REPORT NI 43-101

TECHNICAL REPORT ON THE

MINERAL RESOURCES AND RESERVES OF THE

PANASQUEIRA MINE, PORTUGAL

Prepared for

Almonty Industries

by

Qualified Person:
Adam Wheeler, B.Sc, M.Sc, C. Eng, FIMMM

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Mining Consultant
Cambrose Farm, Redruth
Cornwall, TR16 4HT
England.
E-mail: adamwheeler@btinternet.com

TABLE OF CONTENTS

Page

LIST OF TABLES

Page

LIST OF FIGURES

Page

APPENDICES

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 Resource and Reserve Estimation for the Panasqueira Mine, as of the end of September 2016. The Panasqueira mine is located in central Portugal, in the Distrito de Castelo Branco, on the southern edge of the Serra da Estrela, a Portuguese mountain range approximately 300 km north-east of the Portuguese capital city of Lisbon and 200 km southeast of the port city of Porto.

The first prospecting licence at Panasqueira was granted in 1886 and the first reference to wolframite was two years later. A mining company was founded in 1896 to mine tungsten at Panasqueira, and the underground mine has been operating more or less continuously since that time, except for a brief period at the end of World War II and a second closure in the mid-1990s.

During the period 1937-2016, a total of approximately 40 million tonnes of rock has been mined which has produced approximately 128,000 tonnes of tungsten concentrate, 6,600 tonnes of tin concentrate and 32,000 tonnes of copper concentrate.

This report was prepared by Adam Wheeler, at the request of Almonty Industries Inc. ("Almonty"). Assistance and technical detail were supplied by the technical personnel of Beralt Tin and Wolfram (Portugal), S.A., ("BTW"), a company incorporated and existing under the laws of Portugal. In connection with the preparation of this report, Adam Wheeler visited the Panasqueira site from November 23rd - 25th, 2016.

1.2 Ownership

Almonty owns 100% of Beralt Ventures Inc. ("BVI"), a body corporate pursuant to the laws of British Columbia, Canada. BVI owns 100% of BTW, which in turn is the 100% owner of the various rights and interests comprising the Panasqueira tungsten mine in Covilhã, Castelo Branco, Portugal (the "Panasqueira Mine"). Almonty is therefore the indirect owner of 100% of the Panasqueira Mine.

"Almonty" is a body corporate pursuant to the laws of Canada under the Business Corporations Act.

1.3 Geology and Mineralisation

Panasqueira is a vein type deposit located in the Center Iberian Zone of Portugal, where several tungsten mines have been worked during the 20th century. These are generally accompanied by granite outcrops intruding schist and slates. There are different kind of tungsten-host structures, but the more frequent are sub-vertical quartz veins close the contacts with granites, or even inside them. At the current time, the Panasqueira mine is the only active tungsten mine in Portugal. There are however, several active Sn and WO_3 exploration licenses (DGEG internet site: www.dgeg.pt).

The Panasqueira deposit consists of a series of stacked, sub-horizontal, hydrothermal quartz veins intruding into the Beira schists containing wolframite mineralisation, which occurs as very large nugget-like crystals of large crystal aggregates, usually concentrated towards the margins of the quartz veins or, occasionally, closer to the central portion of the veins. The overall mineralized zone has dimensions of approximately 2,500m in length; 400m to 2,200m in width and at least 500 m in depth.

Historically, mining has progressed from the upper levels to lower levels, which are spaced 60 – 90m apart. Typically seven or eight flat dipping veins occur from one level to the next, with an average thickness of 0.3 m (range 0.1-1.0m). These host the economic mineralization over continuous strike lengths of 40 - 100 m. These mineralized quartz veins located throughout all mine levels, typically pinch out and later re-occur. Resources occur over five levels – Level 0 to Level 4.

Even though the mine has been in operation for more than 100 years, very little primary exploration has been done outside the active or past mine workings. The hills surrounding the mine contain many old pits and shafts left from old small tungsten vein hand mining operations. A regional stream sediment geochemical survey carried out between 1982-1984, some exploration drillholes and a lithogeochemical survey over selected areas in and adjacent to the Panasqueira returned areas of tin and tungsten anomalies.

Exploration drilling for additional resources and reserves, in advance of production, continues as the normal course of mine activities. To date, more than 80 diamond drillholes have been completed from surface, but these holes commonly flatten considerably as they deepen and are therefore limited for assistance with vein location. Underground drilling has now covered over 4,000 drillholes, mostly of 46mm diameter. A combination of a historic fire and core dumping has left the operation with a relatively small collection of core available for review. The company, through its past experience, considers quartz veins exceeding 18cm in width to be significant and so future underground development is generally based on those intercepts.

1.4 Database and Resource Estimation

Two main types of samples are taken for resource and reserve estimation purposes: diamond drillhole samples and face mapping of wolframite crystals. Diamond drillhole core is left intact, but is logged by a geologist and all quartz vein intersections have a width measurement and a qualitative index recorded for up to 24 different minerals. An internally developed empirical (D9) formula is also used to convert the measured quartz vein thickness into a %WO_3 grade figure. These data are used for the estimation of indicated resources, which stem from at least two drillhole intersections, and inferred resources if there are isolated individual drillhole intersections. This resource estimation involves blocking out plan areas around drillhole quartz intersections, greater than 18cm thickness, and utilises mining recovery factors and confidence factors that have been developed at the mine over many years. This 18cm thickness criteria, based on the mine's empirical factors, is equivalent to a resource cut-off of 10.8kg/m^2 or 0.13% WO_3.

The current drillhole database contains data from 3,870 diamond drillholes, over a total drilled length of approximately 156,900 m. The majority of the data for resource estimation comes from underground drillhole data, which are generally either level to level vertical holes, 120m holes drilled down from the deepest available levels, or much shorter 13m holes drilled vertically up and down from current stope workings. These underground holes generally produce 47.6mm (NQ) core.

Face sampling involves measuring the area of wolframite crystals exposed on quartz veins. The areas of wolframite are accumulated for a specific length of exposed vein. Another internally developed empirical (Pintas) formula is then used to convert these crystal areas into wolframite grades in units of kg/m^2. Another formula is then applied to convert these grades into %WO_3 grades, which are effectively diluted according to the minimum stope height of 2.2m. These data are plotted on Autocad plans for each identified vein. Measured resources are then blocked out according to these measurements, using prescribed extension distances and aligned with the mine's planning grid system (80m x 100m on Level 3 and 100m x 100m on Level 0 to Level 2) and the mine's room and pillar block system (11m x 11m). The current cut-off applied in these resource calculations is 10kg/m^2, which is equivalent to approximately 0.12 %WO_3. These resources are calculated from these block definitions, along with an 84% mining recovery, representing the end of exploitation with remnant 3m x 3m stope pillars to support the roof. The resources assigned as either 'Pillar' resources if they have been developed, and therefore sampled, on at least three or four sides, or 'Virgin' resources, if they are extrapolated from one or two sets of face samples, and not yet developed into 11m x 11m pillars. All of the 'measured resources' blocked out at the mine are converted into reserves. There are no measured resources which are external to the reported reserves.

With 100 years of operating experience in a statistically difficult orebody, Beralt has derived a method of resource and reserves estimation that appears to be effective.

1.5 Mine Planning

Mining at the Panasqueira mine has evolved from labour intensive hand operations in the early 1900's through mechanized longwall methods to the mechanized room and pillar operation currently used. This mining method is possible in part due to the very competent host rock, and underground rock support is rare.

Blocks of ore are laid out initially in 100m by 100m sections by driving 5 m wide galleries, 2.2 m in height. The planned height of the stopes is nominally 2.2m, but increased slightly in areas where ore bearing veins are more variable in their dip, strike or thickness. A major emphasis in the stoping operation is to strive towards the 2.2m mining height in all working areas.

Indicated and inferred resources are initially picked up from drillhole intersections. Potential ore/vein intersections are categories according to approximately 10m vertical slices between each main level. Stope development ramps are then driven from level to level, and approximately horizontal sub-development is used to access the highest ore intersection. When the ore intersections have been found by lateral development, and verified by face samples, 5m wide galleries are driven to create roughly 11m by 11m pillars. This development is laterally aligned to the mine grid system, but vertically the development is inclined up or down so as to follow changes in ore dip. Faults, divisions and other string variations in the ore intersections sometimes necessitate additional in-stope diamond drilling. Following yet more face samples, further ore extraction is achieved with more development, to ultimately leave 3m by 3m pillars, which corresponds to an overall extraction rate of 84%.

Between each main level, within large overall mining blocks, veins are stoped out from top to bottom. A minimum of 3m is also required for the sub-horizontal pillars which are left between stope excavations vertically. The room and pillar grid system is regular over the whole mine, so all ultimate 3m x 3m pillars precisely line up vertically. Additional barrier pillars are left to preserve the main drives and panels on each main level.

The final 3m x 3m pillars generally collapse approximately 4-5 months after stope completion. Control points in each stoping area are monitored once a month. This monitoring data, together with observations of pillar conditions, are used in demarcating locally bad ground areas, so as to stop further stoping in these regions.

The mine has two main haulage levels (Level 2 at 560 mRL and Level 3 at 470 mRL) currently in use, with rail haulage of ore from 1.8 m diameter bored raises in the stopes to either the vertical rock hoisting shaft, connecting Level 3 to Level 2 and designed to transport the 6-ton wagons (4 t net weight) , or the orepass where all ore from the mine is stored prior to being crushed and transported along the 1,203m long ,17% inclination, Santa Barbara conveyor belt. . This belt discharges into 4 large coarse ore bins, 3 located under the main office and another in front of the office. In 2014 the mine produced 775 kt of underground ore (ROM) plus waste.

1.6 Mineral Processing

The underground jaw crusher delivers minus 100mm Run-of-Mine (ROM), to the conveyor to the crushing, washing and screening (CWS) plant at a rate of about 160 tph. Plus 0.8mm material is fed to the Heavy Media Separation (HMS) section, which generally accounts for approximately 80% of the original ore feed. The fines material (approximately 20% of the original ore feed) from the CWS plant passes on to the sand and slimes shaking tables. The reject material is conveyed out to the waste dump area, where it is either dumped or sold as gravel.

The HMS concentrate, is crushed in twin roll crushers. One of the roll crushers is dedicated to +3 –5 mm material from the HMS concentrate and this material is re-circulated to the HMS plant. The minus 3mm material is hydrosized prior to concentration by gravity shaking tables. The table concentration eliminates all the gangue minerals, particularly quartz and silicates. The sand tables' concentrate, referred to as Pre-Concentrate, contains all the dense minerals, which besides wolframite, includes sulphides, cassiterite and siderite.

The pre-concentrate produced by the sand tables is then screened and the two different fractions are passed over individual shaking tables, where sulphides are removed - assisted by flotation. These table tailings then become feed for a copper circuit. The table concentrates, without sulphides, are dried and screened to prepare three sized fractions for dry high-intensity cross-belt magnetic separators. This produces a high grade wolframite concentrate, and a non-magnetic cassiterite concentrate, which goes onto a tin circuit.

The overall WO_3 plant recovery averages 81%, producing over 90% of the recovered MTUs in a high grade concentrate averaging over 75% WO_3, and the remainder in another high grade concentrate of 74% WO_3. In 2016, approximately 69,000 MTUs of WO_3 were produced, along with 384 tonnes of copper concentrates and 69 tonnes of tin concentrates.

The mine is in the process of currently planning two metallurgical pilot studies for re-processing purposes:

- Using an XRF ore sorter for the processing of HMS rejects. It is anticipated that the sortable material will represent approximately 40% of the ROM feeding the HMS (size 10-25mm), with an average grade of approximately 0.025% WO_3.

- Re-processing of tailings material. The resources associated with accumulated tailings material have been estimated in the current report.

1.7 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 May, 2014. The current in-situ resource estimation for measured and indicated resources is shown in Table 1-1, and inferred resources are shown in Table 1-2 and Table 1-3. These resources are inclusive of the reported reserves. There are no measured resources external to reserves, as all resources classified as measured have been converted into proven or probable reserves. There has also been an evaluation of inferred resources within two tailings areas, as there is a potential for re-processing this material.

Table 1-1. Panasqueira Mine –Measured and Indicated Mineral Resources As of 30th September, 2016 (INCLUSIVE OF RESERVES)

LEVEL	*Measured*			*Indicated*			*Measured + Indicated*		
	Tonnes *Kt*	WO_3 *%*	WO3 *MTU x1000*	Tonnes *Kt*	WO_3 *%*	WO_3 *MTU x1000*	Tonnes *Kt*	WO_3 *%*	WO_3 *MTU x1000*
L0	51	0.18	9	1,038	0.23	236	1,089	0.22	245
L1	706	0.20	139	1,314	0.21	272	2,020	0.20	411
L2	468	0.20	92	2,984	0.24	726	3,452	0.24	818
L3	727	0.21	153	2,396	0.25	610	3,123	0.24	763
L4	-	-	-	343	0.22	76	343	0.22	76
Total	**1,951**	**0.20**	**393**	**8,076**	**0.24**	**1,920**	**10,027**	**0.23**	**2,313**

Notes

- . Resources shown are inclusive of reserves
- . Minimum thickness = 2.2m
- . Mining recovery = 84%

Measured Resources

. Cut-off = 0.12% WO_3 (Equivalent to 10 kg/m^2)

. Evaluation based on:

- Face mapping of wolframite exposed areas
- Areas converted to grade using Pinta's formula
- Blocks laid out on mine planning grid system

Indicated Resources

. Cut-off = 0.13% WO_3 (Equivalent to 10.8 kg/m^2)

. Evaluation based on:

- Drillhole quartz intersections
- Conversion to grade using D9 formula
- Blocks based on at least 2 drillhole intersections

. Additional factor applied:

- Confidence factor = 60%

Table 1-2. Panasqueira Mine – Inferred Mineral Resources

As of 30th September, 2016

Mine Region	Tonnes Mt	WO_3 %
Panasqueira Deep	0.18	0.22
North	2.73	0.23
South	2.25	0.20
Total	**5.16**	**0.22**

Notes

. Resources shown are exclusive of reserves

. Cut-off = 0.13% WO_3 (Equivalent to 10.8 kg/m^2)

. Evaluation based on:

- Drillhole quartz intersections
- Conversion to grade using D9 formula
- Blocks can be based on single drillhole intersections

. Additional factors applied:

- Minimum thickness = 2.2m
- Mining recovery = 84%
- Confidence factor = 40%

Table 1-3. Overall Property – Inferred Mineral Resources

As of 30th September, 2016

CATEGORY		Tonnes *Kt*	WO_3 *%*	WO_3 *MTU x1000*	Cu %	Sn %
Mine		5,158	0.22	1,110		
Tailings Areas	BL1 *	1,817	0.29	521	0.30	0.027
	BL2A *	3,347	0.24	802	0.21	0.022
Total		**10,322**	**0.24**	**2,433**		

Notes

. Inferred Mine resources based on a cut-off of 0.13% WO_3

* Tailings resources have no cut-off applied

. Resources shown are exclusive of reserves

Mineral Reserves have been determined. These reserves are part of the reported Mineral Resources. The reserves are based on face samples, and have been blocked out as part of the mine's on-going stope planning process. The areas blocked out as 'Pillar resources' have been sampled on all four sides, and have been classified by CIM guidelines as Proven Reserves. The areas blocked out as 'Virgin resources' have been extrapolated from one to three sets of face samples, and have been classified by CIM guidelines as Probable Reserves. These reserves are summarised in Table 1-4.

Table 1-4. Panasqueira Mine–Mineral Reserves
As of 30th September, 2016

	Proven Reserves		*Probable Reserves*		*Total Reserves*	
	Tonnes	WO_3	Tonnes	WO_3	Tonnes	WO_3
Level	Kt	%	Kt	%	Kt	%
0	25	0.19	26	0.17	51	0.18
1	238	0.22	468	0.18	706	0.20
2	216	0.21	251	0.19	468	0.20
3	297	0.24	431	0.19	727	0.21
Total	**775**	**0.22**	**1,176**	**0.19**	**1,951**	**0.20**

Notes

. Cut-off = 0.12% WO_3 (Equivalent to 10 kg/m^2)

. Evaluation based on:

- Face mapping of wolframite exposed areas

- Areas converted to grade using Pinta's formula

- Blocks laid out with stope planning process

. Additional factors applied:

- Minimum thickness = 2.2m

	Virgin Areas	11m x 11m Pillars	11m x 3m Pillars
- Mining recoveries:	**84%**	**67.3%**	**45%**

. Proven reserves are within (11 or 3m) pillars which have been sampled on at least 3 sides

. Probable reserves are within virgin areas which have been sampled on 1-2 sides

1.8 Conclusions

In the opinion of the QP, the following conclusions have been reached:

a) The empirical formulae developed at the mine, for evaluation purposes, have been used for decades and are supported by a very large amount of reconciliation data. The QP considers that these formulae, along with the other parameters and guidelines applied, do provide reliable methods of resource and reserve estimation.

b) The current resource and reserve estimations shown in this report have been reviewed by the QP. In the opinion of the QP, this review supports the estimation results presented.

c) The same resource/reserve cut-off grades have been in use since 2011. Since that time, the total reserve quantity has been maintained, although the overall total resource base has generally declined. This means that the mine's on-going stope development has elevated resource categories as planned, although drilling levels have declined, which has led to a reduction in overall resources.

d) The most important areas of the mine which offer the most scope for overall resource expansion are the Panasqueira deep area and Level 4 (below 470mRL).

e) The tailings dump BL2A resources have currently been classified with an **Inferred** resource category. The principal reason for this classification if chiefly the lack of assay data covering a major part of the area. But when the remaining assay data from all of the 2016 tailings-drilling samples becomes available, some areas within the BL2A should be able to be classified as **Indicated** resources.

f) The use of an XRF ore sorter on the HMS reject material presents an opportunity to recover additional amount of WO_3 product that is currently being rejected.

g) Similarly, re-processing of tailings presents another important opportunity for potential recovery of WO_3 product, as well as some copper and tin. Testwork connected with tailings re-processing is currently being done in both Spain (at the National Energy and Geology Laboratory in Porto) as well as CRONMET in South Africa.

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 an updated Resource and Reserve estimate for the Panasqueira mine, as of the end of September 2016. The project is an active underground operation, producing primarily a tungsten concentrate. Additional secondary copper and tin concentrates are also produced.

This report was prepared by Adam Wheeler, at the request of Almonty Industries. Assistance and technical detail were supplied by the technical personnel at Panasqueira. In connection with the preparation of this report, Adam Wheeler visited the Panasqueira site from November 23rd - 25th, 2016.

2.2 Terms of Reference

Adam Wheeler was retained by Almonty to provide an independent Technical Report on the Mineral Resources and Reserves at Panasqueira, as at September 30th, 2016. 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 Qualified Person responsible for the preparation of this report is Adam Wheeler (C.Eng, Eur.Ing), an independent mining consultant. In addition to the site visit, Adam Wheeler has carried out studies of relevant parts of the available literature and documented results concerning the project and held discussions with technical personnel of Panasqueira who have been doing exploration work at Panasqueira from 2013 – 2016.

The purpose of the current report is to provide an independent Technical Report and update of the resources and reserves of the Panasqueira mine, in conformance with the standards required by NI 43-101 and Form 43-101F1. 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.

Adam Wheeler has reviewed and analysed data provided by BTW and has drawn his own conclusions therefrom. Adam Wheeler has not performed any independent exploration work, drilled any holes or carried out any sampling and assaying.

2.3 Sources of Information

In conducting this study, Adam Wheeler has relied on reports and information connected with the Panasqueira mine. 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 BTW, 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$1.0364 to 1.00 Euros (€), unless otherwise stated. This exchange rate is the average value from Jan-September 2016.

3 RELIANCE ON OTHER EXPERTS

No other experts have been relied upon in the development of this report.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The Panasqueira mine is located in central Portugal, in the Distrito de Castelo Branco, one of eighteen administrative regions in the country. It lies on the southern edge of the Serra da Estrela, the highest Portuguese mountain range approximately 300km NE of the capital city of Lisbon, 200km SE of the port city of Porto and 40km WSW of Covilhã (Figure 4.1, Figure 4.2 and Figure 4.3).

Figure 4-1. Portugal Map

Figure 4-2. Central Portugal – Panasqueira Location

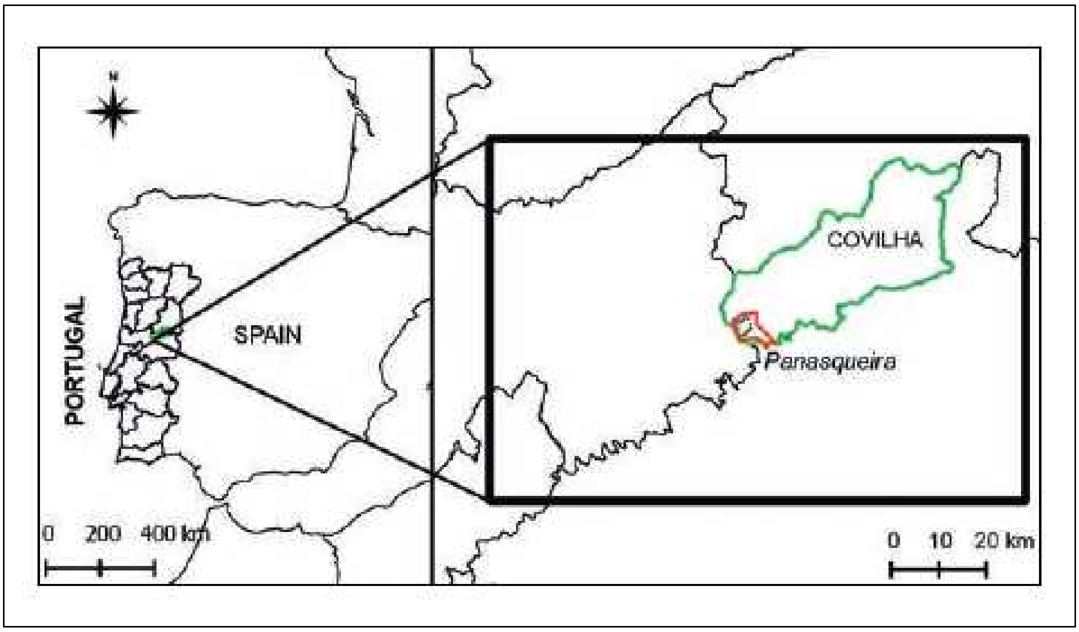


Figure 4-3. Panasqueira Mine – Satellite Image

4.2 Mining Concession

The Panasqueira mining concession named "Contract of Exploitation No. C-18", is owned and operated by Beralt Tin & Wolfram (Portugal) S.A. It covers an irregular, roughly "keyhole" shaped area trending NW-SE and is approximately 7.5km in length and 1.5km wide at the south-eastern end and 5.0km at the north-western end where the mine workings and mill facilities are located. The total area of the Panasqueira concession, Contract of Exploitation No. C-18, is 1913.5983ha.

BTW is the 100% owner of the various rights and interests comprising the Panasqueira tungsten mine in Covilhã, Castelo Branco, Portugal (the "Panasqueira Mine").

A number of smaller concessions were combined in 1992 to form the present day concession, which was officially granted on Dec 16, 1992, for a period of 60 years, and so is valid until Dec 16, 2052, with one or two possible extensions of periods of up to 30 years per extension. The concession has been legally surveyed.

Portugal uses the Hayford–Gauss system and not UTM or another more universally accepted coordinate system. Portugal is in the process of changing to the ETRS89 system. The coordinates of polygonal vertices are referred to an origin which is the central point for the Portuguese cartographic maps. This central point is at the approximate geometric centre of the country and is about 5km south and 31km west of the Panasqueira Mine.

The geographic coordinates of the town of Barroca Grande within the Panasqueira concession using the Hayford-Gauss system and the conventional Latitude and Longitude systems are shown in Table 4.1.

Table 4-1. Barroca Grande Coordinates

[Within Panasqueira Mining Concession]

Location (Portugese Term)	Hayford-Gauss	Conventional (approx)
Latitude (Perpendicular)	54,000.00	40° 9' 16" **N**
Longitude (Meridiana)	33,000.00	7° 44' 49" **W**

In addition to the government issued maps, there are also Portuguese Military topographic maps that use the same system of coordinates but with a different point of origin. The coordinates of this point of origin in relation to the Central Point are Meridiana (M)=200km and Perpendicular (P)=300km. Conversion from Military map coordinates to the normal coordinates requires subtracting 200 and 300 respectively from the military coordinate and converting the metres to kilometres. Table 4.2 lists the co-ordinates of all the survey corners for the irregular shaped Panasqueira concession, and shown in Figure 4-4.

Table 4-2. Panasqueira Mine - Concession Coordinates

Survey Point	Meridiana (East) (Hayford-Gauss)	Perpendicular (North) (Hayford-Gauss)	UTM (Easting)	UTM (Northing)
P	34,961	51,227	608,806	4,442,846
Q	33,636	52,554	607,468	4,444,159
R	31,016	51,873	604,856	4,443,453
A	29,385	54,890	603,196	4,446,453
B	31,227	56,251	605,024	4,447,832
C	33,639	56,566	607,432	4,448,170
D	33,679	55,324	607,484	4,446,929
E	33,570	54,823	607,380	4,446,427
F	36,631	51,760	610,447	4,443,355
G	36,506	51,561	610,322	4,443,156
H	36,510	51,624	610,326	4,443,219
I	36,216	51,986	610,032	4,443,581
J	35,936	51,911	609,752	4,443,506
L	35,718	51,752	609,535	4,443,347
M	35,726	51,461	609,542	4,443,056
N	35,371	51,074	609,187	4,442,669
O	35,294	50,792	609,110	4,442,387

Notes

. UTM coordinates F-O not supplied, so recalculated

4.3 Surface Rights

BTW has surface rights to a total surface area of 875.5886 ha, of which 728.9706 ha is within the concession. The BTW areas outside of the concession are in Rio, with 68.34777 ha and in Argemela, with 78.2703 ha. These areas are depicted in Figure 4-5.

Figure 4-4. Panasqueira Mine – Current Concession Boundary

(North - Vertical Grid Line; Scale – Grid Square = 1km^2)

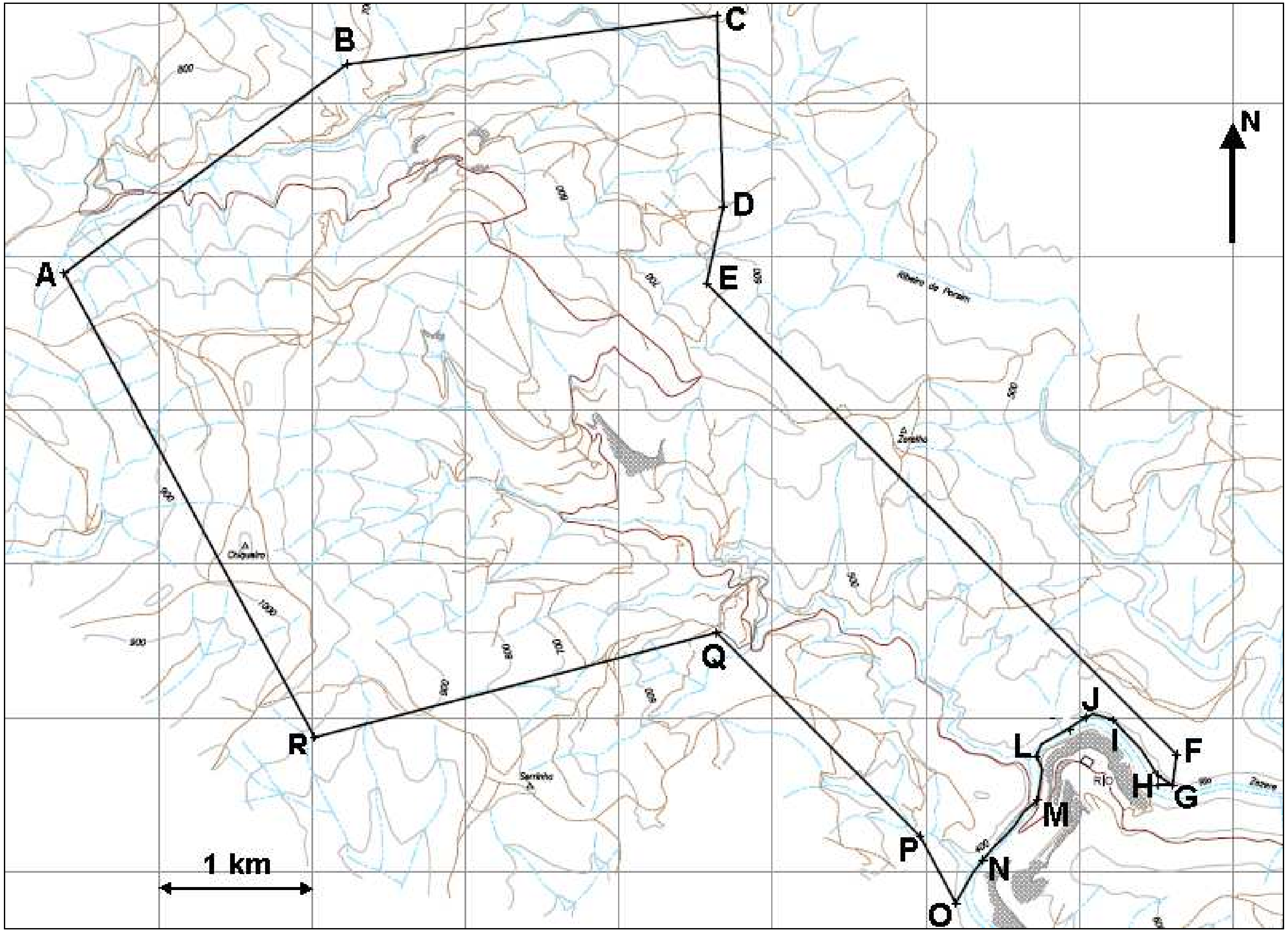
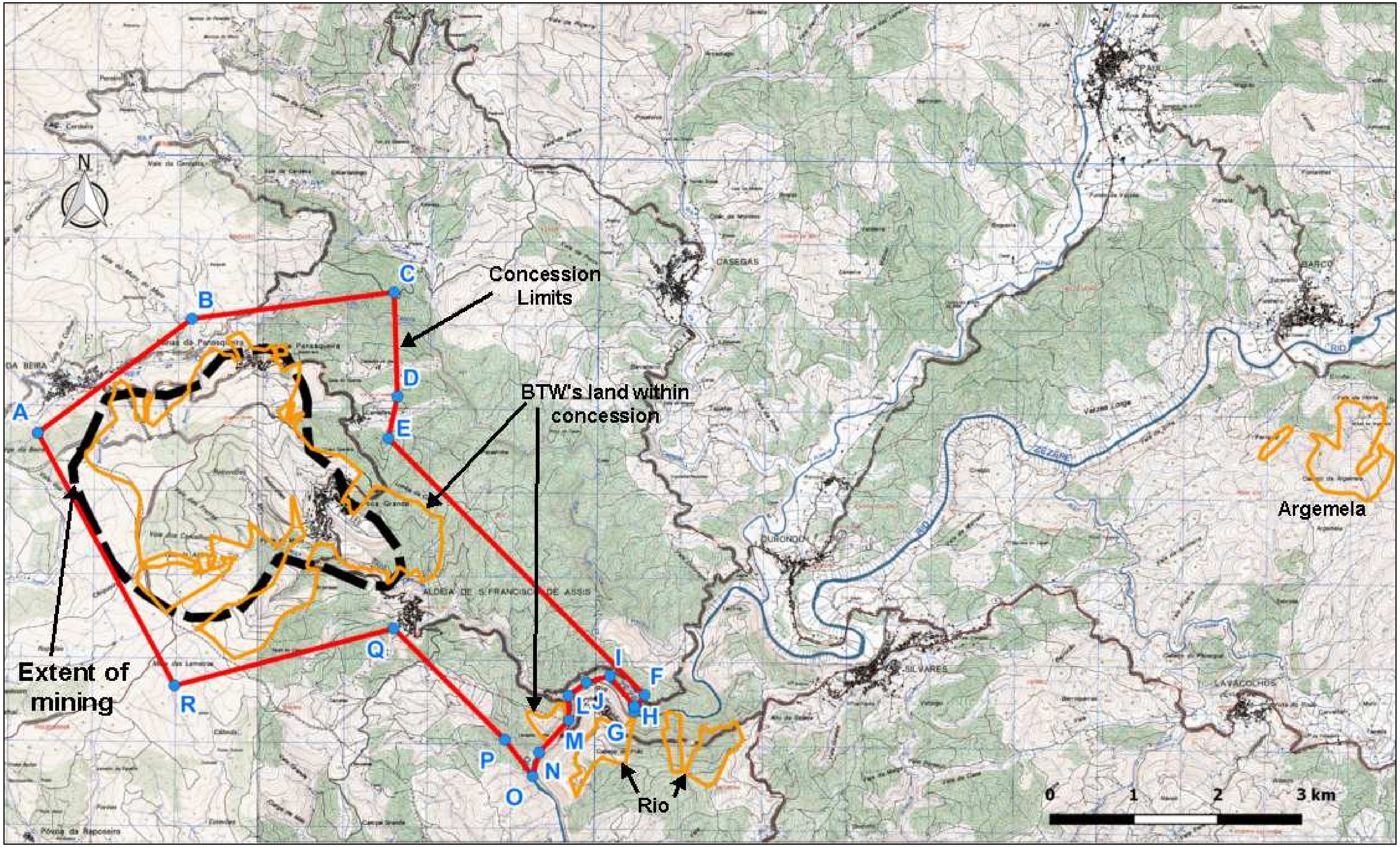


Figure 4-5. Limits of BTW's Surface Rights



4.4 Legal Obligations

The following is a general description of the requirements for the Mining Contract, as prepared by the State Geological Authority, Direção Geral de Energia e Geologia (DGEG):

- The concession contract entitles the Licencee the sole right to exploit the specified resources within the area set forth in the contract. The duration of the contract is stipulated on the basis of the calculated duration of the resources under normal operating conditions in accordance with the geological report, preliminary feasibility study and mining plan submitted together with the application. It covers an initial period and 1 or 2 prorogations.

