm	Tonnes Kt	WO_3 ppm
Waste	950	367	1,217	45
Min Waste	342	955	55	921
Low Grade	460	1,874	276	1,541
High Grade	9	4,005	87	6,474
	1,761	**893**	**1,633**	**667**

Notes

. WO_3% Categories used in evaluation:

Waste	**<800**
Mineralised Waste	**800-1200**
Low Grade	**1200-3000**
High Grade	**3000+**

. Resource block model built up using current parameters

. Production data derived from grade control (GC) data

Table 14-15. Historical Comparison

				Measured/Indicated			Inferred			All Resource Categories		
		Cut-Off		Tonnes Kt	WO3 %	MoS2 %	Tonnes Kt	WO3 %	MoS2 %	Tonnes Kt	WO3 %	WO3 Contained t
2007	QOL		0.10% WO3 Eq	710	0.42	0.17	238	0.4	0.20	948	0.41	3,934
2010	Golders for PML	.25% Weq	0.32% WO3 Eq	780	0.55	0.22	640	0.66	0.18	1,420	0.60	8,525
Feb-11	Golders for Hazelwood Resources		0.05% WO3	2,873	0.23	0.05	2,213	0.253	0.04	5,086	0.24	12,207
Apr-11	Martlett for DRAG	.1% W	0.13% WO3	1,890	0.47	0.12				1,890	0.47	8,890
Mar-14	AW Resource		0.12% WO3	427	0.20	0.04	1,482	0.23	0.07	1,909	0.22	4,200
Aug-15	AW Resource		0.12% WO3	495	0.24	0.07	2,220	0.31	0.08	2,715	0.29	7,953

Notes

. Resource categories are combined purely for comparative purposes

- **It should be noted that all of the above (pre-2012) historical estimates precede reopening of open pit mining operations in 2012.**
- **Subsequent reconciliation of mill and mine production results indicate that major changes in resource estimation methodology have been required.**
- **Therefore the QP is not treating these historical estimates as being particularly relevant to the current updated resource or reserve estimation work.**

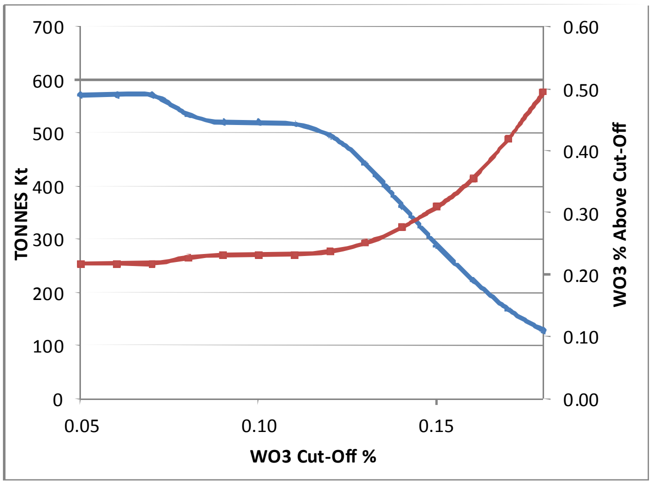
14.11 Mineral Resource Reporting

A grade-tonnage table of the in situ contents, for *Indicated* resources, is shown in Table 14-16. This is based on the selectivity in the inherent parent block size of the resource block model: 5m x 5m x 5m, although also heavily influenced by the zonal extrapolation applied, which will break blocks down within mineralised greisens and pipes potentially to 1m x 1m x 2.5m sub-blocks. The mined contents within the modelled historical underground excavations were removed.

Table 14-16. Grade-Tonnage Table – Indicated In-Situ Resources

As of August 31st, 2015

WO3 Cut-Off	Tonnes	WO_3	MOS_2
%	Kt	%	%
0.05	572	0.22	0.06
0.06	572	0.22	0.06
0.07	572	0.22	0.06
0.08	535	0.23	0.07
0.09	519	0.23	0.07
0.10	519	0.23	0.07
0.11	517	0.23	0.07
0.12	495	0.24	0.07
0.13	442	0.25	0.07
0.14	364	0.28	0.08
0.15	289	0.31	0.08
0.16	224	0.36	0.09
0.17	167	0.42	0.09
0.18	128	0.49	0.10



The overall in-situ resource evaluation results' breakdown is shown in Table 14-17, at a cut-off grade of 0.10% WO_3.

Table 14-17. Resource Evaluation – In-Situ

As of August 31st, 2015

	Greisen			Pipe Material			Total		
Category	Tonnes	WO_3	MoS_2	Tonnes	WO_3	MoS_2	Tonnes	WO_3	MoS_2
	Kt	%	%	Kt	%	%	Kt	%	%
Indicated	442	0.15	0.06	77	0.69	0.12	519	0.23	0.07
Inferred	1,829	0.15	0.07	602	0.69	0.11	2,431	0.29	0.08

Notes

. Cut-off = 0.10% WO3

. Historic underground mined material removed

Consistent with standard practice, a pit-constrained resource evaluation has also been developed, using the pit optimisation parameters summarised in Table 14-18. The corresponding resource figures are shown in Table 14-19. It should also be noted that there is a mineralised waste stockpile, containing 200,000t. Industrial tests completed last year indicate higher grades than originally estimated. The material is being treated through use of the XRF ore sorter and screened fines going directly to the processing plant.

Table 14-18. Optimisation Parameters – Pit Constrained Resources

Description			Unit	Values
Resources Enabled				**Ind+Inf**
General				
		$US:A$ conversion		**0.76**
Metal Prices				
		APT Price	*$/mtu WO_3*	**400**
		Contract reduction	*%*	**79%**
		Metal Price - received	*$/mtu WO_3*	316
			$/t WO_3	31,600
			$/t MoS2	**25,000**
		Transport cost	*$/tWO3*	**395**
Processing				
		Plant WO3 Recovery	*%*	**70.0%**
		LG sorter WO3 recovery	*%*	**85.0%**
		LG overall WO3 recovery		59.5%
		Plant MoS2 Recovery	*%*	**50.0%**
		Processing Cost	*$/t ore*	**15.56**
		G & A	*$/t ore*	**3.34**
		Ratio mill tonnes/crushed tonnes		**2/3**
		Total Applied Ore Cost	*$/t ore*	12.60
		(Processing+G&A+OreMining-WasteMining)		
Mining				
		Ore mining	*$/t ore*	**3.46**
		Waste mining	*$/t waste*	**3.46**
Mining Parameters				
		Mining Recovery		**90%**
		Dilution		**10%**
		Breakeven Economic WO3 Cut-Off - Low Grade		0.07%
		Breakeven Economic MoS2 Cut-Off		0.11%
Pit Parameters				
		Overall Pit Slopes	*To NE*	**48°**
			To S and W	**58°**

Key	
Bold	**Value supplied**
Normal	Derived
Yellow	Values used directly in optimisation process

Table 14-19. Resource Evaluation – Pit-Constrained
As of August 31st, 2015

	Greisen			Pipe Material			Total		
Category	Tonnes	WO_3	MoS_2	Tonnes	WO_3	MoS_2	Tonnes	WO_3	MoS_2
	Kt	%	%	Kt	%	%	Kt	%	%
Indicated	438	0.15	0.06	76	0.69	0.12	514	0.23	0.07
Inferred	1,337	0.16	0.07	541	0.70	0.11	1,879	0.31	0.08

Notes

. Cut-off = 0.10% WO3

. Historic underground mined material removed

. Prices used in optimisation:

US \$/mtu WO_3	400
US \$/t MoS_2	25,000

15 MINERAL RESERVE ESTIMATES

15.1 Pit Optimisation

The approach taken in the current study was to run through the updated resource block model with a series of pit optimisation runs. A base case set of optimisation parameters were developed with reference to current operating cost levels and parameters. It should be noted that these optimisation runs were not constrained by the mining lease, tailings dam or plant location. Nor do they fully reflect the potential of additional low grade greisen material in the northern half of the pit, owing to the sparser exploration data in this area. As well as the base case set of parameters, a set of sensitivities were also run. All of these parameters are summarised in Table 15-1. At the current time the recovery of MoS_2 has been not been considered.

Mining factors were also applied, of 10% dilution and 10% losses (90% mining recovery). The pit slope parameters were derived from geotechnical studies by Golder.

The sensitivities applied have been to enable the Inferred resources, and then to vary +/-10% the base case parameters for the metal prices, the mining cost and the processing cost. The results from all these optimisation runs, for the maximum cashflow pit in each case, are summarised in Table 15 2. A plan of the base optimal pits, along with the subsequently design pit, is shown in Figure 15-1. The optimal pit extents from runs 1, 2 and 4 are shown in Figure 15 2.

The pit base for the run based on Indicated resources only is at 525mRL. With Inferred resources also enabled, the pit base is at 495mRL, with a marked increase in the extent of the pit towards the north-west.

Table 15-1. Open Pit Optimisation Parameters

Description		Unit	Base Case	With Inferred	Price -10%	Price+10%	MC-10%	MC+10%	PC-10%	PC+10%
Run			1	2	3	4	5	6	7	8
Resources Enabled			**Ind Only**	**Ind+Inf**	**Ind+Inf**	**Ind+Inf**	**Ind+Inf**	**Ind+Inf**	**Ind+Inf**	**Ind+Inf**
Metal Prices										
	APT Price	*$/mtu WO_3*	**370**	**370**	**333**	**407**	**370**	**370**	**370**	**370**
	Contract reduction	*%*	**79%**	**79%**	**79%**	**79%**	**79%**	**79%**	**79%**	**79%**
	Metal Price - received	*$/mtu WO_3*	292.3	292.3	263.07	321.53	292.3	292.3	292.3	292.3
		$/t WO_3	29,230	29,230	26,307	32,153	29,230	29,230	29,230	29,230
		$/t MoS2	**25,000**	**25,000**	**25,000**	**25,000**	**25,000**	**25,000**	**25,000**	**25,000**
	Transport cost	*$/tWO3*	**395**	**395**	**395**	**395**	**395**	**395**	**395**	**395**
Processing										
	Plant WO3 Recovery	*%*	**70.0%**	**70.0%**	**70.0%**	**70.0%**	**70.0%**	**70.0%**	**70.0%**	**70.0%**
	LG sorter WO3 recovery	*%*	**85.0%**	**85.0%**	**85.0%**	**85.0%**	**85.0%**	**85.0%**	**85.0%**	**85.0%**
	LG overall WO3 recovery		59.5%	59.5%	59.5%	59.5%	59.5%	59.5%	59.5%	59.5%
	Plant MoS2 Recovery	*%*	**0.0%**	**0.0%**	**0.0%**	**0.0%**	**0.0%**	**0.0%**	**0.0%**	**0.0%**
	Processing Cost	*$/t ore*	**15.56**	**15.56**	**15.56**	**15.56**	**15.56**	**15.56**	**14.00**	**17.11**
	G & A	*$/t ore*	**3.34**	**3.34**	**3.34**	**3.34**	**3.34**	**3.34**	**3.34**	**3.34**
	Ratio mill tonnes/crushed tonnes		**2/3**	**2/3**	**2/3**	**2/3**	**2/3**	**2/3**	**2/3**	**2/3**
	Total Applied Ore Cost	*$/t ore*	12.60	12.60	12.60	12.60	12.60	12.60	11.56	13.64
	(Processing+G&A+OreMining-WasteMining)									
Mining										
	Ore mining	*$/t ore*	**3.46**	**3.46**	**3.46**	**3.46**	**3.11**	**3.80**	**3.46**	**3.46**
	Waste mining	*$/t waste*	**3.46**	**3.46**	**3.46**	**3.46**	**3.11**	**3.80**	**3.46**	**3.46**
Mining Parameters										
	Mining Recovery		**90%**	**90%**	**90%**	**90%**	**90%**	**90%**	**90%**	**90%**
	Dilution		**10%**	**10%**	**10%**	**10%**	**10%**	**10%**	**10%**	**10%**
	Breakeven Economic WO3 Cut-Off - Low Grade		0.08%	0.08%	0.09%	0.07%	0.08%	0.08%	0.07%	0.09%
	Breakeven Economic MoS2 Cut-Off		-	-	-	-	-	-	-	-
Pit Parameters										
	Overall Pit Slopes	*To NE*	**48°**	**48°**	**48°**	**48°**	**48°**	**48°**	**48°**	**48°**
		To S and W	**58°**	**58°**	**58°**	**58°**	**58°**	**58°**	**58°**	**58°**

Key

Bold	**Value supplied**
Normal	Derived
Yellow	Values used directly in optimisation process
Green	Values changed for sensitivity analysis

PC = Processing Cost
MC = Mining Cost

Table 15-2. Summary of Optimisation Results

Run	Description	Profit $M	Revenue $M	Processing Cost $M	Mining Cost $M	Rock Mt	Ore Mt	WO3 %	Waste Mt	Strip Ratio	WO3 Total Product t
1	Base Case	6.3	14.0	4.3	3.4	1.0	0.34	0.24	0.65	1.8	486
2	With Inferred	38.1	97.7	26.8	32.7	9.5	2.13	0.27	7.33	3.4	3,388
3	Price -10%	28.7	79.5	23.1	27.7	8.0	1.83	0.28	6.17	3.3	3,067
4	Price+10%	48.5	117.5	32.6	36.3	10.5	2.59	0.24	7.91	3.0	3,699
5	MC-10%	41.6	101.1	27.8	31.7	10.2	2.21	0.27	7.99	3.6	3,508
6	MC+10%	35.1	92.6	25.2	32.3	8.5	2.00	0.27	6.51	3.2	3,212
7	PC-10%	40.6	102.4	28.5	33.3	9.6	2.46	0.24	7.17	2.9	3,551
8	PC+10%	36.0	95.0	27.1	31.8	9.2	1.99	0.28	7.20	3.6	3,293

Notes

. BC Base case

. MC Mining cost

. PC Processing cost

. In each case the maximum cashflow pit results are shown

Figure 15-1. Plan of Base Case Optimal and Designed Pits

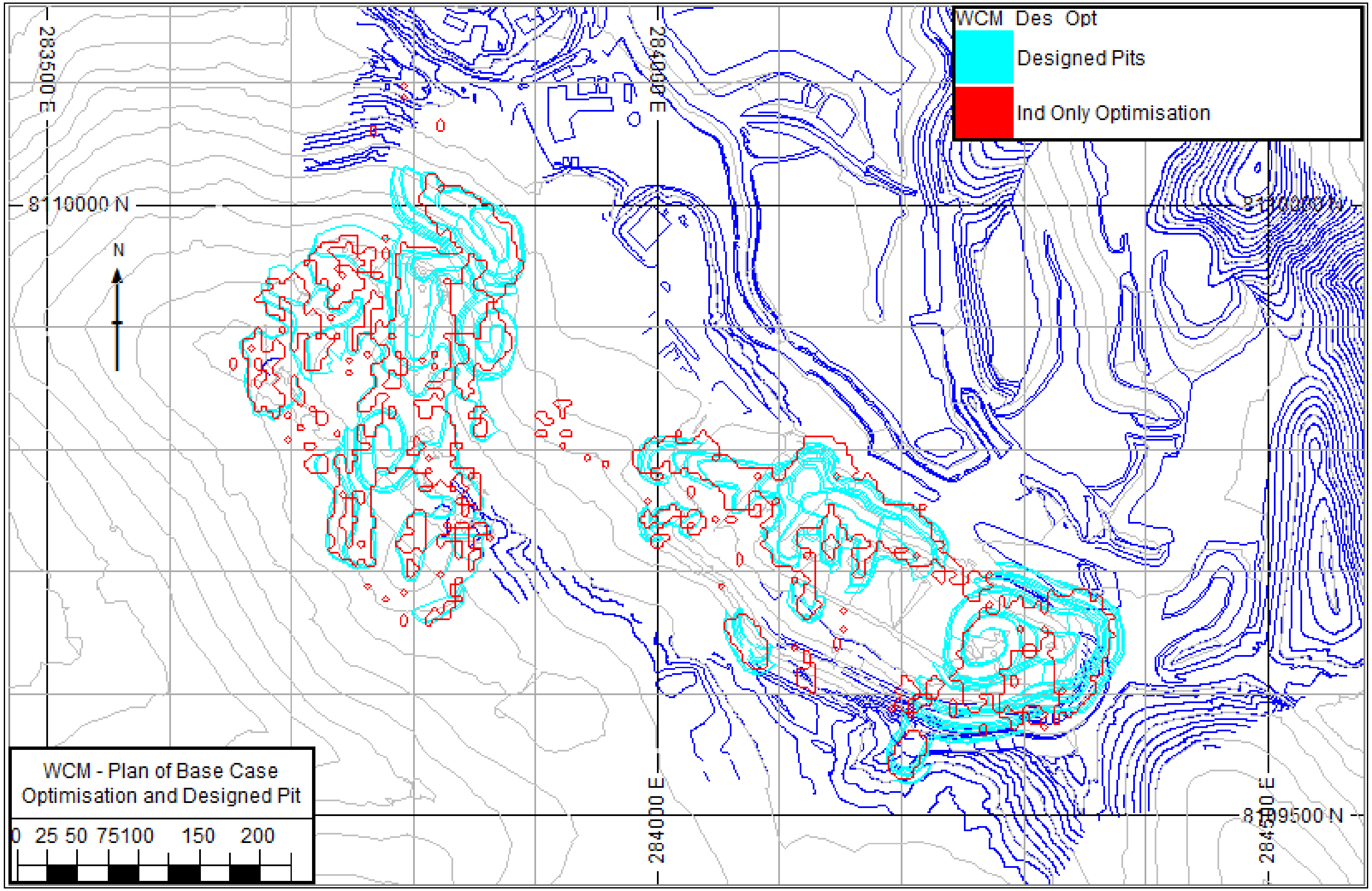
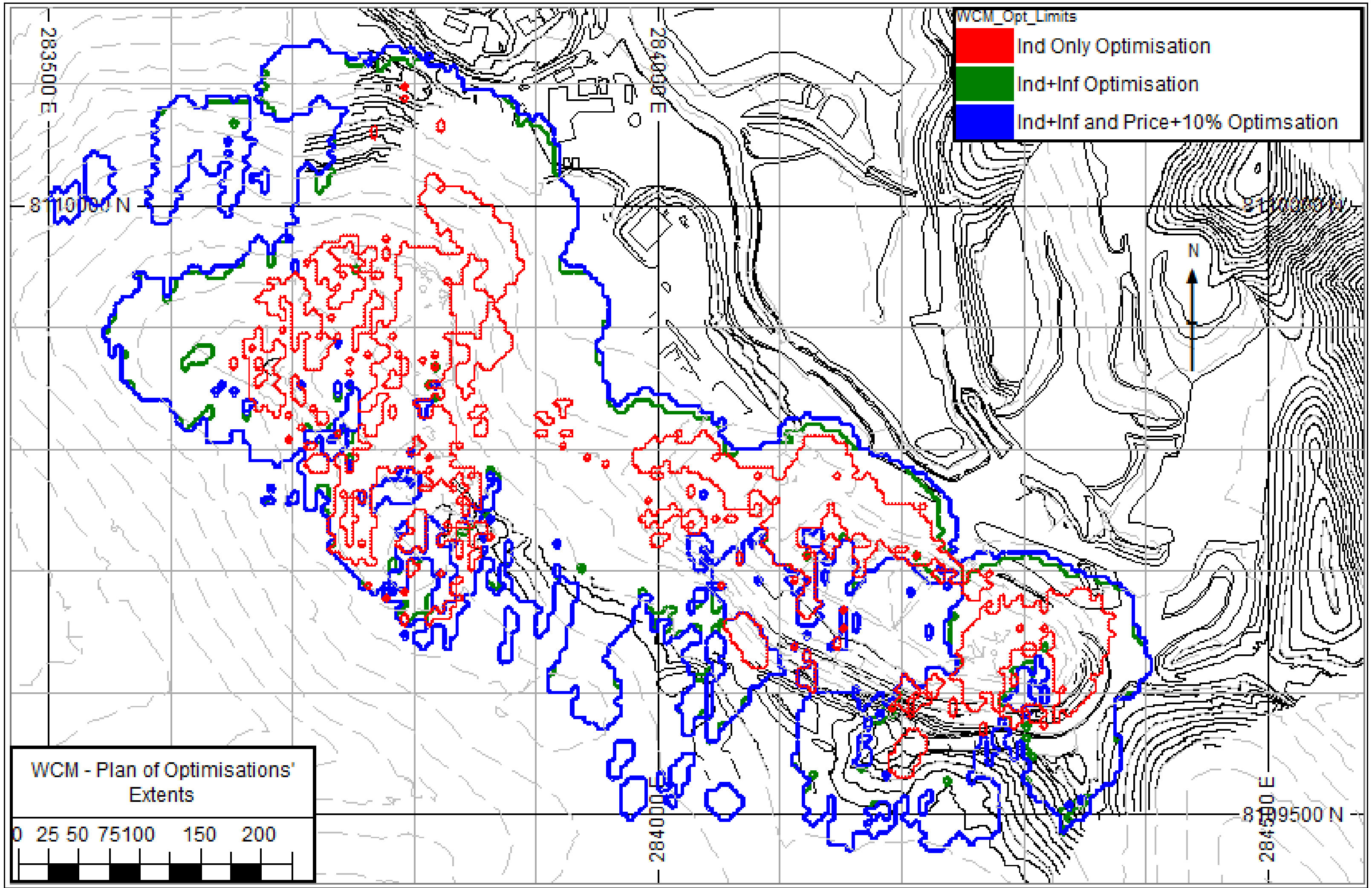


Figure 15-2. Plan of Optimal Pit Extents



15.2 Pit Design

The maximum cashflow pit shell, from the base case optimisation, has been used as the basis of an open pit design, as shown in the plan in Figure 15-3. This plan also shows spot elevations. The parameters used in the generation of this pit design are summarised in Table 15-3.

Table 15-3. Pit Design Parameters

	Unit	Value
Bench Configuration		
Face Angle		70^{o}
Berm Width, every 20m	*m*	5.5
Bench Height	*m*	10
Inter-ramp angle without ramps		57^{o}
Haul Road		
Gradient		10%
Width - Single lane, last 2 benches	*m*	10
Width - Double lane	*m*	15

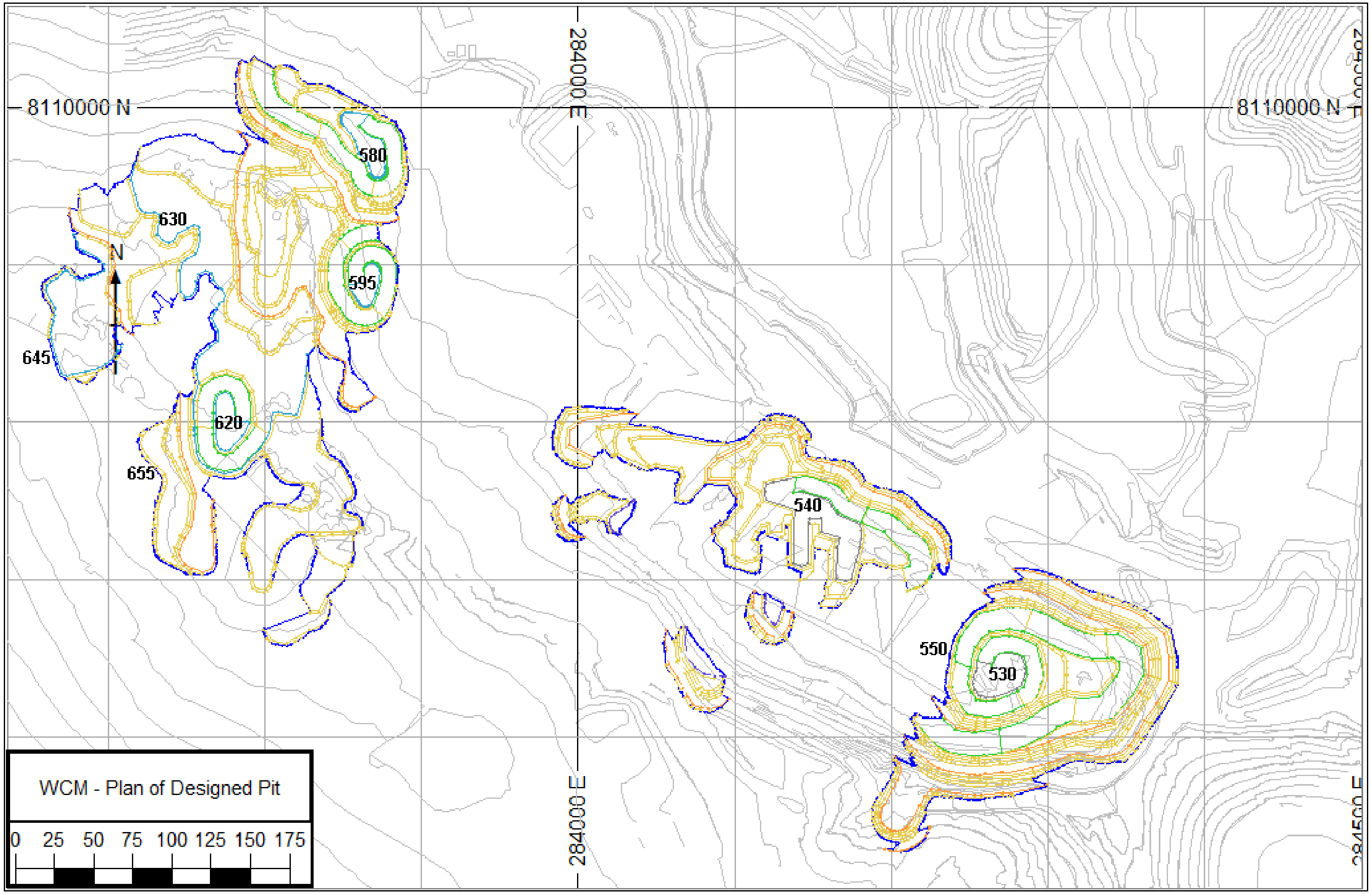
There are three main pit areas:

a) **Parrots.** This is the western most pit area, at higher elevations up to 645mRL.

b) **Central Area.** This is the central part, including three very small cuts into the southern slopes. The main central part goes down to 2 x 5m benches to 540m.

c) **Eastern Area.** This is the eastern part, an deepens the current pit floor at 550mRL down to 530mRL.

Figure 15-3. Plan of Pit Design



15.3 Mineral Reserves

The optimisation resulting from the base case set of parameters, with *Indicated*-only resources enabled, has been used as the basis for an open pit design and updated pit reserve evaluation. A plan of this design is shown in Figure 15-4. This pit design does not contain haul roads, as haul roads will be temporary and not affect reserve extraction.

An overall summary of the mineral reserves corresponding with this design is shown in Table 15-4. A summary of these reserves by bench is shown in Table 15-5. The pit zones used for the bench breakdown are depicted in Figure 15-4.

Table 15-4. Reserve Evaluation Summary
As of August 31st, 2015

	Probable Reserves				
	Tonnes Kt	WO_3 %	Waste+Inf Kt	Rock Kt	Strip Ratio
Parrotts	229	0.18	642	872	2.8
Central	82	0.25	373	455	4.5
East	63	0.30	541	605	8.6
TOTAL	**375**	**0.22**	**1556**	**1,931**	**4.2**

Notes . Cut-off = 0.08% WO_3
. Mining factors of applied of
Dilution = 10%
Losses = 10%
. Pit design also contain 187Kt of inferred resources at an economic grade of 0.25% WO_3

Table 15-5. Bench Reserve Summary

As of August 31st, 2014

PITZONE 1

BENCH	Probable reserves Kt	WO3	Inferred Kt	WO3	Waste Kt	Total Rock WO3
655	0.08	0.12	0.06	0.55	0.45	0.6
650	3.49	0.20	0.33	0.26	7.54	11.4
645	13.91	0.18	1.42	0.43	25.54	40.9
640	8.79	0.21	1.49	0.46	25.68	36.0
635	12.77	0.25	2.15	0.28	40.39	55.3
TOTAL	39.04	0.21	5.46	0.37	99.60	144.1

PITZONE 2

BENCH	Probable reserves Kt	WO3	Inferred Kt	WO3	Waste Kt	Total Rock WO3
645	1.46	0.38	-		2.34	3.8
640	1.84	0.31	0.06	0.65	6.94	8.8
635	9.52	0.24	0.66	0.38	29.27	39.5
630	23.13	0.25	1.96	0.32	72.95	98.0
625	38.77	0.18	6.61	0.32	67.78	113.2
620	23.67	0.16	10.51	0.32	40.07	74.3
615	13.65	0.15	4.09	0.39	12.55	30.3
610	1.03	0.23	0.45	0.56	3.63	5.1
605	1.45	0.28	0.56	0.22	1.67	3.7
TOTAL	114.52	0.20	24.91	0.34	237.20	376.6

PITZONE 3

BENCH	Probable reserves Kt	WO3	Inferred Kt	WO3	Waste Kt	Total Rock WO3
625	0.43	0.13	1.31	0.19	1.17	2.9
620	2.07	0.13	6.38	0.18	6.83	15.3
615	6.26	0.12	7.61	0.14	9.79	23.7
610	3.78	0.12	13.22	0.17	20.06	37.1
605	6.33	0.13	18.65	0.18	30.97	55.9
600	10.45	0.14	20.27	0.22	34.09	64.8
595	15.45	0.16	17.91	0.23	24.42	57.8
590	14.56	0.14	13.40	0.20	19.14	47.1
585	12.27	0.14	7.99	0.22	13.85	34.1
580	4.29	0.15	2.79	0.28	5.48	12.6
TOTAL	75.89	0.14	109.51	0.20	165.81	351.2

PITZONE 4

BENCH	Probable reserves Kt	WO3	Inferred Kt	WO3	Waste Kt	Total Rock WO3
595	0.21	0.11	0.24	0.24	2.08	2.5
590	0.74	0.16	0.39	0.32	4.61	5.7
585	2.58	0.22	0.12	0.33	7.69	10.4
580	2.01	0.21	0.33	0.48	6.91	9.2
575	0.70	0.20	0.02	0.63	4.58	5.3
570	3.66	0.24	0.20	0.39	28.83	32.7
565	9.76	0.21	0.39	0.39	49.31	59.5
560	15.53	0.23	0.92	0.41	43.16	59.6
555	17.38	0.26	1.27	0.45	42.45	61.1
550	12.54	0.29	2.01	0.44	43.80	58.4
545	10.18	0.26	2.92	0.39	54.00	67.1
540	3.01	0.24	2.31	0.32	21.98	27.3
TOTAL	78.29	0.25	11.17	0.39	309.79	399.2

Notes

. Cut-off = 0.08% WO_3

. Mining factors of applied of

Dilution = 10%

Losses = 10%

PITZONE 5

BENCH	Probable reserves Kt	WO3	Inferred Kt	WO3	Waste Kt	Total Rock WO3
625	-		-		2.28	2.3
620	0.001	0.54	-		7.17	7.2
615	-		0.01	0.15	4.67	4.7
610	-		0.05	0.21	5.42	5.5
605	0.18	0.64	1.42	0.33	6.05	7.7
600	1.15	0.40	0.27	0.31	4.45	5.9
590	0.10	0.22	-		0.54	0.6
585	0.65	0.34	0.25	0.22	3.83	4.7
580	0.39	0.27	0.05	0.09	1.47	1.9
570	-		0.02	0.69	2.43	2.4
565	0.10	0.59	0.18	0.42	4.27	4.6
560	0.76	0.34	0.30	0.32	4.75	5.8
555	0.65	0.39	0.12	0.22	1.41	2.2
TOTAL	4.00	0.38	2.66	0.31	49.04	55.7

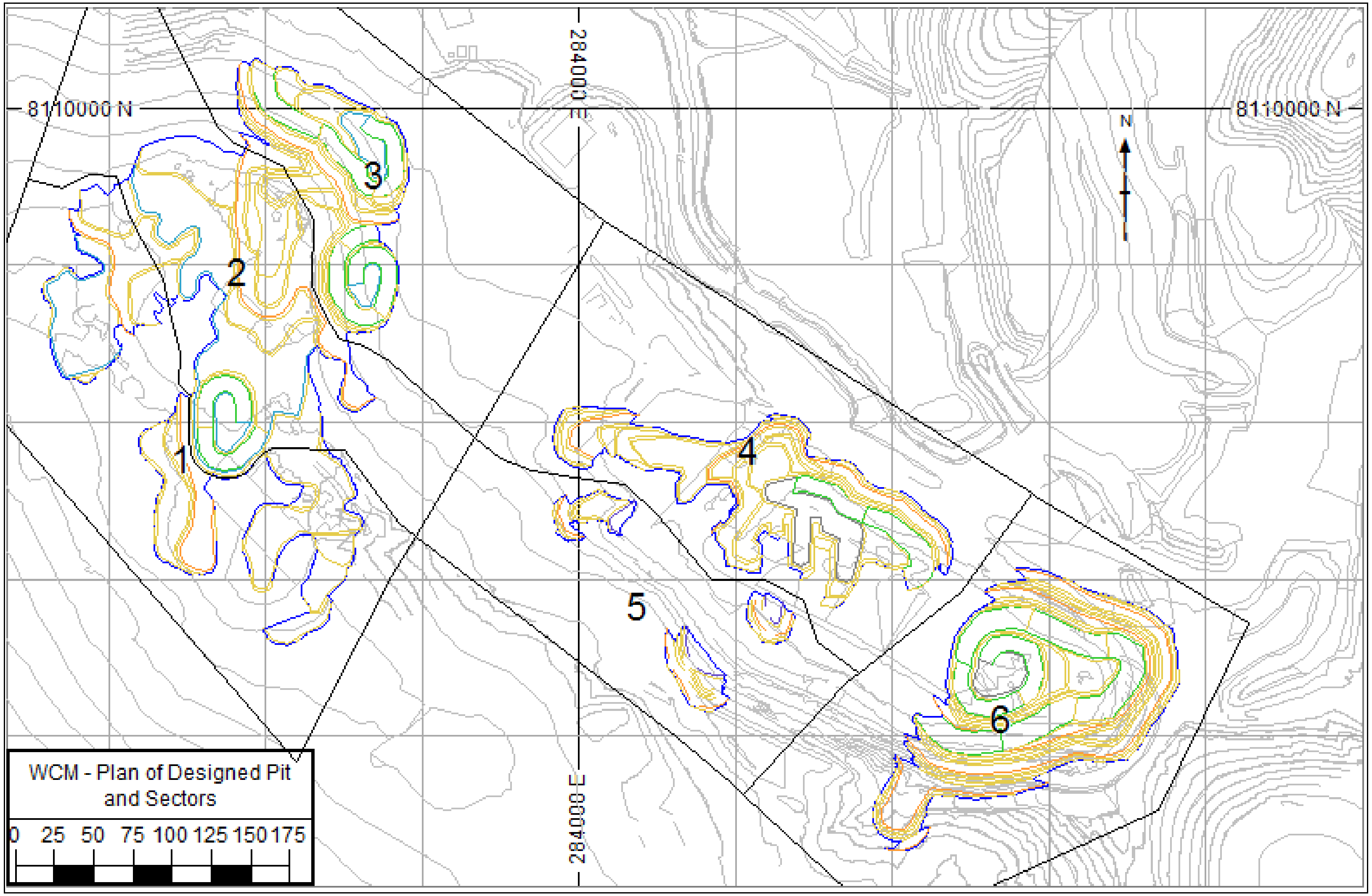
PITZONE 6

BENCH	Probable reserves Kt	WO3	Inferred Kt	WO3	Waste Kt	Total Rock WO3
615	-		0.06	0.58	2.66	2.7
610	-		1.41	0.13	10.27	11.7
605	0.64	0.72	3.73	0.28	12.56	16.9
600	2.03	0.64	3.58	0.25	14.03	19.6
595	0.01	0.79	0.10	0.34	5.90	6.0
590	0.32	0.38	1.24	0.18	15.73	17.3
585	1.65	0.22	2.99	0.21	36.81	41.4
580	1.80	0.22	2.01	0.19	47.58	51.4
575	2.25	0.21	0.74	0.21	35.06	38.0
570	3.49	0.21	1.20	0.29	38.17	42.9
565	5.07	0.24	3.44	0.36	51.02	59.5
560	6.27	0.27	3.31	0.30	48.43	58.0
555	6.00	0.26	2.81	0.26	32.95	41.8
550	4.85	0.25	1.89	0.35	28.60	35.3
545	11.93	0.30	1.15	0.33	58.39	71.5
540	8.63	0.30	1.11	0.39	36.58	46.3
535	5.64	0.33	0.82	0.44	21.54	28.0
530	2.59	0.48	1.81	0.32	11.21	15.6
TOTAL	63.16	0.30	33.37	0.28	507.98	604.5

PITZONE All

BENCH	Probable reserves Kt	WO3	Inferred Kt	WO3	Waste Kt	Total Rock WO3
655	0.08	0.12	0.06	0.55	0.45	0.6
650	3.49	0.20	0.33	0.26	7.54	11.4
645	15.37	0.20	1.42	0.43	27.88	44.7
640	10.63	0.23	1.55	0.46	32.62	44.8
635	22.29	0.24	2.81	0.31	69.66	94.8
630	23.13	0.25	1.96	0.32	72.95	98.0
625	39.21	0.18	7.92	0.30	71.23	118.3
620	25.74	0.16	16.89	0.26	54.57	97.2
615	19.91	0.14	11.76	0.23	29.67	61.3
610	4.81	0.15	15.13	0.18	39.38	59.3
605	8.60	0.21	24.36	0.21	51.26	84.2
600	13.64	0.24	24.18	0.23	52.95	90.8
595	15.67	0.16	18.24	0.23	32.69	66.6
590	15.72	0.15	15.03	0.20	40.02	70.8
585	17.15	0.17	11.34	0.22	62.19	90.7
580	8.49	0.18	5.18	0.26	61.44	75.1
575	2.95	0.21	0.75	0.22	39.65	43.4
570	7.16	0.22	1.41	0.31	69.43	78.0
565	14.93	0.22	4.01	0.37	104.60	123.5
560	22.57	0.24	4.53	0.32	96.34	123.4
555	24.02	0.26	4.20	0.31	76.82	105.0
550	17.39	0.28	3.90	0.40	72.40	93.7
545	22.11	0.28	4.07	0.38	112.39	138.6
540	11.64	0.29	3.42	0.34	58.56	73.6
535	5.64	0.33	0.82	0.44	21.54	28.0
530	2.59	0.48	1.81	0.32	11.21	15.6
TOTAL	374.9	0.22	187.1	0.25	1,369.4	1,931.4

Figure 15-4. Plan of Design Pit and Sectors



16 MINING METHODS

16.1 Operation

Mining at Wolfram Camp is by open cut, over a distance of approximately 800m along strike. The previous operations in 2007 conducted by Queensland Ores commenced mining to the south of the pit and only managed to mine several benches before placing the project on Care & Maintenance due to the falling price of tungsten and process design issues in the plant.

Clay and topsoil overburden from the mine is stockpiled separately from other waste dumps, and is to be used upon closure for rehabilitation. As the mine progresses to the north, additional material will be added to the existing stockpiles.

Wolfram Camp conducted further geotechnical evaluation of the open cut after the purchase by Deutsche Rohstoff AG through additional drilling and logging by Golders. A report was issued and subsequent cutback of the highwall was initiated. Mining production to date is approximately 900,000t of waste rock and 360,000t of combined low and high grade ore.

The open pit operations are conventional drill and blast, based on mining 5m benches in both waste and ore. Drilling and blasting on site is carried out by a specialist D&B contractor using one or two blasthole drill rigs, as shown in Figure 16-1. Blast patterns are drilled at 2.7m x 2.4m on 5m benches, using 89mm diameter blast holes. A down-the-hole blasting service is supplied by a local explosives manufacturer subcontracted by the D&B contractor, who transports the ANFO to site for each blast. The emulsion can be used in wet or dry holes. The drill and blast contractor provides the licensed shotfirer. Blasts are initiated using signal tube based detonators and primers.

The main explosives used is ANFO, with emulsions only the case of rain. Primers are 150g. Initiation is done using bellwire to electric detonators at initiation point to signal cord and then to connectadets.

The final high walls of the pit are defined by using perimeter blasting techniques to minimise potential wall damage. Presplitting in the granite rock involves drilling 89mm diameter holes parallel to the final wall face, 900mm apart. Every second hole is charged with 26mm diameter continuous length decoupled packaged explosives. The holes are not stemmed. Groups of presplit holes are fired simultaneously with consideration given to vibration and overpressure. In softer rock such as the metasediments on the east wall, trim blasting techniques are used.

The design of each blast considers power factor (quantity of explosives per tonne of rock), maximum instantaneous charge (MIC – maximum amount of explosive detonated per delay), sequence and order of initiation to manage heave, throw, direction of movement and vibration. This is carried out to minimise ore or metal loss and maximise recovery when mining, as well as minimising any potential damage to final wall and infrastructure.

An integral part of short-term mine planning is the use of grade control samples taken from blasthole drill cuttings. Blasts are planned over 5m bench heights, with combined ore and waste partitions. With sub-drilling, blastholes are usually 5-5.5m in length. Over each hole two samples are taken. The first from 0-2.5m, the second from 2.5m to final hole depth. A hydrocyclone removes fines (back dust) from the cuttings (approximately 20%), leaving the coarser material to be collected through a rig-mounted riffle splitter. Samples are taken from riffle splitter only. These samples generally contain material less than 9.5mm particle size.