- New mining operations shall proceed in accordance with a general plan of work approved or revised (as necessary) and with yearly work programmes, and in a manner consistent with applicable regulations and good mining and environmental practice. For operations larger than 15ha or 200,000t (and all scale of operations if located in areas of protected landscape identified in the law) an environmental impact assessment and a protection plan must be submitted to be approved by the Environment Ministry as a condition for the award of the contract. In accordance with the provisions of the law, if the environmental impact assessment is refused by the Minister of Environment the mining licence should not be granted.

- The Licencee shall appoint and register the technical manager in charge of the mining operations.

- Notice of suspension of mining operations must be given to the Minister and consent obtained, and the suspension may not last longer than authorised, unless renewal thereof shall have been requested and granted, if justified.

- Unauthorised suspension of mining operations may lead to cancellation of the contract if the Licencee, following a reasonable period of notice, shall not have put an end to the suspension or presented acceptable justification.

- Cancellation of the contract for non-compliance with legal and contractual conditions is subject to the rules and procedures established by law.

- The contract also stipulates annual payment of a royalty, in the form of a percentage of the values of the F.O.B/Concession sales. The last review of these rates was made in March, 2012 and pertains for 12 years. After deductions of transportation costs and 3% of marketing charges, the royalty due is on a sliding scale, with no payment for wolfram prices less than $250/mtu, 0.5% for prices $250-300/mtu, 1.0% for prices $300-350/mtu, 1.5% for $351-400/mtu and so on. Royalties were not paid from 2012-2014, and for this period the mine is required to pay €80,000 during 2016. No royalties are due from 2015.

- Assignment of mining rights is subject to the prior consent of the Minister, as mentioned above in connection with assignment of prospecting and exploration rights.

- The Licencee is entitled to apply for expropriation for public service of the land necessary to the mining operations in the event that agreement cannot be reached with the respective land owners.

4.5 Taxation

The mine is subject to three principal taxes: property taxes, income tax and VAT sales tax. Property taxes are approximately €15,000 per year on the plant site, office and many houses. The company pays 21% income tax, and an additional 1.2% municipal tax, on net profits. Outstanding tax payments for 2010-2012 are €162,639, which is the company is requesting to pay in 24 instalments, from 2017 and 2018. As the tungsten concentrate product is sold overseas, the VAT paid can be reclaimed.

4.6 Environmental Liabilities

The Panasqueira mine, already in operation, is fully permitted with all the necessary requirements under Portuguese mining law. Under Portuguese mining law, the company is required to file an annual report that is due at the end of March for the previous calendar year's work. The company must also file an annual work plan outlining the company's proposed work for the upcoming year, a document that must be filed by the end of September of each year.

The known environmental liabilities are:

- **Barroca Grande Waste Dump.** As described in Section 20.6, the Barroca Grande dump lies adjacent to and above Bodelhão Creek that flows south-easterly to the Rio Zêzere and comprises primarily HMS tailings (gravel), waste rock and sand tails, all of which are very benign and possibly marketable as aggregate.

- **Fine Tailings Disposal Areas(TMFs).** There are two disposal areas – TMF 1, which is completely finished and TMF 2A, into which fine tailings is currently being discharged. TMF 2A is nearly full, with approximately 4 year's more capacity. A new disposal area, TMF 2B, is under construction, and is planned to be in used for tailings storage from 2019 onwards, as described in Section 20.6.

- **Underground Waste Discharge Treatment**. Acid mine water drainage is treated in the Salgueira Water Treatment plant, as described in 20.5. This plant was expanded and upgraded in late 2011, so it can now handle 500m^3/hr.

4.7 Licences and Permits

BTW holds all necessary environmental licences:

• Environmental Permit (in process of being renewed until 2026);
• Licence for using water resources (License for holes AC1, AC2 and AC3 have not expired; otherwise covered by issue of Environmental License, which is in the process of being renewed);
• Licence for rejecting wastewater (Authorised to be covered by of Environmental License, which is in the process of being renewed).

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, PHYSIOGRAPHY

5.1 Topography, Elevation and Vegetation;

The Panasqueira mine is situated on the southern edge of the Serra da Estrela, the highest Portuguese mountain range; Mt. Estrela at 1993m.a.s.l, lies approximately 20km northeast of the Panasqueira mine.

The concession is situated in moderately rugged, pine and eucalyptus covered hills and valleys with elevations ranging from 350m in the southeast to a peak of 1,083m in the north-western corner of the concession.

5.2 Accessibility

Easy all-weather road access is provided by a combination of two-lane and four-lane divided highways, for approximately 280km from the capital Lisbon to the town of Fundão. The mine is located approximately 35km from Fundão along a two-lane windy, paved road to Barroca Grande, the community that supports the mine and contains the Panasqueira plant and offices. Government maintained paved roads, or dirt roads maintained by the company can access most areas of the concession. All essential services such as food and lodging are available from the numerous nearby towns and villages including all heavy-duty equipment. Alternative access to the mine is via two and four lane highways to Porto, a port city located approximately 200km to the north-west. The railway from Lisbon reaches the town of Fundão.

5.3 Local Resources

Local population density is low. Most of the population work in agriculture (pasture, grain, timber, fruit, oil and wine), and at the mine and small service industries. The village of Barroca Grande is notable for its long history of mining, with a significant number of people being employed by BTW and currently engaged in the activities of operation and maintenance of the mine, concentration plant and administrative services.

5.4 Climate

The climate is pleasant, with average temperatures of 24°C in July – August and 4°C in December. Rainfall is most common in November – January, with seasonal averages of 200mm in December to 10mm in August.

Weather statistics for Covilhã, about 40km north of Panasqueira, are shown in Figure 5.1 and Figure 5.2

Figure 5-1. Covilhã -Average Temperatures

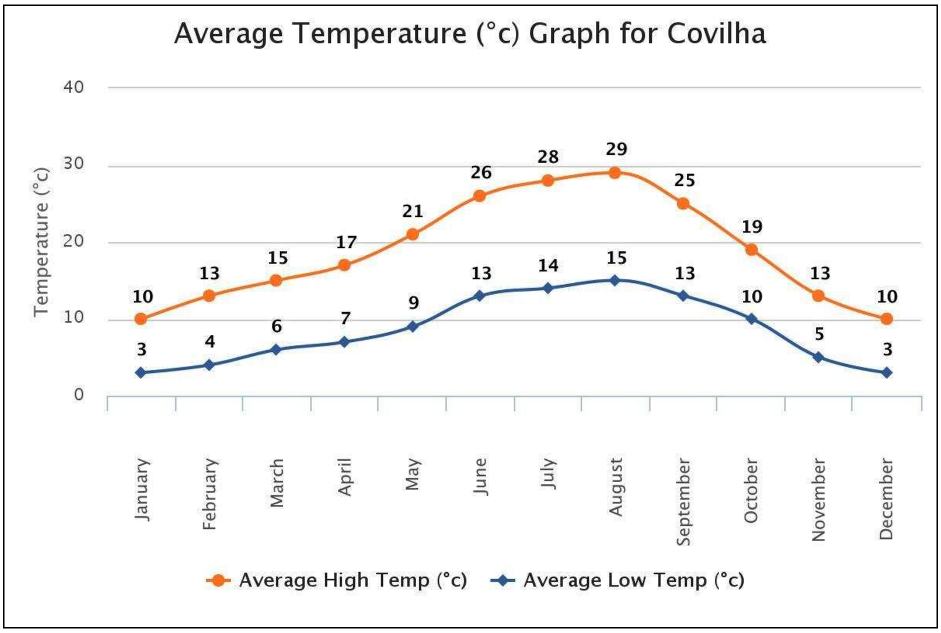
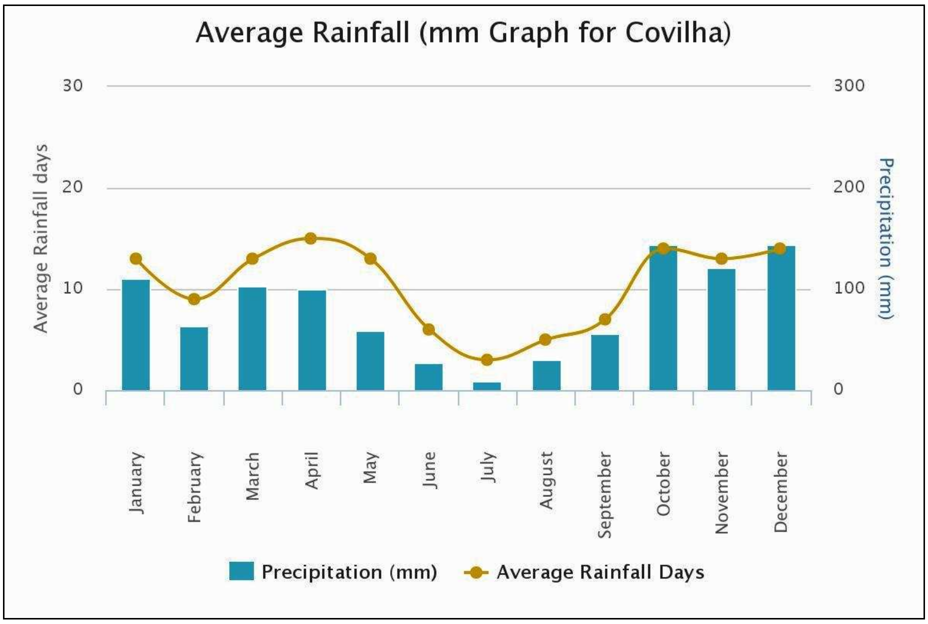


Figure 5-2. Covilhã – Average Rainfall



5.5 Infrastructure

The mine is serviced by the national electricity grid and has transformers on site to fully utilise the electricity supply. The mine has modern telecommunication, with touch-tone dialling, modem connected data communication and international cellular coverage.

The concession contains abundant water sufficient to continue to support future mining operations. The mine is situated in a natural cirque at the top of a valley which ends at the Rio Zêzere (Zêzere River). Numerous unnamed creeks and underground water sources feed the mine and drain into the small Bodelhão Creek that flows south-easterly to the Rio Zêzere. The Rio Zêzere is the major source of water for the city of Lisbon, so the mine conducts constant water monitoring. Mining can take place year-round.

6 PROJECT HISTORY

6.1 Ownership

The mining company was founded in 1896 to mine tungsten at Panasqueira as the industrial uses of the commodity were first being developed throughout the world. The first area where wolframite ore was recovered was from Cabeço do Piao (now known as Rio); the first areas mined near present day workings were from Vale das Freiras, Vale da Ermida, Panasqueira and Barroca Grande. All the individual concessions were grouped into one single mining area known as the “Couto Mineiro da Panasqueira” which covered almost the same ground as the present day concession.

In 1904, a new mechanised treatment plant was built near Cabeco do Piao (Rio), which was situated on the Zezere River for water supply. There is a report of the delivery of 41t of ore to various buyers. The first underground drifts were opened at Rio but mining activity decreased as richer veins were discovered at nearby Panasqueira. Milling and treatment of the Panasqueira ore at Rio continued until September 1996, when the final concentration equipment began to be moved to Barroca Grande.

In 1911 the Wolfram Mining and Smelting Company was formed and purchased all the rights to the concessions including the buildings, the equipment and 125ha of rural land. In 1912, the new company made major investments in machinery and equipment, upgrading the Rio treatment plant and installing the first aerial 5,100m rope-tramway that brought the ore from different mining sites at Panasqueira to the Rio plant. In 1912, the production of wolframite concentrates was reportedly 267t of 65% WO_3 mined by 244 workers from 10,791t of vein material as well as 86,063t of host rock.

World War I in 1914 saw a period of accelerated expansion and growth of the mining operation. The production rate was increased, the plant was enlarged and a furnace was installed. The number of workers at the mine increased to 800. In addition, the company allowed individuals to work small surface veins exposures in the concession area, an activity that involved approximately 1,000 people recovering small quantities of ore for sale back to the company. The hills surrounding the present day operation contain many old pits and shafts left from these small operations.

From the end of the World War I to 1928, the mining activity was contingent on the price of tungsten. During this period of uncertainty in the tungsten price, the search and the recovery of tin was intensified. In 1927 approximately 110t of cassiterite concentrate as well as 190t of wolframite concentrate were produced. In 1928, the Wolfram Mining and Smelting Ltd. reorganised and changed its name to become Beralt Tin & Wolfram Company (BTW - the name, Beralt, being derived from Beira Alta, the local region). The new company increased production to 30t of concentrates per month but unfortunately, the tungsten price fell sharply in 1931 that required the mine to intensify the production of the more valuable tin concentrates

The tungsten price recovered in 1934 and stayed high through to the end of World War II. These were the years of peak production at the mine. Manpower increased from 750 workers in 1933 to 3,300 in 1940 and nearly 5,800 in 1943. Portugal was neutral during the war and the mine could count on a steady supply of workers and sales to both sides in the conflict. In addition, there were approximately 4,800 individual miners working the small veins on the surrounding hills.

The tungsten price fell sharply again after the end of the war and only increased in 1950 due to the Korean conflict. Steady production was maintained with increased mechanisation and increased production of tin and the introduction of the recovery of copper from the plant tailings. It was also during the 1950's that the company recognised the importance of water quality and installed the water treatment plant at Salgueira, located 1km downstream from the main mining operations. In 1962, the plant began the production of copper concentrates by treatment of chalcopyrite from the plant tailings.

From 1957 to 1965 the tungsten price continued to fall, and as a consequence, production was severely curtailed to cut costs. In 1967, Charter Consolidated plc of the UK acquired a large block of shares. Tungsten prices began to rise in 1966, peaked in 1970 which lead to an expansion programme at the mine and mill, but again the price fell, rose in the mid 70's and then dramatically fell in the early 1980's. In an attempt to counter the low tungsten prices, production from the mine was stockpiled rather than sold, but the financial strain created from low cash flow forced the company to again reorganise. A new Portuguese company was formed under the name Beralt Tin & Wolfram (Portugal) S.A., with IAPME, a Portuguese state holding company, as the new 20% shareholder and Charter Consolidated holding the balance of the shares.

Since 1974, the company has accelerated the mechanisation of the underground operations in order to further reduce labour costs and changed the mining method from largely longwall stoping to the more mechanised room and pillar method. The opening of a new inclined conveyor shaft from the deeper Level 2 began in 1977 and the extraction of ore from Level 2 began in 1982. In 1983 the tungsten price began to weaken so the owners of the operating company, Charter Consolidated plc, sold its 80% share to Minorco S.A. in 1990. Unfortunately, the tungsten price did not recover and after heavy losses, Minorco closed the mine at the end of 1993 and put the assets up for sale. Avocet Ventures Inc. of Canada, the predecessor of Avocet Mining PLC, and then listed on the Vancouver Stock Exchange, acquired Minorco's shares in June 1994 and the remaining Portuguese state holding company's interest in March 1995.

Avocet reopened the mine in January 1995, and in 1998 moved the remainder of the plant from Rio so that all the milling operations would be located at its present day site at Barroca Grande. A new underground shaft connecting Level 2 with Level 3, 90m in depth, was completed and a new 284kW winch was installed. The complete shaft system with automated mine car handling began operation in April 1998. The main part of the Level 3 haulage infrastructure was completed, allowing the ore mined from the Level 3 stopes to be extracted from the shaft.

Primary Metals Inc. held a 100% interest in the Panasqueira concession, Contract of Exploitation No.C-18, through its subsidiary companies, which were acquired from Avocet Mining PLC in March 2003. Primary held a 100% interest in Primary Mining Canada Inc., (formerly Avocet Mining Canada Inc.), which owned a 100% interest in Beralt Ventures Inc. (formerly Avocet Ventures Inc.), which in turn held a 100% interest in Beralt Tin and Wolfram (Portugal) S.A., ("Beralt"), the company owning the concession.

The concession was granted to Beralt on December 16, 1992 for a period of 60 years with one or two possible extensions of periods of up to 30 years per extension and was amended and approved on November 17, 2005 to remove part of the old mining area which was of no further interest to Beralt.

In 2005, certain members of Almonty Industries Inc.'s current board and management team, through Almonty Partners LLC, acquired a 55% controlling interest in Primary Metals Inc.("Primary") . At the time, Primary was a Canadian public company with its shares listed for trading on the TSX-Venture Exchange (ticker symbol PMI). Primary held a 100% interest in the Panasqueira concession, Contract of Exploitation No.C-18, through its subsidiary company BVI, which were acquired from Avocet Mining PLC in March2003. Primary held a 100% interest in Primary Mining Canada Inc., (formerly Avocet Mining Canada Inc.), which owned a 100% interest in BVI (formerly Avocet Ventures Inc.), which in turn held a 100% interest in BTW, the company holding the concession to the Panasqueira Mine.

In late 2007, Sojitz Corporation acquired 100% of the shares of Primary, and the name of the subsidiary companies were changed to BVI (from Avocet Ventures Inc.) and to Sojitz Beralt Tin & Wolfram (Portugal) S.A. from BTW.

In 2016 Almonty acquired ownership of 100% of BVI from Sojitz. BVI owns 100% of BTW, which in turn is the 100% owner of the various rights and interests comprising the Panasqueira tungsten mine in Covilhã, Castelo Branco, Portugal (the "Panasqueira Mine"). Almonty is therefore the indirect owner of 100% of BTW and the Panasqueira Mine.

The mine is one of the largest operating tungsten mines in the Market Economy Countries (MEC). Production quantity is variable but currently is in the range of 85,000 – 95,000mtu WO_3 (1mtu – metric tonne unit – is equal to 10kg) per year, depending on the ore grade extracted from different areas of the mine. The tungsten concentrate produced has a WO_3 content of 74 – 75%, which is one of the highest grades available on the market.

6.2 Historical Exploration

Even though the mine has been in operation for more than 100 years, very little in the way of “true” exploration has been conducted on the property outside the active or past mine workings. The hills surrounding the present-day operation contain many old pits and shafts left from small tungsten vein hand mining operations.

True exploration of the concession and the area immediately surrounding the concession seems to have been fairly limited. A regional stream sediment geochemical survey carried out between from 1982 to 1984 by Beralt outlined several anomalous areas of elevated tungsten and tin geochemistry in the concession and along the borders.

Beralt also completed a lithogeochemical survey over selected areas of the Panasqueira concession and in the immediate area adjacent to the concession which returned areas of tin and tungsten anomalies. Two areas appear to stand out as potential sites for follow-up work as defined by both anomalous lithogeochemical and stream sediment sample results

6.3 Historical Resource Estimates

With 100 years of operating experience, Beralt has developed internal empirical (D9 and Pintas) formulae to determine the grade of the deposit. These formulae have been validated by external consulting companies several times and have been considered the one reliable method in light of the nugget mineralisation of the wolframite in the vein. They are described in detail in Section 11.

Resources are estimated from a formula based on vein thickness: the D9 formula. This formula is used to estimate Inferred and Indicated Resources by in house geologists. Measured resources – Virgin Area & Pillars – are estimated by measuring the area of wolframite crystals and applying the Pintas Formula.

All resources are reduced to 84% to allow for resources left in pillars that need to be maintained, because of the mining method, for stability. For Inferred and Indicated Resources safety factors of 40% and 60%, respectively, are applied.

For Measured resources (reserves), recoverable resources are estimated to be 84% in unworked areas, 67.3% in areas with 11m x 11m pillars and 45.45% in areas with 11m x 3m pillars. All of the measured resources evaluated in this way are equivalent to reported CIM reserves. There are no measured resources external to reserves.

Historical resource estimates from 2011 up to 2016 are summarised in Table 6.1. The issuer is not treating these historical estimates as current mineral resources or mineral reserves – the table below is just shown for comparative purposes.

Table 6-1. Historical Resource Estimates

Date	Measured Resources		Indicated Resources		Inferred Resources	
	Tonnes *Mt*	WO_3 *%*	Tonnes *Mt*	WO_3 *%*	Tonnes *Mt*	WO_3 *%*
2011 January	1.25	0.25	10.93	0.23	6.07	0.22
2011 July	1.29	0.24	10.93	0.23	6.03	0.22
2012 January	1.20	0.24	11.05	0.23	6.04	0.22
2012 July	1.22	0.23	10.82	0.23	5.96	0.22
2013 January	1.23	0.22	9.68	0.23	5.92	0.22
2013 July	1.26	0.21	9.43	0.23	5.88	0.22
2014 January	1.28	0.21	8.48	0.24	5.03	0.22
2014 July	1.57	0.20	8.14	0.24	5.01	0.22
2015 January	1.54	0.20	7.94	0.23	4.93	0.22
2015 July	1.66	0.21	7.88	0.24	4.91	0.22
2016 September	1.95	0.20	8.08	0.24	5.16	0.22

Notes

. Cut-offs applied:

Measured Resources = 10kg/m^2 = 0.12% WO3

Ind+Inf Resources = 18cm > 10.8kg/m^2 = 0.13% WO_3

6.4 Historical Production

Historical production figures (from 1934 to 2016) are summarised in Table 6.2.

Table 6-2. Historical Production

Year	Concentrate Produced			ROM	Year	Concentrate Produced			ROM
	WO_3	Sn	Cu	Ore		WO_3	Sn	Cu	Ore
	t	*t*	*t*	*Kt*		*t*	*t*	*t*	*Kt*
1934	262	68			**1976**	1,597	75	1,440	436
1935	433	158			**1977**	1,287	58	1,176	405
1936	675	167			**1978**	1,450	62	1,101	435
1937	957	134		294	**1979**	1,783	88	1,818	455
1938	1,485	114		***375***	**1980**	2,145	133	2,524	522
1939	1,830	135		582	**1981**	1,808	147	2,131	538
1940	2,212	101		***605***	**1982**	1,849	156	1,753	689
1941	2,232	41		807	**1983**	1,580	126	1,511	558
1942	2,083	44		***514***	**1984**	2,085	158	1,427	666
1943	2,521	77		499	**1985**	2,539	90	932	805
1944	802	27		***455***	**1986**	2,667	66	858	675
1945					**1987**	2,011	60	607	475
1946	199				**1988**	2,300	57	582	467
1947	2,041			444	**1989**	2,296	59	665	593
1948	1,850			456	**1990**	2,343	51	530	613
1949	1,690	205		426	**1991**	1,619	43	455	412
1950	1,697	202		558	**1992**	1,864	37	498	491
1951	2,271	69		676	**1993**	1,280	28	418	332
1952	2,281	137		689	**1994**	100	2	37	7
1953	2,287	110		791	**1995**	1,467	14	0	335
1954	2,105	69		693	**1996**	1,305	15	550	303
1955	2,054	178		724	**1997**	1,729	44	483	431
1956	2,227	211		799	**1998**	1,381	24	279	344
1957	2,129	305		639	**1999**	750	7	77	179
1958	1,314	664		615	**2000**	1,269	12	132	332
1959	1,740	353		690	**2001**	1,194	23	118	378
1960	2,095	59		578	**2002**	1,179	21	81	346
1961	2,135	46		539	**2003**	1,213	20	99	355
1962	1,714	56	103	306	**2004**	1,277	50	138	432
1963	940	89	184	174	**2005**	1,405	44	187	574
1964	1,026	52	202	182	**2006**	1,342	28	235	642
1965	897	11	175	195	**2007**	1,456	48	258	762
1966	1,117	10	250	193	**2008**	1,684	32	186	782
1967	1,261	14	337	261	**2009**	1,410	36	164	720
1968	1,442	19	429	357	**2010**	1,364	25	198	792
1969	1,356	25	472	401	**2011**	1,399	45	238	905
1970	1,600	34	696	538	**2012**	1,303	47	228	830
1971	1,423	26	459	492	**2013**	1,174	103	352	789
1972	1,539	31	601	539	**2014**	1,131	98	732	775
1973	1,860	49	682	519	**2015**	799	53	361	518
1974	1,827	70	843	481	**2016**	926	69	384	643
1975	1,742	87	1,034	490	**Total**	**128,110**	**6,576**	**32,410**	**40,317**

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

Portugal's geology can be divided into two major groups, the Hesperian Massif and the Epi-Hercynian cover rocks with the principal metallic mineral deposits of Portugal being located in the former. The Epi-Hercynian Covering consists of Carboniferous aged granites related to the Hercynian Orogeny.

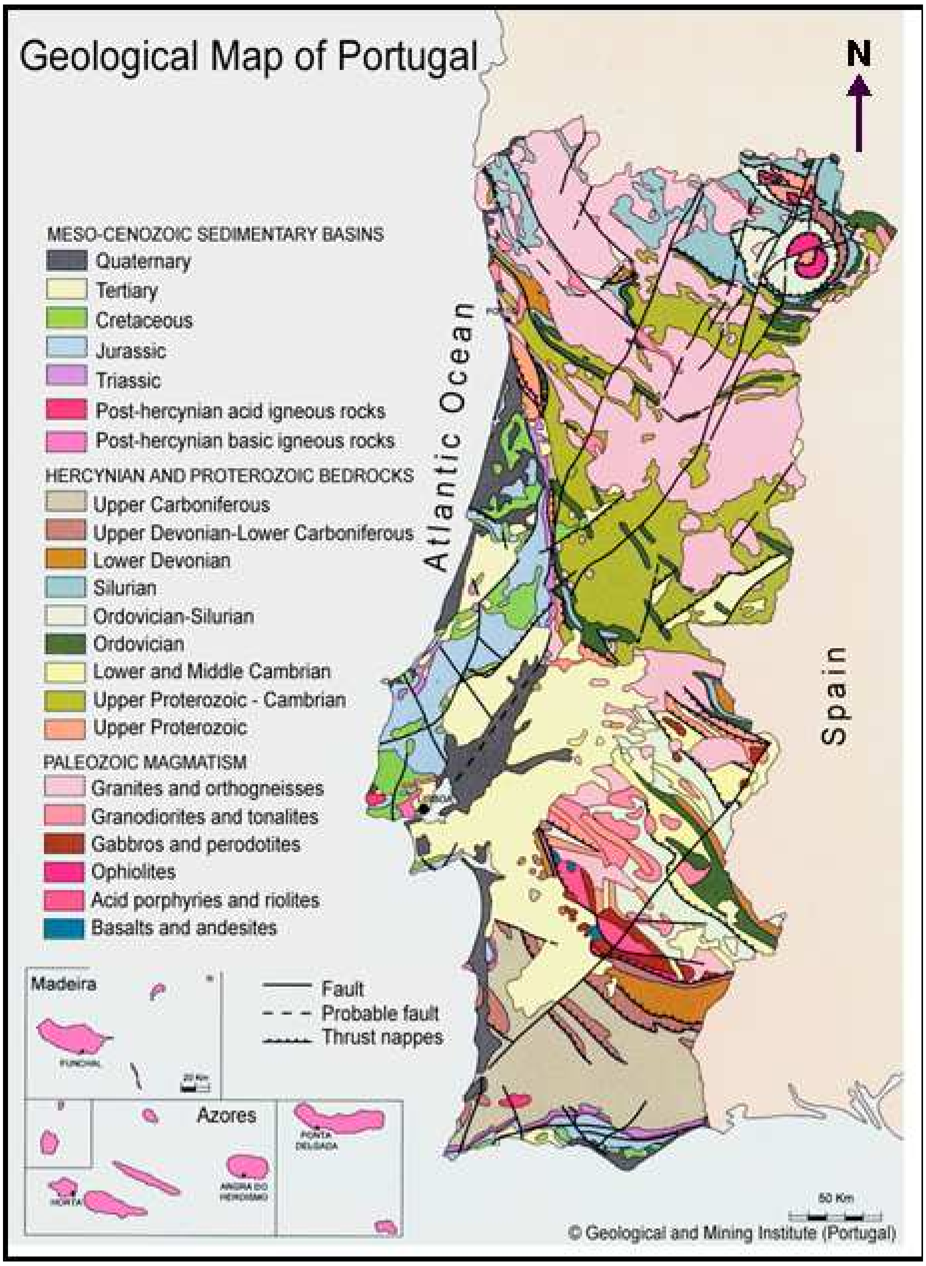
The Hesperian Massif can be further divided into four principal tectonic domains (from north to south); the Galicia-Trás-os-Montes Zone, the Central Iberian Zone, the Ossa-Morena Zone and the South Portuguese Zone. The Galicia-Trás-os-Montes Zone lies at the northern tip of Portugal and comprises principally mafic and ultramafic metamorphic complexes (Figure 7.1).

The Central Iberian Zone (CIZ), which hosts the Panasqueira Mine, is one of the most important metallogenic provinces of Europe. The CIZ is composed of a thick sequence of flysch-type units primarily composed of greywackes, shales and schists of late Precambrian to Cambrian age. Intruding this flysch sequence are the Epi-Hercynian syn-metamorphic muscovite-biotite granites or post-metamorphic biotite rich granites. The tin-tungsten deposits, such, as Panasqueira, are spatially related to the contacts between the flysch-type units and the syn-metamorphic muscovite-biotite granites.

The Ossa-Morena Zone (OMZ) is located to the south of the CIZ and separated by the large Blastomylonitic Belt shear system which extends across Portugal from the city of Porto, into southern central Spain near the city of Cordoba. The OMZ contains a complex series of metamorphic rocks aged from Precambrian through to late Devonian, including a sequence of both calc-alkalic and basic intrusives.

The South Portuguese Zone is composed of volcano-sedimentary units of late Devonian to early Carboniferous age. The acid volcanic rocks in this sequence host the massive sulphide deposits that are characteristic of the Iberian Pyrite Belt such as the Neves Corvo (Somincor - Lundin Group) and the Aljustrel deposits (Almina). The world famous Iberian Pyrite Belt is host to at least thirty currently-producing or past-producing massive sulphide mines, 11 in Portugal and 19 in Spain.

Figure 7-1. Map of the Geology of Portugal

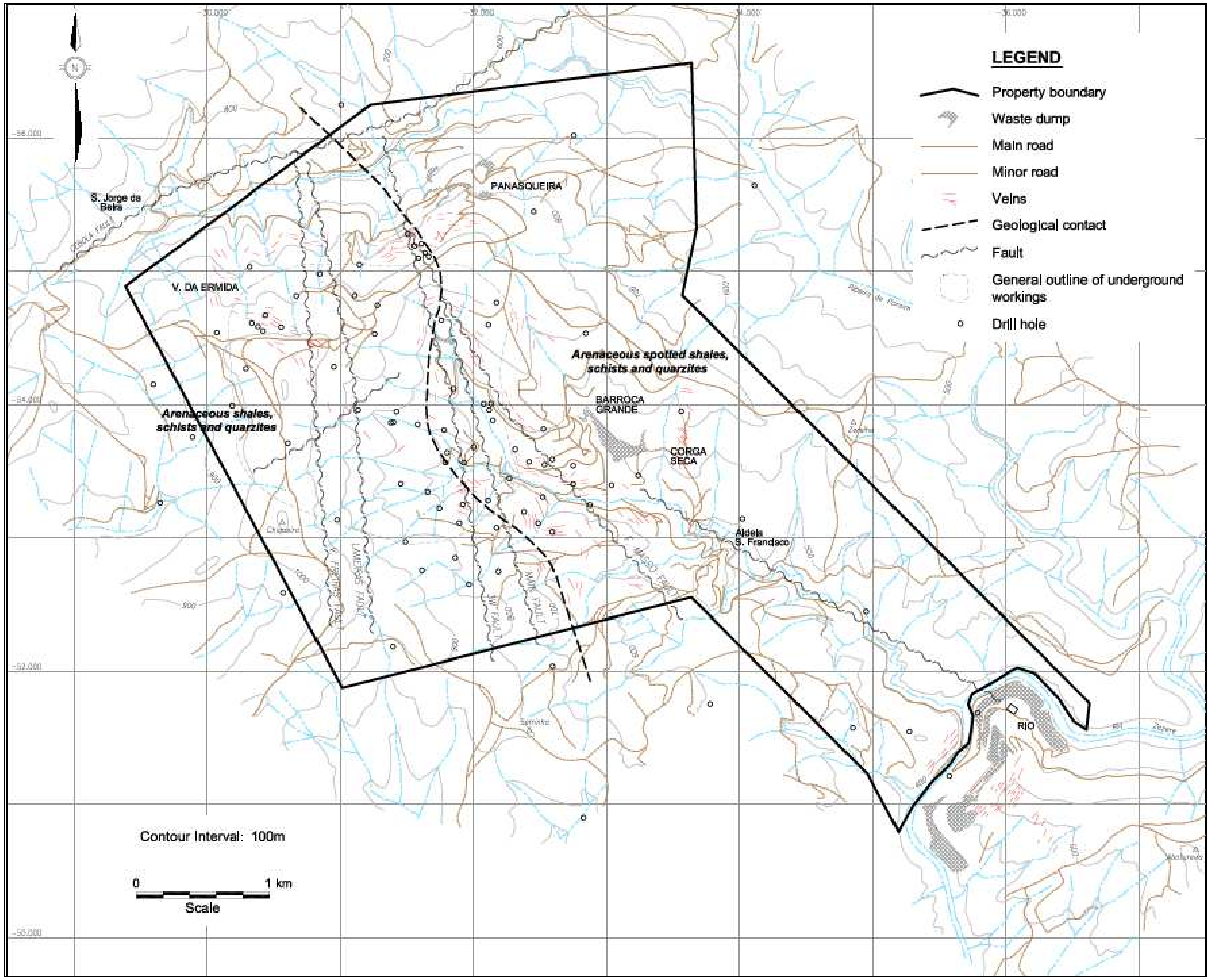


7.2 Local Geology

The Panasqueira deposit lies in a folded metasedimentary sequence known as the upper Precambrian-Cambrian aged Beira-Schist Formation which is composed of a several thousand metre thick sequence of lower marine flyschoid schists, greywackes, lenticular, thinly bedded mudstones, shales and arenites. The concession has been divided into two general geological units (Figure 7.2).The western half of the concession contains arenaceous shales, schists and quartzites. The eastern half is underlain dominantly by arenaceous spotted shales, schists and quartzites. There is interfingering between both major geological units.

Figure 7-2. Panasqueira Concession Geological Map

(After Orequest 2006)



7.3 Local Geology

The Beira Schist Formation was subjected to lower greenschist grade regional metamorphism during the early compressive stages of the Hercynian Orogeny. During the primary deformation event, the rocks were folded into a sequence of tight, upright isoclinal folds generally striking NW-SE and accompanied by a sub-vertical slaty cleavage sub-parallel to bedding.

The Panasqueira deposit consists of a series of stacked, sub-horizontal, hydrothermal quartz veins intruding into the Beira schists and shales. A second set of non-wolframite bearing quartz veins (veins contain minor chalcopyrite, galena and pyrite) also exists at the Panasqueira deposit, and are aligned with the vertical foliation and cut by the later tungsten-bearing hydrothermal vein system. Intrusives are an important component of the mineralising events at Panasqueira.

Regionally, the Serra da Estela batholith lies to the northeast, a post tectonic intrusive. A slightly older syn- to late-tectonic intrusive lies to the east. As the result of the regional metamorphism, the sediments changed to biotite-chlorite schists and phyllites, and the more arenaceous units converted to dark, fine-grained quartzites. Metamorphism of the sediments related to these intrusions, also resulted in the generation of distinctive spotted schists, a thermal metamorphic overprint characterised by elliptical spots composed of biotite and chlorite.

The Serra da Estrela batholith does not outcrop on the Panasqueira concession; however greisenised granite, thought to be the roof of the batholith, has been intersected in several of the exploration diamond drill holes and in a few stoped areas in the north of the deposit. The granite is thought to be the principal source of the mineralising fluids responsible for the economic wolframite vein system. The roof rocks range from a more or less strongly greisenised granite to muscovite-quartz unit with associated pyrite, chalcopyrite and is also cut by the later tungsten bearing quartz veins. The greisen-granite dome is capped by about 15m of massive unmineralised quartz (silica cap), which is thought to have been emplaced in the void left by the contraction of the intrusive material. The granites have been Rb-Sr dated as 289 ± 4 Ma.

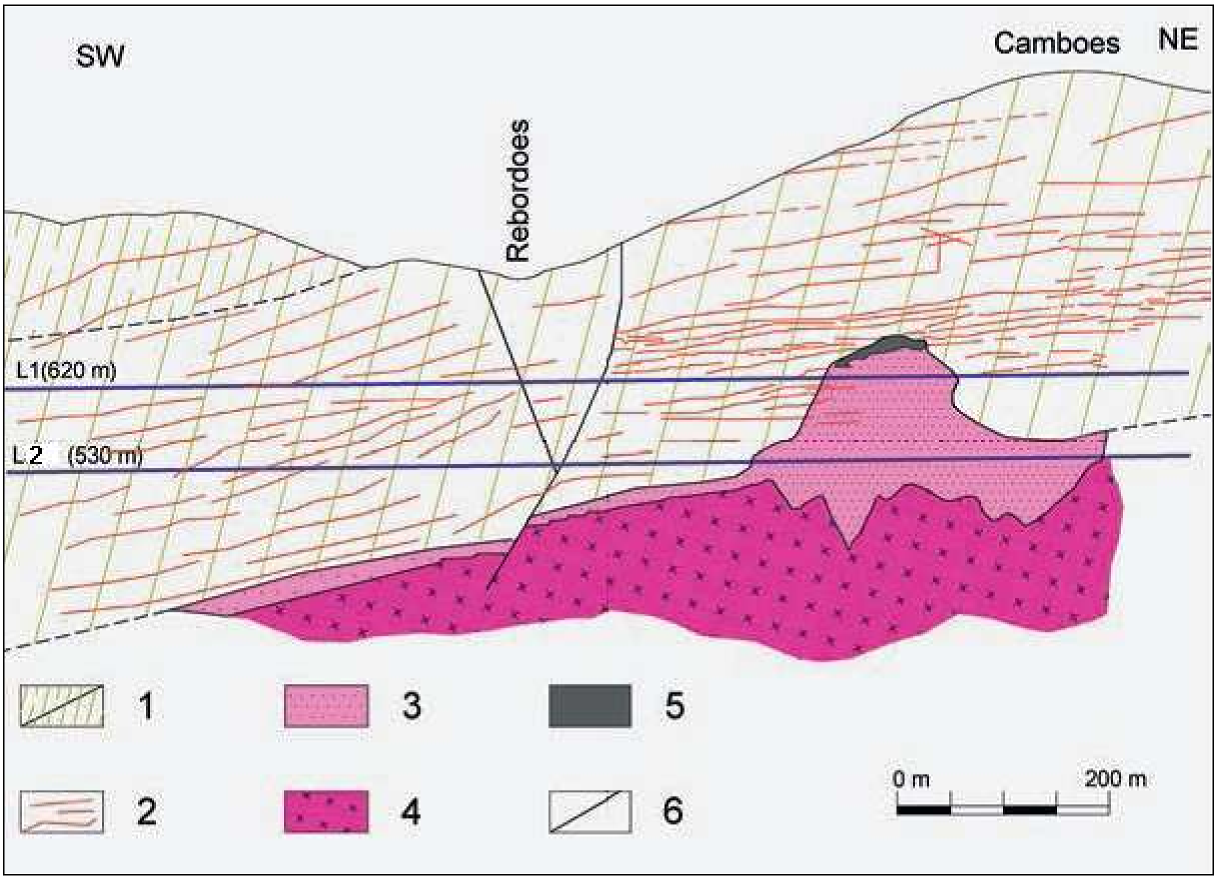
Diabase dykes are abundant in the region, but rarely intersected in underground workings. The dykes had no influence on the tungsten mineralisation and are cut by the tungsten vein system. The dykes may occupy pre-existing faults, are sub-vertical to vertical, and vary in thickness from approximately 0.5-3.0m and reportedly can be traced for more than 1km strike at surface. The rock is fine grained and micro-porphyritic and mainly consists of labradorite and amphibolitised pyroxene.

7.4 Mine Geology

The Panasqueira tungsten deposit comprises a series of thin, flat-lying quartz-wolframite veins developed in the joints of a phyllite within the contact zone of a concealed granite intrusion (Figure 7.3). The largest concentration of veins occurs immediately over a granite-greisen high (cupola) on the No 2 Level of Panasqueira Deep. The horizontal extent of the veins, which pitch to the southeast, appears to decrease with depth (Figure 7.4). Individual veins average about 0.30m in thickness and can persist over hundreds of metres. Frequently, when one vein thins out, another forms close to it, with a small over-lap (Figure 7.5). There is considerable dilution during mining because the veins are so thin, which is why the mine goes to great lengths to not exceed the 2.2 m stope height.

Figure 7-3. Schematic Section of Granite at Panasqueira

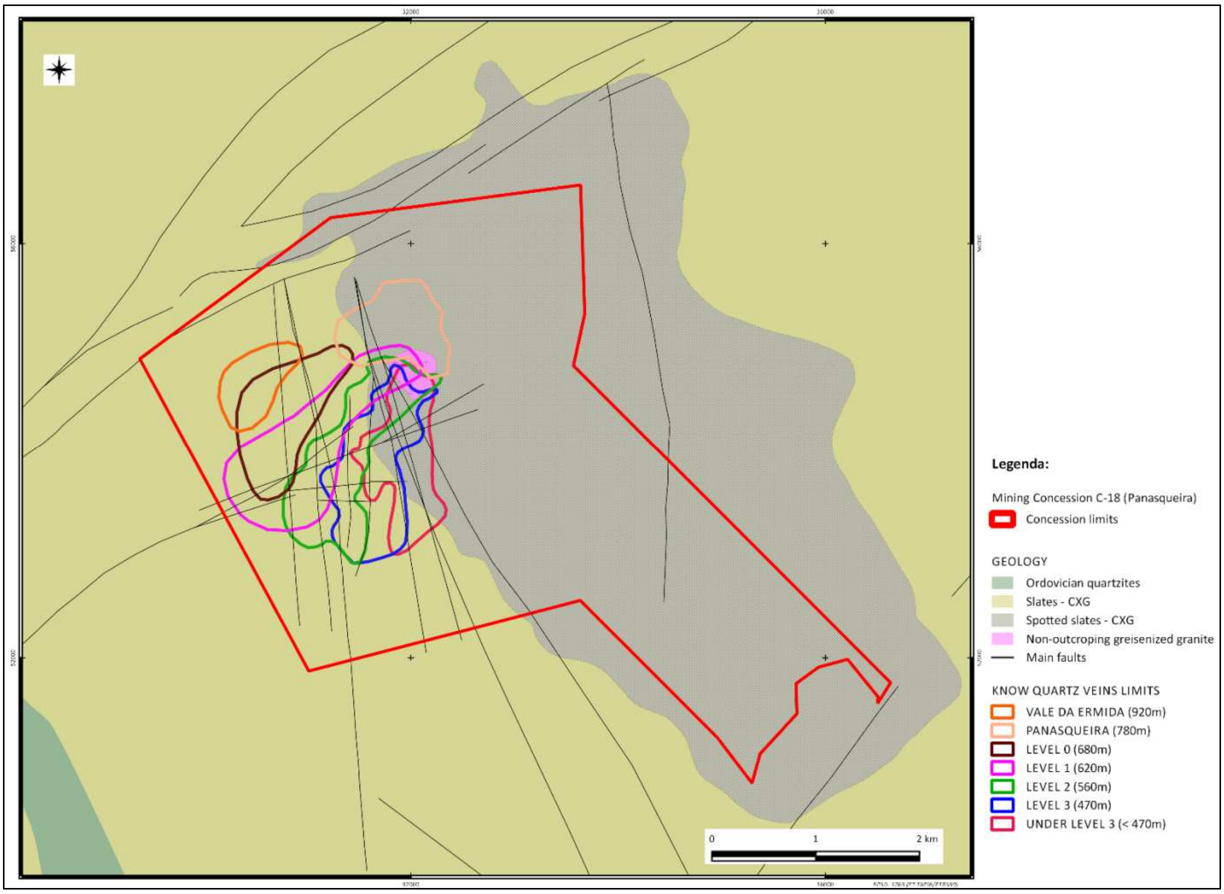
Non-outcropping Greisenised Two-Mica Granite of Panasqueira: (1). Schists Spotted Schists; (2) Quartz Veins; (3) Greisen; (4) Two-mica granite; (5) Silica Cap; and (6) Main Fault



The ore minerals comprise wolframite, cassiterite and chalcopyrite, which are associated with arsenopyrite and pyrite. The mineralisation is generally coarse-grained and very erratically distributed in the quartz veins. This has necessitated the adoption of a resource estimation procedure specific to the deposit. Only in the Panasqueira Deep area, close to the greisen, is a more disseminated "fly-dirt" style of mineralisation developed.

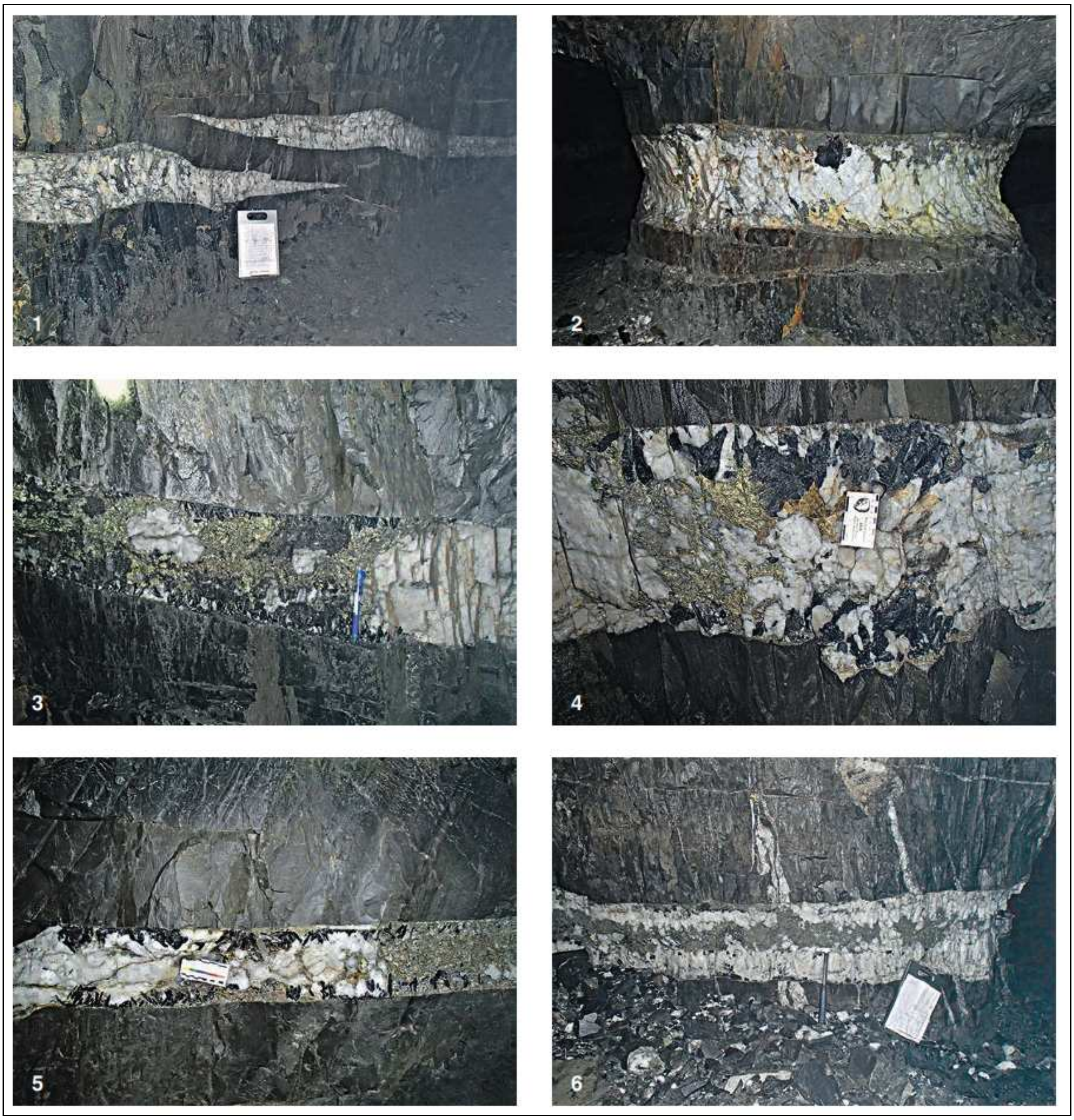
Figure 7-4. Horizontal Projection of Veins

- Successively lower levels illustrating pitch to SE



The mineralisation and joint development in the phyllite is clearly associated with the granite intrusion, which was either the source of the hydrothermal veins or a heat provider to drive a mineralising system. The veins are assumed to have developed as a consequence of successive pulses of mineralisation resulting in the vertical dilation of pre-existing joint sets. The veins are known to penetrate the greisen on Level 1 in the north of the mine, where they have a steeper dip, but they have only been worked in the phyllite. Few drill holes penetrated the greisen, but its potential for hosting mineralisation is considered to be low. Quartz veining in the northern part of the mine is noticeably thicker than elsewhere, while wolframite observed in cores is very weak or absent. It was historically assumed that these thicker, barren veins represent a late-phase pulse of quartz mineralisation, that is untypical of the rest of the mine, with the result that the customary D9 grade estimation formula was not applicable in these areas. However, recent stopes in the North of the mine (near the greisen) and are showing different relationships in some of the intersected veins. This implies that probably in the North, different veins have been fed in different periods by hydrothermal fluids.

Figure 7-5. Examples of Quartz-Mineralised Veins



1. Typical eel-tail morphology of quartz veins.
2. 3m x 3m pillar with wolframite nugget.
3. Highly mineralised vein, with wolframite on contacts, and sulphide rich central zone.
4. Thick vein with coarse wolframite, siderite and sulphides.
5. Crystals of wolframite, perpendicular to the walls.
6. Sulphide rich vein in the central zone.

7.5 Mine Structural Geology

The Panasqueira area contains two main fault systems; a series of N to NW and S to SE trending, near vertical clay filled faults and a carbonate cemented ENE-WSW set. These faults are believed to have been initiated with strike-slip movements during the Hercynian Orogeny. The tungsten mineralisation does not migrate into these faults which are exposed in the mine workings and commonly displace the mineralised veins for short distances (metres). Both sets of faults are post mineralisation and have no influence on grade.

The most dominant and important structural feature at the Panasqueira mine is a flat open joint system prevalent throughout the mine workings. The flat open joint system occurs over the whole of the area surrounding the mine, but only in the vicinity of the Panasqueira granite are there tungsten mineralised veins developed. The flat joint system was structurally prepared before the granite intruded the sediments. The granite used the prepared fissure-flat joint system, opened it and penetrated the sediments as sills. A general summary of the paragenesis of the deposit follows (after Breiter):

- Intrusion of a granite melt, building of the granite dome, opening of some joints, and intrusion of the residual granite magma into the joints.
- Supply of fluids from the deeper part of the granite pluton into the roof and then into some of the non-opened joints. The fluids greisenised the interior of the granite dome and some of the granite sills.
- After the opening of the majority of the flat joints, the fluids migrated outside the granite, building the flat ore veins.

There has been considerable debate about the origin of the flat lying open joint system and the unusual orientation of the vertical dilation that caused the emplacement of the tungsten mineralised vein system within the joints. The joints may have been produced as sub-horizontal fractures resulting from the vertical pressure release caused by slight sagging of the underlying granite mass when it contracted during cooling or by tectonic unloading or possibly by hydrothermal dilation. Further research will be required to fully understand how the important joint system was developed.

7.6 Mineralisation

The Panasqueira deposit consists of a series of stacked, sub-horizontal, hydrothermal quartz veins intruding the Beira schists (from the Schist-Greywacke Complex – CXG), hosted within a flat, open set of joints. The flat open joint system occurs over the whole of the area surrounding the mine, but only in the vicinity of the Panasqueira granite are there tungsten mineralised veins developed (Figure 7.6 and

Figure 7.7). A second set of non-wolframite bearing quartz veins containing minor chalcopyrite and pyrite also exists at the Panasqueira deposit. These earlier quartz veins are aligned with the vertical foliation and cut by the later tungsten-bearing hydrothermal vein system. These sub-vertical veins are locally called “Seixo Bravo”.

The sub-parallel stacked quartz veins contain principally, wolframite, arsenopyrite, pyrite, chalcopyrite and cassiterite. The tungsten mineralised quartz veins have an average dip of 8°-10°SW. The mineralised zone has dimensions of approximately 2,500m in length and varies in width from 400m to 2,200m, continuing to 500m in depth. Previous mining was done in the upper levels and the existing reserves are from Level 0 and Level 4. The current reserves lie above Level 2 and Level 3 (90m below Level 2). The deposit, between Level 1 and Level 2 (60m vertical separation) consisted of seven or eight flat dipping veins with an average thickness of 0.3m (range 0.1-1.0m) that typically hosted the economic mineralisation over continuous strike lengths of 40 to 60m. These mineralised quartz veins located throughout all mine levels, typically pinch out and later reoccur. It is also common to have one vein pinch out while another vein is beginning, creating a short overlap of two veins in the same mining stope.

Wolframite mineralisation occurs as very large crystals or large crystal aggregates, usually concentrated towards the margins of the quartz veins or, occasionally, close to the central portion of the quartz veins. The quartz veins commonly contain open spaces and vugs (Figure 7.7) that are commonly filled with spectacular crystal growth. The mode of occurrence of wolframite crystals (Level 2) includes:

- Pods (5-20cm);
- Large scattered crystals (10-40cm);
- Small scattered crystals;
- Mixed (comprising a mixture of the three previous styles);
- Selvedge wolframite (3-10cm bladed crystals);
- Fibrous wolframite (1cm wide and 5-10cm long);
- Barren (no or little wolframite – fine crystal selvedges or isolated large (5-10cm) crystals).

The Panasqueira Mine is renowned throughout the world for the extraordinary size and quality of the minerals wolframite, apatite, arsenopyrite, cassiterite and quartz crystals that occur in cavities in the quartz veins. Wolframite crystals of this size are reportedly rare in other tin-tungsten occurrences; at Panasqueira the large crystal size of the wolframite and other minerals is directly attributable to the extremely slow rates of crystal growth and slow velocities of the mineralising fluids in the vein system. The mineralisation is commonly accompanied by intense biotite alteration.

Figure 7-6. Quartz Veins on Stope Wall

Flat-lying White Quartz Veins along the Wall of a Stope with Abundant Black Crystalline Wolframite. The vein is approximately 1m at the thickest point In the photo.

Figure 7-7. Wolframite Crystals in a Void

The paragenesis of Panasqueira mineralised veins is complex; nevertheless four stages of mineral formation are generally accepted as the classic development by most of the authors who have studied this deposit (Table 7.1).

Table 7-1. Panasqueira Mineral Paragenesis

Stage	Event	Minerals
1	Oxide-silicate	Quartz, wolframite, cassiterite
2	Main sulphide	Pyrite, arsenopyrite, pyrrhotite, sphalerite, chalcopyrite
3	Pyrrhotite alteration	Marcassite, siderite, galena, Pb-Bi-Ag sulphosalts
4	Late carbonate	Dolomite, calcite

Sixty five minerals (listed alphabetically not by abundance) have been identified in the Panasqueira deposit (after Correa de Sá and Naique, 1998):

ARSENATES:	arseniosiderite, scorodite, pharmacosiderite
ARSENIDE :	lollingite
CARBONATES:	ankerite, calcite, dolomite, siderite
HALOIDS:	fluorite
NATIVE ELEMENTS:	antimony, bismuth, gold, silver
OXIDES:	cassiterite, goethite, hematite, magnetite, rutile
PHOSPHATES:	althausite, amblygonite, apatite, isokite, panasqueirite, thadeuite, vivianite, wagnerite, wolfeite
SILICATES:	beryl, bertrandite, biotite, chlorite, quartz, muscovite, topaz, tourmaline
SULPHATES :	gypsum
SULPHIDES and SULPHOSALTS:	acanthite, arsenopyrite, bismuthinite, chalcocite, chalcopyrite, canfieldite, covellite, cubanite, freibergite, galena, gudmundite, mackinawite, marcassite, matildite, molybdenite, pavonite, pentlandite, pyrite, pyrrhotite, pyrargyrite, sphalerite, stannite, stephanite, stibnite, tetrahedrite
WOLFRAMATES:	hydrotungstite, scheelite, tungstite, wolframite

The mineral panasqueirite and thadeuite only occur at the Panasqueira mine. Their names comes from the name of the mine and from the name of a recognised professor (Décio Thadeu) who worked there and found it for the first time. The mineralisation has been dated in several studies, suggesting that the tungsten mineralising events took place over a four million year time period.

8 DEPOSIT TYPES

The Panasqueira tungsten-tin deposit is reported to be the largest quartz vein deposit in Europe (2006). The major tungsten deposits of the world are classified into seven general groups, vein/stockwork, skarn, porphyry, strata-bound, disseminated, placer and brine/evaporite (Werner, Sinclair and Amey, 1998).

The vein/stockwork deposits account for approximately 50% of the world tungsten production. The deposits are not necessarily large; the following table summarises the 12 largest tungsten deposits of the world. The largest vein/stockwork deposit in the world (the recently opened Drakelands Mine at Hemerdon UK) is number 4 on the list. The vein deposits are generally not large, most contain a few hundred thousand tonnes of contained tungsten, and deposits rarely exceed one million tonnes in size , as shown in Table 8.1 (2006).

Table 8-1. World Tungsten Deposits

Deposit Name (Location)	Country	Deposit Type	Contained Tungsten (t)
Verkhne-Kayrakty (Dzhezkazgan Oblast)	CIS	Stockwork	872,000
Mactung (Yukon & Northwest Territories)	Canada	Skarn	617,000
Shizhuyuan (Hunan)	China	Skarn/stockwork	502,000
Hemerdon (Devon)	UK	Stockwork/sheeted vein	309,000
Tynyauz (former Kabardin-Balkar ASSR)	CIS	Skarn/stockwork	244,000
Norther Dancer Project (Yukon Territory)	Canada	Porphyry	168,000
Yangchuling (Jiangxi)	China	Porphyry	160,000
Xingluokeng (Fujian)	China	Porphyry	144,000
O’Callaghan’s (W.Australia)	Australia	Skarn	135,000
Damingshan (Guangxi)	China	Stratabound/stratiform	116,000
Vostok-2 (Primorskye)	CIS	Skarn	102,000
Ta’ergou (Gansu)	China	Vein/skarn	100,000

The classic tungsten vein deposits typically consist of tungsten bearing quartz veins that occur near granitic intrusions with the larger deposits containing hundreds of parallel to sub-parallel mineable veins. The principal tungsten-bearing mineral is wolframite, whilst other common minerals recovered from these deposits include tin, copper, molybdenum and bismuth.

Panasqueira is a typical vein type deposit but would be considered one of the largest economic vein type deposits in the world since it has been in production for over 100 years. The mine has produced over 128,000t of WO_3 from nearly 40Mt of rock over the period 1934 - 2016, (refer to Table 6-2 above).

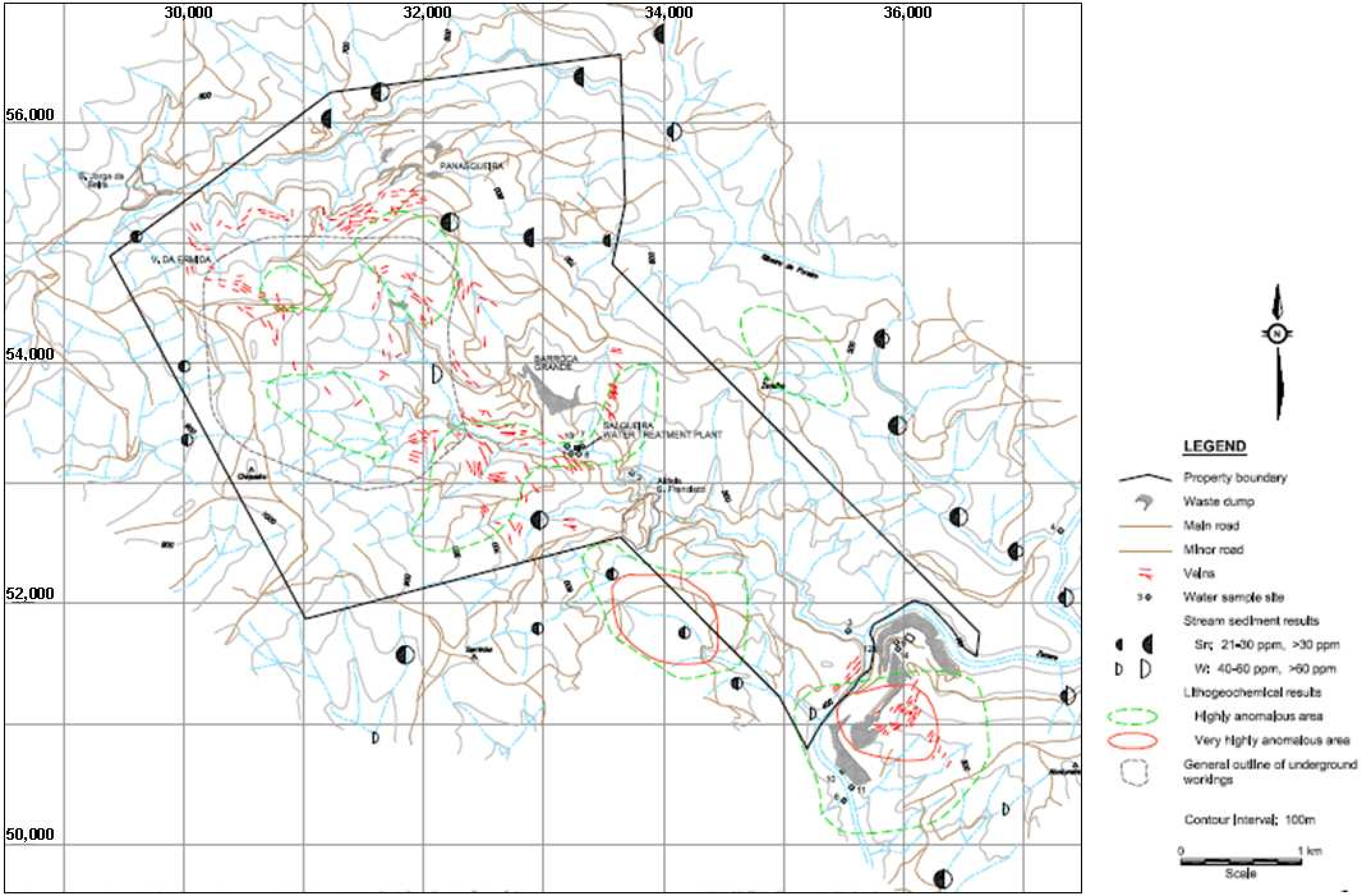
9 EXPLORATION

Even though the mine has been in operation for more than 100 years, very little in the way of “true” exploration has been conducted on the property outside the active or past mine workings. Exploration for additional reserves in advance of production continues as the normal course of the mine activities.