The short-term planning block model, with WO_3 grades derived predominantly from GC drilling, as well as geological pit floor mapping, is used to delineate 3 different categories of potential ore material for mining:

- Mineralised waste, 0.07-0.12% WO_3
- Low Grade Ore, 0.12-0.3% WO_3
- High Grade Ore, >=0.3% WO_3

In addition to the blastholes for the basic 2.7m x 2.4m pattern, additional holes are also drilled, in which plastic hoses are placed for blast displacement monitoring purposes. Subsequent to blasting, the positions of the plastic hoses are re-surveyed, and the original ore/waste delineations are modified according to the measured displacements, as well as by visual assessment by geologists. Different colour ribbons are used to demarcate the different ore/waste categories.

A typical dig plan is shown in Figure 16-2. This shows the original ore outlines overlaid on the short-term planning model, as well as the displaced outlines after blast displacement. Digging of material is done with a backhoe-configured excavator, sitting on top of the broken muckpile, loading 40t trucks. Digging is done in 3 vertical passes: the first for the heave above the original bench floor, the second for the 0-2.5m depth cut and the third for the 2.5-5m depth cut. Ribbons are marked up individually for each cut prior to mining, based on the blast displacements at the top of each cut. Any additional high grade material spotted visually by geologists is also mined and stockpiled separately.

Figure 16-1. Blasthole Drilling Operations

Figure 16-2. Typical Dig Plan

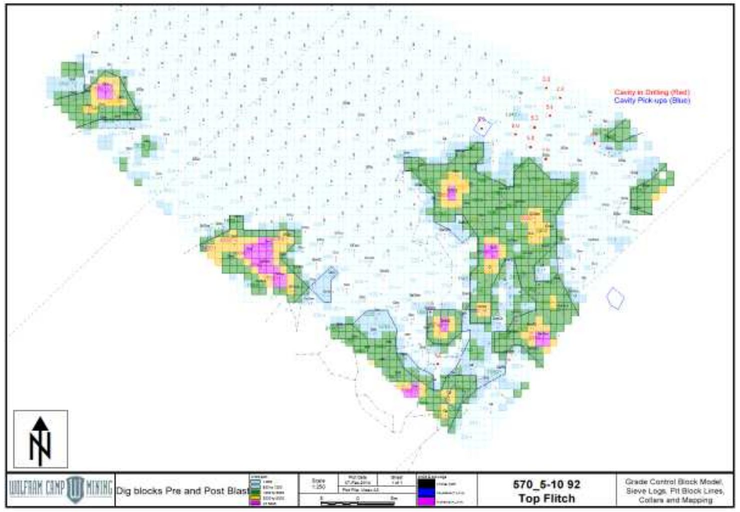


Figure 16-3. Muckpile After Marking Up

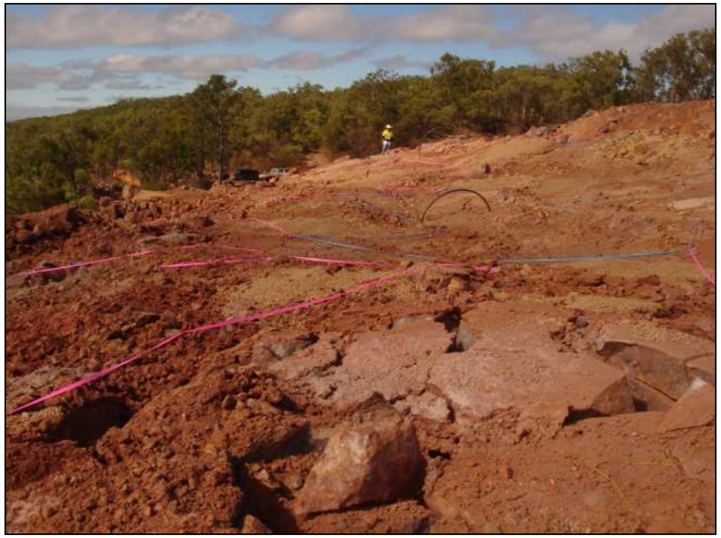


Figure 16-4. In-Pit Loading Operations

Waste and mineralised waste to hauled to stockpiles, and ore is trucked to the ROM pad adjacent the processing facilities. Mineralised waste is screened. Material sized 15-50mm is sent to the ore sorter, and -15mm material is direct fed to the mill.

Mined tonnages are reconciled against monthly stockpile surveys and these in turn are used to reconcile against the GC and resource block models.

The highwall on the west is excavated within competent granite at 70^{o}, while the eastern wall passes from sediments through the contact into granite and is excavated to a much shallower depth.

The target ore production rate is 500,000 tpa. The current reserve estimate gives a mine life of approximately 9 months. However, owing to the very erratic nature of mineralisation, and the relatively wide spacing of available exploration drilling, compared to the scale of mineralised structures, the proportion of *Inferred* to *Indicated* resources is high. As the pit advances with more blasthole sampling, progressively more reserves can be determined, approximately 25m beneath the base of the open pit at any time. Based on the optimisation results, where *Inferred* resources have been enabled, an open pit life of 4 years is suggested, before the additional contribution of potential extension zones.

The smallest possible selective mining unit size can be considered to be 1m x 1m x 2.5m. During reserve estimation, an additional mining dilution factor of10% has been applied, along with a mining recovery of 10%.

16.2 Equipment

Current mobile equipment at the mine consists of:

- 1 x 85 tonne excavator – Cat 385
- 1 x 87 tonne excavator – Hitachi 870
- 1 x 30 tonne Sumitomo excavator
- 4 x 40 tonne articulated dump trucks – Bell B40D
- 1 x water truck
- 2 x front loaders
- 1 x IT loader

Most of the mine equipment is now owned by WCM. Any excess over and above the agreed hours is paid on an hourly rate used. The supplier is responsible for all servicing of the equipment with the mine responsible for general wear & tear including GETs, fuels and oils.

17 RECOVERY METHODS

17.1 Introduction

The process plant is primarily based on gravimetric separation, aimed at recovering a high grade wolframite concentrate. During 2013 it was able to crush 369kt of material and (after ore-sorting) process 259kt of ore, with an average feed grade of 0.25% WO_3. With the planned processing improvements it is anticipated that the processing plant recovery will be 71%; and allow a mill capacity of over 518 ktpa.

17.2 Crushing and Grinding

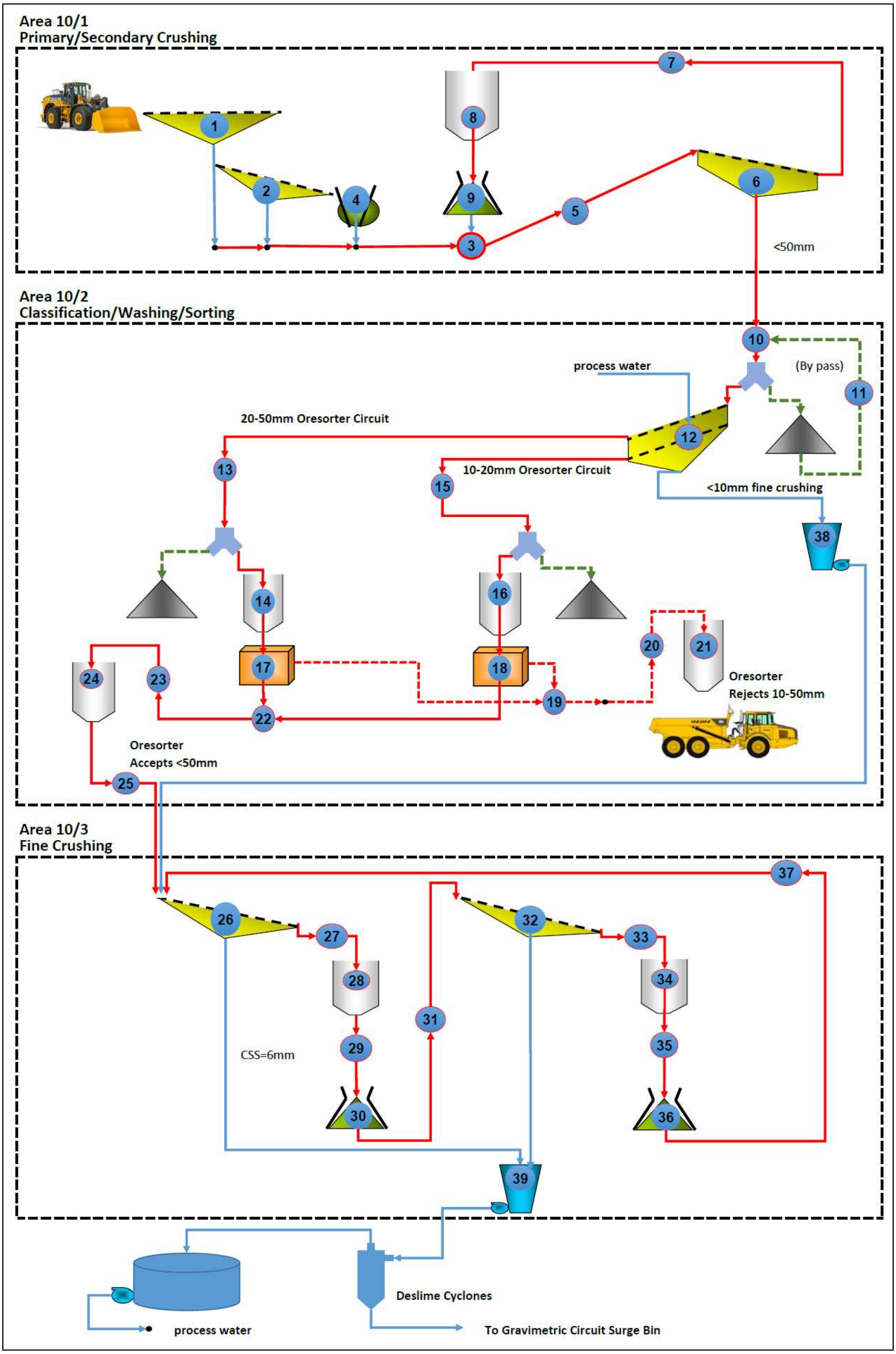
The design focus for recent updates to the plant process has been to improve the recovery of wolframite by minimising over-grinding of the ore. This will be achieved by increasing the number of ore crushing and screening stages, thereby improving the control of the grinding process. By removing the ball mill (as used previously) and using instead additional cone crushers, it is planned to avoid generating excessive quantities of ultra-fines. This is anticipated to significantly increase the recovery of wolframite in the gravimetric circuit and reduce operational costs. A flowsheet of the crushing and ore sorting circuit is shown in Figure 17-1.

The primary crushing circuit starts by screening the ore and feeding the jaw crusher with a grizzly feeder. The ore is then passed through a screen. The +50mm is fed to a secondary cone crusher working on close circuit with this screen, remaining ore is then classified on a washing screen. From the washing screen, the 25-50mm fraction is fed onto one XRT ore sorter, and the 10-25mm is fed onto the second XRT ore sorter. The ratio of the biggest to smallest particles, that are fed to each ore sorter, is kept below 3, to improve the efficiency and recovery of ore sorting. By feeding the ore sorters with washed ore, some operational issues from the past can also be improved.

The -10mm material from the washing screen will be fed to a wet screen of the fine crushing circuit. The ore sorter rejects are transported by dumpers to waste disposal. Studies are currently being carried to allow this and already stockpiled material to be sold as an aggregate to the construction industry, thereby minimizing the volume of waste to be managed on site.

The accepts from the ore sorters and the -10mm from the washing screen are fed to the fine crushing circuit. This circuit consists of two tertiary cone crushers, interleaved with two wet screens, working on close circuit. The objective is to obtain the required granulometry, while avoiding the generation of ultra-fines and the consequent losses in the gravimetric circuit. The accepted material is then pumped to the gravimetric circuit through deslime cyclones.

Figure 17-1. Crushing and Ore Sorting Flowsheet



17.3 Gravimetric circuit

The fine and coarse fractions pass onto two parallel banks of triple start spiral classifiers and from there onto Wilfley shaking tables. Recoveries from the tables have been recently further improved with the use of flotation frames with Xanthate to assist in sulphide removal.

A series of plant upgrades have occurred since May 2015, lifting the recovery. The upgrade provided with the installation of more10 Wilfley shaking tables and a Hydrosizer to provide separation of sizes through the middlings from tables to improve recovery. The gravimetric flowsheet, with the hydrosizer and shaking table arrangements, are shown in Figure 17-2.

Prior to 2014, material was also passed through a flotation circuit to produce molybdenite concentrate. This equipment is still in place, but has been by-passed since the end of 2013, as it was deemed uneconomic at that time.

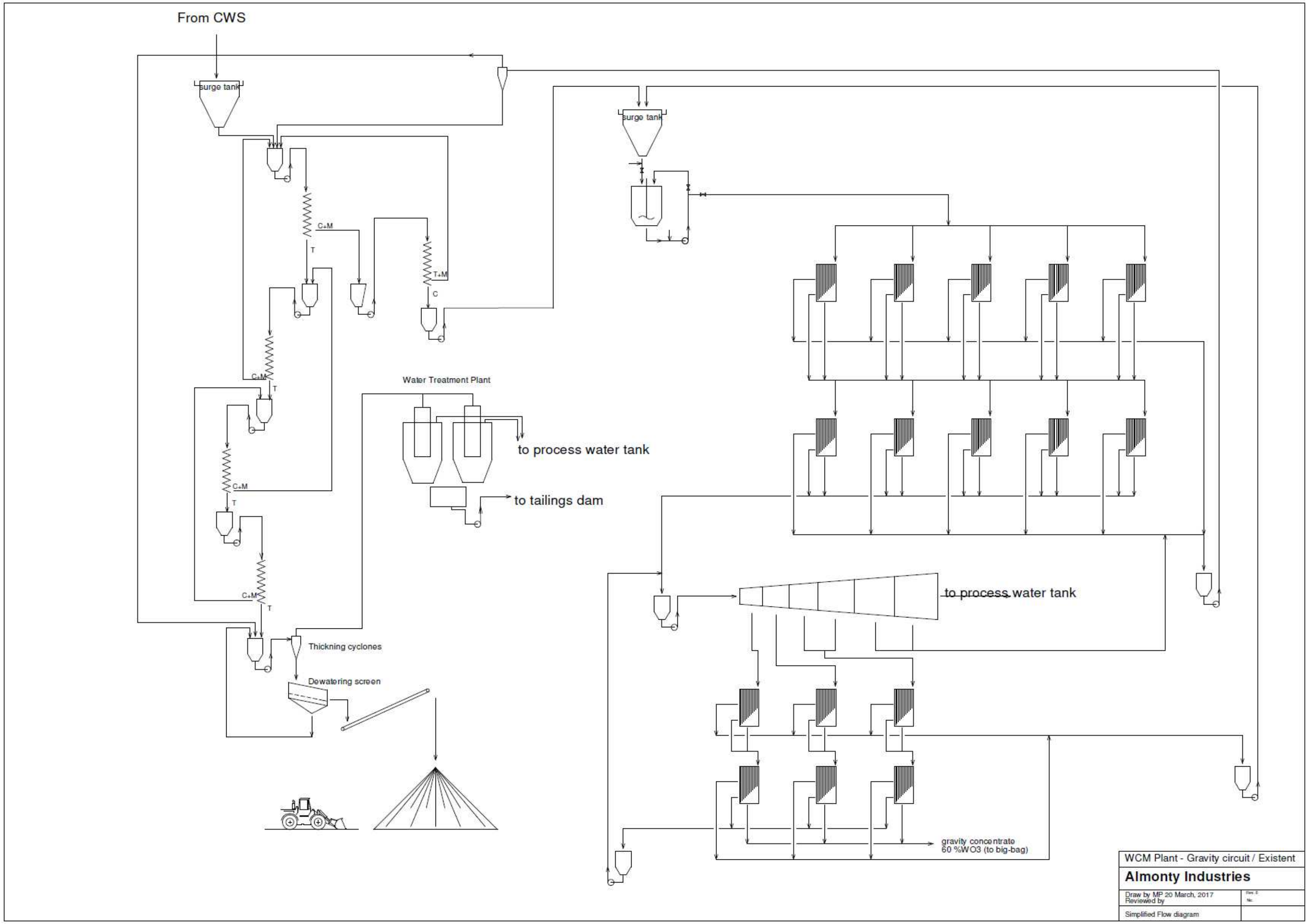
17.4 Dressing Plant

The concentrate from the shaking tables is subjected to batch flotation to reduce the fine sulfide content. The sulphide reduced concentrate is dried and cooled.

The accepted material is then transferred to the dressing plant. Here the material goes through a rotary diesel dryer, and from there onto a rare earth roll (RER) magnetic separator. The material is passed through the RER three times. The rejects from the RER, containing scheelite, are currently stored, but will be processed in the future with regrinding and flotation. The RER accepts are split into 3 streams. One stream with relatively high iron is passed through an electromagnetic (EM) unit at low magnetic settings. Low Fe material from the EM is blended back with the accepts from the RER. The high Fe material is retained and blended back when possible. The other 2 streams from the RER are bagged and assayed. Any material with high uranium and thorium (U+Th) is separated, and blended to allow the sale of acceptable concentrates.

Concentrate grades are typically 63% WO_3. This final saleable concentrate is bagged (weighed and sampled) and transported by semi-trailer to Brisbane. Once checks are finalised, the concentrate is loaded into sea containers and shipped direct from Brisbane to the port of New York in the USA. Sailing time is approximately 6 weeks, with sail schedules every 2 weeks.

Figure 17-2. Overall Flowsheet



17.5 Tailings and Fines

Tailings slimes from tables go back onto spiral classifiers to reduce losses. Final tailings from the spirals are pumped to cyclones and then to a de-watering screen. The course tailings are then transported to a stockpile and final deposition is dry staked with dumper trucks. It is planed that water with the fine tailings fraction goes thru a process water plant. Clean water is to be reused on the plant. The wet tailings material is pumped by pipeline to the tailings dam, just to the north of the mine and plant area. Material enters the dam via a perimeter discharge. The dam is also for used disposal of other waste processing water. Decanted water from the tailings dam is pumped back directly to the plant and used as process water. The mine water dam to the north is used to supply make-up water. There is zero discharge for the mine overall.

With new plant updates, a different type of tailings will be produced. Rather than a homogenous tailings, the tailings will be classified into a fine and a coarse fraction. The coarse fraction can then be dry-stacked efficiently, with the advantage of not occupying volume within the tailings dam. This material may also potentially be used to facilitate site rehabilitation works. It is expected that the fine fraction will represent approximately 10% of the tailings generated. Along with the mass reduction obtained using the ore sorters, means that the gravimetric circuit will only be fed with 55.6 % of the overall processed tonnage. Therefore, the fine tailings fraction tonnage will be 10% x 55.6% = 5.56 % of the overall plant feed. In the current model, the plant feed will be 518,400 tonnes/annum, so the fine tailings fraction will be 28,823 tonnes/annum. This much reduced fine tailings output reduces the priority in terms of developing a new tailings storage facility (TSF3).

Alternative methods for disposal of fine tailings are also being considered, which include use of a filter press to produce a filter cake, and possible use of the current Main Pit when it has been exhausted.

90% of the total tailings produced will be coarse tailings, which represents approximately 50% of the total plant feed. It is planned that the coarse tailings fraction will be stored downstream of the TSF2 embankment. This will initially permit the collection of the residual process water in the existing mine water dam (MWD) for re-use in the plant. During this stage, another water dam that is already approved as the Mine Water Management Dam (MWMD) will be constructed downstream of both the MWD and TSF3. This would permit the former MWD to be backfilled with tailings and subsequently permit use of the entire area between the TSF3 and TSF2 walls to be used to store coarse tailings. It would also provide a more suitable dam for water storage and recovery. A plan diagram of the TSF1/2 and TSF3 facilities are shown in Figure 17-4 and
Figure 17-5.

It is planned to utilise a water treatment plant to thicken the fine tailings fraction and to provide clarified water for re-use in the processing plant. Historically, the lack of clean recycled water has caused problems in the processing plant, with accelerated wear of equipment and diminished separation performance in the flotation and gravity circuits. It is anticipated that in addition to improving the physical properties of the water, there will be an increased water recovery efficiency, as the tailings are better dewatered prior to disposal.

Figure 17-3. Tailings Disposal Area

Regulations in Australia, require to have an available storage volume on 1 November (DSA - design storage allowance) for rainstorms, which is calculated as a function of the catchment area. This volume cannot be used to store tailings. In the amendment to the EA approved on 1 November 2016, the Blair pit (east end of the current Main pit) has been accepted and included as an authorized water storage structure.

The mine is using the TSFs and pit as an integrated containment system, which allows a reduction in the DSA volume on the TSFs. The current TSF1-2 has capacity of 10,000m^3. The first stage of TSF3 has a capacity of 203,550t of tailings. With the calculated annual fine tailings production of 28,823 tpa, this corresponds to 7 years of production. Further construction stages, lifting the embankment level of TSF3, will allow further increases in its capacity of up to 400%.

Figure 17-4. Plan - Tailings Storage Facility 1/2

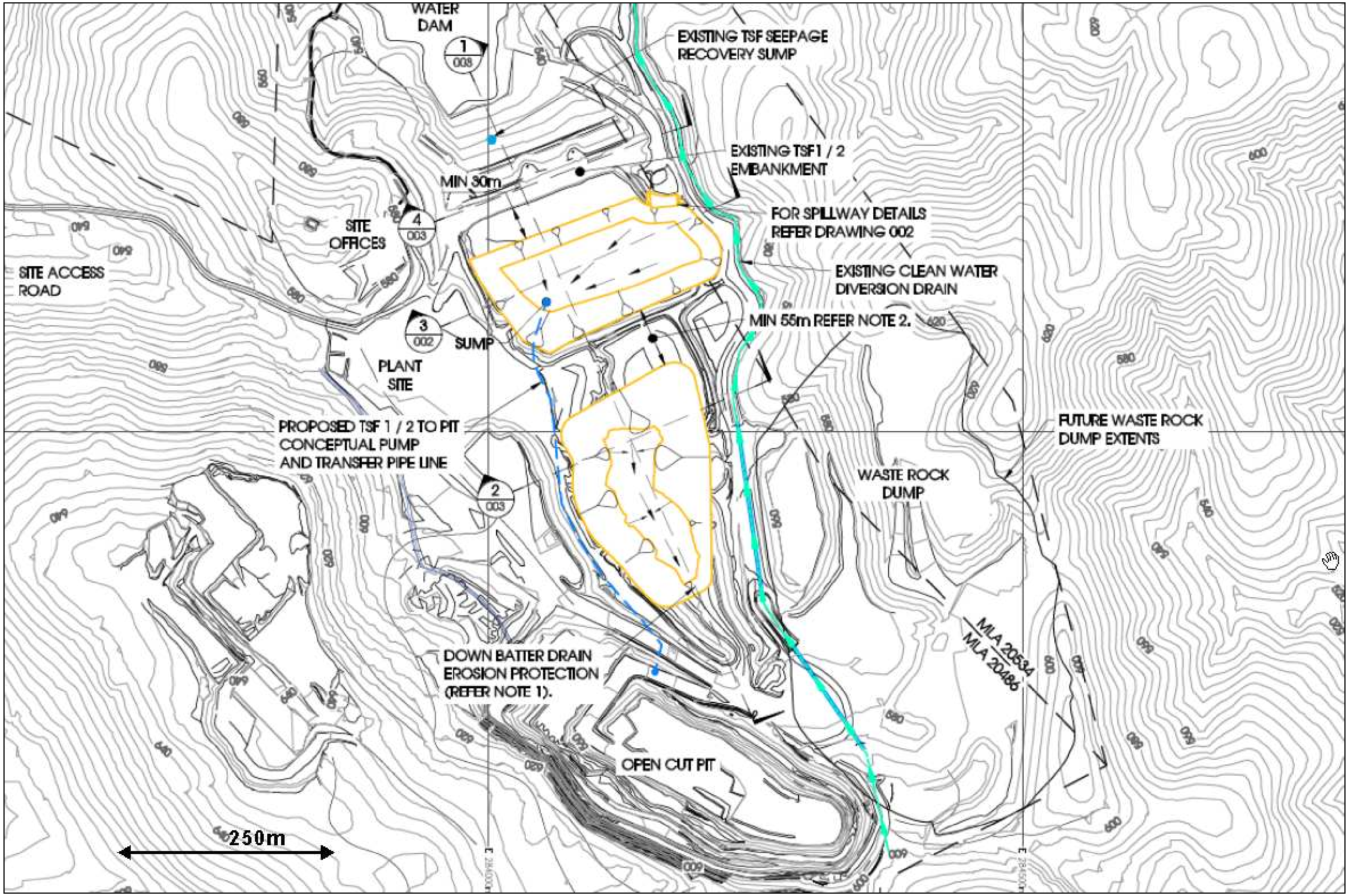
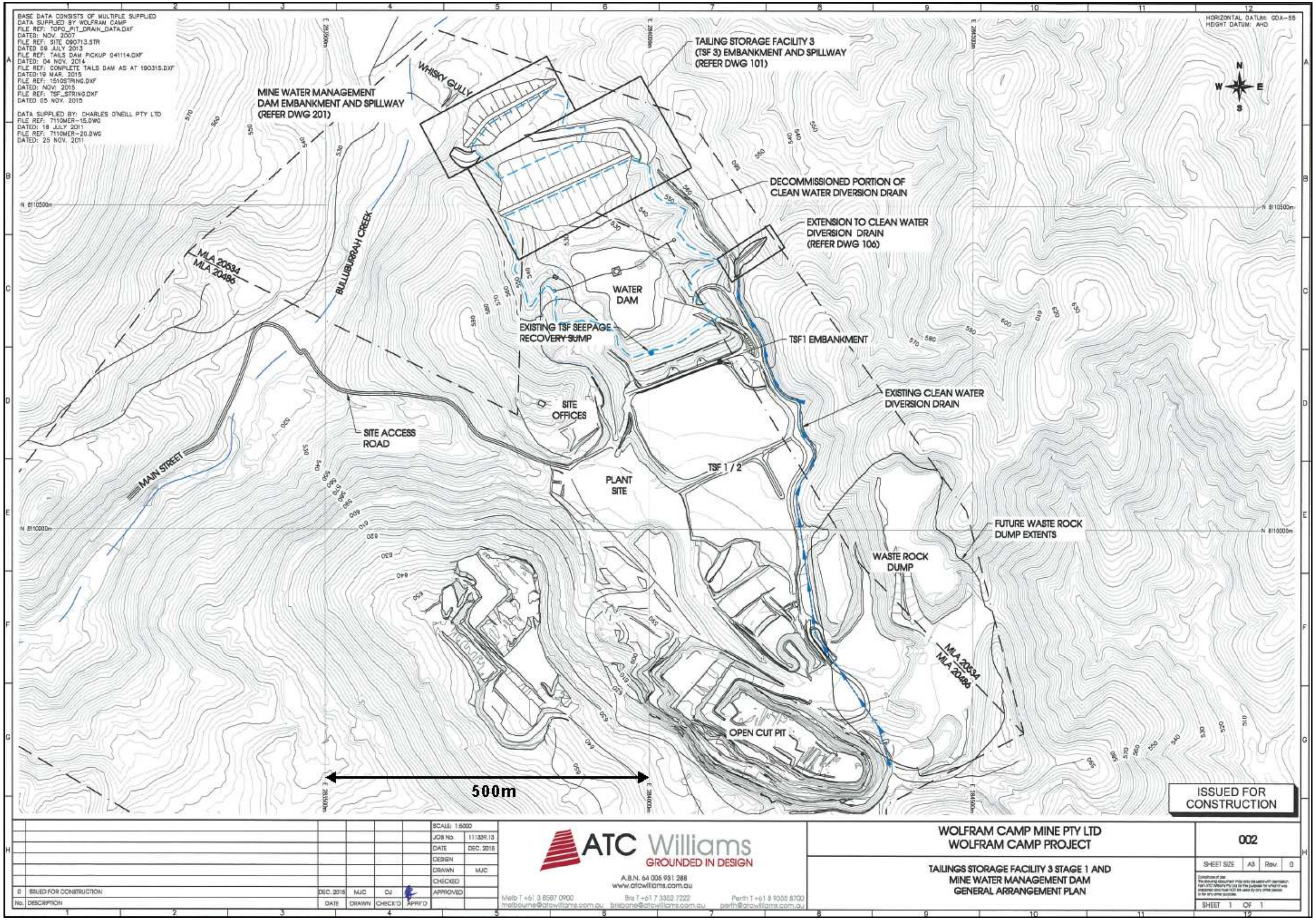


Figure 17-5. Plan - Tailings Storage Facility 3



18 PROJECT INFRASTRUCTURE

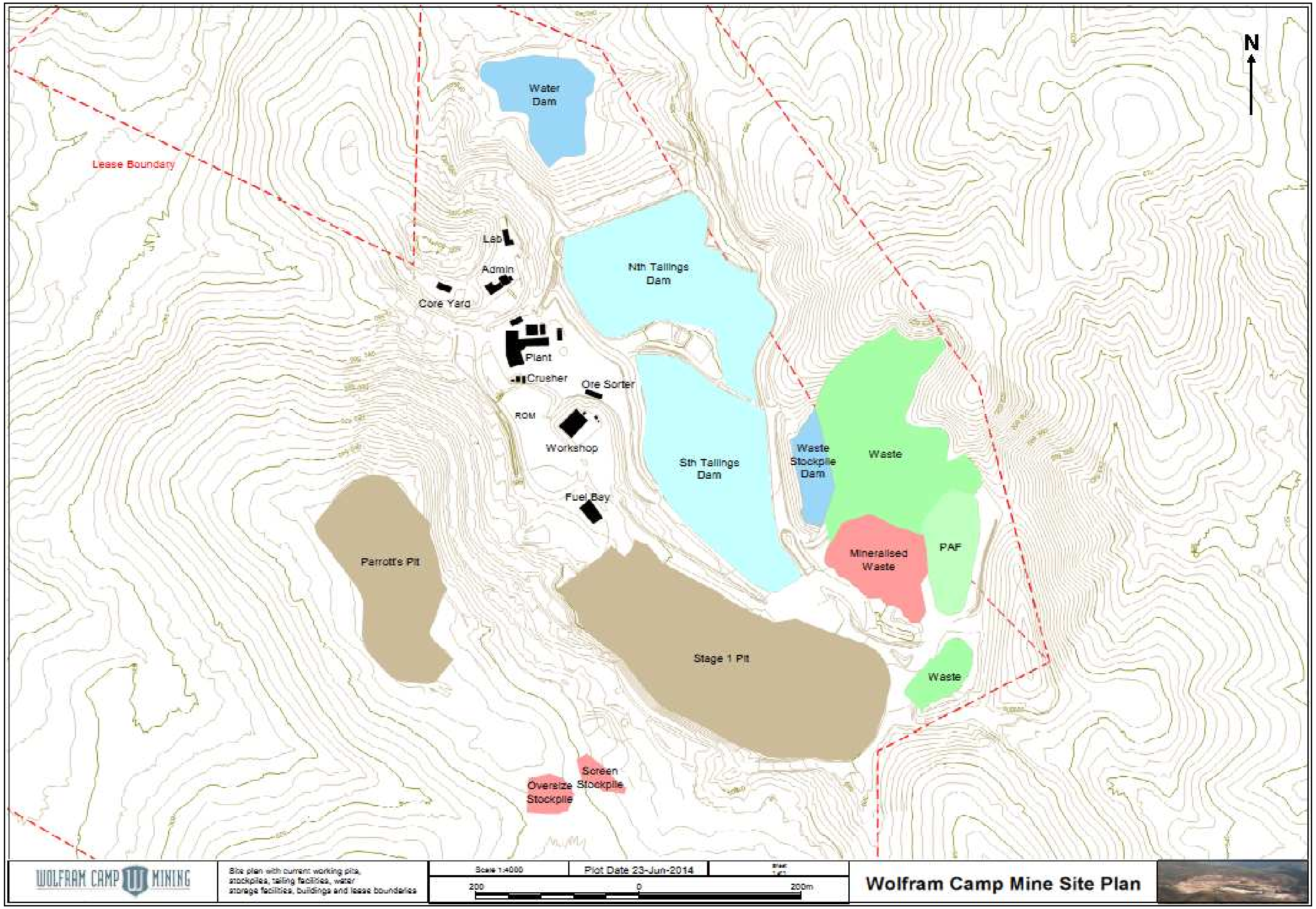
The site is serviced by an established 22 kV grid power line, waterline, telephone and road. The main transformer capacity is 1,500 kVA, which meets the installed electric capacity at the plant and also covers additional demand. The water allocation is covered by a resource operations licence, for 140 Mlpa, with the Mareeba Dimbulah Water Supply Scheme, as part of the Barron River Resource Operations Plan. The water supply for the plant is recycled water at a design rate 35% of supply water from the tailings storage facility before fresh water is used. The water dam capacity on site is 20 Ml.

Road transportation is used for all material, equipment and personnel transport to and from the mine. There is excellent road infrastructure to the mine site during the North Queensland dry season, which lasts from May to November each year. The road route from Cairns is 62 km from Cairns to Mareeba, 48 km from Mareeba to Dimbulah, 16km sealed road from Dimbulah to the old Wolfram Camp town site, and then a 9km dirt road to site. Many of the mine workers live in Mareeba and a work bus is provided every day for transport to and from the mine. During the wet season (December to April), very rare interruptions (1 day in 5 years) are possible at the Walsh River Bridge crossing in Dimbulah and on the gravel road at two causeway crossings (twice in the last wet season). When these interruptions do occur there is an emergency road which still allows site access. A compensation agreement between Mareeba Shire Council and Wolfram Camp covers the repair and maintenance the road up to the Old Wolfram Camp cemetery gate.

A site plan of the open pit operations, plant, tailings and stockpile areas is shown in Figure 18-1. The current open pit mining areas are in two parts: the principal open pit, measuring approximately 500m x 200m over the main Wolfram Camp deposit, the other in the area of the Parrott's pit which is about 100m to up the hill from the main pit. The mill buildings, laboratory and mine services are located just to the north of the main open pit. The tailings area is located in natural valley, which lies approximately 50m north of the main open pit, and runs roughly parallel to it. The tailings area is divided into two separate regions, with separate north and south tailings dams.

The main waste dumps are located off to the north of the western end of the main pit area. A separate potentially acid forming (PAF) waste dump is on the eastern side of the waste area. A separate mineralised waste dump area lies to the south of the dump area. This material has estimated grades of between 0.08 and 0.12% WO_3. A mobile screening plant is used to screen this material. The -15mm fine material is fed direct to the mill. Middlings +15mm – 50mm material is passed in a separate batch through the ore sorter. Oversize +50mm material is stockpiled, and then set to ROM and crushed.

Figure 18-1. Site Plan



19 MARKET STUDIES AND CONTRACTS

Wolfram Camp Mining entered into an exclusive agreement in 2011 with Global Tungsten & Powders for all tungsten concentrate produced from the mine. This contract calls for delivery, in 2t Bulk Bags, in standard container lots per consignment, of tungsten concentrate grading plus 60-65% WO_3.

As per standard industry practice, the price paid per tonne of concentrate is based on the number of contained metric tonne units ("mtu") of tungstic oxide (WO_3). This unit price varies for individual consignments according to the prevailing ammonium paratungstate (APT) price as published during the week of shipment in Metals Bulletin magazine ("the Metals Bulletin price").

The details of the contract, including the APT discount rate applicable, are strictly commercial-in-confidence and may not be disclosed, but equate to industry norms.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT

20.1 Environmental Management Plan (May 2007)

20.1.1 Introduction

This Environmental Management Plan, produced by **Natural Solutions Environmental Consultants**, covered **ML20486 and ML20534**; the information provided in the report was derived from a number of specialist technical reports including:

- Preliminary Waste Rock Assessment for the Wolfram Camp Tungsten-Molybdenum Project, Environmental Licensing Professionals, May 2006**;**

- Baseline Environmental Report for the Wolfram Camp Tungsten- Molybdenum Project, Environmental Licensing Professionals, March 2006**;**

- Potential Aboriginal and Historic Cultural Heritage reports, Gordon Grimwade and Associates, March 2006;

- Baseline Flora and Fauna Study- Wolfram Camp Project, Landline Consulting February 2006 and

- Various Reports from Coffey Mining and Coffey Geotechnics as listed below:

  • Mine Pit Water Management
  • TSF Water Management Strategy report
  • Conceptual Pit & Waste Rock Dump Design Report
  • Preliminary Water Supply Report
  • Design Report Tailings Storage Facility

- Radiation Advice and Solutions Report

- Environmental Impacts from Mining Activities

The following activities were listed for the Mining Lease area:

- Pit excavation;
- Product and topsoil/overburden stockpiling;
- On-site processing;
- Sediment control works;
- Limited fuel, diesel and explosive storage; and

- Access tracks.

All the activities listed above were considered to have only a minor impact on the environmental values of the area.

20.1.2 Description of Environmental Values and Potential Impacts

20.1.2.1 General

Sewage would be disposed of using a septic system and will be designed and operated in accordance with Mareeba Shire Council guidelines. Should the capacity require expansion to in excess of 21 equivalent persons, a registration certificate would be sought from EPA.

20.1.2.2 Air Quality

Local airshed (the volume of air receiving emissions which predominantly affects a specific watershed or catchment) is of generally good quality, compromised only infrequently by dust and smoke from burning off from surrounding grazing lands.

It was not considered likely that the mining operation would significantly impact upon the air quality of the surrounding land users. In addition, the scattered vegetation would possibly aid in the passive control of dust.

20.1.2.3 Water Management

Wolfram Camp lies amongst granite-sediment hills to the north of the Walsh River. The Walsh River is the main drainage in the region and supports intensive farming operations. There are no such activities near the Wolfram Camp site, the nearest being 3km from site. Bulluburrah Creek flows through the western end of the ML, whilst small gullies at the south eastern corner of the MLAs lead eventually into James Creek. Both Bulluburrah Creek and James Creek are ephemeral.

Over the years, large amounts of waste rock have been dumped across Whiskey Gully so that waters from the upper part of the catchment flow into a series of water/waste management structures consisting of a sediment pond, tailings storages and waste rock dumps/embankments and a water dam. Flood waters and seepage from the water dam flow to the northwest *via* a downstream dam to Bulluburrah Creek. However, waste rock is expected to present a low risk of net acid production and insignificant source of contaminants such as arsenic.

Fluoride levels exceeded stock watering limits in the Water Dam and seepage also triggered the EPA hazardous dam criterion. The presence of fluoride can be attributable to the granitic nature of the geology in the area including the ore previously processed on site. It is likely that the high levels of fluoride in stored surface waters on the site are due to the acid produced during the mining/processing of the ore/waste rock lowering the pH and subsequently resulting in the increased solubility of the fluoride (occurring in the ore).

Arsenic which is present in the granites as arsenopyrite becomes mobilised under alkaline conditions; however the resulting concentrations in surface waters were at the time of the site inspection below stock watering guidelines.

Wet season groundwater discharge from the site *via* the Forget-Me-Not Decline was within accepted water quality criteria; however the Forget-Me-Not Tunnel was high in fluoride. The fluoride concentration at the decline (1.9mg/l) was slightly below the stock water limit of 2mg/l whilst in the Forget- Me-Not Tunnel (4.5mg/l) it was above the stock water limit.

The water quality in Bulluburrah Creek did not appear to be impacted by the historical mining and mineral processing operations and waters discharging from the former mining and processing areas.

Also contamination of both surface and groundwater was not expected to occur as a result of mineral processing, with appropriate management controls in place

The baseline studies concluded that whilst existing stockpiles of ore and possibly some waste rock and tailings exhibit acid generating potential, there is sufficient dissolved carbonate within local catchment runoff to maintain acceptable pH conditions (>6.5) within the affected part of the Whiskey Gully catchment.

Overall, the analytical results from soil, rock and water samples identified that although there is acid producing ore and possibly potential acid producing waste rock and tailings stored on the mining lease, their presence did not appear to have significantly impacted on water quality in stored waters, apart from fluoride levels. This is likely to be due to:

- the small degree of acid production;
- the small amounts of actual acid producing material exposed to water and oxygen; and sufficient buffering capacity in surface water as a result of natural carbonate content in the granitic rocks and the run-off volume in the Whiskey Gully catchment.

It was expected that laboratory determination of Net Acid Generating Potential (NAGP) and Non Acid Generating Potential (NAG) should reliably predict net acid formation behaviour.

Thorium and Uranium levels were consistent with the natural average level in granites, and therefore radioactivity was not considered to be a significant human health or environmental issue at the operation. A report by Radiation Advice and Solutions confirmed that *radiation* is unlikely to present a hazard to workers and the environment at the site.

20.1.3 Noise and Vibration

Due to the small-scale and remote location, it was considered unlikely that the mining operation would impact upon the background noise level of the surrounding land users. In addition, the scattered vegetation would aid in natural control of noise.