BTW has historically allowed individuals to work small surface veins in the concession area, an activity that involved approximately 4,800 people during the peak development period created by the demand for tungsten generated by WWII. These small time miners working at their own expense recovered small quantities of ore that were sold back to BTW. The hills surrounding the present day operation contain many old pits and shafts left from these small operations. In 1946, BTW completed a detailed review of the veins but the information gained has not apparently been made available. Neither BTW, nor any of its predecessor companies completed a modern, detailed study, other than that produced in 1946, of the veins, to determine if any potential existed for future mine development.

True exploration of the concession and the immediately surrounding area seems to have been fairly limited. Partial results from a regional stream sediment geochemical survey carried out between 1982-1984 by BTW under contract with the Portuguese government have been located (Figure 9.1). During the first phase of geochemical sampling, an area of 650km^2 was covered; sample density was approximately one sediment sample per km^2. The Panasqueira mine is located in approximately the west central portion of the 650km^2 sampling area. A total of 622 stream sediment samples, each sample comprising 2-3kg of sediment plus an additional 100g of fine sediment, were collected and analysed for Cu, Pb, Zn, W, and Sn. Previous investigators were only able to locate poor quality photocopies of maps with dots indicating relative anomalies; exact numerical results were not available.

Figure 9-1. Panasqueira Regional Geochemical Map



Several anomalous areas of elevated tungsten and tin geochemistry were identified outside the concession, although no data identifying the other elements analysed are available. Some anomalous areas can be attributed to known veins worked on by the small miners in the 1940's but others lie in areas that do not appear to have been thoroughly examined using any modern exploration techniques.

A number of the streams, located less than 5km south of the Panasqueira concession returned tungsten stream sediment values of greater than 60ppm tungsten and greater than 30ppm tin. Other streams located less than 1km east of the concession also returned similar values. BTW consultants determined that these values are as anomalous as values collected from streams draining the mining areas at Panasqueira. Lower values, but still considered moderately anomalous are also scattered around the concession, including in the two areas containing the more anomalous values. These moderately anomalous values range from 40-60ppm tungsten and from 21-30ppm tin. No apparent follow-up has been completed on any of the anomalous stream geochemical samples. Figure 9.1 summarises the known location of the anomalous and moderately anomalous stream sediment samples.

BTW also completed some sort of litho-geochemical survey over selected areas of the Panasqueira concession and in the immediately adjacent area. Data relating to the distribution, sample size, density of the samples in any particular area or actual analytical values are not available. The only reference to the survey is contained in a report by Breiter (Nov 2001). Breiter included a map in his report which outlined area of anomalous litho-geochemistry with anomalous areas divided into high and very high anomaly areas.

Figure 9.1 also shows the high anomalous litho-geochemical areas, although specific analytical values are not available. Some anomalous areas within the concession can be attributed to known veins worked on by the small miners in the 1940's but the other areas are located in territory that do not appear to have been thoroughly examined by any modern exploration techniques.

Two areas appear to stand out as potential sites for follow-up work as defined by both anomalous litho-geochemical sample results and anomalous stream sediment results. One area lies along and slightly outside the concession south of the village of Aldeia de. São Francisco. The second area of combined litho-geochemical and stream sediment anomalies lies just east of the concession boundary east of topographic control point of Zerelho. Figure 9-1 also shows the high anomalous lithogeochemical and anomalous stream sediment areas. It is not known if any follow-up work was completed in these two areas.

10 DRILLING

Several BQ sized boreholes were drilled outside the main area of known veining, either outside or very close to the edge of the concession boundary as part of the regional exploration completed by BTW from 1982-1984. Borehole 6-G(L), 506.52m in depth, sited on the western border intersected a total of 54 9cm wide quartz veinlets and one 29cm veinlet that contained visible wolframite. The hole is located close to one of the tungsten stream sediment anomalies, noted above, and may indicate that the source of the stream anomaly may comprise the mineralised quartz veins seen in the drill hole. Most of the other drill holes from this period also intersected quartz veinlets, generally less than 30cm in width.

A combination of a historic core storage fire and the practice of dumping the core to reuse core boxes has left the operation with a small collection of core available for review. In November 2005 through to March 2006, BTW completed an additional eight diamond drill holes (2,313m) from surface (Table 10.1). The drilling was completed using a drill contractor and was successful in controlling the hole deviation common during the previous holes drilled previously from 1946-2005.

BTW, through the past experience of geologists at the mine, considers quartz vein exceeding 18cm in width to be significant and targets future underground development based on those intercepts. In the 2005-2006 campaign, a total of 80 veins were intercepted, 15 of which exceeded 18cm in thickness.

Table 10-1. Summary of BTW Diamond Drilling – 2005 - 2006

Hole	Depth of Hole (m)	Number of Quartz Veins Intercepted	Number of Veins > 18cm Thick
A	332	10	2
B	372	19	4
C	328	14	1
D	314	4	2
E	277	18	4
F	292	6	0
G	326	9	2
H	72	0	Fault
Total	2,313	80	15

74 diamond drill holes were drilled from surface during the period 1946 to Nov 2005. These holes commonly flatten up to 60^{o} over a 200-300m deep borehole and are therefore not considered very useful for vein location. Figure 7.2 shows the drill collars for all the surface drill holes completed by BTW, including those drilled outside the main area of mining.

BTW owns its own diamond drill equipment, but does not own or use modern downhole drill surveying instrumentation nor any wedging equipment. Instead, mine geologists have relied on underground drilling for vein location, which has provided much shorter vertical holes. Up to the end of 2005, underground drilling had encompassed 2,567,mostly 46mm core sized, diamond drill holes for a total of approximately 131,000m. These holes contained more than 20,000 quartz vein intercepts which were used to guide the underground development.

In 1995, Avocet retained the services of two in-house consultants who attempted to enter all the drill hole data from both the surface holes and underground holes into a Datamine software programme. In 2006 Primary re-entered all the historic drill hole data located in a collection of manually-entered notebooks into a new digital database. This work has been carried on, and currently the drillhole database contains data from 3,780 diamond drillholes, over a total drilled length of approximately 155 km. Since 2006, core has been stored in a modern warehouse next to the main offices.

The current diamond drilling rigs at the mine, all for underground drilling, are:

- Diamec 242, up to 120m holes
- Hilti DD750, up to 13m holes
- Boyles, up to 13m holes

A summary of the drill completed since 2006 is shown in Table 10-2.

In 2010 a geophysical campaign was completed over part of the permit, using Magneto telluric and TEM methods. The first method supported a Master's Thesis in the Lisbon University and some cored drill holes (5+3) were completed to confirm the anomalies without significant results, except the attempt of Greisen with disseminated sulphides at North of the mine, even to the North of the East-West fault that closes the known mine at North (Cebola fault).

The post-2000 drillhole data is shown in Appendices B (collars) and C (intersection data). 96% of these drillholes are vertical, and so the intersections with the approximately horizontal mineralised structures gives true thicknesses.

Table 10-2. Drilling Summary, 2006-2016

YEAR	UNDERGROUND		SURFACE	
	Machine	(m)	Machine	(m)
2006		2,147		1,383
2007		75		-
2008		1,644		-
2009		-		-
2010	Short (Hilti)	30		-
2011	Short (Hilti)	1,100		-
	Long (Contracted company)	302		-
2012	Short (Hilti)	775	LongYear DB1200 + DIAMEC 282+ Sandvik ONRAM1500	3,093
	Long (Diamec 252)	193		
	Long (Contracted company)	2,310		
2013	Short (Hilti)	911		-
	Long (Diamec 252)	1,022		-
2014	Short (Hilti)	496	Christensen CS14	1,921
	Long (Diamec 252)	1,241		
2015 (Up to Sept)	Short (Hilti)	515		-
	Long (Diamec 252)	1,241		-
Sept 2015 to Sept 2016	Short (Hilti)	567		-
	Long (Diamec 252)	1,096		-
Total		**15,665**		**6,397**

10.1 Lower Level Exploration

The resources are distributed between Level 0 and Level 4 but there are only main development down as far as Level 3. Part of the veins in the upper levels were mined in ancient times.

The location of the veins was determined by vertical diamond drill holes from both surface and underground. Long diamond drill holes drilled from surface tended to deviate making vein intercept locations difficult to accurately determine. The 1.8m diameter raises bored between levels for use as ore passes were also mapped for the vein elevation and thickness. Underground holes required access that was not always available for the plunging sections of the orebody. For these reasons along with the large lateral extent, the orebody was not well defined by drilling.

Predictions of grade and vein variability were not possible using simple geological mapping techniques a consequence of the unpredictable distribution of the wolframite and the resultant “nugget” effect distribution and the variation in the nature of the enclosing quartz veins. Drill holes were inadequate for the accurate determination of the grade of the vein and were not assayed; however vein thickness and elevation were recorded on all maps and the vein intercepts provided some assistance in production decisions.

In some cases as a consequence of pinching and swelling of the veins individual holes did not provide accurate information on location of veins; however, in most cases, within the boundaries of the mineralisation, a thick vein intercept would become part of a future stope.

Underground drilling below level 2 and 3 has allowed progressive increases in the amount of resources evaluated for level 3 and level 4.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The distribution of the wolframite crystals is extremely erratic and results in a “nugget effect” similar to what is commonly seen in high-grade, coarsely crystalline, gold mines. The mine does not utilize conventional sampling techniques such as channel sampling or drill core assaying to provide material to estimate grades. Instead, empirically derived factors are used, within formulae that have proved reliable and accurate for many decades of mining. In the author’s opinion, the sample preparation, security and analytical procedures are adequate for the resource and reserve estimations presented in this report.

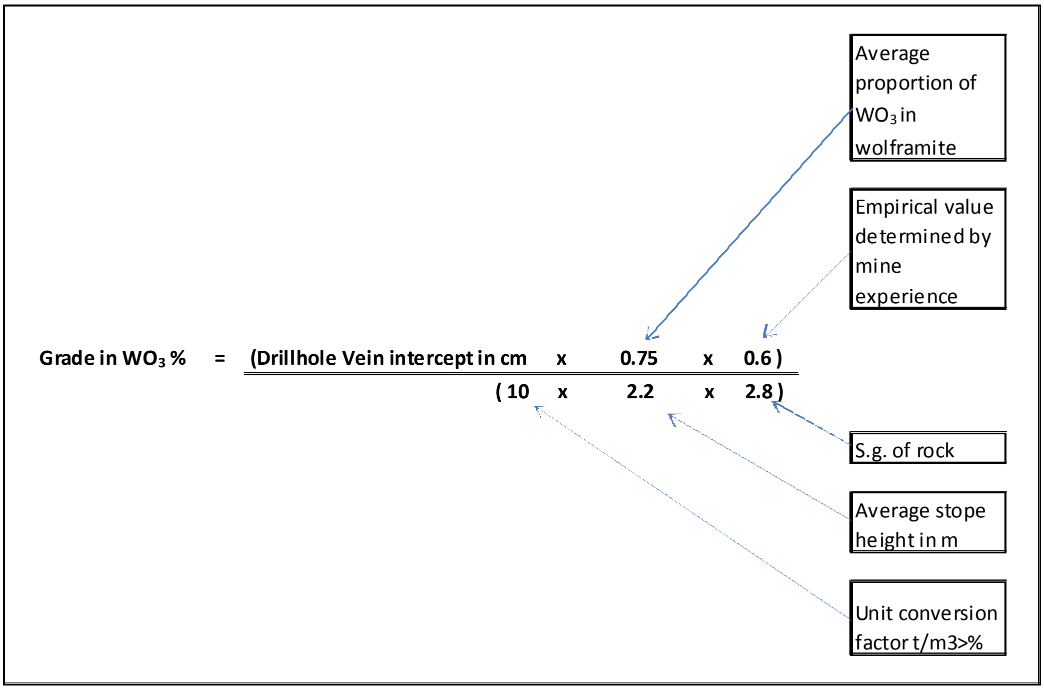
11.1 Diamond Drilling

Underground holes produce 47.6mm (NQ) core. An example underground diamond drilling is shown in Figure 11-3. Diamond drillhole core is left intact, and is logged by a geologist to record the following data in separate tables:

- **Mineralogy.** For the quartz intersections, a qualitative index (1-6) is recorded for up to 24 different minerals, which are summarised in Table 11-1. The quartz vein intersection length is also recorded. An example core quartz intersection is shown in Figure 11-4.
- **Lithology.** Lithology codes were recorded, as summarised in Table 11-2.

An internally developed empirical (D9) formula is also used to convert the measured quartz thickness into a %WO_3 grade figure. This formula is depicted in Figure 11-1.

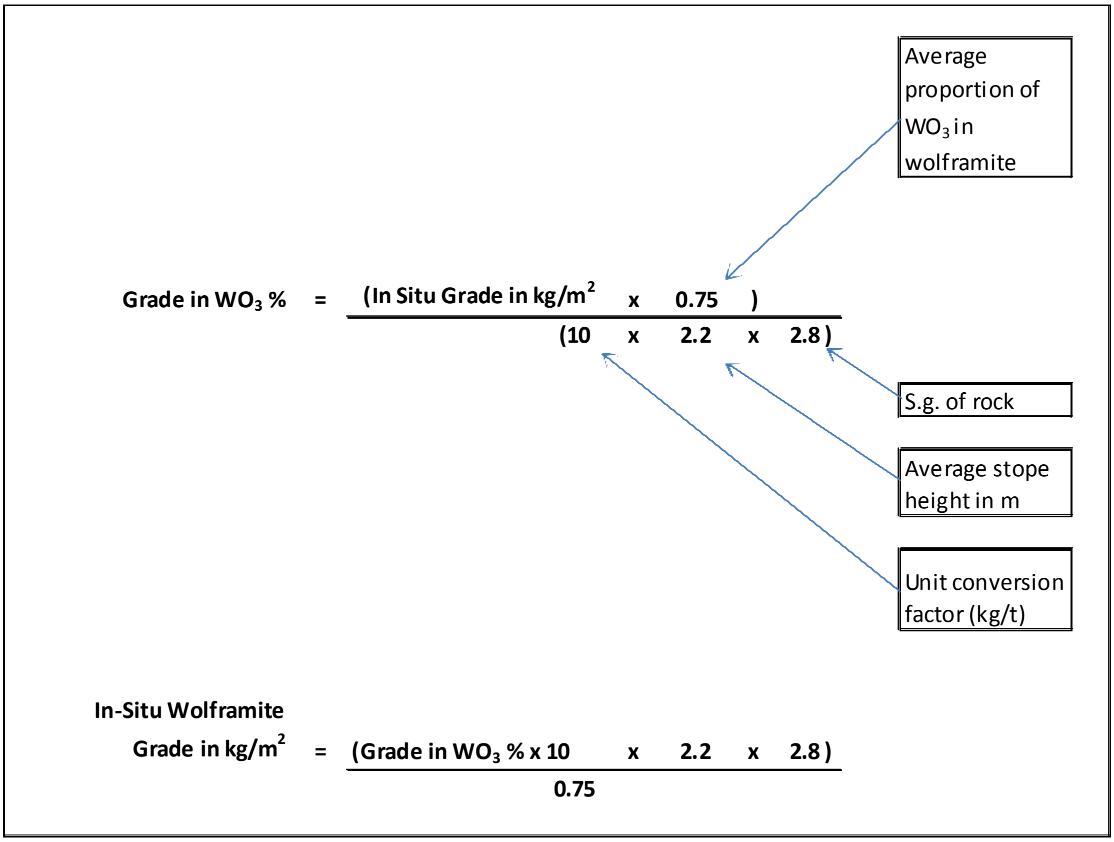
Figure 11-1. D9 Formula



This D9 formula is basically a relationship between the frequency and thickness of quartz vein intersections and the expected grade of wolframite present. Given that mine has been in production for so many decades, the geology department have developed a solid understanding between mined grades and quartz vein occurrence, as reflected in the D9 formula.

There is also a relationship between the in-situ grade of wolframite in the vein (expressed in kg/m^2) and the overall mineable grade in WO%, over an effective stope width of 2.2 m, as shown in Figure 11-2.

Figure 11-2. WO_3 – kg/m^2 Relationship



This resource estimation involves blocking out plan areas around drillhole quartz intersections, greater than 18cm thickness. Using these formulae, this 18 cm limit is equivalent to a resource cut-off of 10.8kg/m^2 or 0.13% WO_3. As the resource estimate uses the quartz intersection data, rather than laboratory assays, drillhole samples are not prepared as is normally the case for metal assays.

Planned diamond drilling for 2017 is shown in Figure 11-5. All core logging and storage is carried out in secure facilities, next to the mine offices. These facilities are locked and only geological personnel have access.

Table 11-1. Mineralogy Codes

Code	Mineral	Code	Mineral
Qz	Quartz	**Fl**	Fluorite
Wolf	Wolframite	**Mrc**	Marcassite
Cst	Cassiterite	**Tur**	Tourmaline
Ccp	Chalcopyrite	**Cal**	Calcite
Py	Pyrite	**Bn**	Bornite
Apy	Arsenopyrite	**Tpz**	Topaz
Sp	Sphalerite	**Po**	Pyrrhotite
Gn	Galena	**Chl**	Chlorite
Sd	Siderite	**Bt**	Biotite
Cb	Carbonates	**Dol**	Dolomite
Ms	Muscovite	**Pan**	Panasqueirite
Ap	Apatite	**Stn**	Stannite

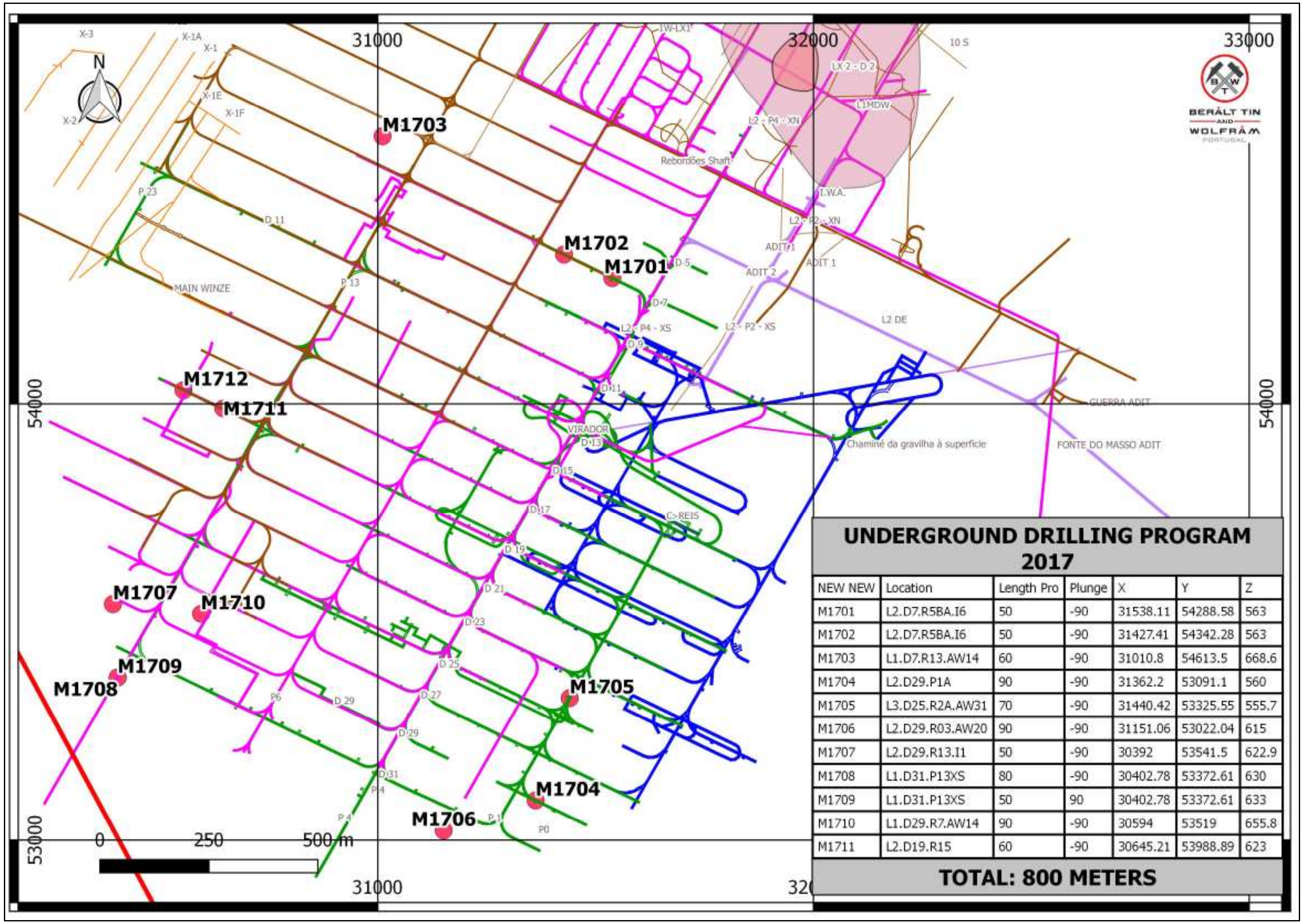
Table 11-2. Lithological Codes

Code	Description
XA	Argillaceous schist
XG	Silicified schist
XAM	Argillaceous schist - spotted
XGM	Silicified schist - spotted
XAG	Argillaceous/silicifed schist
F	Vein
SB	Seixo Bravo - local barren quartz system
FA	Fault - with argillaceous filling
FC(B)	Fault - with carbonate filling (or as breccia)
GS	Greisen
GR	Granite
DOL	Dolerite
AP	Aplite
APG	Aplite pegmatite

Figure 11-3. Underground Diamond Drilling

Figure 11-4. Example of Diamond Drillhole Quartz Intersection

Figure 11-5. Planned Diamond Drilling – 2017



UNDERGROUND DRILLING PROGRAM 2017

NEW NEW	Location	Length Pro	Plunge	X	Y	Z
M1701	L2.D7.R5BA.I6	50	-90	31538.11	54288.58	563
M1702	L2.D7.R5BA.I6	50	-90	31427.41	54342.28	563
M1703	L1.D7.R13.AW14	60	-90	31010.8	54613.5	668.6
M1704	L2.D29.P1A	90	-90	31362.2	53091.1	560
M1705	L3.D25.R2A.AW31	70	-90	31440.42	53325.55	555.7
M1706	L2.D29.R03.AW20	90	-90	31151.06	53022.04	615
M1707	L2.D29.R13.I1	50	-90	30392	53541.5	622.9
M1708	L1.D31.P13XS	80	-90	30402.78	53372.61	630
M1709	L1.D31.P13XS	50	90	30402.78	53372.61	633
M1710	L1.D29.R7.AW14	90	-90	30594	53519	655.8
M1711	L2.D19.R15	60	-90	30645.21	53988.89	623

TOTAL: 800 METERS

11.2 Face Sampling

Face sampling involves measuring the area of wolframite crystals within exposed veins on the faces or walls in stoping or development areas. The areas of wolframite are accumulated for a specific length of exposed vein and recorded along with the length sampled and the average vein thickness, as shown in Figure 11-6.

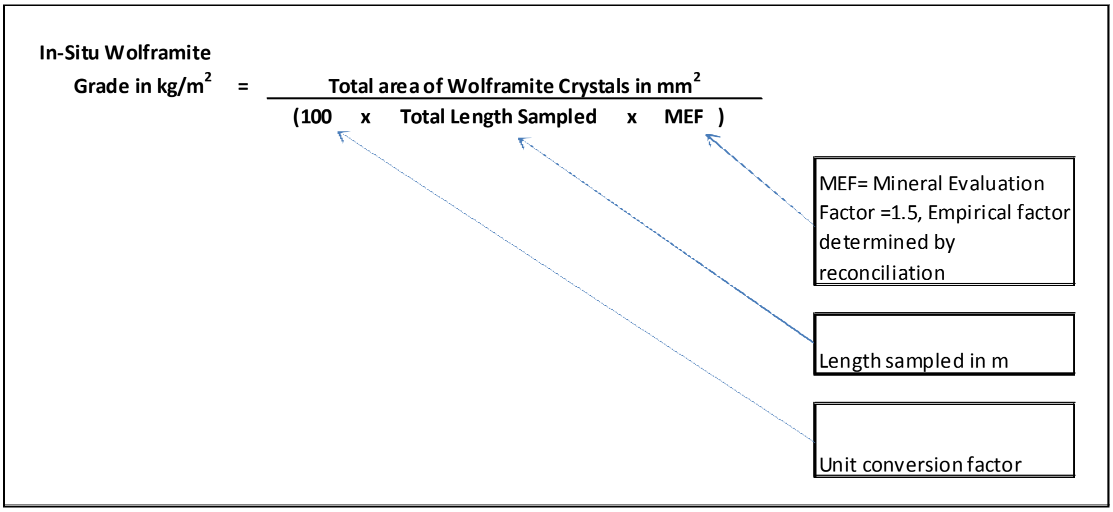
Figure 11-6. Face Measurements of Wolframite Crystals' Area

For each vein length measured in this way, the total area of wolframite crystals is used with the empirical Pintas formula, to estimate the recoverable wolframite per horizontal square metre of vein, as shown in Figure 11-7. For each face sample, therefore, a calculated kg/m^2 value can be estimated. These values can then also be converted into %WO_3 values, effective over the stope thickness, using the formula shown in Figure 11-2.

This measuring system has been used successfully for decades and provides a reliable method for calculating grade. The frequency of taking these samples is mainly limited by the number of samples that can be taken per shift by the (currently) two-man sampler team. There are many production faces over an extensive mine layout, so there is a limit as to how many faces may be reached per day.

Underground channel samples are not taken for assaying, although the results obtained from the use of face sample measurements and the Pintas formula are analogous to the type of results typically obtained from channel samples.

Figure 11-7. Pintas Formula



For face samples collected since April 2016, an updated MEF factor of 1.0 has been applied, which is based mine-mill reconciliation results in recent years.

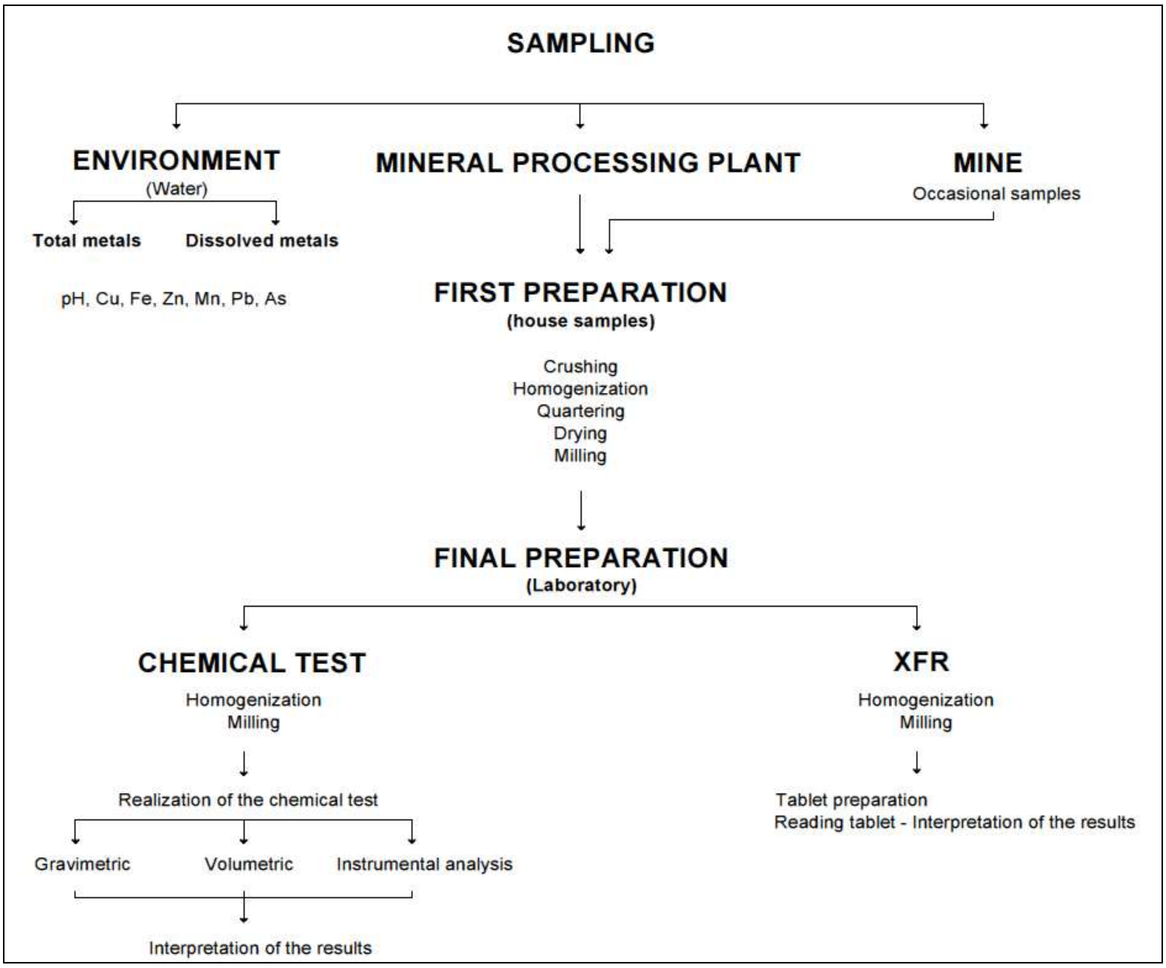
11.3 Quality Control

With respect to diamond drillhole or underground sample measurements, there are no quality controls measures currently in place, associated with diamond drilling and underground sampling. Quality control measures are employed in the laboratory, for mill sample measurements, involving blanks, certified samples and external samples.

11.4 On-Site Laboratory

The mine has its own on-site sample preparation facility, and its own on-site laboratory, for assaying of samples taken at various points in the mill circuit, as well as final concentrate measurement. A general flowsheet for the lab procedures is shown in Figure 11-8.

Figure 11-8. Flowsheet of Laboratory Procedures



11.4.1 Initial Sample Preparation

Samples are collected hourly during the various shifts of operation in the mill. The collected sample passes through several stages of preparation: any fragments greater than 5 mm are broken down in the jaw crusher, and later in a rolling mill, so that the sample has better homogenization. The final sample selection is done by quartering on a rubber mat, which consists in forming a circular wafer, which is then divided into four parts. Then two opposite parts are removed in relation to any diagonal axis, and the other two parts are used to be re-homogenized and quartered until a desirable final sample is obtained.

Drying follows, in a calibrated drier with moderate temperatures, so as not to modify some physical and chemical characteristics of some minerals. Finally, the dried sample may or may not have a further reduction of quantity, using the Jones splitter or quartered again depending on the volume and type of sample. Final grinding in a mortar mill reduces the product to less than 75 microns.

In addition to the daily samples, screening is also done to obtain the granulometric characterization of certain flows within the Plant, in order to check and obtain a greater efficiency of all concentration processes.

Wolframite and Cassiterite samples are collected in an automatic sampler during final bagging. They are prepared and homogenized in a rotary splitter, which consists of a cylinder with a conical termination, leading to an opening. At the opening, the sample falls into a rotating disk containing 4 cups, each representing a quarter fraction. Grinding of the sample follows, and then sent to the laboratory for analysis.

A summary of the equipment used in sample preparation is shown below:

- Jaw crusher
- Rolling mill
- Mortar Grinder
- Vibrating screens
- Screens
- Jones divider
- Rotating separator
- Scales
- Rubber mat
- Spatulas / Brushes
- Probes / samplers
- Trays / containers
- Drying oven

11.4.2 XRF Analysis

A pressed powder tablet is prepared, comprising a portion of the sample powder and a wax binder and pressed to 15 Tons / cm^3. XRF is used in routine analysis to control the entire process in the Plant, allowing the analysis of various elements of the same sample quickly and at a lower cost. Among many elements that can be analysed, the most common are: WO_3, Sn, Cu, Zn, As, S, Ag, Si.

For tungsten assaying, the laboratory uses a Panalytical Axios XRF Spectrometer, which utilises Dispersion by wavelength (WDXRF). Regular maintenance and calibration of the equipment is done by PANALYTICAL technicians , checking instrument accuracy in respect to repeatability and reproducibility. Calibration patterns have been established that approximates the composition and physical characteristics of Panasqueira samples.

Samples involved in daily plant monitoring are:

- Medium separation (HMS) feeding
- Feeding of coarse tables
- Feeding of slimes
- Feeding of Final Concentration
- Copper Feeding
- Fines Feeding
- Copper Concentrates
- Wolframite concentrate (Low and High Grade)
- Siderite
- Medium separation tailings
- Coarse table tailings
- Slimes tailings
- Final concentration tailings
- Copper Tailings
- Concentrate from medium separation
- Duplex concentration
- Cassiterite concentrate
- Punctual Mine Samples

11.4.3 Chemical Analyses

For analyses of the end products of Wolframite, Chalcopyrite and Cassiterite concentrates, chemical analyses are used.

The wet process is used for Chemical Analysis. A pulverized sample undergoes a dissolution with acids, and the chemical reactions allow the separation, identification and determination of the relative amounts of sample components. Certified reference materials (CRMs) are used for quality control.

Chemical separation, followed by gravimetric analysis (weighing using an electronic scale XS 205DU METTLER) is used for the analysis of:

- Tungsten trioxide (WO_3)
- Bismuth (Bi)
- Molybdenum (Mo)
- Sulfur (S)
- Silica(Si)
- Aluminum(Al)
- Silver(Ag)
- Titanium(Ti)

Chemical separation, followed by a volumetric analysis of elements in solution using a burette, is applied for a range of elements:

- Tin (Sn)
- Arsenic (As)
- Copper (Cu)
- Phosphorous (P)
- Lead (Pb)
- Calcium(Ca)
- Iron(Fe)
- Manganese(Mn)
- Antimony(Sb)
- Zinc(Zn)

11.4.4 UV-VIS Spectrophotometer

Sometimes it is necessary to quantify very small concentrations of $WO_{3.}$ A fused sample is created with carbonates, in which tungsten is reduced to its pentavalent form, in which it is "complexed" with potassium thiocyanate. This allows measurement of the "absorbance" in spectrophotometer UV Helios - UNICAM. These tests are always accompanied by blanks tests and certified standards to ensure the accuracy of the results.

11.4.5 Atomic Absorption Spectrophotometry

The sample in liquid form is introduced into a flame where it undergoes chemical and thermal reactions, giving rise to a number of "free" atoms capable of absorbing wavelengths characteristic of each element. The measured concentrations may be [ppm] or [ppb]. This method is used for monitoring of the water treatment plant - ETAM - as well as all sewage and wastewater. The pH of the water is determined using the Potentiometric method, thus verifying the treatments applied in order to minimize the environmental impact.

The equipment is a UNICAM-THERMO-ICE 3300. Maintenance and calibration is done periodically by UNICAM technicians. Metals (total and dissolved) that are analysed are: Cu, Fe, Zn, Mn, Pb and As.

12 DATA VERIFICATION

The QP has completed an independent review of the supplied resources, reserves and economic analysis. The work completed for this review included:

- An underground visit to active stoping areas.
- Inspection of core storage and logging facility at the mine site.
- Visits to the mill, sample preparation, on-site laboratory and tailings areas.
- An overall check on the compiled resource and reserve figures for the whole mine.
- Check calculations of the economic cashflow results from mine monthly reports.
- Import and combination of all available drillhole data into Datamine. During the combination an desurveying process, 21 sequencing or overlap errors were found, out of 52,551 records, which represent just 0.04% of the database. Section, plans and 3D views of this drillhole data were checked against mine topographic data, and did not show any major discrepancies with corresponding mine plans.
- Check calculations of tonnage and grades were made for 139 resource blocks on Levels 1, 2 and 3, representing 14% of the measured resources and the reserves. These results were satisfactory. The data for these blocks were contained on 9 different Autocad plans. The blocks included 117 pillar blocks and 22 virgin blocks. Checks were made of evaluated areas and tonnages, block grade and mining recovery calculations.

In the opinion of the QP, the verification results obtained support the resource, reserve and economic results that have been presented in the current study. However, it is recommended that various quality control steps are introduced into certain aspects of the sampling and evaluation procedures, to improve on-going estimation practices.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

The mineral processing facilities have been in their current configuration for several decades. Improvements have been introduced by small changes to the existing plant while it is in production, rather than major expansions with specific testing studies. The current mill flow sheet is explained in Section 17.

14 MINERAL RESOURCE ESTIMATES

14.1 Panasqueira Mine

14.1.1 Methodology

An overview of the resource estimation methodology for the Panasqueira mine resources is given below:

1. **Diamond Drillhole Intersections.** Quartz intersections in drillholes, greater than 18cm thickness are grouped as individual vein structures according to the approximate vertical position between main levels. Plans are prepared of these intersections for each common vein structure.

2. **D9 Formula > kg/m^2 grades**. The D9 formula is applied to convert the quartz intersection thicknesses into equivalent wolframite grades, in terms of kg of wolframite per unit plan area (kg/m^2), for the parent vein structure intersected.

3. **Resource Perimeters.** For each common vein structure, polygons are created around drillhole intersections, taking into account mined-out areas, blocked-out reserves areas and general mine guidelines for extrapolation distances. In this blocking out process, an effective cut-off grade of 10kg/m^2 (0.13%WO_3) is applied, as with the D9 formula this is equivalent to 18cm. Individual intersections with lower grade can be accepted to assist continuity, but not included as ultimate resources. Polygons are extended a distance of 30.5m if the vein intersection thickness between 18cm and 29cm. An extension distance of 50.5m is used if the vein thickness is greater than or equal to 30cm.

4. **Indicated/Inferred Resource Classification.** The perimeters are categorised as pertaining to **indicated** resources if they contain more than one drillhole intersection over 18cm, and for **inferred** resources if they only contain one drillhole intersection of over 18cm.

5. **Resource Factors.** A common mining recovery is applied to all of the resource perimeters, of 84% which takes accounts of the non-recoverable 3m x 3m pillar system that applied throughout the mines. Additional confidence factors are applied, of 60% to indicated resources, and 40% to inferred resources. These factors stem from historical resource and production data.

6. **Face Sampling.** Following development into unmined vein areas, and with level to level ramps and then lateral development onto intersected vein structures, face samples are taken with area mapping of exposed wolframite crystals, as outlined in Section 11.2.

7. **Pintas Formula > kg/m^2.** As described in Section 11.2, the Pintas formula is utilised to convert the mapped crystals areas into a grade, in terms of the kg of wolframite per unit of plan area (m^2).

8. **Plotting of Sample Data.** The face sample data are then added to Autocad plans which are generated and maintained for each identified vein structure within each level.

9. **Blocking Out > Virgin/Pillars.** Based on all the mining and sample data shown on the level/vein Autocad plans, reserves are outlined in the form of polygons. For initial vein areas opened up by drive or panel development, blocks are allocated as 'Virgin' areas. When development has advanced to develop at least 11m x 11m, blocks are allocated as 'Pillar' areas. The blocks are outlined so as to achieve average grades higher than 10kg/m^2. For CIM classification of reserves in this report, 'Pillar' areas have been allocated Proven Reserves and 'Virgin' area have allocated as Probable Reserves.

10. **Block Evaluation > Measured Resources.** Areas, average grades (in kg/ m^2) and vein thicknesses are collared for each defined block, based on those face samples in or adjacent to each block. This collation is done in separate Excel spreadsheet tables, along with other information related to mining activity over the last 6 months: stoped quantities, eliminated quantities and internal transfer from Virgin to Pillar blocks. These data are used with the previous results, to also determine a Not-In-Reserve (NIR) adjustment. The final estimated result is a mineable area (after mining recovery applied), the average grade in kg/ m^2 and average vein thickness for each block.

11. **Resource Summary.** The data for all the blocks within each level are combined. The average grades in kg/m^2 are converted to %WO_3, and the stope tonnages is calculated assuming an average stope height of 2.2m. The grade and area data is also collated by vein structure, and mine region for each level.

12. **Resources Overall.** All of these resources are then collated by level and mine region, to produce an overall summary of indicated and inferred resources. The plan areas of vein structures are converted to a tonnage using an average height of 2.2m and an s.g. of 2.8.

14.1.2 Drillhole Data

The current drillhole database contains data from 3,870 diamond drillholes, over a total drilled length of approximately 157 km. A plan of all drillholes is shown in Figure 14-1. The majority of the data for resource estimation come from underground drillhole data, which are generally either level to level vertical holes, 120m holes drilled down from the deepest available levels, or much shorter 13m holes drilled vertically up and down from current stope workings.

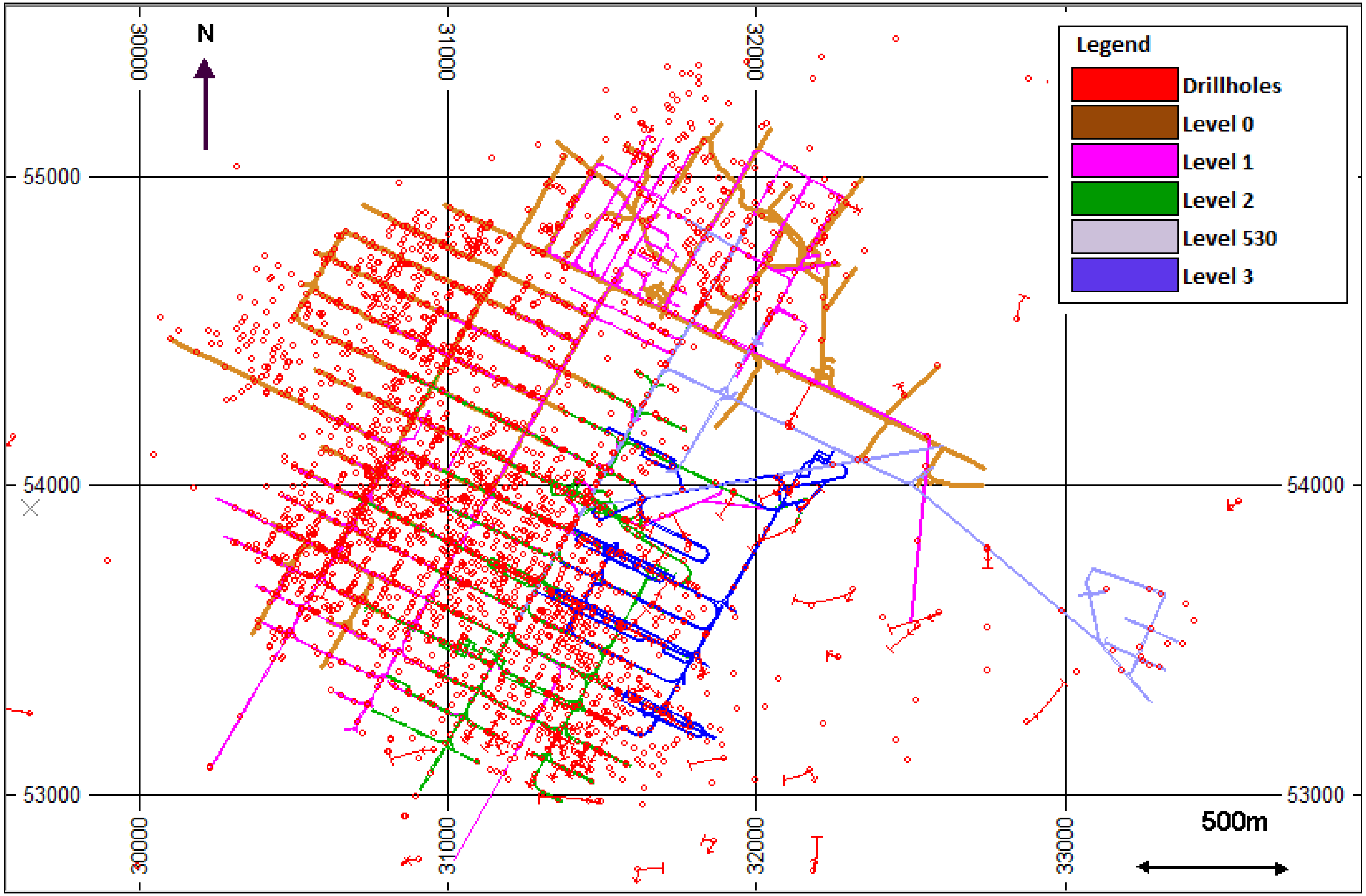
A breakdown of the drillhole data by level, for quartz intersection greater than 18cm, is summarised below in Table 14-1.

Table 14-1. Drillhole Quartz Intersections By Level

LEVEL	Number of Qtz Intersections >18cm	Total Number of Drillholes
0	608	1,056
1	1,526	1,873
2	1,522	1,222
3	646	596
4	76	139

These demonstrate the much more focussed mining activity, and therefore diamond drilling, on levels 1 and 2 in the preceding decades. It also demonstrates that the proportions of quartz vein intersections generally mirror the amount of drilling, rather than some fundamental change in the intensity of quartz vein occurrence with depth.

Figure 14-1. Plan of Drillhole Data



14.1.3 Resource Estimation

At the Panasqueira mine, the evaluation of measured resources is synonymous with reserve estimation. An example of a level/vein Autocad plan is shown in Figure 15-1. The example shown is for level 3, adjacent to drive D25, and for the top-most AW31 vein. This plan displays several important aspects of the measured resource estimation process:

- Plotting of face sample data.

- Definition of Pillar blocks in pillars sized 11m or smaller, and which have face samples on at least three sides of the pillars.

- Definition of Virgin blocks, in areas which have only been intersected by development, so laterally the block outwards from face sampled areas. Such blocks may only have one side with face samples and/ or one set of internal samples.

- Sometimes originally identified veins may split, so that two potentially mineable veins are apparent, as shown by the cross-hatched H- shaped block in Figure 15-1.

A summary of the extrapolation distances used from drillhole quartz intersections is shown in Table 14-2, which are applied to both indicated and inferred resource blocks. The key assumptions associated with this resource estimate are:

- A cut-off grade of 0.12%WO_3 (equivalent to 10 kg/m^2). This cut-off level is supported by breakeven calculations (showing a breakeven cut-off of 0.10%WO_3) based on parameters related to average cost levels that have been estimated for 2017. These parameters include:

Underground mining cost	=	\$12.65/t ore
G & A cost	=	\$5.24/t ore
Processing cost	=	\$4.04/t ore
Mill recovery	=	80.1%
Received metal price	=	\$280/mtu WO_3

- A minimum mining height of 2.2m.

- Applied losses consistent with the mine room-and-pillar mining method, leaving non recoverable 3m x 3m pillars.

These assumptions support the reasonable prospect of these resources' economic extraction.

The level reference system at the mine is summarised in Table 14-3. The blocks are generally laid out in parallel with the mine's drive and panel system, and modified to reflect local mined out areas. This drive and panel system is shown in a plan of level 3 in Figure 14-2, and overall for the mine relative to surface infrastructure in Figure 14-3. This whole system is oriented at approximately 25° to true north. The drives run approximately NW to SE and are sequentially numbered from north to south. The panels run approximately SW to NE, and are numbered sequentially from east to west. East of Panel 0 the panel numbering continues P-1, P-2 etc. The drives and panels are 86° and 94° to each other. An example of resource blocks laid out for one vein structure on level 3 is shown in Figure 14-4.

A summary of the resource classification criteria, and associated parameters, are shown in Table 14-4.

Table 14-2. Resource Extrapolation Distances

Intersection Thickness *cm*	Extrapolation Distance *m*
<30 cm	34.5
>30 cm	50.5

Table 14-3. Level Reference

Level	Elevation mRL
0	680
1	620
2	560
3	470
4	380

An overall summary of the measured and indicated resources evaluated is shown in Table 14-5 with more detailed breakdowns in Table 14-6 and Table 14-7, for indicated and inferred resources, respectively. The levels are broken down by vein structure, which are approximate 10m vertical intervals in between each main level. These intervals are assigned colour codes, are shown from the top down for each of the level intervals Table 14-6 and Table 14-7.

All of the measured resources are converted into reserves, so there are no measured resources external to reserves.

Table 14-4. Resource Classification Criteria and Parameters

Resource Category	Criteria	Confidence Factor	Estimation	Mining Recovery Factor
Measured	Vein accessed by development, and then evaluated by face samples	not applicable	. Pintas formula used to convert measured wolframite crystal areas into kg/m^2 grades: . Cut-Off = 10kg/m2 wolframite (0.12% WO3)	84%
Indicated	>= 2 +18cm quartz drillhole intersections	60%	. D9 formula used to convert quartz thicknesses into kg/m^2 grades. 18cm is equivalent to a cut-off of 10.8 kg/m2 wolframite (0.13% WO3)	
Inferred	At 1 least +18cm quartz drillhole intersection	40%		

Notes

. Tonnage Calculations all use:

- Average s.g. = 2.8

- Average mining height = 2.2m

Figure 14-2. Level 3 – Drive and Panel System

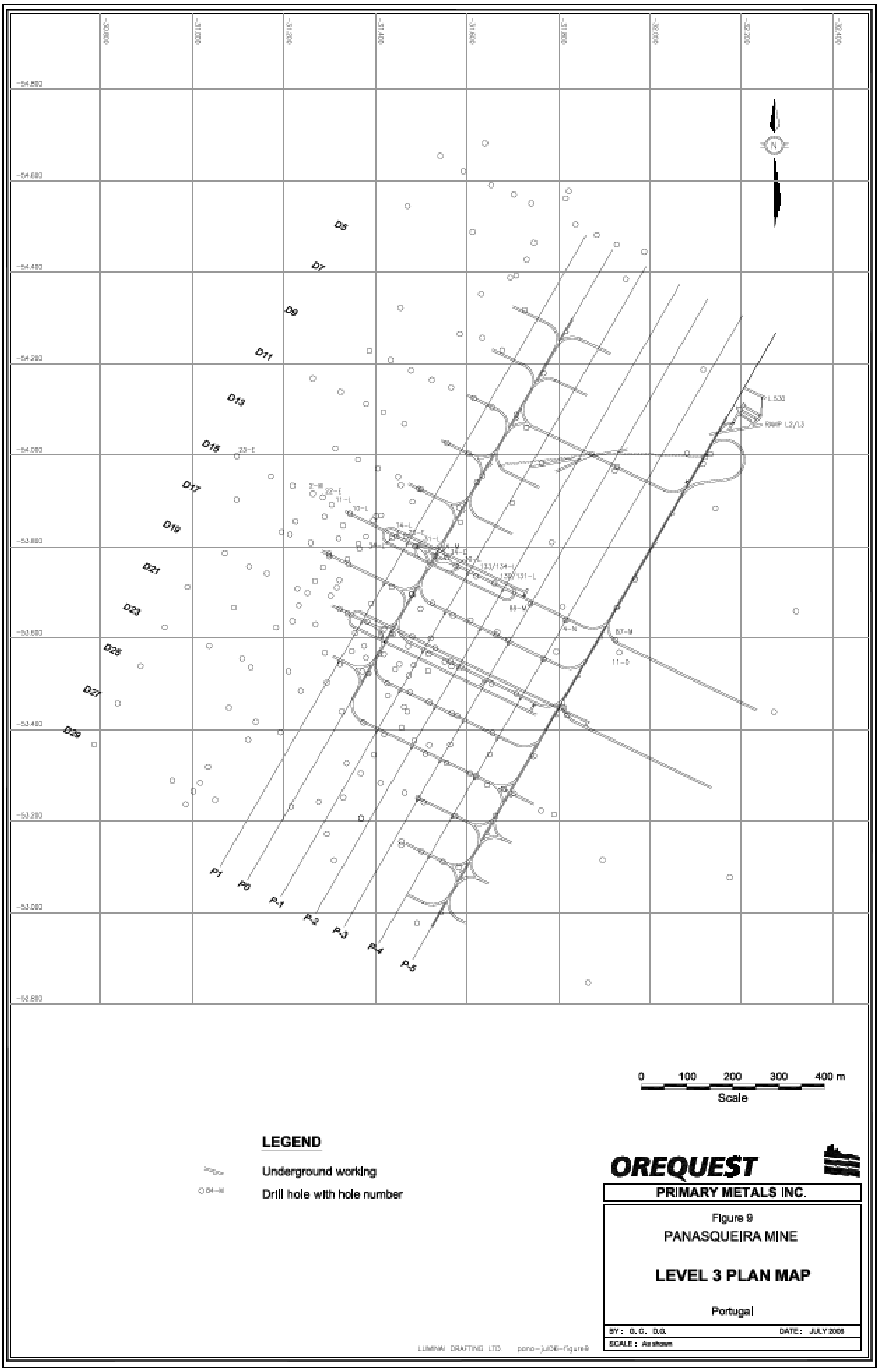


Figure 14-3. Mine Development Plan

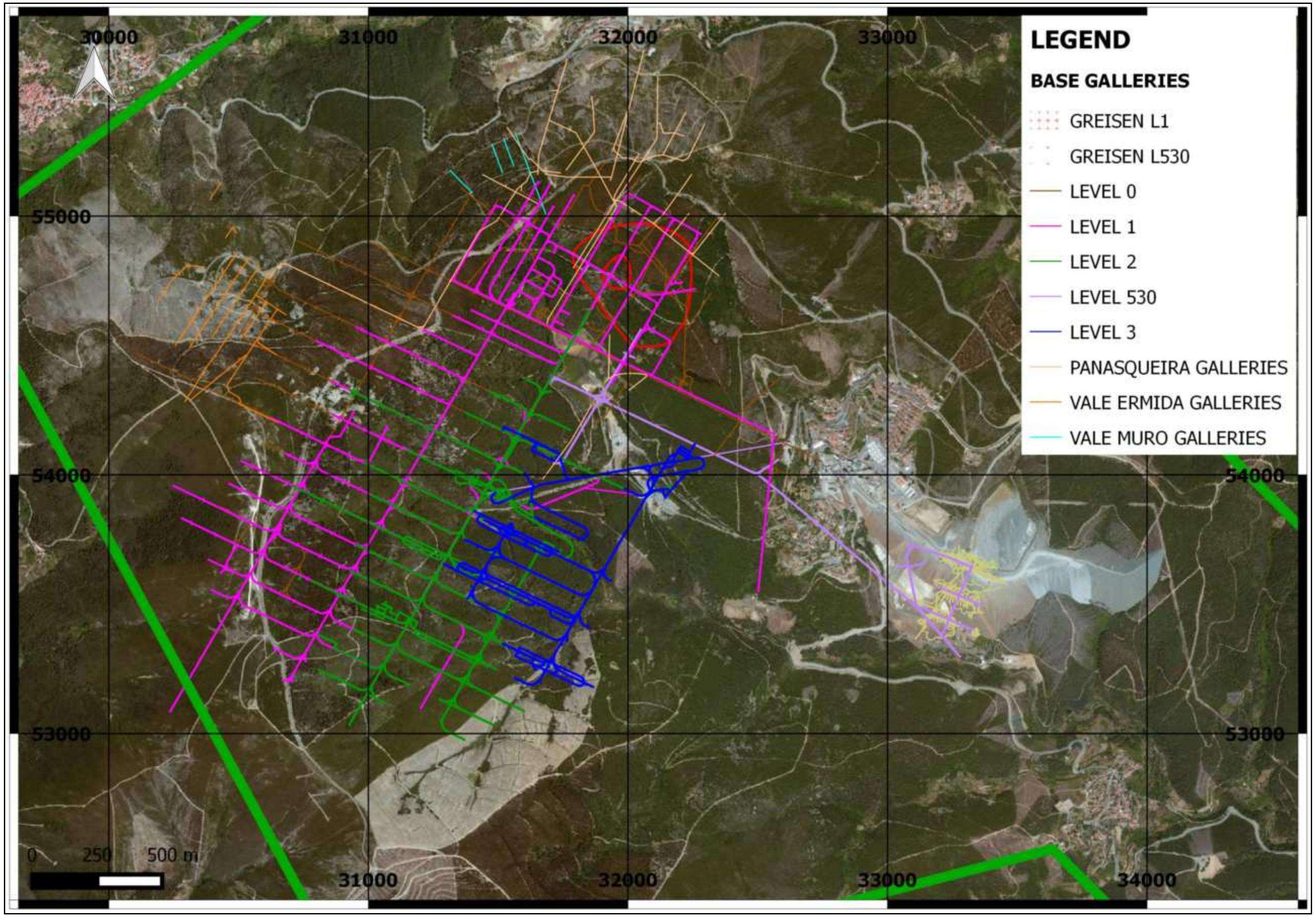


Figure 14-4. Example of Resource Blocks – Level 3 Vein Structure AW31

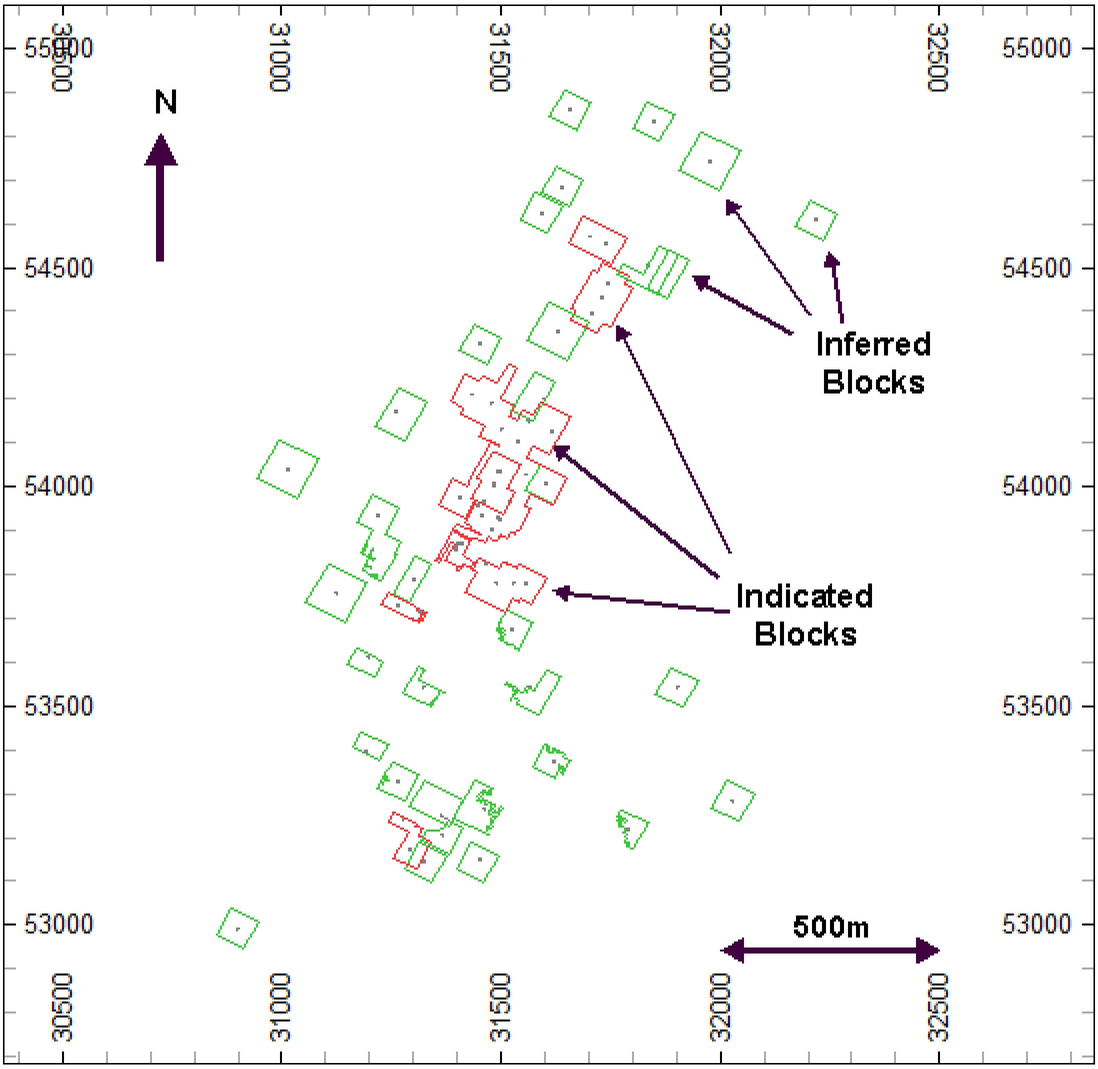


Table 14-5. Measured And Indicated Resources
As of 30th September, 2016

LEVEL	*Measured*			*Indicated*			*Measured + Indicated*		
	Tonnes	WO_3	WO3	Tonnes	WO_3	WO_3	Tonnes	WO_3	WO_3
	Kt	*%*	*MTU x1000*	*Kt*	*%*	*MTU x1000*	*Kt*	*%*	*MTU x1000*
L0	51	0.18	9	1,038	0.23	236	1,089	0.22	245
L1	706	0.20	139	1,314	0.21	272	2,020	0.20	411
L2	468	0.20	92	2,984	0.24	726	3,452	0.24	818
L3	727	0.21	153	2,396	0.25	610	3,123	0.24	763
L4	-	-	-	343	0.22	76	343	0.22	76
Total	**1,951**	**0.20**	**393**	**8,076**	**0.24**	**1,920**	**10,027**	**0.23**	**2,313**

Notes

. Resources shown are inclusive of reserves
. Minimum thickness = 2.2m
. Mining recovery = 84%

Measured Resources

. Cut-off = 0.12% WO_3 (Equivalent to 10 kg/m^2)
. Evaluation based on:
- Face mapping of wolframite exposed areas
- Areas converted to grade using Pinta's formula
- Blocks laid out on mine planning grid system

Indicated Resources

. Cut-off = 0.13% WO_3 (Equivalent to 10.8 kg/m^2)
. Evaluation based on:
- Drillhole quartz intersections
- Conversion to grade using D9 formula
- Blocks based on at least 2 drillhole intersections