20.1.4 Waste Management

The environmental protection objective for waste management would be managed to avoid any direct or indirect impacts on the health and well-being of the people and the environment surrounding the mine site. This objective was to be achieved through the implementation of the following control strategies:

• Recycle and reuse waste material where practicable;
• Appropriate disposal of waste off-lease in designated facilities; and
• Use a licensed specialist facility for off-lease disposal of waste oil

20.1.5 Land and Management

The majority of the project area was considered to be Class 4 pastoral lands at best, with the steep lands in the area of the pit Class 5 unsuitable for any agricultural purposes.

A series of unrehabilitated previous mining operations left the land in a degraded state, with very little soil cover in the proposed pit area resulting in significant erosion in some areas and weed infestation.

Due to the extensive disturbance created by historical mining in the area, it was expected that little impact would occur to habitats and flora and fauna species. The areas of proposed disturbance were characterised by various stages of regrowth.

Currently, a Post Mine Land Use Plan (PMLUP) is being developed. This plan will include these strategies to manage the tailings (coarse and fine fractions) and other waste materials on site. Concurrent with the development of the PMLUP, we are performing an assessment of the acid forming potential of the materials currently stocked on the site. It seems very likely that the rejects from the ore sorter will be considered non-acid forming, considering old analyses and the feedback of the environmental scientist that is conducting the study. This will provide an opportunity that will allow us to commercialise the sale of material locally with the benefit of increasing revenue, recycling a waste material and reducing the volume of material to be stored on site. A further benefit would be that the residual material can be deposited with less restrictions on site, minimising the cost of rehabilitation and future FA.

The construction of the TSF3 will not be unnecessary in the short term. In 2016, WCM designed an integrated containment system that could utilise existing containment structures on the site with only minor modifications to serve the immediate purposes. At a significantly lower cost than the construction of a new TSF, an integrated containment system was constructed utilising the Main Pit and the existing TSF2. A suitable volume of material was excavated from the current TSF2 and dry stacked on the southern end of it. The volume created within the existing TSF2 was used in conjunction with Main pit to form an integrated containment system. A passage in the underground historical works under the main pit was sealed to render the Main pit fit to contain mine affected water. The TSF2 and Main Pit structures were then connected by pumps and pipework to permit the transfer of water as required. This avoided the premature construction of a new TSF, and addressed the site EA requirements relating to providing sufficient design storage allowance (DSA) volume on site, to contain wet season run-off of mine affected water. A number of in-house designed and fabricated water evaporation systems have been implemented to reduce the volume of mine affected water stored on site whilst there is no production occurring. When production resumes it is anticipated that there will be a net consumption of water.

20.1.6 Community

Due to the moderate scale of the proposed mining operation, it was not considered that there would be any detrimental impact upon social or economic scenarios. In fact, the local community of Dimbulah was likely to be boosted by an increase in potential job opportunities and ancillary services as a result of the mining operation.

Searches conducted at the EPA indicated that the proposed mining area is not contained on any heritage registers. A heritage place does exist approximately one kilometre south west of the southern boundary of the MLAs and was not expected to be affected by mining activities.

20.1.7 Monitoring

A comprehensive monitoring programme to measure variables relevant to licence conditions was to be developed to track the environmental performance of the mining activity which might impact upon compliance with the EA conditions. This is detailed in the Plan of Operations (refer 20.2 below) which is based on the commitments and strategies provided above.

20.2 Plan of Operations

(January 2017 and valid for 5 years)

The current Plan of Operations (PoO) was prepared consistent with the following:

- Schedule of Conditions***** of the WCM Environmental Authority (EA) EPML00831213 (previously MIN102648011, dated 7/8/12).
- Section 288 of the Queensland Environmental Protection Act 1994.
- Queensland Department of Environment and Heritage Protection (EHP) guideline EM1010..
- Financial assurance under the Environmental Protection Act 1994, Version 2 (EHP 2014).

*****_Schedule of Conditions. The environmental authority consists of the following schedules of conditions relevant to various issues:_

Schedule A	*General*
Schedule B	*Air*
Schedule C	*Land*
Schedule D	*Waste*
Schedule E	*Noise*
Schedule F	*Water*
Schedule G	*Definitions*
Schedule H	*Maps*

[*Note: The Environmental Authority* EPML00831213 *has been seen by Adam Wheeler]*

20.3 Environmental Audit Statement 03/09/2012

20.3.1 General

NRA Environmental Consultants (NRA) was requested by Wolfram Camp Mining Pty Ltd (WCM) to undertake an audit to accompany the current Plan of Operations (PoO).

Reference was made to relevant Department of Environmental and Heritage Protection (DEHP) guidelines noted above in the PoO *viz*:

- Preparing a plan of operations and audit statement for level 1 mining projects (09/07/2012)
- Calculating financial assurance for mining projects (23/12/2011).

The DEHP also conducts annual audits.

20.3.2 Rehabilitation Programme

Existing areas of disturbance have been determined by WCM. No rehabilitation is planned during the term of this PoO. In the event of site closure during the term of the PoO a rehabilitation schedule has been prepared and is summarised below. For the purpose of costing rehabilitation activities, seven disturbance categories are recognised. The disturbance categories together with estimates of the cost of rehabilitating each to final rehabilitation status are provided in the PoO.

The rehabilitation costing for each disturbance category is based on the following rehabilitation strategies:

- Pit – Rehabilitation will involve drainage to minimise catchment area and construction of safety bund and fencing.

- Ore Stockpile – remaining ore will be removed to the TSF or pit.

- Waste Rock Dump (WRD) – Rehabilitation will involve a cover of NAF waste (run of mine) topped by 1m capillary break and 2m inert waste (EMPlan 2007). Topsoil will be applied if available. Drainage will be to natural ground to north and north-west. No chutes are proposed. The surface will be ripped and seeded. NOTE: A conservative area of 2.9ha has been allowed for WRD.

- Tailings Storage Facility (TSF) – Rehabilitation will involve pumping water to the pit or evaporation (depending on quality), 1m capillary break, 0.5m compacted low permeability overburden/clay, 1m capillary break and 2m paddock dumped inert overburden/soil (EMPlan 2007). Drainage will be provided to the existing spillway. The surface will be ripped and seeded. NOTE: A conservative area of 5.5ha has been allowed for the TSF, including the historic tails deposition area.

- Plant Site and Administration – Plant and administration buildings will be removed from site for sale.

- Water Dam – Water Dam pumped out and decommissioned. Walsh River pumping system sold to local landholders.

- Roads, Access Tracks and Exploration – Some tracks required for monitoring and maintenance will be retained. Exploration areas and access ways no longer required will be stabilised and revegetated.

20.3.3 Financial Assurance

The Financial Assurance (FA) calculation was provided by WCM. The following comments are reported.

- The list of rehabilitation items to be addressed appears appropriate.

- WCM reports that third party rates applied and allowance for maintenance and monitoring (2%), CPI (2.1% per year2) and GST (10%) are included. The CPI rate is not consistent with the current guideline (CPI of 3%), but at the time it was the actual CPI in Australia and so was accepted by the statutory authority.

- The total calculated rehabilitation liability presented in the PoO was AUD2,528,500.

  The applicable performance category (with reference to the DEHP guideline) is considered to be Category 4 (i.e. nil).

- The State currently holds a financial assurance of AUD$ 1,308,767 for the WCM operation; an additional amount of AUD$ 1,219,733 is required to be lodged as an additional security bond.

20.3.4 Conclusions

The certified Principal Environmental Auditor (NRA Environmental Consultants) made their assessment from the available evidence of:

a) Compliance of the Plan of Operations with the conditions of the EA, the qualifications comprising:

1. The planned activities in the PoO in relation to waste management are not consistent with EA.

- Bioremediation of hydrocarbon waste will be undertaken on-site. This is considered to be best practice environmental management and more appropriate than disposal of the material as a regulated waste.
- Volumes of general waste less than 50tpy will be disposed of on-site. These volumes are less than the threshold nominated for ERA in the *Environmental Protection Regulation 2008*.
- Explosive packaging material will be incinerated on site, as is practiced widely in Queensland.

2. The Action Programme presented in the PoO presents an acceptable approach to addressing the requirements of the EA.

3. The TSF operating plan requires review, and should be updated as required, with consideration of the TSF remediation works undertaken and any future alterations to the TSF.

b) The Accuracy of the Third Party Costs for Rehabilitation, the qualifications being:

1. Cost estimates have been made by WCM and have been accepted by WCM.

2. A contingency amount has not been included but is recommended (10%).

The project operates under EA MIN102648011, (7 August 2012) which describes the three plans to be prepared and submitted with the PoO. These are summarised in Table 20-1, together with status descriptions. The action programme outlining the approach for meeting the conditions of the EA is provided in the PoO.

In November 2016, an amendment to WCM's EA was made, that authorises the use of the Main Pit as a water storage structure. It is expected that it will be possible in the future to extend this approval to include tailings containment, as a component of its rehabilitation.

Table 20-1. Plan/Programmes Listed in the EA to be Associated with the PoO

Plan Programme	EA Condition	Status
Emergency response contingency plan	A3-3	Provided in the site Environmental Risk Management System (Natural Solutions 2007b)
Site Water Management Plan	F6-1	Provided in PoO 2007 reviewed in August 2016
Erosion and Sediment Control Plan	F6-1(b)	Prepared in 2007 (Natural Solutions 2007a) reviewed in September 2016

The following documents are required to be developed, though are not necessarily attached to the PoO.

a.Risk Management System (A8)
b.Post Mine Land Use Plan (C11)
c.REMP Receiving Environment Reporting Plan (G9)

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21 CAPITAL AND OPERATING COSTS

A summary of planned ongoing operating costs are shown in Table 21-1, along with historical operating costs.

Table 21-1. Operating Unit Costs

Description	Unit	2012	2013	2014	2015	2017+
		Year				
WO_3 price	*US$/mtu*	317	363	372	290	364
Effective WO_3 price	*$US/t product*	25,010	28,698	27,400	23,516	28,000
Mining operations	*$US/t rock*	8.32	7.22	5.93	4.91	3.69
Processing Cost	*$US/t ore*	55.33	19.41	18.57	15.05	11.44
Administration/environmental Cost	*$US/t ore*	8.98	3.28	3.73	4.40	2.64

Notes

. 2015 Costs are for Jan-Sept 2015

. 2014/2015 Costs are converted back from AUD based on exchange rates of :

2014	AUD/USD = 0.93
2015	AUD/USD = 0.79

. Planned 2017+ costs based on AUD/USD = 0.755

In the current reserve estimation, the recovery of MoS_2 has not been included, owing to current price levels.

Capital investments planned for 2017 are summarised in Table 21-2.

Table 21-2. Planned Capital Costs

Item	Capital Cost $M
Tailings Storage Facility (TSF3)	0.723
Water Treatment Plant (WTP)	0.432
Tailings Dry Stack	0.107
Scalping	1.872
Crushing and Screening (CWS)	1.063
Total	**4.197**

22 ECONOMIC ANALYSIS

The main economic parameters assumed in this economic modelling are those base case parameters shown in Table 22-1. An economic analysis, based on revenue derived from reserves only, is summarised in the mining schedule in Table 22-2. This mining schedule the lowest value ore category (MW- mineralised waste), is stockpiled and then processed when the all the higher grade material has been treated. A total operating margin of approximately $5.1M has been estimated. The pit sectors used in the development of the mining schedule are shown in Figure 15-4.

Owing to the very erratic nature of mineralisation, and the relatively wide spacing of available exploration drilling, compared to the scale of mineralised structures, the proportion of *Inferred* to *Indicated* resources is high. As the pit advances with more BEX drilling, progressively more reserves can be determined, approximately 25m beneath the base of the open pit at any time. Based on the optimisation results, where *Inferred* resources have been enabled, an open pit life of 4 years is suggested, before the additional contribution of potential extension zones described in section 24.1.

Table 22-1. Economic Modelling Parameters

Description		Unit	Base Case
Resources Enabled			**Ind Only**
Mining Data			
	Waste density	*t/m3*	2.7
	Ore density	*t/m3*	2.70
Metal Prices			
	APT Price	*$/mtu WO_3*	**364**
	Contract reduction	*%*	**77%**
	Metal Price - received	*$/mtu WO_3*	280
		$/t WO_3	28,000
	Conc Transport cost	*$/tWO3*	**532**
	Royalty		**2.7%**
Processing			
	Plant overall WO3 Recovery	*%*	**71.03%**
	Processing Cost	*$/t mineral*	**11.44**
	G & A	*$/t mineral*	**2.64**
	Mill circuit - proportion of original feed ore		**56%**
Mining			
	Ore mining	*$/t mineral*	**3.69**
	Waste mining	*$/t waste*	**3.69**
Mining Parameters			
	Mining Recovery		**90%**
	Dilution		**10%**
	Breakeven Economic WO3 Cut-Off		0.08%

Key	
Bold	**Value supplied**
Normal	Derived
Yellow	Values used directly in economic model

Table 22-2. Mining Schedule

WOLFRAM CAMP MINE	Totals	Month 1	Month 2	Month 3	Month 4	Month 5	Month 6	Month 7	Month 8	Month 9
---- MAIN PIT EAST ---- ZONE 6										
MPE TOTAL ORE (t)	**62,275**	**10,414**	**29,793**	**22,068**						
MPE WO3 (%)	**0.30**	**0.34**	**0.27**	**0.33**						
MPE RATIO (waste m3/ ore m3)	8.71	21.01	7.51	4.51						
---- MINE PIT WEST ---- ZONE 4										
MPW TOTAL ORE (t)	**76,421**	50,140	26,281							
MPW WO3 (%)	**0.25**	**0.24**	**0.27**							
MPW RATIO (m3/m3)	4.22	3.85	4.94							
---- CLAY PIT ---- ZONE 5										
CLAY TOTAL ORE (ton)	**4,002**		4,002							
CLAY WO3 (%)	**0.38**		**0.38**							
CLAY RATIO (m3/m3)	12.91		12.91							
---- PARROTS_1 PIT ---- ZONE 1										
PAR_1 TOTAL ORE (t)	**38,528**					**38,528**				
PAR_1 WO3 (%)	**0.21**					**0.21**				
PAR_1 RATIO (m3/m3)	2.74					2.74				
---- PARROTS_2 PIT ---- ZONE 2										
PAR_2 TOTAL ORE (ton)	**112,864**			**38,318**	**60,094**	**14,452**				
PAR_2 WO3 (%)	**0.20**			**0.25**	**0.17**	**0.17**				
PAR_2 RATIO (m3/m3)	2.34			3.16	2.02	1.45				
---- PARROTS_3 PIT ---- ZONE 3										
PAR_3 TOTAL ORE (ton)	**75,607**					**7,148**	**60,271**	**8,188**		
PAR_3 WO3 (%)	**0.14**					**0.12**	**0.14**	**0.15**		
PAR_3 RATIO (m3/m3)	3.65					4.00	3.85	1.86		
---- MINE PRODUCTION ----										
WASTE (m3)	578,417	152,483	150,196	81,745	45,069	57,439	85,858	5,628		
ORE (m3)	136,927	22,427	22,250	22,366	22,258	22,270	22,323	3,032		
TOTAL (m3)	**715,344**	**174,909**	**172,446**	**104,111**	**67,327**	**79,709**	**108,181**	**8,660**		
Total Rock (t)	**1,931,429**	**472,255**	**465,605**	**281,099**	**181,783**	**215,214**	**292,089**	**23,383**		
MW (ton)	78,120	11,903	7,441	6,803	20,822	17,574	12,553	1,025		
LG (ton)	228,971	34,374	35,285	36,535	33,248	35,008	47,359	7,163		
HG (ton)	62,606	14,277	17,350	17,049	6,025	7,546	359			
TOTAL ORE (t)	**369,697**	**60,554**	**60,075**	**60,387**	**60,094**	**60,128**	**60,271**	**8,188**		
WO3 MW (%)	0.109	0.10	0.11	0.11	0.11	0.11	0.11	0.12		
WO3 LG (%)	0.144	0.14	0.15	0.15	0.14	0.14	0.15	0.15		
WO3 HG (%)	0.626	0.65	0.61	0.64	0.59	0.59	0.67			
TOTAL WO3 (%)	**0.218**	**0.25**	**0.28**	**0.28**	**0.17**	**0.19**	**0.14**	**0.15**		
TOTAL MTU	80,637	15,424	16,636	16,993	10,441	11,409	8,546	1,188		
RATIO (m3/m3)	**4.22**	**6.80**	**6.75**	**3.65**	**2.02**	**2.58**	**3.85**	**1.86**		
---- STOCKPILE ----										
Stockpiled Ore (t)	102,309	17,354	16,875	17,187	16,894	16,928	17,071			
WO3 Stockpiled (%)		0.25	0.21	0.21	0.15	0.16	0.10			
Stockpile Accumulated (t)		17,354	34,230	51,416	68,310	85,238	102,309	67,297	24,097	
WO3 Stock Accumulation (%)		0.25	0.23	0.22	0.20	0.20	0.18	0.18	0.18	
Stockpile Processed (t)								35,012	43,200	24,097
---- PROCESSING ----										
Processed Ore (t)	369,697	43,200	43,200	43,200	43,200	43,200	43,200	43,200	43,200	24,097
WO3 Processed (%)	0.216	0.26	0.30	0.31	0.18	0.20	0.16	0.17	0.16	0.18
WO3 Product (t)	566	79.17	93.41	95.20	56.23	61.37	49.10	52.16	49.10	30.59
WO3 Product (MTU)	56,632	7,917	9,341	9,520	5,623	6,137	4,910	5,216	4,910	3,059
Revenue - WO3 Sales (US$ x 1000)	15,857	2,217	2,615	2,665	1,575	1,718	1,375	1,461	1,375	856
Cash Costs										
Mining (US$ x 1000)	7,127	1,743	1,718	1,037	671	794	1,078	86		
Mill (US$ x 1000)	2,352	275	275	275	275	275	275	275	275	153
Admin (US$ x 1000)	543	63	63	63	63	63	63	63	63	35
Conc Transport (US$ x 1000)		42	50	51	30	33	26	28	26	16
Royalty (US$ x 1000)		60	71	72	43	46	37	39	37	23
Operating Cash Costs (US$ x 1000)	10,750	**2,183**	**2,177**	**1,498**	**1,081**	**1,211**	**1,479**	**492**	**401**	**228**
Operating Margin (US$ x 1000)	5,106	**34**	**439**	**1,167**	**493**	**507**	**-105**	**969**	**973**	**628**

23 ADJACENT PROPERTIES

23.1 Bamford Hill Deposit

The source of the information for the description shown in this section include:

- Communication with WCM and Almonty geological staff.
- Reports from the Geological Survey of Queensland, including open file exploration reports.
- BMR Bulletin 70 Geology Publication (Keyser and Wolff, 1964)
- Queensland Government Mining Journal Extracts (1914, 1955).

This area is completely separate and to the south of the Wolfram Camp deposit. The geology of this deposit is described in Section 9.3.

23.2 Other Deposits

The most comprehensive overview of all the key historical tin-tungsten prospects within the Wolfram Camp exploration tenements is the 1964 Geological Survey Publication 317. There is limited information for the numerous small prospects and workings plotted on the regional maps as they have not been systematically mapped or sampled, but they warrant evaluation to determine if they could be associated with significant mineralisation.

23.2.1 Scardon's Mining Area

The area may be divided into a western group of mines around Shannons Hill – Scardon's Bottom Camp - and an eastern group on Convict Creek – Scardon's Top Camp around which there has been the greatest amount of historical activity. The workings lie close to the contact of granodiorite and leucogranite with folded sandstone of the Mount Garnet Formation. Wolframite and molybdenite occur in white to clear glassy quartz veins which appear to be controlled by joint systems. Boundaries between the quartz lodes and the granite are gradational.

The first production came from detrital deposits but when these were exhausted shafts were sunk into the primary ore (maximum depth 70ft (21m)). Workings are concentrated along a zone trending 280°.

Scardon's Bottom Camp mines exploited detrital deposits but two shallow shafts were eventually sunk on a narrow quartz lode in the granite.

The total production up to 1911 was about 30t of wolframite concentrate.

23.2.2 Koorboora Mining Area

Several of the tin mines in the Koorboora area, e.g. Two Jacks contained varying amounts of wolframite in the upper part of their lodes (refer below), but in only three mines was the concentration sufficient to justify exploitation. The mines are situated in steeply dipping micaceous sandstone and shaly siltstone of the Mount Garnet Formation, and are heavily iron stained near surface.

23.2.2.1 Neville Mine

The Neville, located over one mile (1.6km) SSE of Koorboora was an important wolfram mine between 1904-1909 yielding 550t of wolframite; further production in 1918 increased the total output to 580t from 8,583t of ore. (WMC record the production as 9,000t @ 5% WO3).

Nearly all the ore came from one small but very rich shoot shaped more or less as a spiralling pipe, 10 – 20ft (3-6m) in diameter and extending down to 140ft (43m). At greater depth only low grade mineralisation was found. The average grade of wolframite in the upper 80ft (24m) was 10%, between 80 – 120ft (24-37m) 20% and from 120 – 140ft (37-43m) 4%; in the lower workings cassiterite appeared and sulphides became abundant below the 200ft (53m) level.