. Additional factor applied:
- Confidence factor = 60%

Table 14-6. Indicated Resources Breakdown
As of 30th September, 2016

LEVEL	VEIN STRUCTURE	OLD PANASQUEIRA: NORTH D1				NORTH: D15 - D1				SOUTH: D15				TOTAL	
		Tonnes	WO3	Wol'te	Av'age Thick	Tonnes	WO3	Wol'te	Av'age Thick	Tonnes	WO3	Wol'te	Av'age Thick	Tonnes	WO3
		Kt	%	kg/m²	cm	Kt	%	kg/m²	cm	Kt	%	kg/m²	cm	Kt	%
L0	GREEN	-	-	-	-	49	0.22	18	30	117	0.26	22	37	166	0.25
	ORANGE	-	-	-	-	141	0.18	15	25	-	-	-	-	141	0.18
	VIOLET	-	-	-	-	244	0.22	18	30	45	0.20	16	27	289	0.22
	MAGENTA	-	-	-	-	182	0.23	19	31	41	0.19	16	26	223	0.22
	BLACK	-	-	-	-	60	0.31	25	42	122	0.23	19	32	182	0.26
	BROWN	-	-	-	-	-	-	-	-	38	0.25	21	34	38	0.25
	TOTAL	**-**	**-**	**-**	**-**	**676**	**0.22**	**18**	**31**	**362**	**0.24**	**19**	**32**	**1,038**	**0.23**
L1	GREEN	-	-	-	-	94	0.19	16	26	187	0.16	13	22	281	0.17
	ORANGE	-	-	-	-	29	0.15	12	20	223	0.19	16	26	252	0.19
	VIOLET	-	-	-	-	77	0.25	20	34	159	0.17	14	23	236	0.19
	MAGENTA	-	-	-	-	86	0.22	18	30	138	0.29	24	40	224	0.26
	BLACK	-	-	-	-	25	0.25	20	34	113	0.20	17	28	138	0.21
	BROWN	-	-	-	-	132	0.22	18	31	52	0.28	23	38	183	0.24
	TOTAL	**-**	**-**	**-**	**-**	**443**	**0.22**	**18**	**30**	**871**	**0.20**	**17**	**28**	**1,314**	**0.21**
L2	GREEN	118	0.23	19	32	229	0.25	20	34	265	0.20	17	27	612	0.23
	ORANGE	155	0.32	26	44	187	0.27	22	36	64	0.16	13	22	406	0.27
	VIOLET	43	0.33	27	45	204	0.21	17	27	276	0.19	15	26	524	0.21
	MAGENTA	174	0.32	26	44	241	0.22	18	30	196	0.22	18	30	610	0.25
	BLACK	121	0.47	39	65	211	0.24	20	34	96	0.17	14	24	428	0.29
	BROWN	-	-	-	-	289	0.24	20	33	114	0.21	17	29	404	0.23
	TOTAL	**611**	**0.33**	**27**	**46**	**1,362**	**0.24**	**19**	**32**	**1,011**	**0.20**	**16**	**27**	**2,984**	**0.24**
L3	GREEN	-	-	-	-	293	0.24	19	32	178	0.21	18	29	471	0.23
	ORANGE	-	-	-	-	277	0.23	19	31	112	0.24	19	29	389	0.23
	VIOLET	-	-	-	-	141	0.24	20	33	144	0.21	17	28	284	0.22
	MAGENTA	-	-	-	-	141	0.29	23	38	77	0.24	19	32	218	0.27
	BLACK	-	-	-	-	73	0.25	21	34	56	0.24	20	33	129	0.25
	BROWN	-	-	-	-	162	0.37	30	50	66	0.33	27	45	228	0.36
	DARK BLUE	-	-	-	-	146	0.27	22	37	53	0.26	21	36	199	0.27
	YELLOW	-	-	-	-	185	0.25	21	35	8	0.15	12	20	194	0.25
	LIGHT BLUE	-	-	-	-	212	0.29	24	40	71	0.21	18	30	284	0.27
	TOTAL	**-**	**-**	**-**	**-**	**1,630**	**0.26**	**22**	**36**	**766**	**0.23**	**19**	**31**	**2,396**	**0.25**
L4	GREEN	-	-	-	-	86	0.24	20	34	80	0.18	15	25	166	0.21
	ORANGE	-	-	-	-	53	0.29	24	39	37	0.21	17	28	90	0.25
	VIOLET	-	-	-	-	18	0.14	11	19	-	-	-	-	18	0.14
	MAGENTA	-	-	-	-	-	-	-	-	37	0.20	16	27	37	0.20
	BROWN	-	-	-	-	-	-	-	-	31	0.23	19	31	31	0.23
	TOTAL	**-**	**-**	**-**	**-**	**157**	**0.25**	**20**	**34**	**186**	**0.20**	**16**	**27**	**343**	**0.22**
GRAND TOTAL		611	0.33	27	46	4,268	0.24	20	32	3,197	0.21	17	27	8,076	0.24

Notes
. D9 formula used to convert quartz thicknesses into kg/m² grades
. 18cm cut-off applied, which is equivalent to :
= 10.8 kg/m² wolframite
= 0.13% WO_3
. All the resources evaluated are exclusive of reserves
. Tonnage Calculations all use:
- Average s.g. = 2.8
- Average mining height = 2.2m
. Mining recovery = 84%
. Indicated confidence factor = 60%

Table 14-7. Inferred Resources Breakdown – Panasqueira Mine
As of 30th September, 2016

LEVEL	UNIT	OLD PANASQUEIRA: NORTH D1				NORTH: D15 - D1				SOUTH: D15				TOTAL	
		Tonnes	WO3	Wol'te	Av'age Thick	Tonnes	WO3	Wol'te	Av'age Thick	Tonnes	WO3	Wol'te	Av'age Thick	Tonnes	WO3
		Kt	%	kg/m²	cm	Kt	%	kg/m²	cm	Kt	%	kg/m²	cm	Kt	%
L0	GREEN	-	-	-	-	122	0.20	17	28	60	0.19	16	27	183	0.20
	ORANGE	-	-	-	-	187	0.19	16	27	10	0.18	15	25	197	0.19
	VIOLET	-	-	-	-	30	0.16	13	21	30	0.15	13	21	59	0.15
	MAGENTA	-	-	-	-	76	0.16	13	22	49	0.17	14	25	125	0.16
	BLACK	-	-	-	-	38	0.19	15	26	97	0.20	16	27	135	0.19
	BROWN	-	-	-	-	73	0.27	22	37	62	0.24	20	33	134	0.26
	TOTAL	**-**	**-**	**-**	**-**	**525**	**0.20**	**16**	**27**	**308**	**0.20**	**16**	**27**	**833**	**0.20**
L1	GREEN	-	-	-	-	20	0.15	12	20	170	0.20	17	28	190	0.20
	ORANGE	-	-	-	-	106	0.26	21	37	105	0.20	16	27	211	0.23
	VIOLET	-	-	-	-	100	0.19	16	26	55	0.23	19	31	154	0.21
	MAGENTA	-	-	-	-	73	0.20	17	28	104	0.22	18	30	177	0.21
	BLACK	-	-	-	-	59	0.20	17	28	59	0.21	18	29	118	0.21
	BROWN	19	0.45	37	62	26	0.15	12	20	101	0.12	10	16	146	0.17
	TOTAL	**19**	**0.45**	**37**	**62**	**383**	**0.21**	**17**	**29**	**595**	**0.20**	**16**	**27**	**996**	**0.21**
L2	GREEN	32	0.22	18	30	99	0.18	15	25	149	0.19	16	27	279	0.19
	ORANGE	20	0.16	13	22	87	0.18	15	27	164	0.17	14	25	271	0.17
	VIOLET	32	0.15	12	20	74	0.31	25	45	80	0.18	15	25	186	0.23
	MAGENTA	6	0.19	16	26	148	0.24	19	32	131	0.21	17	29	285	0.22
	BLACK	22	0.18	15	25	98	0.18	15	24	108	0.19	16	27	228	0.18
	BROWN	46	0.21	17	29	41	0.27	22	37	16	0.15	13	21	103	0.23
	TOTAL	**157**	**0.19**	**15**	**26**	**547**	**0.22**	**18**	**31**	**647**	**0.19**	**16**	**26**	**1,351**	**0.20**
L3	GREEN	-	-	-	-	171	0.23	19	31	94	0.26	21	35	264	0.24
	ORANGE	-	-	-	-	101	0.22	18	31	30	0.17	14	23	131	0.21
	VIOLET	-	-	-	-	179	0.24	20	33	96	0.17	14	23	275	0.22
	MAGENTA	-	-	-	-	101	0.22	18	29	60	0.18	15	25	162	0.21
	BLACK	-	-	-	-	16	0.18	15	25	89	0.21	17	28	106	0.20
	BROWN	-	-	-	-	49	0.19	16	26	10	0.15	13	22	59	0.18
	DARK BLUE	-	-	-	-	90	0.25	21	34	35	0.18	15	25	125	0.23
	YELLOW	-	-	-	-	76	0.27	22	36	36	0.34	28	46	112	0.29
	LIGHT BLUE	-	-	-	-	83	0.25	20	34	52	0.17	14	24	135	0.22
	TOTAL	**-**	**-**	**-**	**-**	**867**	**0.24**	**19**	**32**	**502**	**0.21**	**17**	**28**	**1,369**	**0.22**
L4	GREEN	-	-	-	-	31	0.27	22	36	26	0.17	14	23	57	0.22
	ORANGE	-	-	-	-	52	0.25	21	34	27	0.14	12	20	79	0.21
	VIOLET	-	-	-	-	56	0.17	14	23	35	0.24	20	33	91	0.20
	MAGENTA	-	-	-	-	51	0.21	17	29	-	-	-	-	51	0.21
	BLACK	-	-	-	-	10	0.20	16	27	-	-	-	-	10	0.20
	BROWN	-	-	-	-	38	0.28	23	39	30	0.15	12	21	67	0.23
	DARK BLUE	-	-	-	-	20	0.19	15	26	10	0.15	13	21	30	0.18
	YELLOW	-	-	-	-	31	0.29	24	40	9	0.26	21	35	40	0.28
	LIGHT BLUE	-	-	-	-	48	0.27	22	37	42	0.37	30	51	90	0.32
	GREY	-	-	-	-	10	0.19	16	26	-	-	-	-	10	0.19
	RED	-	-	-	-	21	1.55	127	212	6	0.13	11	18	28	1.22
	LIGHT GREEN	-	-	-	-	10	0.21	17	29	-	-	-	-	10	0.21
	LIGHT BROWN	-	-	-	-	34	0.21	17	28	10	0.15	13	21	43	0.19
	TOTAL	**-**	**-**	**-**	**-**	**410**	**0.30**	**25**	**41**	**196**	**0.22**	**18**	**30**	**606**	**0.27**
GRAND TOTAL		**176**	**0.22**	**18**	**30**	**2,731**	**0.23**	**19**	**32**	**2,248**	**0.20**	**16**	**27**	**5,155**	**0.22**

Notes
- . D9 formula used to convert quartz thicknesses into kg/m² grades
- . 18cm cut-off applied, which is equivalent to :
  - = 10.8 kg/m² wolframite
  - = 0.13% WO_3
- . All the resources evaluated are exclusive of reserves
- . Tonnage Calculations all use:
  - Average s.g. = 2.8
  - Average mining height = 2.2m
- . Mining recovery = 84%
- . Inferred confidence factor = 40%

14.1.4 Validation

A summary of the comparison of the Panasqueira mine resources over the past 5 years is shown in Table 14-8. For comparison purposes, resources are shown for measured, indicated and inferred categories. The issuer is not treating these historical estimates as current mineral resources or mineral reserves – they are just for comparative purposes.

Table 14-8. Historical Comparison

Date	Measured Resources		Indicated Resources		Inferred Resources	
	Tonnes *Mt*	WO_3 *%*	Tonnes *Mt*	WO_3 *%*	Tonnes *Mt*	WO_3 *%*
2011 January	1.25	0.25	10.93	0.23	6.07	0.22
2011 July	1.29	0.24	10.93	0.23	6.03	0.22
2012 January	1.20	0.24	11.05	0.23	6.04	0.22
2012 July	1.22	0.23	10.82	0.23	5.96	0.22
2013 January	1.23	0.22	9.68	0.23	5.92	0.22
2013 July	1.26	0.21	9.43	0.23	5.88	0.22
2014 January	1.28	0.21	8.48	0.24	5.03	0.22
2014 July	1.57	0.20	8.14	0.24	5.01	0.22
2015 January	1.54	0.20	7.94	0.23	4.93	0.22
2015 July	1.66	0.21	7.88	0.24	4.91	0.22
2016 September	1.95	0.20	8.08	0.24	5.16	0.22

Notes

. Resource and reserves added for comparative purposes only

. Cut-offs applied:

Reserves = 10kg/m^2 = 0.12% WO3

Ind+Inf Resources = 18cm > 10.8kg/m^2 = 0.13% WO_3

These figures indicate that the overall resource level has generally declined from 2012 to 2015, and thereafter has generally stabilised, despite mining approximately 800 ktpa over the same period. The degree to which the overall resource level is preserved is affected by:

- Additional resources encountered with d/d drilling in new zones.
- Not-In-Reserve (NIR) ore which is material added between resource/reserve updates, that simply had not been evaluated before with either normal resources/reserves.
- Material mined as ore.
- Some material lost as 'Damaged Areas' (DA) where subsidence has forced the premature closure of certain stoping areas.

For the Panasqueira mine area, resource block plans have been examined for all levels, and generally the layout and distribution of resource polygons, relative to drillhole intersections, appears logical and follows the general guidelines described previously. The QP has additionally completed his own verification steps, as summarised in Section 12.

14.2 BL1 Tailings Area

The BTW management are considering the re-processing of previously dumped tailings material. The BL1 tailings area is located south of the waste dump area, as shown in Figure 14-5. The BL1 tailings area contains tailings up to the year of 1985. 24 test diamond drillholes were completed in the BL1 area in 1991, and 5 more were completed in 2008. A plan of these holes is shown in Figure 14-6. This plan also shows section lines that were used in the definition of the solid/tailings boundaries.

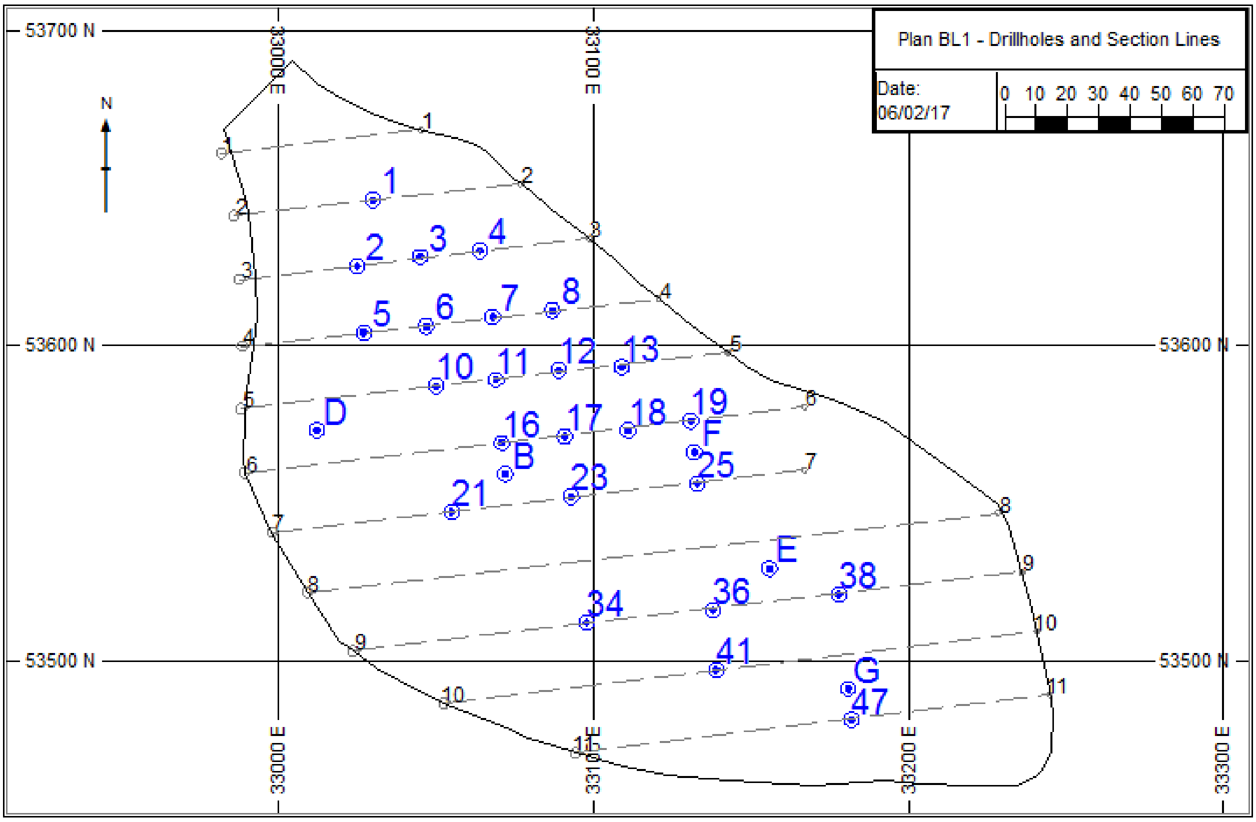
Figure 14-5. Plan of Tailings Areas

A summary of the BL1 drillholes is shown in Table 14-9. The majority of older 1991 samples were 1.5m in length, while the majority of the more recent samples were 3m in length.

Table 14-9. BL1 Drillhole Summary

				Samples			
YEAR	HOLES	LENGTH	Average Length / Hole	WO_3	Cu	Sn	Zn
		m	*m*				
1991	24	525.3	21.9	323	343	330	346
2008	5	104.2	20.8	34			
Total	29	629.5	21.7	357	343	330	346

Figure 14-6. Plan of Drillholes in BL1 Tailings Area



For each section line, profiles of the deposited tailings were built up on each section line, based on previous profiles defined by the mine survey department, as well as the imported drillhole data. These profiles, spaced approximately 20m apart, were then used to build up a 3D wireframe model. 3D plots of these profiles and resultant wireframe models are shown in Figure 14-7 and Figure 14-8.

A statistical summary of the assays from the drilling data is shown in Table 14-10. Histograms and log-probability plots of the sample data are shown in Figure 14-9. Is can be seen from this the most of the grades belong to approximate log-probability populations.

The results of a decile analysis on the WO_3 grades is shown in Table 14-11. From the log-probability plots and decile analyses, a top-cut level of 0.5 WO_3% was applied onto outlier grades, prior to the creation of 3m composites. Summary statistics of the composites are shown in Table 14-12.

Figure 14-7. 3D View of BL1 Tailings Profiles – Viewed from SW

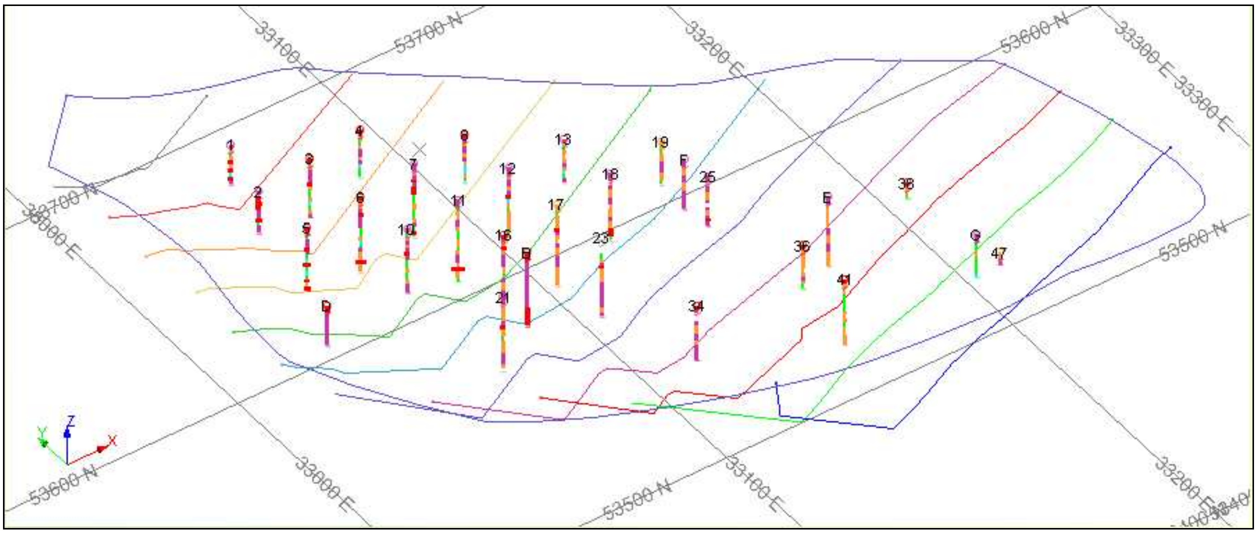


Figure 14-8. 3D View of BL1 Tailings Wireframe Model – Viewed from SW

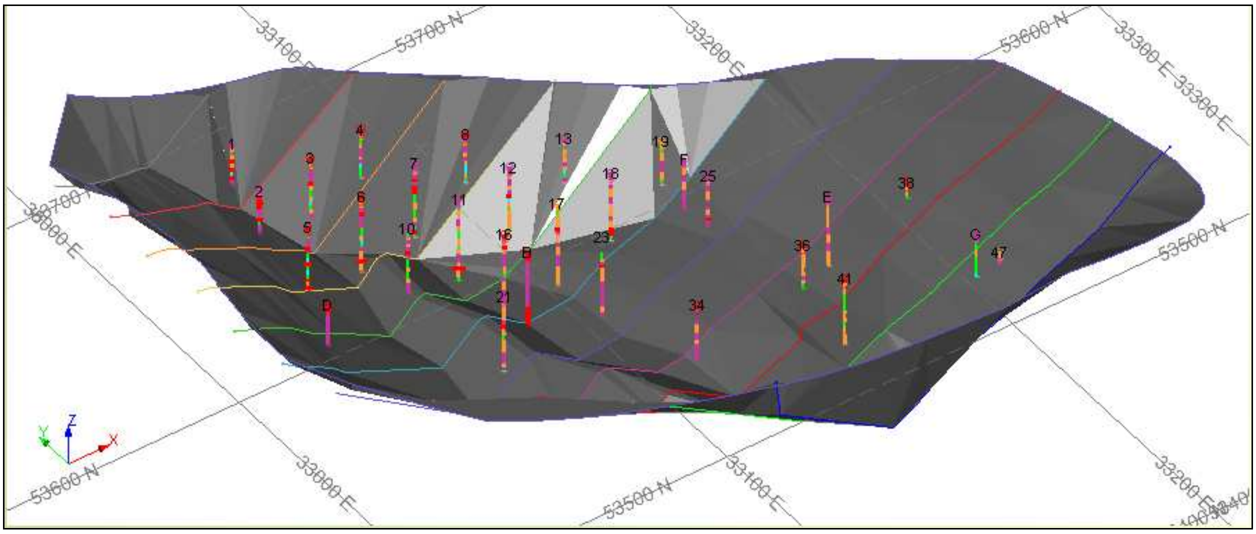


Table 14-10. Statistical Summary of BL1 Assay Data

FIELD	NUMBER	MINIMUM	MAXIMUM	MEAN	STANDDEV	VARIANCE	COVARTN
WO3	391	0	2.41	0.29	0.122	0.015	0.42
CU	354	0.01	0.7	0.33	0.111	0.012	0.33
SN	354	0.01	0.07	0.03	0.008	0.000	0.30
ZN	354	0.02	1.42	0.64	0.269	0.072	0.42

Figure 14-9. Histograms and Log-Probability Plots of BL1 Sample Data

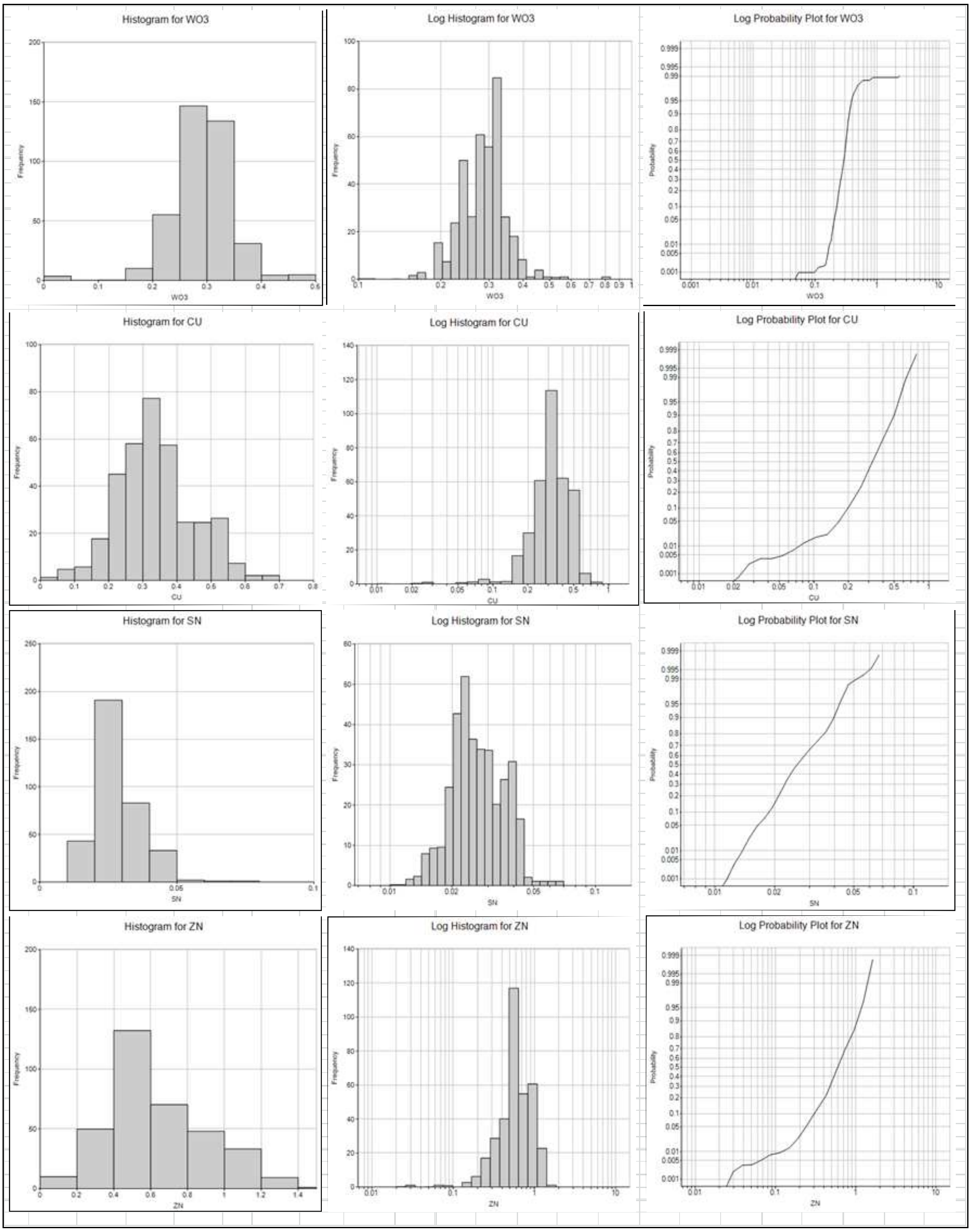


Table 14-11. Decile Analysis – WO_3 Grades – BL1

Q%_FROM	Q%_TO	NUMBER	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
0	10	46	0.20	0.03	0.22	12.1	6.6
10	20	36	0.24	0.23	0.25	14.9	8.1
20	30	40	0.25	0.25	0.26	15.7	8.6
30	40	41	0.27	0.26	0.27	16.5	9.0
40	50	40	0.28	0.27	0.29	17.5	9.6
50	60	40	0.30	0.29	0.3	18.3	10.0
60	70	37	0.31	0.3	0.31	18.9	10.3
70	80	33	0.32	0.31	0.33	19.6	10.7
80	90	40	0.34	0.33	0.35	21.2	11.6
90	100	38	0.45	0.35	2.41	28.0	15.3
90	91	4	0.35	0.35	0.35	2.1	1.1
91	92	3	0.36	0.36	0.36	2.2	1.2
92	93	4	0.36	0.36	0.36	2.2	1.2
93	94	4	0.37	0.36	0.37	2.2	1.2
94	95	3	0.37	0.37	0.37	1.7	0.9
95	96	3	0.38	0.38	0.38	2.3	1.2
96	97	5	0.39	0.39	0.4	3.5	1.9
97	98	4	0.42	0.41	0.43	2.3	1.3
98	99	3	0.46	0.45	0.46	2.7	1.5
99	100	5	0.97	0.49	2.41	7.0	3.8
0	100	391	0.30	0.03	2.41	182.8	100.0

Table 14-12. Statistical Summary of 3m Composites – BL1

FIELD	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	COVARTN
WO3	222	0.10	0.45	0.29	0.002	0.045	0.16
CU	183	0.07	0.58	0.33	0.009	0.096	0.29
SN	183	0.01	0.05	0.03	0.00005	0.007	0.25
ZN	183	0.14	1.27	0.64	0.059	0.243	0.38

A volumetric block model was set up for the BL1 tailings area, based on a parent block size of 20m x 20m x 3m. The model prototype is summarised in Table 14-13. The composite grades were then interpolated into the block model, using an inverse-distance weighting (^2) estimation method. The parameters used are summarised in Table 14-14.

Table 14-13. Model Prototype – BL1

	Min *m*	Max *m*	Range *m*	Size *m*	Number
X	32,900	33,300	400	20	20
Y	53,400	54,000	600	20	30
Z	560	620	60	3	20

Table 14-14. Grade Estimation Parameters – BL1

Search Distances (m)		Search	Minimum	Minimum
Lateral	Vertical		Composites	Drillholes
50 x 50	10	1st	3	3
100 x 100	20	2nd	3	3
250 x 250	50	3rd	1	1

Notes:

. Maximum number of composites used = 15
. All grades estimated using ID (^2)
. For validation, grades aslo estimated using NN

A global density of 2.5 t/m3 was assumed, based on more recent density measurements taken from samples in the BL2A tailings material, as summarised in Section 14.3. Example cross-sections through the estimated BL1 block model, showing the WO3 estimated grades, is shown in Figure 14-10.

As part of the validation process, global average grades were compared, as derived from initial samples, composites and estimated model grades, as summarised in Table 14-15.

Table 14-15. Comparison of Global Averages

			Model Grades	
	Samples	Composites	NN	ID
WO_3	0.29	0.29	0.28	0.29
Cu	0.33	0.33	0.30	0.30
Sn	0.028	0.028	0.026	0.027
Zn	0.64	0.64	0.56	0.57

Notes

. All grades shown in %

All of the BL1 were classified with an **Inferred** resource category. Reasons for assigning this resources category included:

- No core samples are available for current check purposes.
- No QAQC data or information is available from the drilling campaigns.
- No density measurements are available.

An overall evaluation of the estimated BL1 tailings area is shown in Table 14-16. No cut-off level has been applied in this evaluation, as in any exploitation it is unlikely that different areas of the tailings area can be selectively mined. A bench breakdown of the resources is shown in Table 14-17, and a grade-tonnage table in Table 14-18.