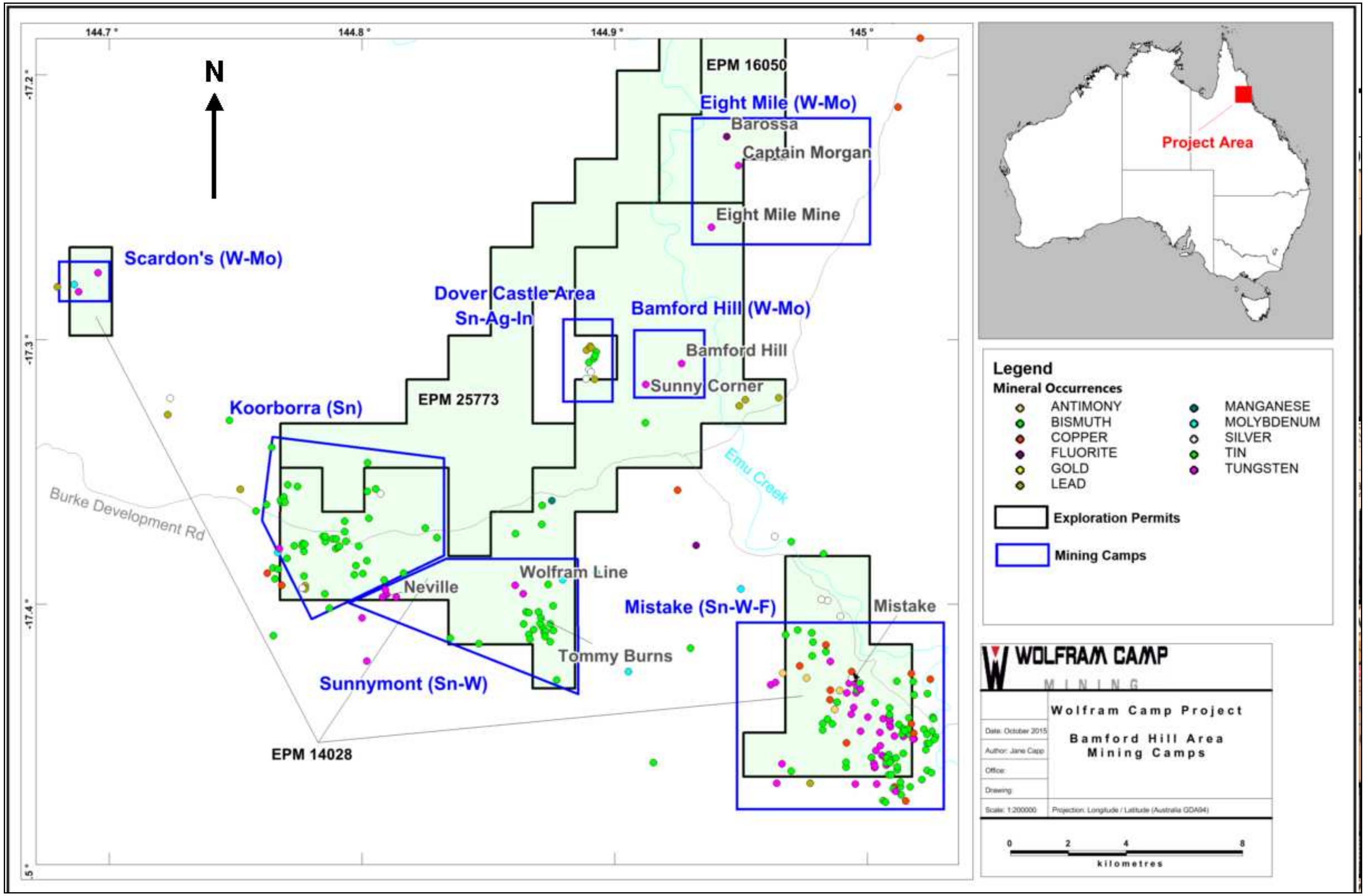
The Neville was considered as one of the greatest single wolfram mines in the world. The Neville was worked intermittently by prospectors long after its demise as a company mine. The location of the Neville deposit is shown in Figure 23-1.

23.2.2.2 Shakespeare Mine

The Shakespeare tin mine, located on the site of the old Koorboora settlement near the Almaden – Petford road, is one of the earliest and most extensively worked mines in the area, but carried comparatively low grade ore, based on the reported production figures of 203t of SnO2 from 14,539t of ore between 1901 and 1947.

Two lodes were opened up: a western lode carrying galena, chalcopyrite, pyrite and arsenopyrite in addition to cassiterite and a more important lode of ferruginous and chloritic material striking NW and dipping NE, which accounted for the majority of the output. Open cuts and shafts yielded payable tin to a depth of 200ft (61m). The host rocks generally contain large amounts of mica and sericite as a result of pneumatolytic action.

Figure 23-1. Location of the Neville Deposit



23.2.2.3 Two Jacks Mine

The Two Jacks lease, including the Three Jacks and New Dalnotter lodes, is located about 2 miles (3.2km) ENE of the former Koorboora settlement and was one of the last worked deposits in the area. The workings are situated in chloritised and sericitic Mount Garnet sediments exposed as an inlier in rhyolites of the Featherbed Volcanics. The sediments comprise fine grained micaceous siltstones, generally strongly brecciated and traversed by N–NW faults which in places appear to displace the lodes. The ore-shoots pitch NW and dip NE and form discontinuous lenses, between which the crushed country rock carries low values.

The cassiterite is very fine grained and, in places, associated with tourmaline. Fine grained pyrite is visible as films on joint planes. 457t of cassiterite were produced from 11,321t of ore between 1906 and 1921. Subsequent attempts to open the mine in 1926 and 1961 failed. The later investigation by Broken Hill Pty Co. Ltd involved dewatering the mine down to the 185ft (56m) level where ore had been extracted over a length of 30ft and an average width of 4.5ft (9 x 1.4m). Large quantities of ore had been mined from a lode measuring 80ft by 8ft (24 x 2.1m) between the 120ft (37m) and 140ft (43m) levels.

23.2.3 Sunnymount Mining Area

The mines in the Sunnymount area, separated by the Tennyson Ring Dyke from the Koorboora area, were discovered 20 years after the latter area.

23.2.3.1 Tommy Burns Mine

The Tommy Burns tin-tungsten deposit is located within the Sunnymount Mining Camp approximately 10km to the southwest of Bamford Hill. The mine was first worked in the early 1900's but became the largest producer in the Sunnymount – Koorboora district with the development of a large pipe-like orebody from 1973 until closure in 1984.

Tin-tungsten mineralization is associated with structurally controlled garnet-chlorite alteration of the host meta-sediment and meta-basalt host rocks within a roof pendant in the surrounding granites. Several discrete orebodies range in size up to 185,000T with total recorded production of over 6,000T of tin and associated tungsten concentrates from 259,000T of ore. The main orebody was up to 25m in diameter and mined to a depth of 280m.

This significant historical production centre highlights the potential for tin-tungsten resources within the host rocks intruded by the granites associated with extensive greisen and quartz-pipe W-Mo-Bi mineralization at Wolfram Camp and Bamford Hill within the exploration tenements. Planned forward exploration programs will incorporate targeting extensions to the known deposits and the potential for concealed granite-hosted mineralization at depth.

23.2.3.2 Wolfram Line Mine

The project has no history of production according to WCM, their target being to delineate 2 – 300,000t @ 0.2% WO3. The mineralisation is apparently hosted in a 'siliceous greisen lode'.

23.2.4 Mineralisation in the Koorboora and Sunnymount Areas

The mineralisation in the Koorboora and Sunnymount areas is closely similar to that in the Herberton district (about 60km SE of Wolfram Camp): the lodes are ill defined bodies of mainly chloritised, kaolinised and, to some extent, pneumatolytically altered sandstone, siltstone and quartz greywacke, and merge gradually with the unaltered host rock. They contain ore shoots that tend to be irregularly pipe-like and are apparently controlled by fissure and joint intersections that have served as migration channels. The ore shoots change gradually to barren rock.

The granite responsible for the hydrothermal-pneumatolytic alteration (chloritisation, formation of muscovite, garnet, tourmaline, cassiterite, wolframite) is present at no great depth in the Koorboora area (blocks of granite occur on the dumps), and in the Sunnymount area the granite batholiths is widely exposed to the east of the mines.

Most of the lodes in both areas carried some wolframite in their upper parts, and a couple of the mines have been worked exclusively for wolframite. In the bottom parts of the lodes an increase in the proportion of base metal sulphides is frequently noticed and as treatment of the complex ore is costly the mining of deeper ore shoots of this type had little attraction.

23.2.5 Eight Mile Area

23.2.5.1 Eight Mile Mine

The Eight Mile Mine lies about 5km N of Bamford Hill in a small granite outcrop on the east bank of Emu Creek and was worked for wolframite and molybdenite. Along with Bamford Hill, Captain Morgan, Four Mile and Wolfram Camp, it is representative of a rare mineralisation style that appears to be unique in that there are no other identical occurrences outside Australia.

The mineralisation is characterised by wolframite, molybdenite and bismuth contained in branching, quartz rich, pipe-like bodies within the greisenised apical and flank portions of high level, fractionated granites. Other minerals include small quantities of scattered sphalerite, pyrite, chalcopyrite, galena, fluorite, kaolin, magnetite and various tungsten, molybdenum and bismuth minerals (scheelite, powellite and bismuthinite). Many of the sulphides are commonly found concentrated in vugs, and the proportion of sulphides seems to increase with depth.

23.2.5.2 Four Mile Area

The Four Mile workings lie about 2 miles (3.2km) WNW of Wolfram Camp in a tongue of granite bounded on the east and west by Featherbed Volcanics, and on the north by sediments of the Hodgkinson Formation. The mineralisation style is described above in section 1.6.1.

23.2.6 Mistake Area

Many of the mines in the Emuford district were worked for wolfram, others for tin, fluorspar and copper, and the lodes were generally small and insignificant. No production figures were available at the time of the GS 1964 publication; however, WCM indicate that <100,000t of tin and wolfram have been produced.

23.2.6.1 Mistake Mine

The mine, located 7.25 miles (12km) SSE of Petford, was opened in 1915 primarily for wolfram. Up to 1964, in addition to fluorspar, wolfram had been produced intermittently.

23.2.6.2 Mystery Mine

Located about 1 mile (1.6km) south of the Mistake workings, the mine occurs in pink granite close to its contact with conglomerate and quartz greywacke of the Mount Garnet Formation. Much of the host rock is greisenous. Wolframite was worked in patches down to 180ft (55m) in the main shaft, and bismuth , galena, chalcopyrite and copper carbonates were noted in small quantities. A parcel of 9% copper was extracted from the mine in its early stage. The quartz gangue contained some fluorite and topaz.

23.2.6.3 Spotted Dog Mine

The mine lies about 1 mile (1.6km) NW of the Mystery mine; wolframite occurs in sediments of the Mount Garnet Formation.

24 OTHER RELEVANT INFORMATION

24.1 Potential Open Pit Resource Extensions

A number of potential resource extensions exist along-strike from the main Wolfram Camp pit, as well as more which are offset and more to depth, as depicted in a plan view in Figure 24-1, which shows the old mines depicted with symbols scaled according to historic production of wolfram metal. This plan has been overlaid on the current pit design, and current drillhole data. A more detailed plan of is shown in Figure 24-2, which shows the resource extensions immediately adjacent to the current open pit area. These are only a subset of all the resource extensions along the whole WCM strike length, and also do not include potential underground resources. The plan in Figure 24-2 also shows a pit design corresponding to the current resource model, related to a pit optimisation in which both Indicated and Inferred resources were enabled. An estimate of the tonnages associated with this pit design and these immediate resource extensions are shown in Table 24-1.

Table 24-1. Summary of Adjacent Open Pit Resource Extensions

	Area *m^2 x 1000*	Tonnes *Mt*
Resource Extensions		
Access	58	0.77
James Hill	49	0.65
James Hilltop	17	0.23
Sub-Total	**124**	**1.64**
Main/Parrotts		**2.10** *
Total		**3.74**

Notes

*** Based on Indicated and Inferred pit optimisation**

. These resource extension estimates are not relevant to the current resources and reserves

The tonnages of these resource extensions were derived from the plan areas and assuming the same proportion of mineralisation as has been demonstrated for the Main and Parrotts open pit areas. These resource extensions are intersected by old historic workings, but do not have recent samples within them, so they have been excluded from *Inferred* resources at the current time.

Figure 24-1. Plan of Old Mines in Wolfram Camp Area

[Symbols for old mines scaled according to production of Wolfram]

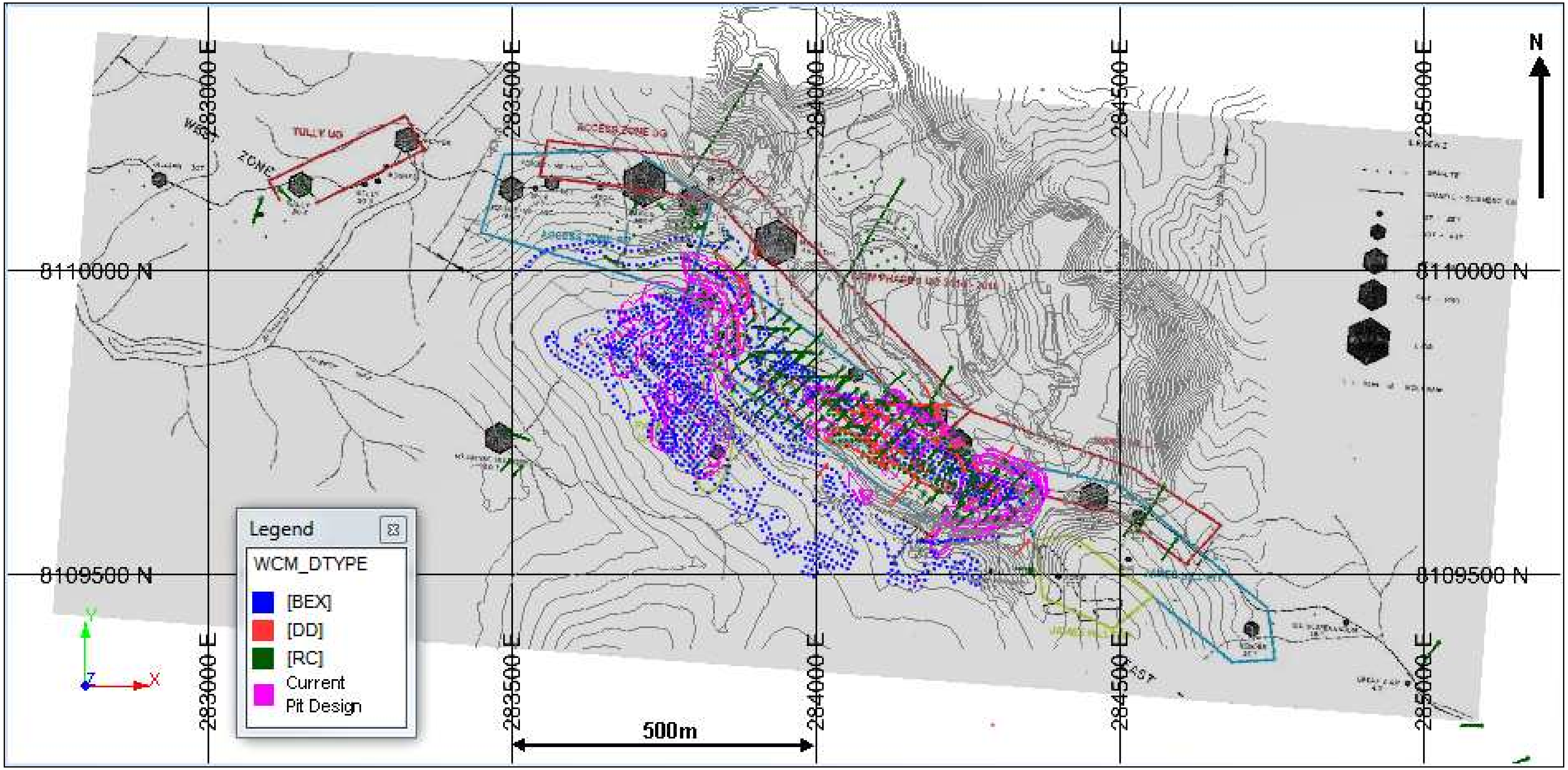
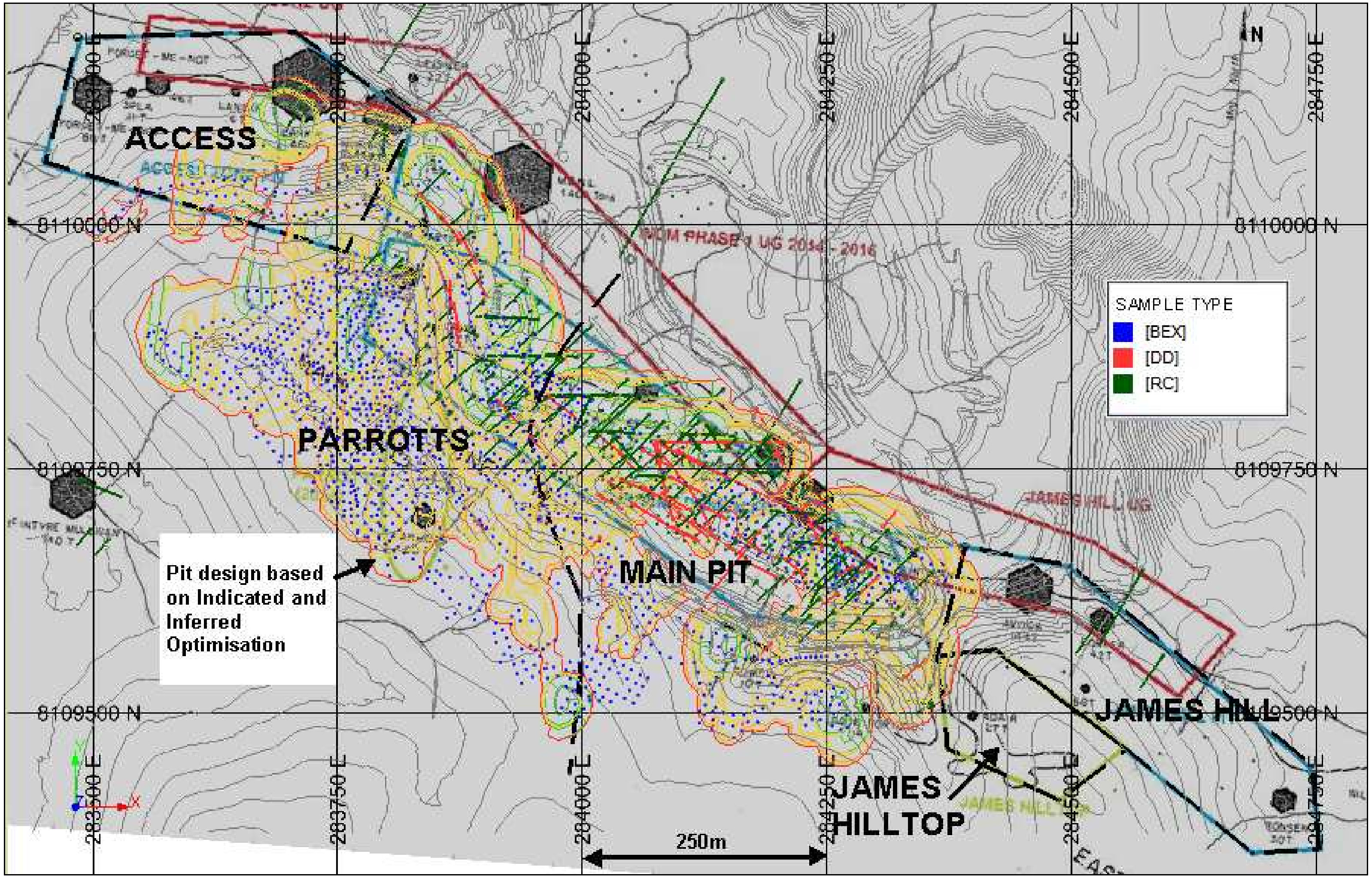


Figure 24-2. Plan of Resource Extensions with Respect to Old Mines

[Symbols for old mines scaled according to production of Wolfram]



24.2 Survey Control

All surface surveying measurements are generated in UTM Coordinates, and are referenced to the MGA55 coordinate system. Survey data is measured using a GPS Leica Viva GS 15, using the geoid model AusGeoid09_QLD.

There is also a Trimble S6 Robotic total station used for ground control monitoring and survey back-up.