Figure 14-10. Model Sections – WO3 – BL1

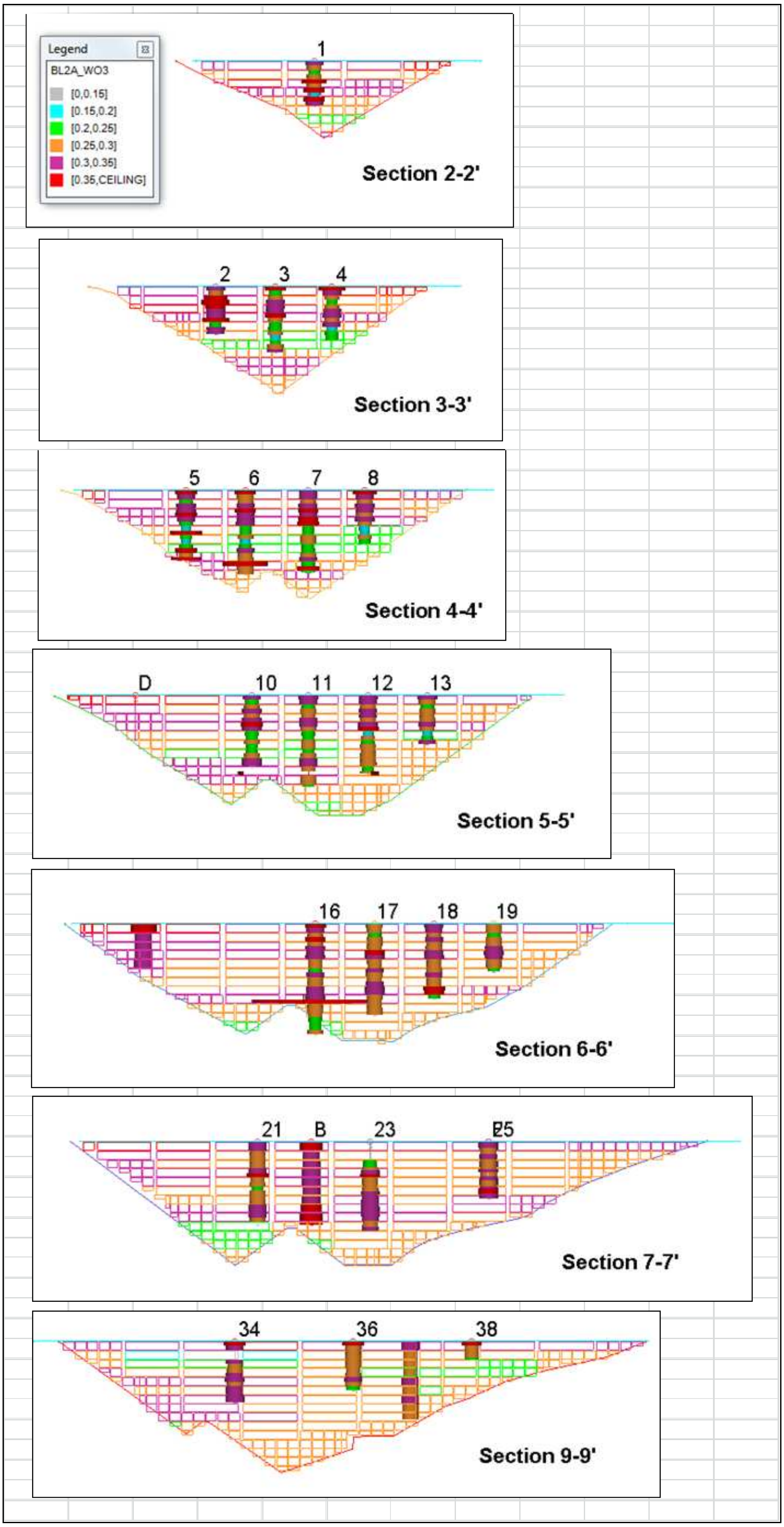


Table 14-16. BL1 Evaluation Summary

Category	Tonnes Kt	WO3 %	Cu %	Sn %	Zn %
3 holes within 50 x 50m	1,474	0.29	0.31	0.027	0.59
3 holes within 100 x 100m	341	0.28	0.27	0.025	0.49
Extrapolated	3	0.27	0.23	0.021	0.36
Total Inferred Resources	**1,817**	**0.29**	**0.30**	**0.027**	**0.57**

Notes

.No cut-off applied

. Assumed dry density = 2.5 t/m^3

. All resources classified as inferred

Table 14-17. BL1 Bench Breakdown

BENCH mRL	Tonnes Kt	WO_3 %	Cu %	Sn %	Zn %
617	255	0.33	0.26	0.023	0.38
614	233	0.27	0.28	0.028	0.65
611	212	0.28	0.34	0.034	0.85
608	191	0.29	0.42	0.035	0.85
605	171	0.27	0.36	0.028	0.60
602	154	0.27	0.30	0.025	0.53
599	139	0.27	0.29	0.024	0.55
596	123	0.30	0.31	0.025	0.55
593	108	0.30	0.27	0.022	0.41
590	88	0.28	0.25	0.021	0.34
587	63	0.27	0.21	0.018	0.29
584	42	0.27	0.21	0.018	0.28
581	25	0.26	0.20	0.017	0.27
578	10	0.27	0.21	0.018	0.28
575	1	0.27	0.22	0.019	0.30
TOTAL	**1,817**	**0.29**	**0.30**	**0.027**	**0.57**

Notes

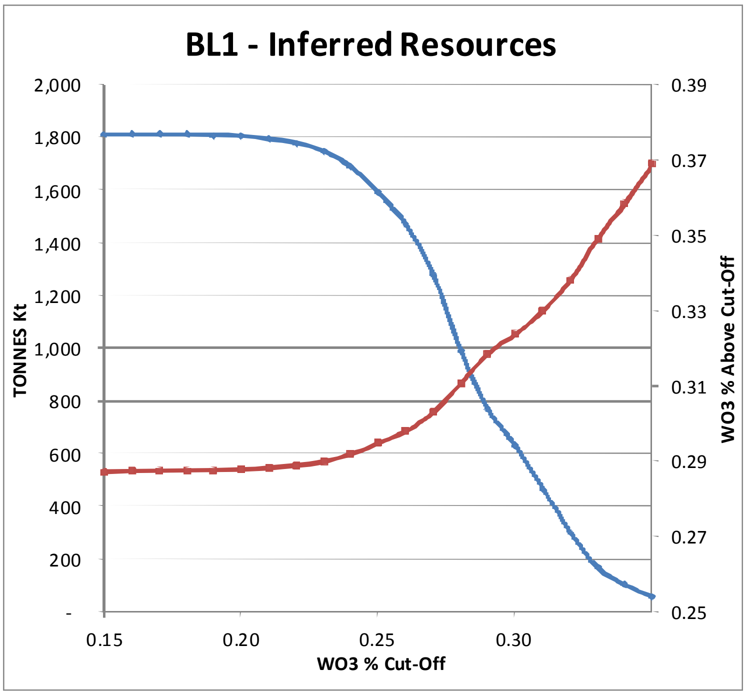
.No cut-off applied

. Assumed dry density = 2.5 t/m^3

. All resources classified as inferred

Table 14-18. Grade-Tonnage Table – BL1

WO_3 Cut-Off %	Tonnes Kt	WO_3 %	Cu %	Sn %	Zn %
0.15	1,814	0.29	0.30	0.027	0.57
0.16	1,811	0.29	0.30	0.027	0.57
0.17	1,811	0.29	0.30	0.027	0.57
0.18	1,811	0.29	0.30	0.027	0.57
0.19	1,808	0.29	0.30	0.027	0.57
0.20	1,805	0.29	0.30	0.027	0.57
0.21	1,794	0.29	0.30	0.027	0.57
0.22	1,779	0.29	0.30	0.027	0.57
0.23	1,747	0.29	0.30	0.027	0.57
0.24	1,688	0.29	0.31	0.027	0.57
0.25	1,589	0.29	0.31	0.027	0.57
0.26	1,469	0.30	0.31	0.027	0.57
0.27	1,279	0.30	0.32	0.027	0.58
0.28	993	0.31	0.33	0.028	0.60
0.29	768	0.32	0.34	0.028	0.61
0.30	630	0.32	0.34	0.028	0.60
0.31	468	0.33	0.34	0.028	0.59
0.32	303	0.34	0.34	0.028	0.57
0.33	169	0.35	0.32	0.027	0.53
0.34	103	0.36	0.34	0.028	0.57
0.35	58	0.37	0.34	0.028	0.56



14.3 BL2A Tailings Area

Two different types of drilling have taken place on the BL2A tailings area during 2016. 7 percussive drillholes were completed, with assaying of 1m sludge samples. 12 diamond drillholes were also completed, with the assaying of 1m length core samples. This assaying work is still in progress, the database used for the current estimate is not complete. 42 of the samples from the diamond drilling campaign have also been split into four different granulometric fractions, and assays were obtained for each of the fractions. The current database of samples for BL2A is summarised in Table 14-19.

Table 14-19. Summary of BL2A Sample Database

					Samples			
Drillhole Type	HOLES	LENGTH	Average Length / Hole	HOLES With Samples	WO_3	Cu	Sn	Zn
		m	*m*					
Percussive	7	82.0	11.7	7	82	82	80	82
Diamond Drillhole	12	396.1	33.0	6	100	100	100	100
Total	**19**	**478.1**	**25.2**	**13**	**182**	**182**	**180**	**182**

Plans showing the BL2A drilling, overall and those holes just currently with assays, are shown in Figure 14-11. These plans also show reference section lines.

A 3D view showing the BL2A assay data is shown in Figure 14-12. This view also shows a basal wireframe created for the deposited tailings, generated from the limit perimeter at the top of tailings deposited, and the bottom points for each of the drilled holes.

A statistical summary of the complete sample assays for BL2A is shown in Table 14-20. A statistical summary of the granulometric fractions are shown in Table 14-21.

The relationships between assays and elevation are depicted in Figure 14-13. Although there are no general relationships apparent, there appear to be distinct clusters of higher grades (WO_3, Cu and Sn) at an elevation of approximately 660m (depth 25m).

Figure 14-11. Plans of BL2A Drilling

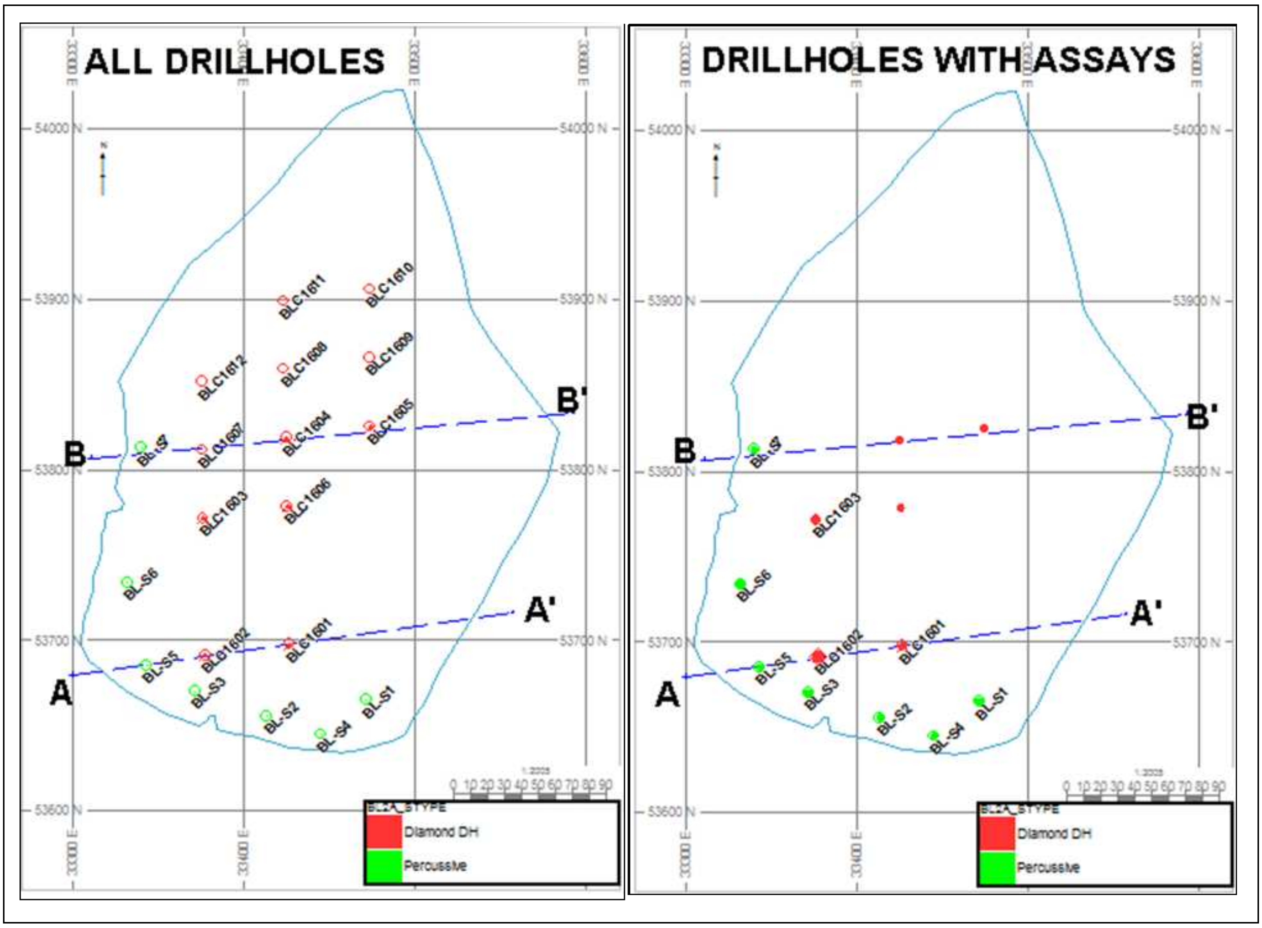


Figure 14-12. 3D View of BL2A Assay Data

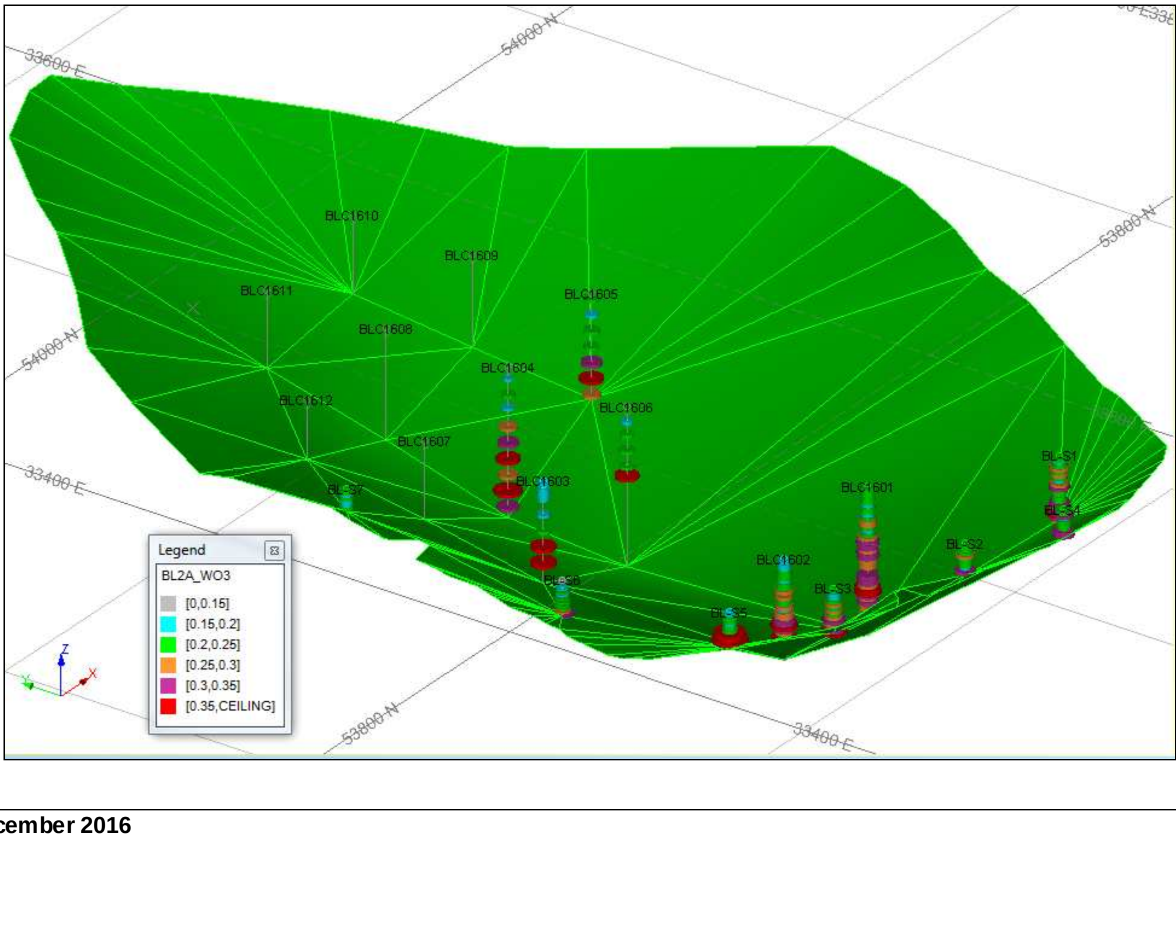


Table 14-20. Summary Statistics –Complete Sample Assays – BL2A

FIELD	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDARD DEVIATION	COVARTN
WO3	182	0.02	0.52	0.24	0.005	0.074	0.30
CU	182	0.03	0.69	0.25	0.015	0.122	0.50
SN	180	0	0.25	0.024	0.000	0.020	0.82
ZN	182	0.02	1.13	0.39	0.064	0.253	0.65

Table 14-21. Summary Statistics of Granulometric Fraction Assays

FIELD	MESH	MICRON	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDARD DEVIATION	COVARTN
WO3	+ 100 mesh	150	42	0.014	0.355	0.07	0.008	0.088	1.21
	+ 200 mesh	75	42	0.02	0.33	0.12	0.006	0.077	0.65
	+ 325 mesh	45	42	0.08	0.53	0.25	0.007	0.082	0.33
	- 325 mesh	-45	42	0.07	0.87	0.36	0.017	0.129	0.36
CU	+ 100 mesh	150	42	0.02	0.84	0.14	0.025	0.158	1.10
	+ 200 mesh	75	42	0.04	0.72	0.34	0.023	0.151	0.45
	+ 325 mesh	45	42	0.04	0.84	0.28	0.020	0.141	0.50
	- 325 mesh	-45	42	0.04	0.5	0.19	0.009	0.093	0.49
SN	+ 100 mesh	150	42	0.003	0.055	0.010	0.000	0.010	0.95
	+ 200 mesh	75	42	0.007	0.085	0.022	0.000	0.018	0.82
	+ 325 mesh	45	42	0.012	0.119	0.030	0.000	0.017	0.59
	- 325 mesh	-45	42	0.011	0.089	0.025	0.000	0.008	0.33
ZN	+ 100 mesh	150	42	0.03	1.76	0.23	0.066	0.258	1.11
	+ 200 mesh	75	42	0.05	1.24	0.52	0.102	0.320	0.62
	+ 325 mesh	45	42	0.05	1.04	0.44	0.072	0.267	0.60
	- 325 mesh	-45	42	0.03	0.66	0.28	0.022	0.150	0.54

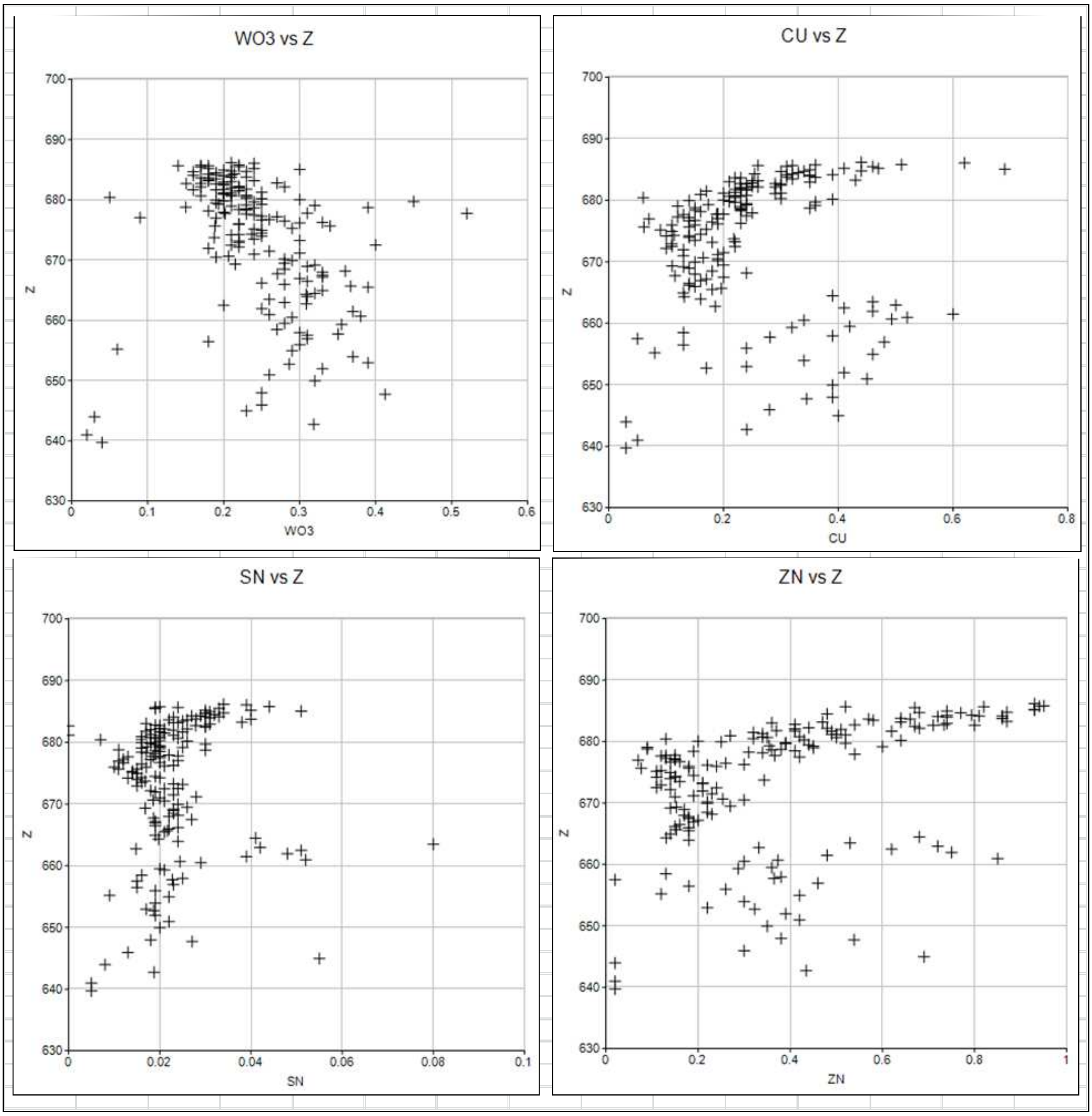
Notes

. Statistics weighted by proportion within each fraction

. Overall proportions of different fractions:

MESH	MICRON	PROPORTION
+ 100 mesh	150	19%
+ 200 mesh	75	15%
+ 325 mesh	45	17%
- 325 mesh	-45	48%

Figure 14-13. Graphs of Metal Assays v Elevation – BL2A



For 17 of the complete sample assays, assays were also available for the individual granulometric fractions. A comparison of the total sample assays against the averaged assays from the granulometric samples is shown in Figure 14-14. With only one outlier and a bias level of 10%, these results are acceptable.

Histograms and log-probability plots of the sample data are shown in Figure 14-15. It can be seen from this the most of the grades belong to approximate log-probability populations. The results of a decile analysis on the WO_3 grades are shown in Table 14-22. From the log-probability plots and decile analyses, it was decided that top-cuts were not required: no particular outliers were apparent.

Figure 14-14. Check of Granulometric Assays Against Complete Sample Assays

Number of Pairs			R^2	m	Error (m)	b	Bias
17			0.97	0.92	0.039	0.000	8.12%

Number Accepted	Outliers	Outliers %	R^2	m	Error (m)	b	Bias
16	1	5.9%	0.94	0.89	0.054	0.007	10.78%

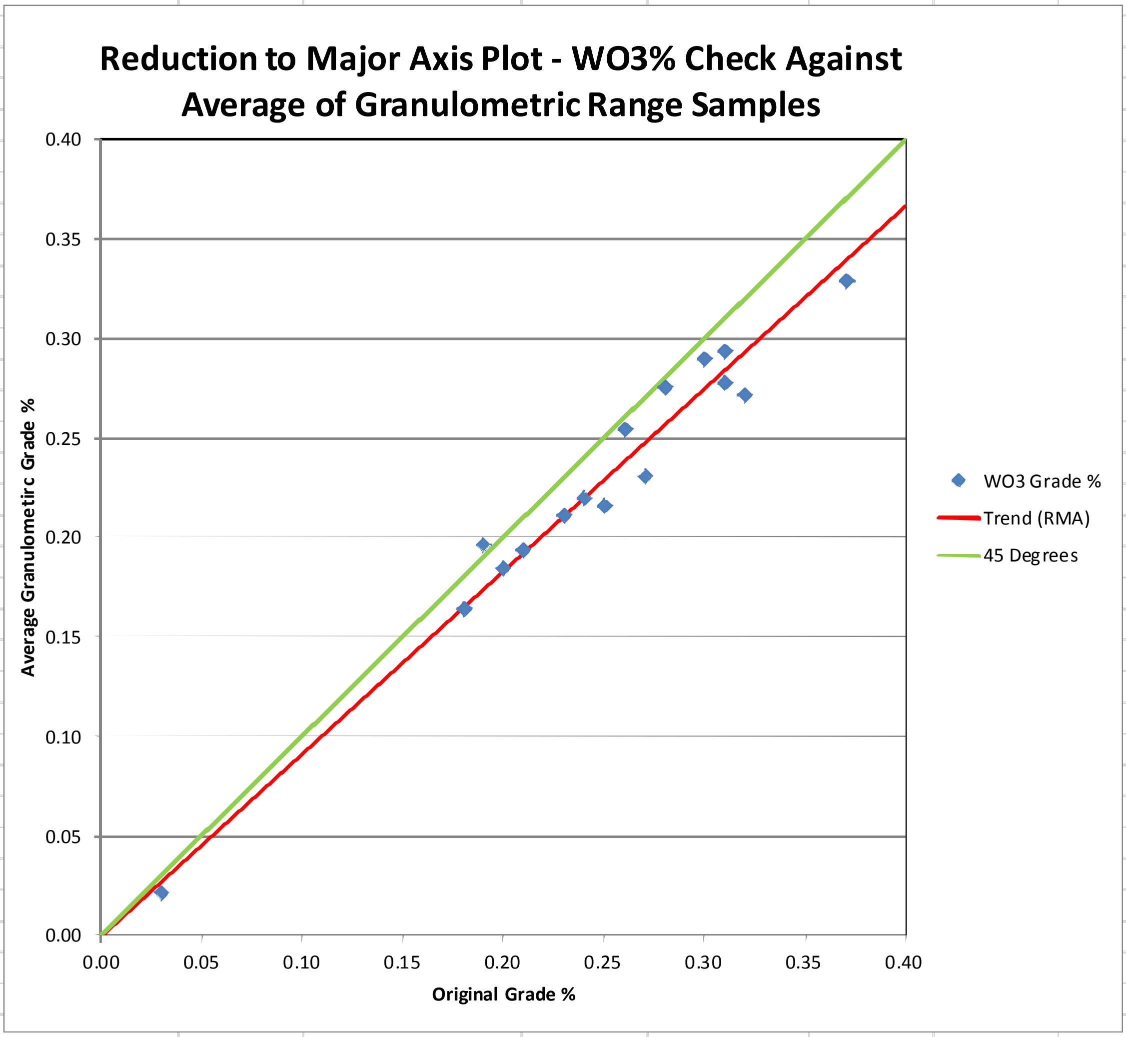


Figure 14-15. Histograms and Log-Probability Plots for BL2A Samples

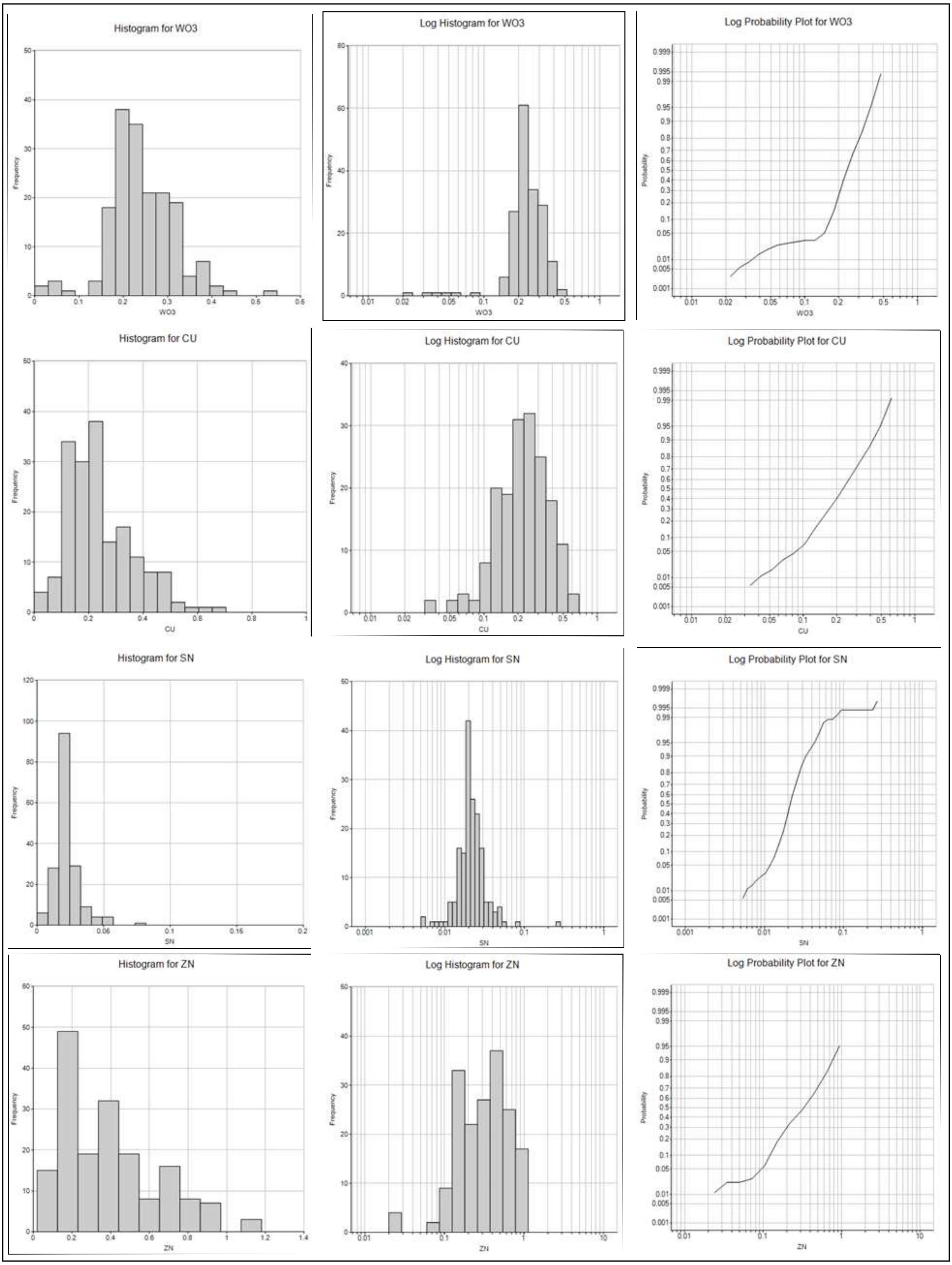


Table 14-22. Decile Analysis for BL2A WO_3 Samples

Q%_FROM	Q%_TO	NUMBER	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
0	10	17	0.12	0.02	0.17	1.90	4.62
10	20	18	0.18	0.17	0.19	3.22	7.84
20	30	17	0.20	0.19	0.20	3.17	7.72
30	40	18	0.21	0.21	0.22	3.71	9.04
40	50	18	0.23	0.22	0.23	3.68	8.96
50	60	19	0.24	0.23	0.25	4.25	10.36
60	70	17	0.26	0.25	0.28	4.35	10.59
70	80	17	0.29	0.28	0.30	4.95	12.05
80	90	17	0.32	0.30	0.33	5.37	13.07
90	100	18	0.38	0.33	0.52	6.47	15.75
90	91	1	0.33	0.33	0.33	0.33	0.80
91	92	2	0.33	0.33	0.33	0.66	1.61
92	93	2	0.35	0.34	0.35	0.52	1.27
93	94	2	0.36	0.36	0.36	0.72	1.74
94	95	2	0.37	0.37	0.37	0.74	1.80
95	96	1	0.37	0.37	0.37	0.37	0.90
96	97	2	0.39	0.38	0.39	0.77	1.88
97	98	2	0.39	0.39	0.39	0.78	1.90
98	99	2	0.41	0.40	0.41	0.61	1.49
99	100	2	0.49	0.45	0.52	0.97	2.36
0	100	176	0.24	0.02	0.52	41.05	100.00

For grade estimation purposes, composites were not created. A volumetric model was set up with a parent block size of 25m x 25m x 1m, corresponding with the 1m sample length used for the primary samples. The model prototype used is summarised in Table 14-23.