25 INTERPRETATION AND CONCLUSIONS

The evaluation work was carried out and prepared in compliance with Canadian National Instrument 43-101, and the mineral resources in this estimate were calculated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council May, 2014. The updated resource estimation is shown in Table 25-1 and Table 20-2.

Table 25-1. Wolfram Camp - Mineral Resources
Pit-constrained resource, as of 31st August, 2015

Resource Category	Tonnes Kt	WO_3 %	MoS_2 %
Indicated	514	0.23	0.07

Notes:

. Cut-off = 0.10% WO3
. Historic underground mined material removed
. Prices used in optimisation:

US $/mtu WO_3	**400**
US $/t MoS_2	**25,000**

. Minimum width = 1m
. Resources shown are inclusive of reserves

Table 25-2. Wolfram Camp – Inferred Mineral Resources
Pit-constrained resource, as of 31st August, 2015

Resource Category	Tonnes Kt	WO_3 %	MoS_2 %
Inferred	1,879	0.31	0.08

Notes:

. Same cut-off and controls as above

The updated reserve estimation, stemming from a plan developed from an updated pit optimisation, is shown in Table 25-3.

Table 25-3. Wolfram Camp – Mineral Reserves

Reserve Category	Tonnes Kt	WO_3 %	Waste + Inferred Kt	Rock Kt	Strip Ratio
Probable Reserves	**375**	**0.22**	**1,556**	**1,931**	**4.2**

Notes

. Cut-off = 0.08% WO_3
. Mining factors of applied of
Dilution = 10%
Losses = 10%
. Pit design also contain 187Kt of inferred resources at economic grades

The following conclusions have been reached:

1. Wolfram Camp has all permits and licenses to operate and remain in compliance with appropriate regulations. It has no restrictions with respect to waste dumping or tailings capacity.

2. Grade control (GC) samples from blasthole drilling in the open pit mining operations have in general corresponded fairly well with previous exploration diamond drilling (DD) and reverse circulation (RC) drilling results for the mined areas. This has supported the use of GC samples in resource estimation, and together with reconciliation information, has provided a very important assistance in the development of parameters for updated resource modelling.

3. In the author's opinion, the current resource and reserves estimates for Wolfram Camp are conservative, because of reasons which include:

    a) Areas within only relatively widely spaced exploration data will have missed some mineralised intersections.
    b) The currently orebody model has been limited to a depth of 490mRL, which represents the approximate base of drilling information, not the geological base of the deposit.
    c) There are known mineralised extensions, both along-strike in both directions as well as at depth, where historical underground workings demonstrate mineralisation. At current metal price levels, these areas also offer potential for future underground reserves.

4. Owing to the very erratic nature of mineralisation, and the relatively wide spacing of available exploration drilling, compared to the scale of mineralised structures, the proportion of *Inferred* to *Indicated* resources is high. As the pit advances with more BEX sampling, progressively more reserves can be determined, approximately 25m beneath the base of the open pit at any time. Based on the optimisation results, where *Inferred* resources have been enabled, an open pit life of approximately 3 years is suggested, before the additional contribution of potential extension zones.

5. Significant improvements are being made to the plant since the shutdown in 2015. The key improvements include the crushing and wet screening facilities, scalping (use of ore sorters), a new tailing storage facility (TSF3), and tailings dry stacking facility and the water treatment plant.

26 RECOMMENDATIONS

The work programme for the mill improvements is already being implemented. The costs associated with all these improvements are summarised in Table 21-2.

27 REFERENCES

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Report 43-101 - Technical Report on the Mineral Resources and Reserves of the Wolfram Camp Project, Australia, prepared by Adam Wheeler, October, 2015.

28 QUALIFIED PERSONS CERTIFICATES

Certificate Of Author

Adam Wheeler, Mining Consultant,

Cambrose Farm, Redruth, Cornwall, TR16 4HT, England.

Tel/Fax: (44) 1209-899042; E-mail: adamwheeler@btinternet.com

As the author of this report on the Wolfram Camp Project, I, A. Wheeler do hereby certify that:-

1. I am an independent mining consultant, based at, Cambrose Farm, Redruth, Cornwall, TR16 4HT, England.

2. I hold the following academic qualifications:-
   B.Sc. (Mining) Camborne School of Mines 1981
   M.Sc. (Mining Engineering) Queen's University (Canada) 1982

3. I am a registered Chartered Engineer (C. Eng and Eur. Ing) with the Engineering Council (UK). Reg. no. 371572.

4. I am a professional fellow (FIMMM) in good standing of the Institute of Mining, Metallurgy and Materials.

5. I have worked as a mining engineer in the minerals industry for over 28 years. I have experience with a wide variety of mineral deposits and reserve estimation techniques.

6. I have read NI 43-101 and the technical report, which is the subject of this certificate, has been prepared in compliance with NI 43-101. By reason of my education, experience and professional registration, I fulfil the requirements of a "qualified person" as defined by NI 43-101. My work experience includes 5 years at an underground gold mine, 7 years as a mining engineer in the development and application of mining and geological software, and 22 years as an independent mining consultant, involved with evaluation and planning projects for both open pit and underground mines.

7. I am responsible for all sections, 1-27, of the technical report, as well as the report preparation. The report is titled "Technical Report on the Mineral Resource and Reserves of the Wolfram Camp Mine Project" and dated October 31st, 2015. I visited the mine site from June 18th-21st, 2014 and October 28th – November 1st, 2014.

8. As of the date hereof, to the best of the my knowledge, information and belief, the technical report, which is the subject of this certificate, contains all scientific and technical information that is required to be disclosed to make such technical report not misleading.

9. I am independent of Almonty Industries Inc, pursuant to section 1.5 of the Instrument. Up until the work described in this report, I have had no involvement with the Wolfram Camp project.

10. I have read the National Instrument and Form 43-101F1 (the "Form") and the Technical Report has been prepared in compliance with the Instrument and the Form.

11. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated this 31st of March, 2017

AWheeler

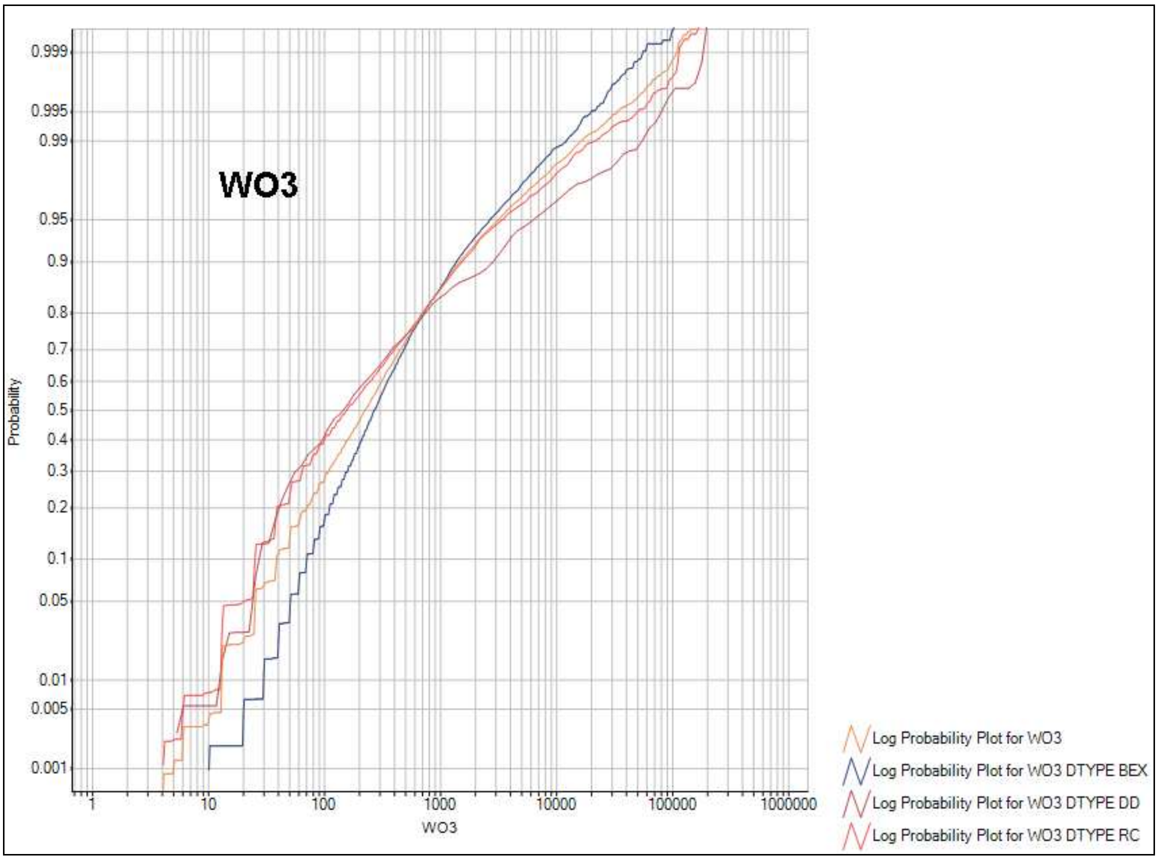
________________________________ A. Wheeler, C.Eng.

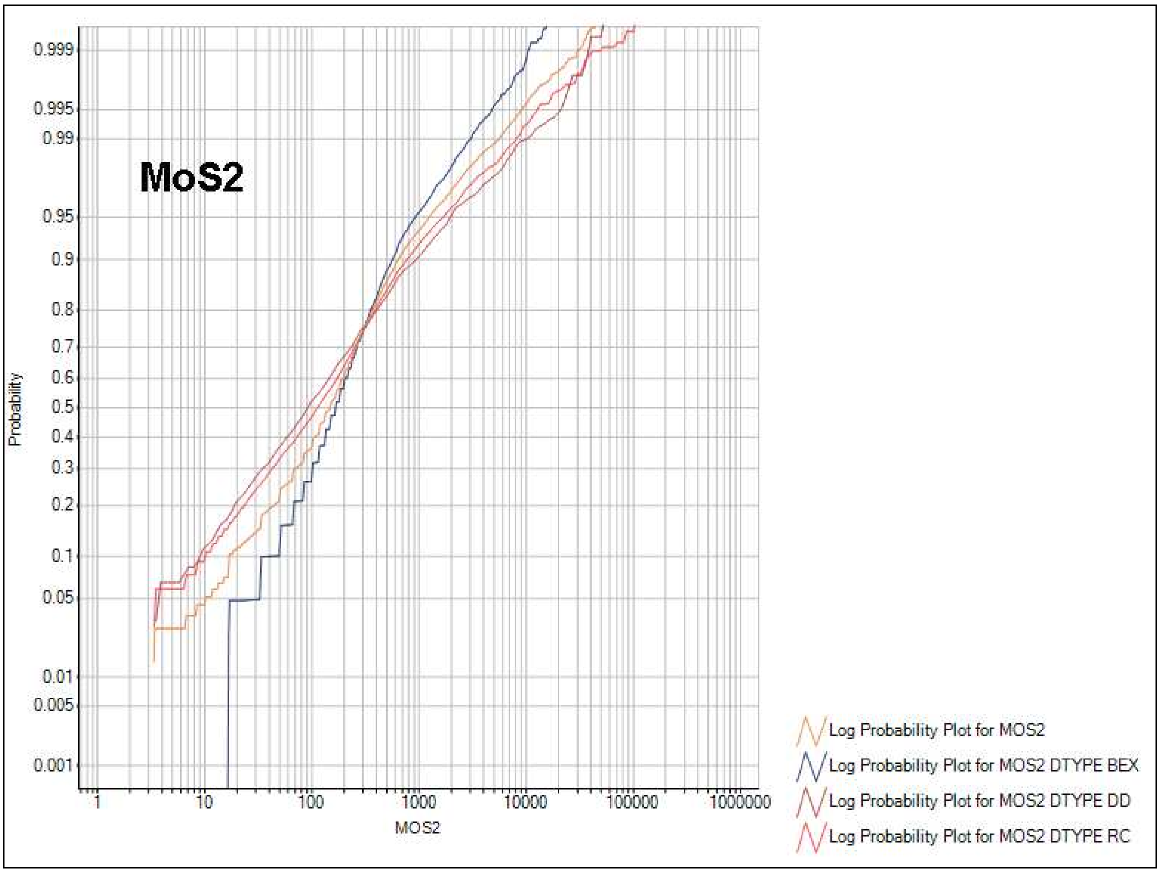
APPENDIX A:

Geostatistical Plots

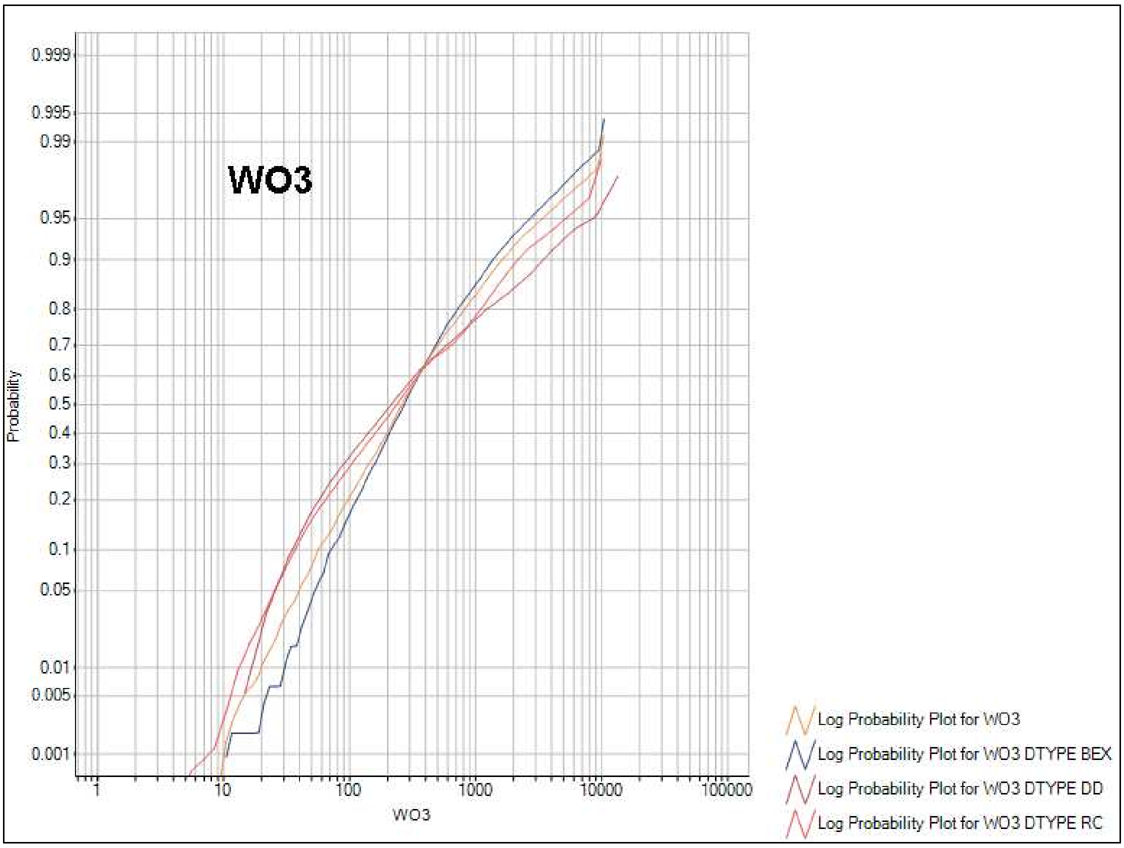
March 2017

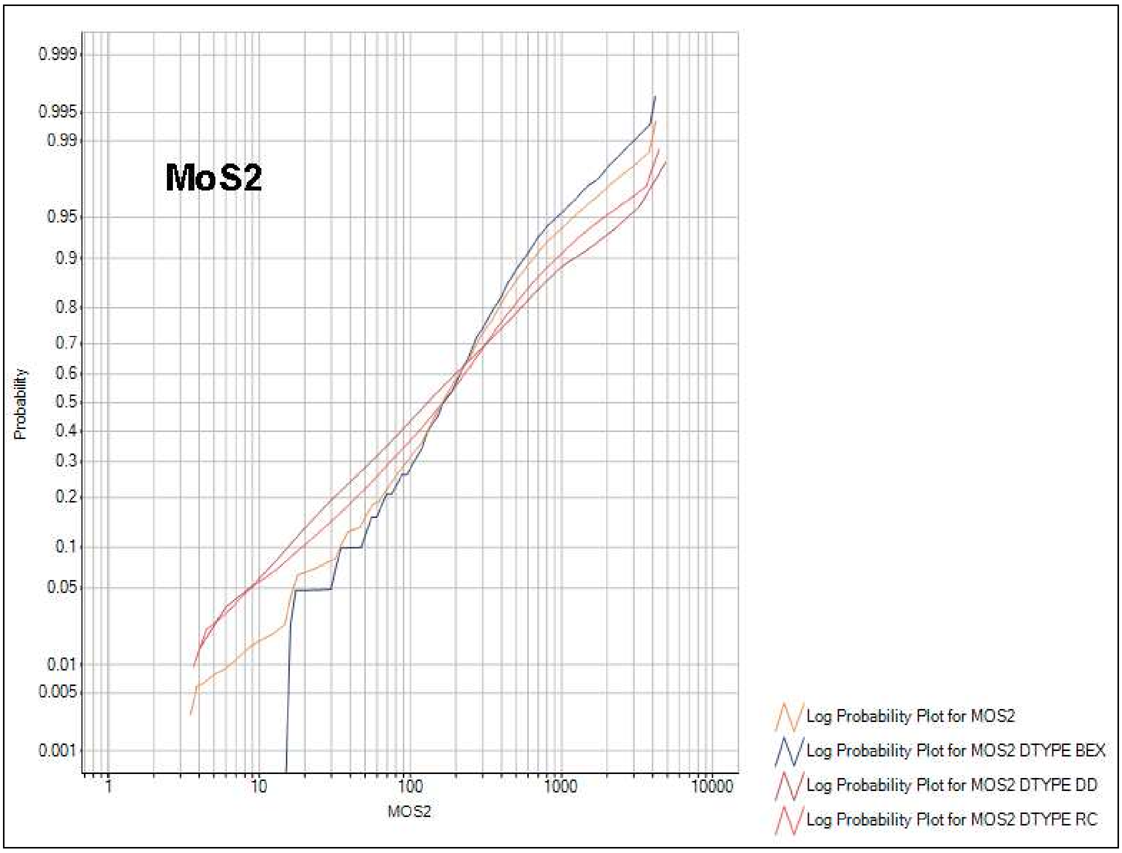
Log Probability Plots – Samples



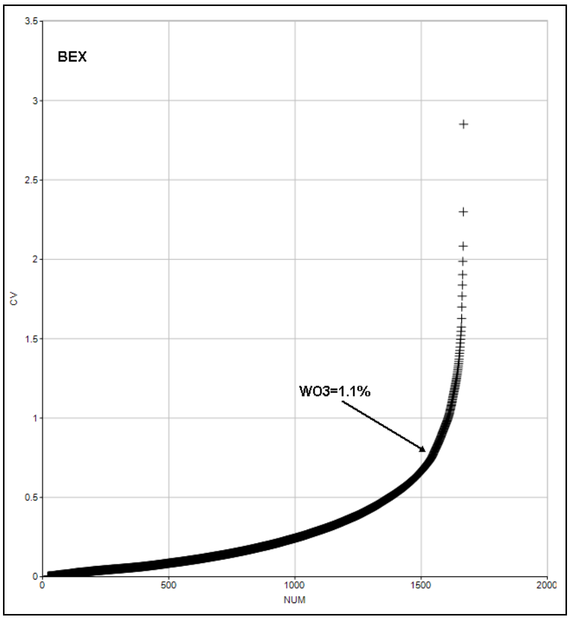


Log Probability Plots – Composites

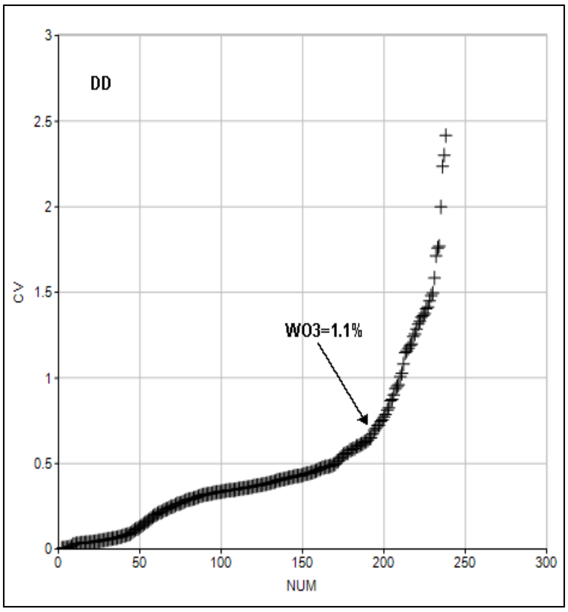




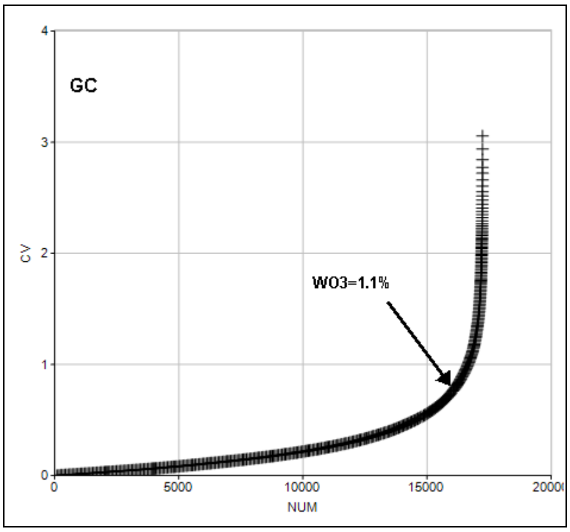
Coefficient of Variation Plots



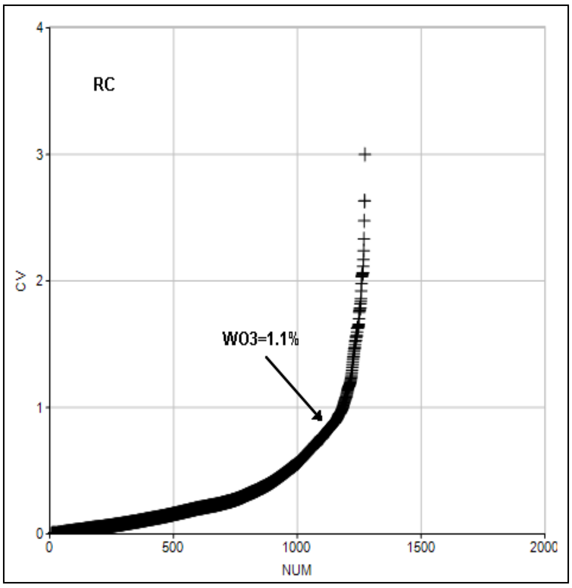
CV Analysis – BEX Samples, WO_3



CV Analysis – DD Samples, WO_3

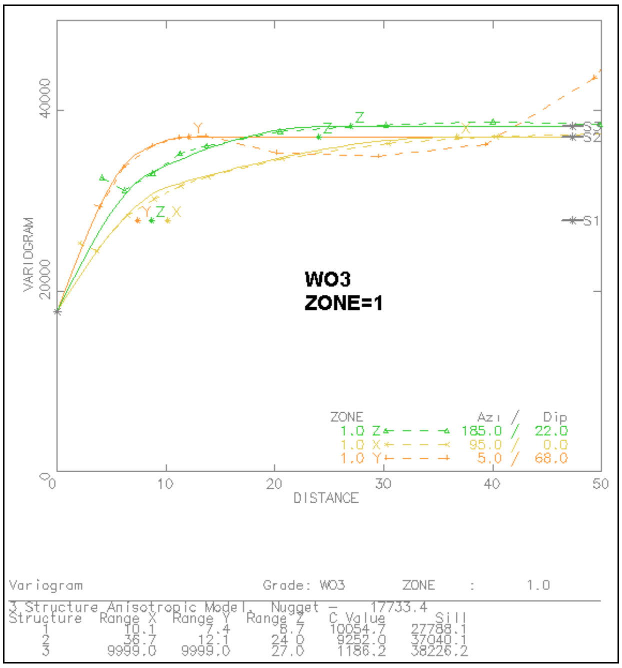


CV Analysis - GC Samples, WO_3



CV Analysis – RC Samples, WO_3

Variograms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VARIOGRAM
DISTANCE
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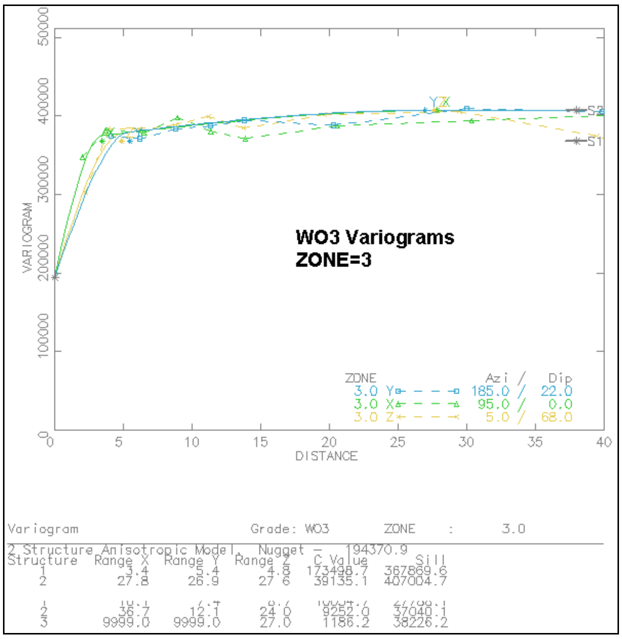
1.0 X 95.0 / 0.0
1.0 Y 5.0 / 68.0
Variogram Grade: WO3 ZONE : 1.0
3 Structure Anisotropic Model, Nugget = 17733.4
Structure Range X Range Y Range Z C Value Sill
1 10.1 7.4 8.7 10054.7 27788.1
2 36.7 12.1 24.0 9252.0 37040.1
3 9999.0 9999.0 27.0 1186.2 38226.2


WO3 Variograms
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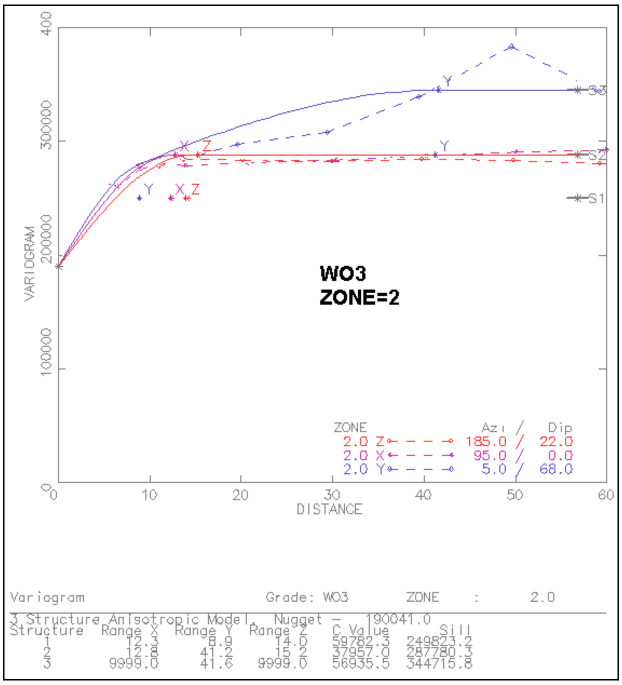
VARIOGRAM
DISTANCE
ZONE Azi / Dip
3.0 Y 185.0 / 22.0
3.0 X 95.0 / 0.0
3.0 Z 5.0 / 68.0
Variogram Grade: WO3 ZONE : 3.0
2 Structure Anisotropic Model, Nugget = 194370.9
Structure Range X Range Y Range Z C Value Sill
1 3.4 5.4 4.8 173498.7 367869.6
2 27.8 26.9 27.6 39135.1 407004.7
2 36.7 12.1 24.0 9252.0 37040.1
3 9999.0 9999.0 27.0 1186.2 38226.2


WO3
ZONE=2
VARIOGRAM
DISTANCE
ZONE Azi / Dip
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2.0 X 95.0 / 0.0
2.0 Y 5.0 / 68.0
Variogram Grade: WO3 ZONE : 2.0
3 Structure Anisotropic Model, Nugget = 190041.0
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1 12.3 8.9 14.0 59782.3 249823.2
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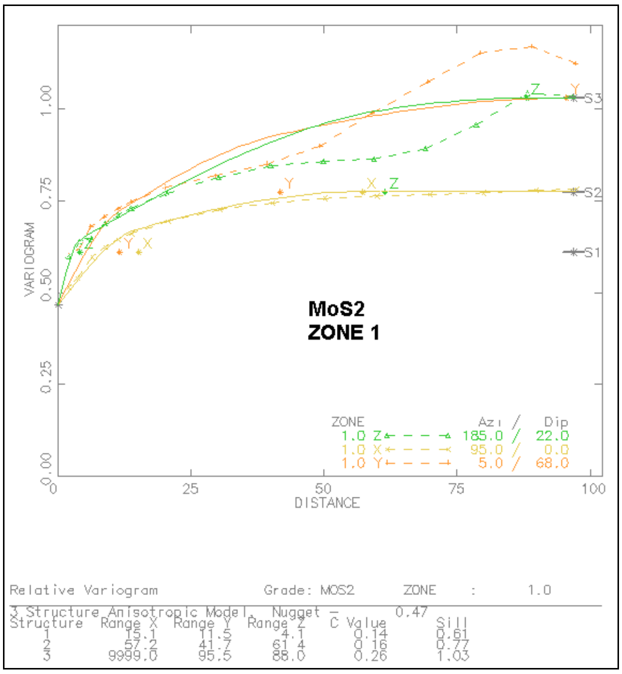
MoS2
ZONE 1
VARIOGRAM
DISTANCE
ZONE Azi / Dip
1.0 Z 185.0 / 22.0
1.0 X 95.0 / 0.0
1.0 Y 5.0 / 68.0
S1
S2
S3
Relative Variogram Grade: MOS2 ZONE : 1.0
3 Structure Anisotropic Model, Nugget = 0.47
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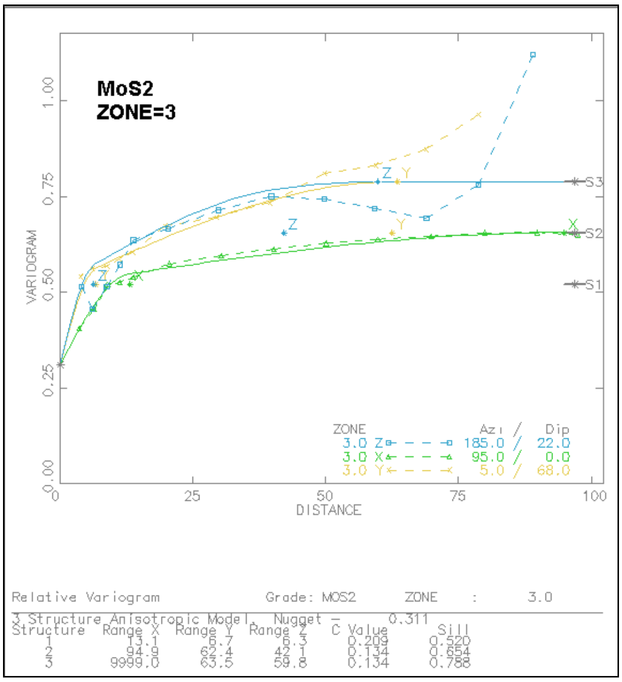
2 57.2 41.7 61.4 0.16 0.77
3 9999.0 95.5 88.0 0.26 1.03


MoS2
ZONE=3
VARIOGRAM
DISTANCE
ZONE Azi / Dip
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3.0 Y 5.0 / 68.0
S1
S2
S3
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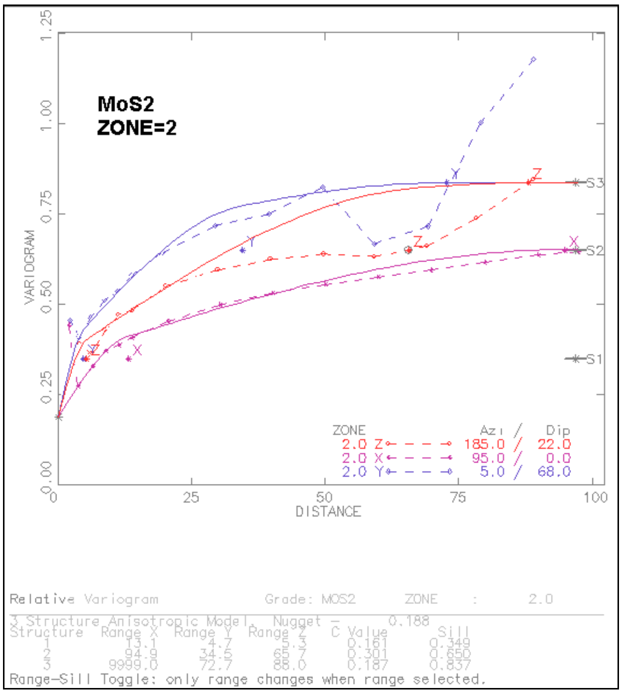
MoS2
ZONE=2
VARIOGRAM
DISTANCE
ZONE Azi / Dip
2.0 Z 185.0 / 22.0
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2.0 Y 5.0 / 68.0
S1
S2
S3
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2 94.9 34.5 65.7 0.301 0.650
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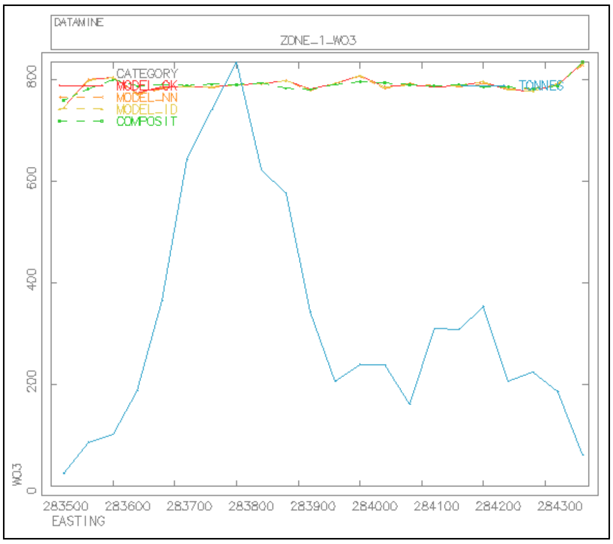
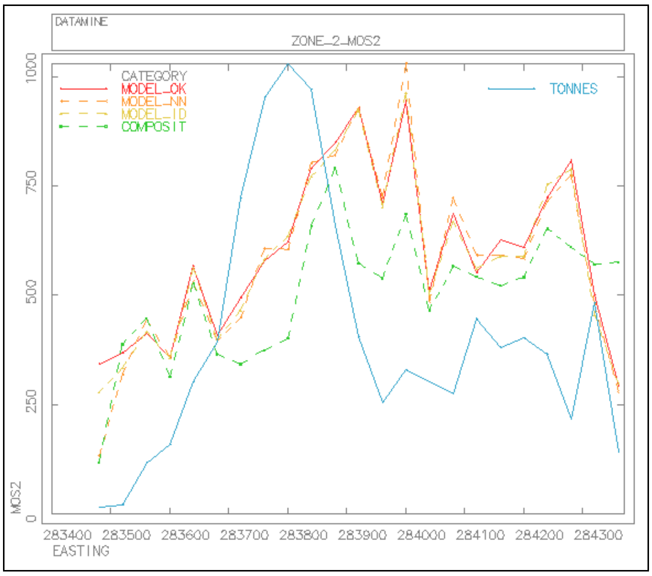
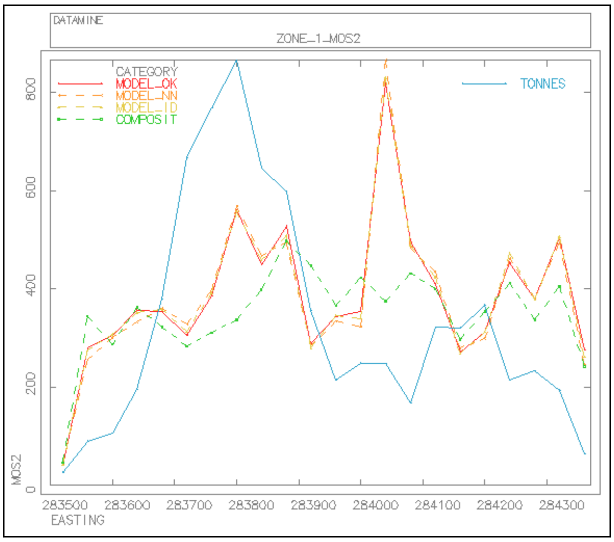
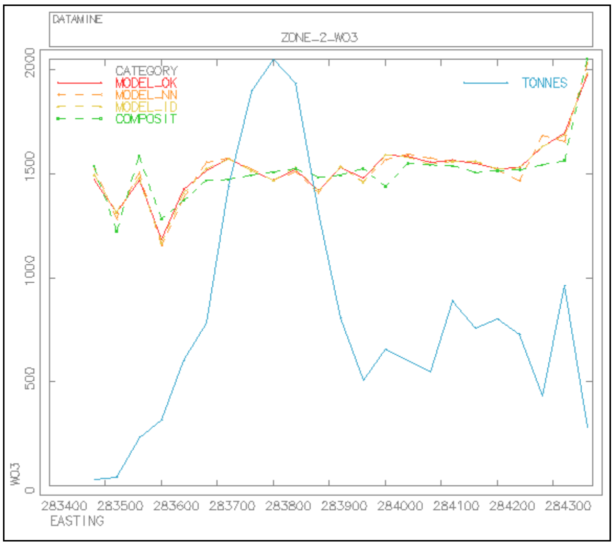
Range-Sill Toggle: only range changes when range selected.

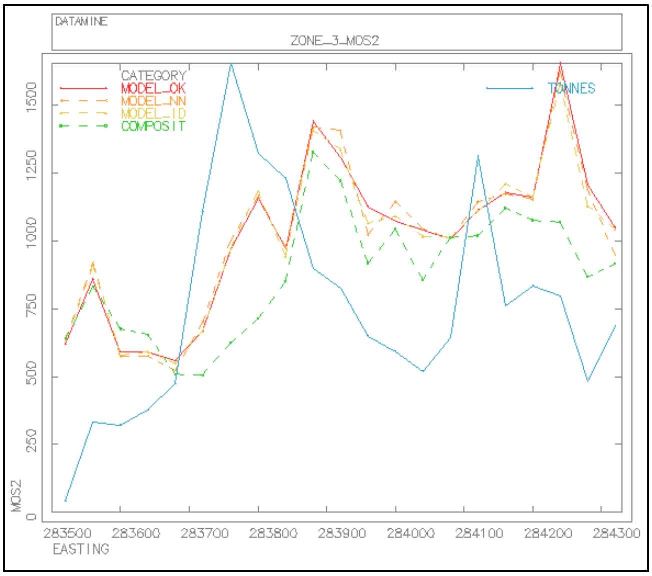
Grade Profiles - Swath Plots











APPENDIX B:

Glossary of Terms

March 2017

UNITS OF MEASURE AND ABBREVIATIONS

AUD	Australian dollars
D&B	Drill and blast
DMT	dry metric tonne
DSA	Design storage allowance
EA	Environmental Authority
FA	Financial Assurance
GC	Grade control
Ktpa	Kilo-tonnes per annum
m	meters
m/h	meters per hour
mtu	metric tonne unit
	1 mtu = 10kg = 0.01t. Normally used to refer to 10kg of WO_3 concentrate at a grade of 100% WO_3.
m^3	cubic meter
m^3/h	cubic meters per hour
Ml	Million litres
MRE	Mineral Resource Estimation
t	Tonne (1,000 kg)
kt	Tonnes x 1,000
Mt	Tonnes x 1,000,000
PML	Planet Metals Limited
PMLUP	Post Mine Lands Use Plan

PoO	Plan of Operation
tph	Tonnes per hour
tpa	Tonnes per annum/year
TMPL	Tropical Metals Pty Ltd
TOMA	Tenneco Oil and Minerals Australia
TSF	Tailings storage facility
QA/QC	Quality assurance/ quality control
QOL	Queensland Ores Ltd's
ha	hectares
US$	US dollars
W	Wolfram
WCM	Wolfram Camp Mining Pty Ltd
WMC	Woulfe Mining Corp

APT Pricing

Mined tungsten concentrates are priced by reference to the price of Ammonium Paratungstate (APT), an intermediate product in the production of tungsten metal, powder, tungsten carbide or other end use tungsten products. Prices are quoted "per metric tonne unit" (mtu) which is equivalent to 10 kg of product. An equivalent price per tonne is therefore the price on an mtu basis multiplied by 100.

The price received for concentrate sales are typically subject to a discount to the APT price to cover the cost of converting mined concentrate to APT as in the case of TC/RC charges for base metals.