Table 14-23. Block Model Prototype – BL2A

	Minimum *m*	Maximum *m*	Range *m*	Size *m*	Number
X	33,250	33,600	350	25	14
Y	53,600	54,000	400	25	16
Z	620	700	80	1	80

Experimental variograms were generated from the drillhole data. Figure 14-16 shows the experimental variograms both horizontally and vertically, as well the defined model variograms, for WO_3, Cu and Sn. These also show a considerable longer horizontal than vertical ranges, as would be expected from the layered way in which the tailings dump area has been built up. The model variogram parameters are summarised in Table 14-24.

The variographic analysis was used to develop grade estimation parameters, as summarised in Table 14-25.

Figure 14-16. Experimental and Model Variograms – BL2A

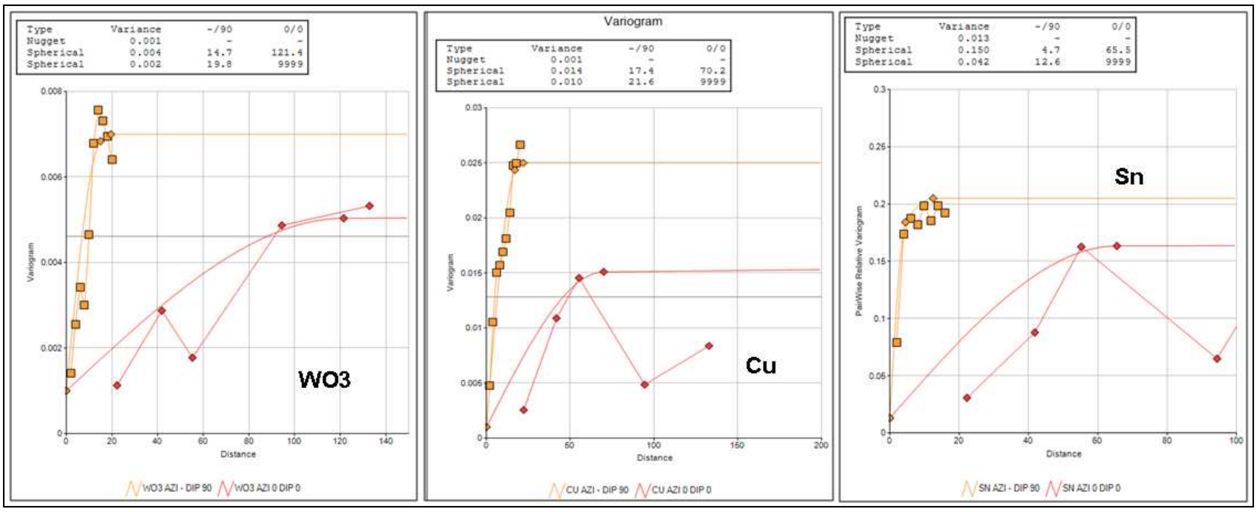


Table 14-24. Model Variogram Parameters – BL2A

Reference Number	Metal	NUGGET	Range 1 (m)			C1	Range 2 (m)			C2
			1	2	3		1	2	3	
1	WO3	0.001	14.7	121.4	121	0	19.8	9999	1	0
2	CU	0.001	17.4	70.2	70.2	0.01	21.6	9999	1	0.01
3	SN	0.013	4.7	65.5	65.5	0.15	12.6	9999	1	0.04
Notes										
	. Direction 1 : Z axis									
	. Directions 2 and 3 : X and Y axes									

Table 14-25. Grade Estimation Parameters – BL2A

Search Distances (m)		Search	Minimum	Minimum
Lateral	Vertical		Samples	Drillholes
50 x 50	10	1st	10	3
100 x 100	20	2nd	10	3
250 x 250	50	3rd	1	1
Notes:				
	. Maximum number of samples used = 20			
	. Maximum number of samples used from any drillhole = 4			
	. For first 3 searches, octant control also applied:			
		. Minimum samples per octant = 2		
		. Maximum samples per octant = 4		
		. Minimum octants required = 5		
	. A further 3 searches then repeated, without octant control			
	. All grades estimated using OK			
	. For validation, grades also estimated using ID (^2) and NN			

Example model sections, showing the estimated WO_3 grade, are shown in Figure 14-17 and Figure 14-18.

Figure 14-17. Section A-A' Through BL2A Block Model

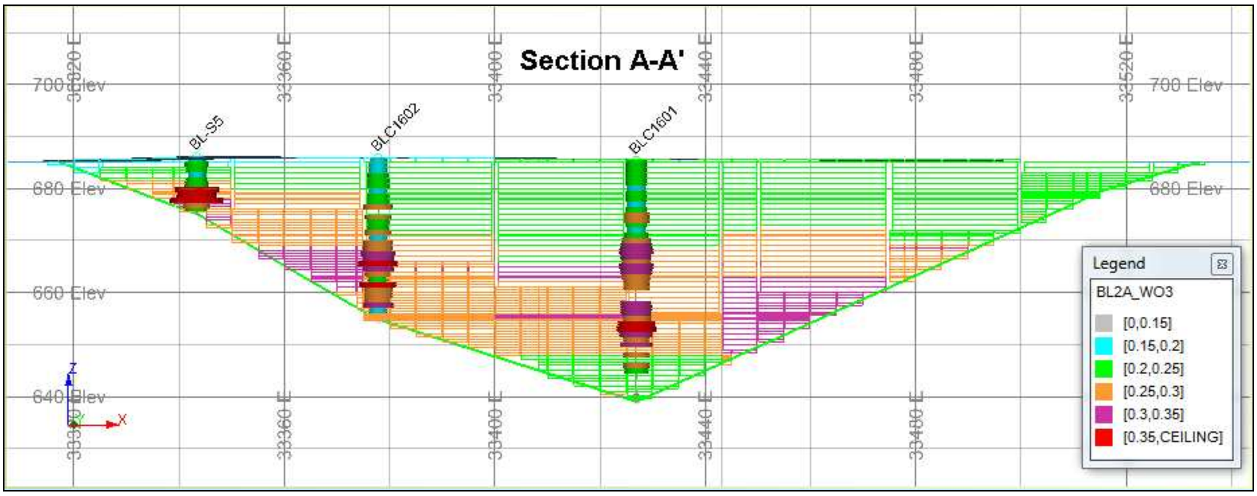
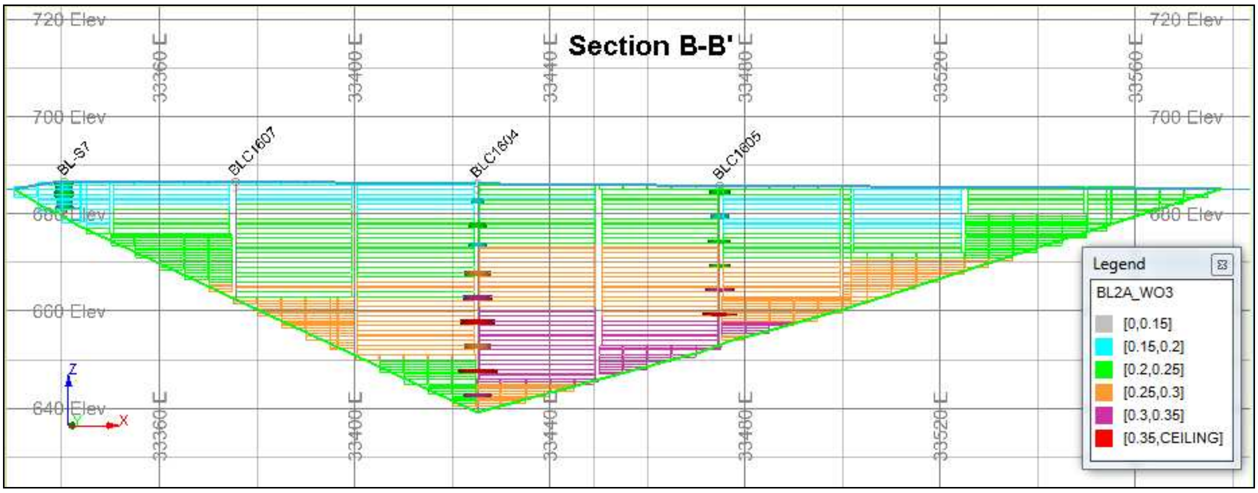


Figure 14-18. Section B-B' Through BL2A Block Model



A global dry density value of 2.5 t/m^3 has been assumed. This value stems from 2016 testwork completed by Laboratorio Nacional de Engenharia Civil (LNEC), as summarised in Table 14-26. Example pictures of the test sample areas and samples taken are shown in Figure 14-19 and Figure 14-20.

Table 14-26. LNEC Density Measurements

Ref. LNEC	Ref. Local	Orientation	Density
			t/m³
5345	B	Vertical	3.09
		Horizontal	3.09
5346	C	Vertical	2.82
		Horizontal	2.81
5347	D	Vertical	2.90
		Horizontal	2.88

Figure 14-19. Positions of LNEC Sample Locations

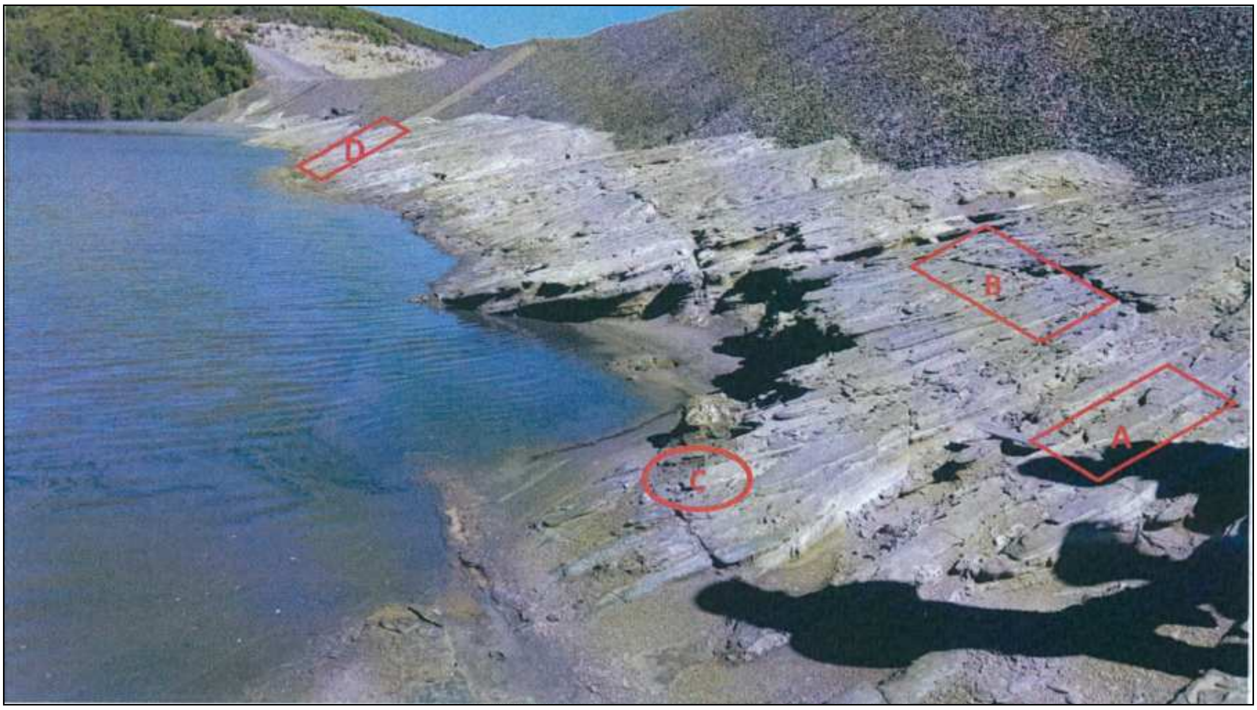


Figure 14-20. Example of Samples Taken from Test Pit D

As part of the validation process, global average grades were compared, as derived from samples and estimated model grades, as summarised in Table 14-27.

Table 14-27. Comparison of Global Averages

		Model Averages		
Field	**Samples**	**OK**	**NN**	**ID**
WO_3	0.24	0.25	0.25	0.25
Cu	0.25	0.23	0.23	0.23
Sn	0.024	0.022	0.022	0.022
Zn	0.39		0.35	0.36

Notes

. OK = ordinary kriging
. NN = nearest neighbour
. ID = inverse-distance weighting (^2)
. All grades shown in %

All of the BL2A resources were classified with an **Inferred** resource category. Reasons for assigning this resources category included:

- Poor coverage of available assay data.
- Limited density measurements.

When the remaining assay data from all of the 2016 drilling samples becomes available, areas which have sufficiently drilled off be classified as **Indicated** resources.

An overall evaluation of the estimated BL2A tailings area is shown in Table 14-28. No cut-off level has been applied in this evaluation, as in any exploitation it is unlikely that different areas of the tailings area can be selectively mined. A bench breakdown of the resources is shown in Table 14-29, and a grade-tonnage table in Table 14-30.

Table 14-28. Evaluation Summary of BL2A

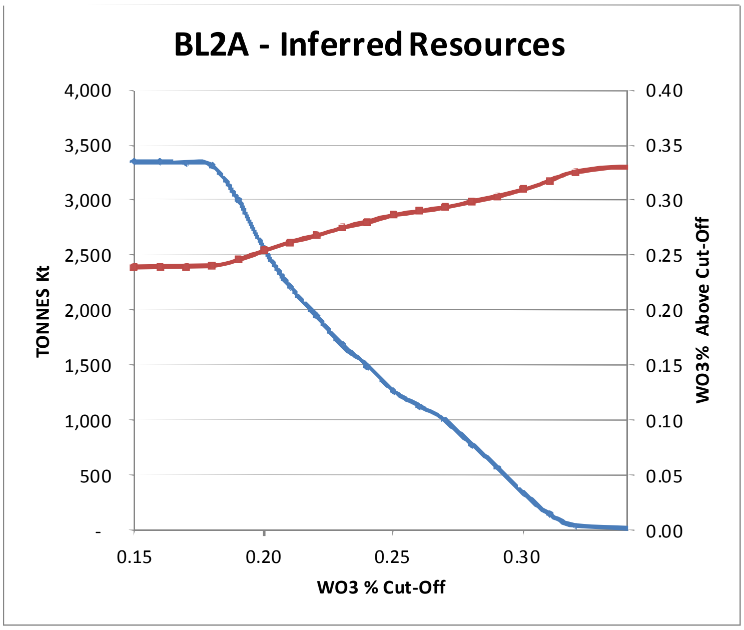
Category	Tonnes	WO3	Cu	Sn	Zn
	Kt	%	%	%	%
3 holes within 50 x 50m	195	0.23	0.22	0.02	0.36
3 holes within 100 x 100m	1,260	0.25	0.22	0.02	0.34
Extrapolated up to 250m	1,892	0.23	0.21	0.02	0.38
Total Inferred Resources	3,347	0.24	0.21	0.02	0.36
Notes					
	.No cut-off applied				
	. All resources above classified as inferred				
	. Assumed density = 2.5 t/m^3				

Table 14-29. Bench Breakdown of BL2A

BENCH	Tonnes	WO_3	Cu	Sn	Zn
mRL	Kt	%	%	%	%
685	123	0.20	0.28	0.03	0.63
680	785	0.20	0.24	0.02	0.52
675	645	0.21	0.19	0.02	0.35
670	523	0.23	0.18	0.02	0.25
665	416	0.27	0.19	0.02	0.22
660	323	0.29	0.24	0.02	0.31
655	234	0.30	0.25	0.02	0.33
650	157	0.30	0.25	0.02	0.33
645	93	0.28	0.23	0.02	0.36
640	42	0.26	0.21	0.02	0.31
635	6	0.26	0.20	0.02	0.27
TOTAL	**3,347**	**0.24**	**0.21**	**0.02**	**0.36**
Notes					
	. All resources above classified as inferred				
	. No cut-off applied				
	. Assumed density = 2.5 t/m^3				

Table 14-30. Grade-Tonnage Table for BL2A

WO_3 Cut-Off %	Tonnes Kt	WO_3 %	Cu %	Sn %	Zn %
0.15	3,347	0.24	0.21	0.02	0.36
0.16	3,347	0.24	0.21	0.02	0.36
0.17	3,343	0.24	0.21	0.02	0.36
0.18	3,314	0.24	0.21	0.02	0.36
0.19	2,999	0.25	0.22	0.02	0.35
0.20	2,562	0.25	0.21	0.02	0.32
0.21	2,216	0.26	0.22	0.02	0.30
0.22	1,950	0.27	0.21	0.02	0.29
0.23	1,686	0.27	0.22	0.02	0.28
0.24	1,495	0.28	0.22	0.02	0.28
0.25	1,263	0.29	0.22	0.02	0.29
0.26	1,129	0.29	0.23	0.02	0.29
0.27	995	0.29	0.23	0.02	0.30
0.28	781	0.30	0.24	0.02	0.31
0.29	572	0.30	0.25	0.02	0.32
0.30	334	0.31	0.25	0.02	0.34
0.31	151	0.32	0.26	0.02	0.35
0.32	41	0.33	0.28	0.02	0.35
0.35	9	0.33	0.27	0.02	0.36



14.4 Resources Overview

An overall summary of resources from both the mine and tailings areas are summarised in Table 14-31.

Table 14-31. Overall Resource Summary

CATEGORY	Tonnes Kt	WO_3 %	WO_3 MTU x1000
Measured	1,951	0.20	393
Indicated	8,076	0.24	1,920
Measured + Indicated	**10,027**	**0.23**	**2,313**

CATEGORY		Tonnes Kt	WO_3 %	WO_3 MTU x1000	Cu %	Sn %
Inferred	Mine	5,158	0.22	1,110		
	BL1 *	1,817	0.29	521	0.30	0.027
	BL2A *	3,347	0.24	802	0.21	0.022
	Total	**10,322**	**0.24**	**2,433**		

Notes

. Measured and Indicated Resources based on a cut-off of 0.12% WO_3

. Inferred Mine resources based on a cut-off of 0.13% WO_3

*** Tailings resources have no cut-off applied**

15 MINERAL RESERVE ESTIMATES

15.1 Reserve Estimation

An example of a level/vein Autocad plan is shown in Figure 15-1. The example shown is for level 3, adjacent to drive D25, and for the top-most AW31 vein. This plan displays several important aspects of the reserve planning process:

- Plotting of face sample data, after use of the Pintas formula to obtain kg/m^2 grades.
- Definition of Pillar blocks in pillars sized 11m or smaller, and which have face samples on at least three sides of the pillars.
- Definition of Virgin blocks, in areas which have only been intersected by development, so laterally the block outwards from face sampled areas. Such blocks may only have one side with face samples and/ or one set of internal samples. Face sample grades are averaged out within defined resource blocks.
- Sometimes originally identified veins may split, so that two potentially mineable veins are apparent, as shown by the cross-hatched H- shaped block in Figure 15-1.

At the mine site, all of the reserves evaluated are also referred to ‘Measured Resources’. There are no Measured Resources determined exclusive to the reserves. However, for CIM classification for the current report, it has been decided to classify the Pillar block material as proved reserves, and Virgin block material as probable reserves. This reflects the lower confidence associated with probable reserves, as compared to proved reserves, as is shown in Figure 15-1, and summarised in Table 15-1.

An overall reserve summary is shown in Table 15-2, with a more detailed breakdown by vein system in Table 15-3. These reserves are part of the mineral resources reported earlier in Table 14-5. Panasqueira is a producing mine, and has been producing continuously for over 50 years. These reserves have been demonstrated to a Feasibility Study level.

In order to convert the resources into reserves, mine plans have been laid out, as shown in Figure 15-1, down to a level of detail such that the individual 11m x 11m pillars are evaluated individually by local face samples. If these pillar evaluation are below cut-off, they are exclude from the reserves. The tonnage calculations are even more detailed, taking into account the mining of the 11m x 11m pillars, to leave the final remnant 3m x 3m pillars.

The key assumptions associated with this reserve estimate are:

- A cut-off grade of 0.12%WO_3 (equivalent to 10 kg/m^2). This cut-off level is supported by breakeven calculations (showing a breakeven cut-off of 0.10%WO_3) based on parameters related to average cost levels that have been estimated for 2017. These parameters include:

Underground mining cost	=	\$12.65/t ore
G & A cost	=	\$5.24/t ore
Processing cost	=	\$4.04/t ore
Mill recovery	=	80.1%
Received metal price	=	\$280/mtu WO_3

- A minimum mining height of 2.2m.

- Applied losses consistent with the mine room-and-pillar mining method, leaving non recoverable 3m x 3m pillars.

- For probable reserves, the largest blocks can be delineated on the same basis as the development grid layout, which is 100m x 80m. This implies a maximum assumed continuity of 50m.

Figure 15-1. Example of Reserve Plan - L3 D25_R02A_AW31

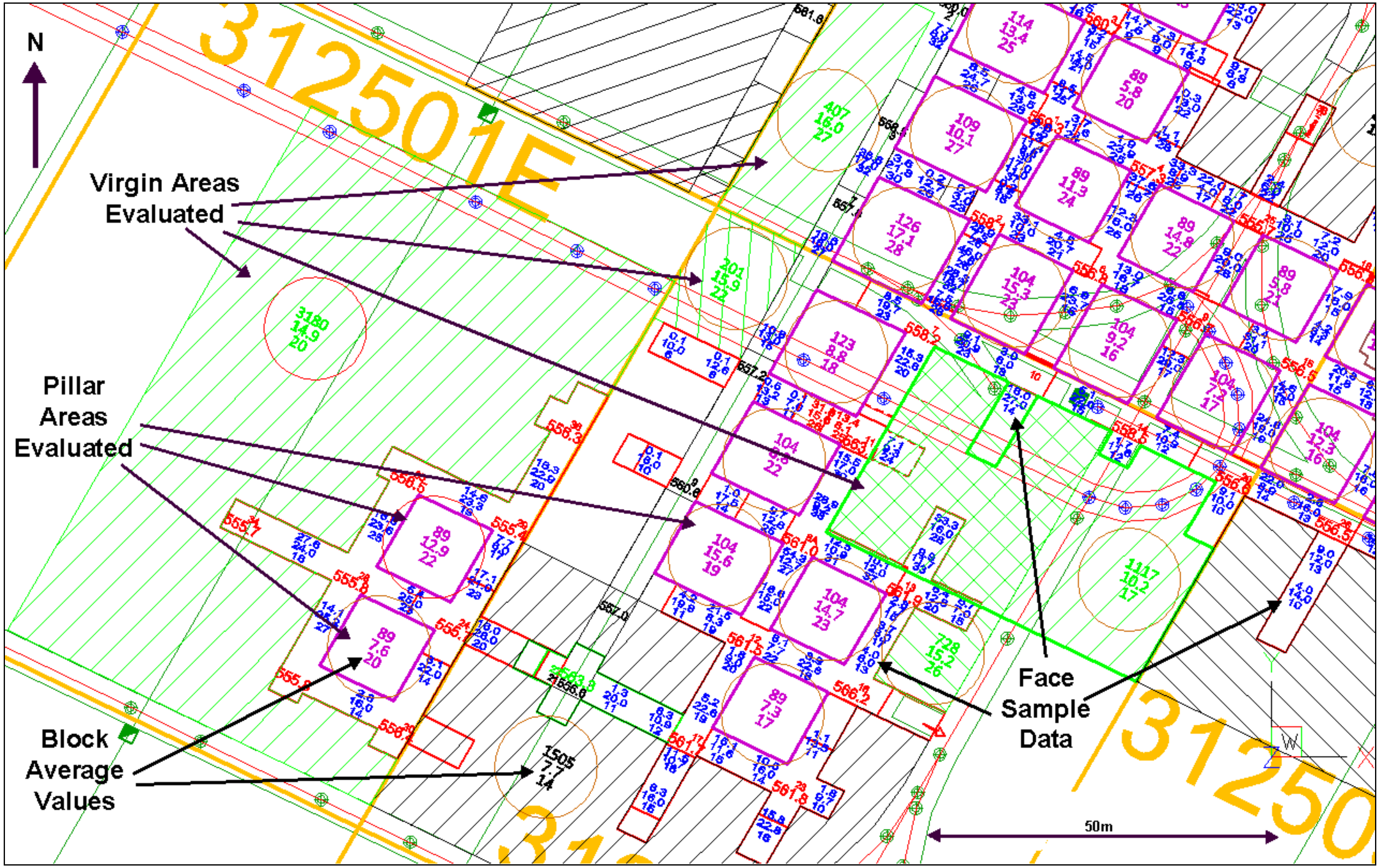


Table 15-1. Reserve Estimation Parameters

Reserve Category	Samples	Criteria	Block Extrapolation	Mining Recovery Factor
Proven	Face Samples on at least 3 sides	Pillars 11m x 11m	not applicable	67.3%
		Pillars 3m x 11m		45%
Probable	Face samples on at least one side or internal	Developed with drive or panels, prior to pillar development	Vein < 30m cm - Max extrapolation 34.5m	84%
			Vein > 30m cm - Max extrapolation 50.5m	

Notes

. Pintas formula used to convert measured wolframite crystal areas into kg/m^2 grades:

$$\text{Grade in kg/m}^2 = \frac{\text{Total area of Wolframite Crystals in mm}^2}{(100 \times \text{Total Length Sampled} \times \text{MEF})}$$

MEF, Mineral Evaluation Factor = 1.5 (1.0 After April 2016)

. Cut-Off used in reserve development:
= 10kg/m2 wolframite
= 0.12% WO_3

. Tonnage Calculations use:
- Average s.g. = 2.8
- Average mining height = 2.2m

. Mining recoveries in accordance with final non-recoverable 3m x 3m pillars

Table 15-2. Reserve Summary
As of 30th September, 2016

	Proven Reserves		*Probable Reserves*		*Total Reserves*	
	Tonnes	WO_3	Tonnes	WO_3	Tonnes	WO_3
Level	Kt	%	Kt	%	Kt	%
0	25	0.19	26	0.17	51	0.18
1	238	0.22	468	0.18	706	0.20
2	216	0.21	251	0.19	468	0.20
3	297	0.24	431	0.19	727	0.21
Total	**775**	**0.22**	**1,176**	**0.19**	**1,951**	**0.20**

Notes

. Cut-off = 0.12% WO_3 (Equivalent to 10 kg/m^2)

. Evaluation based on:
- Face mapping of wolframite exposed areas
- Areas converted to grade using Pinta's formula
- Blocks laid out with stope planning process

. Additional factors applied:
- Minimum thickness = 2.2m

	Virgin Areas	11m x 11m Pillars	11m x 3m Pillars
- Mining recoveries:	84%	67.3%	45%

. Proven reserves are within (11 or 3m) pillars which have been sampled on at least 3 sides

. Probable reserves are within virgin areas which have been sampled on 1-2 sides

Table 15-3. Reserve Breakdown
As of 30th September, 2016

LEVEL	UNIT	Proven Reserves (Pillars) Tonnes Kt	WO_3 %	Wolf'te kg/m²	Av'age Thick cm	Probable Reserves (Virgin Areas) Tonnes Kt	WO_3 %	Wolf'te kg/m²	Av'age Thick cm	TOTAL Tonnes Kt	WO_3 %	WO_3 mtu
L0	AW 04					11	0.13	10.9	17	11	0.13	1,452
	AW 06	25	0.19	15.4	35	15	0.19	16.0	45	40	0.19	7,539
	TOTAL	25	0.19	15.4	35	26	0.17	13.9	33	51	0.18	8,991
L1	AW 10	6	0.16	13.5	30	9	0.16	13.0	22	16	0.16	2,506
	AW 11	31	0.17	13.7	32	7	0.13	10.7	28	38	0.16	6,075
	AW 12	21	0.18	14.5	29	42	0.18	14.9	30	63	0.18	11,349
	AW 13	6	0.16	12.9	35	16	0.13	10.7	29	21	0.14	2,942
	AW 14	47	0.28	22.6	41	108	0.18	14.9	31	155	0.21	32,509
	AW 15	38	0.19	15.8	27	213	0.18	14.7	23	250	0.18	45,355
	AW 16	23	0.20	16.3	28					23	0.20	4,649
	AW 17	66	0.25	20.7	27	74	0.23	18.7	23	139	0.24	33,249
	AW 18	1	0.26	21.3	33	0	0.00	0.0	0	1	0.26	144
	TOTAL	238	0.22	18.0	31	468	0.18	15.2	26	706	0.20	138,779
L2	AW 20	16	0.17	14.2	32	3	0.14	11.9	26	19	0.17	3,170
	AW21	5	0.20	16.3	34	16	0.24	20.0	33	21	0.23	4,833
	AW22	10	0.19	15.2	26	10	0.14	11.9	33	20	0.17	3,364
	AW 23	82	0.18	14.8	30	61	0.17	13.9	27	143	0.18	25,090
	AW 24	41	0.21	17.5	39	55	0.20	16.0	34	96	0.20	19,381
	AW 25	14	0.19	15.5	22					14	0.19	2,542
	AW 26	49	0.28	23.3	32	106	0.19	15.2	34	156	0.22	33,704
	TOTAL	216	0.21	17.3	31	251	0.19	15.2	32	468	0.20	92,084
L3	AW 30	2	0.23	18.7	19					2	0.23	377
	AW 31	19	0.17	14.2	25	37	0.16	12.8	23	56	0.16	8,998
	AW 32	60	0.24	19.7	33	151	0.16	13.5	25	211	0.19	39,184
	AW 33	95	0.23	18.8	39	28	0.20	16.5	43	124	0.22	27,591
	AW 34	4	0.34	27.9	83	14	0.39	31.8	73	17	0.38	6,510
	AW 35	49	0.28	23.2	38	27	0.22	17.7	44	76	0.26	19,664
	AW 36	42	0.27	21.8	36	82	0.22	17.8	36	125	0.23	29,022
	AW 37	21	0.19	15.7	20	44	0.17	13.8	20	65	0.18	11,476
	AW 39	5	0.27	22.4	31	47	0.19	15.2	37	52	0.19	10,101
	TOTAL	297	0.24	19.8	35	431	0.19	15.5	32	727	0.21	152,923
GRAND TOTAL		775	0.22	18.4	33	1,176	0.19	15.3	29	1,951	0.20	392,777

Notes

. Cut-off = 0.12% WO_3 (Equivalent to 10 kg/m^2)

15.2 Reconciliation

One way of assessing the reserve methodology is consider reconciliation results. A summary of the monthly reconciliation results, from July 2014 to September 2016, is shown in Table 15-4.

The grades reported by the Mine are monthly compared with the Plant feed grades. The Plant feed grades are determined from the various grade measurements analysed in the laboratory, and then combined in a metallurgical balance. By this is calculated the Mine Call Factor (MCF), corresponding to Plant grade/Mine grade. This means that a MCF of 150 indicates an actual feed grade of 15 kg/m^2, corresponds to a mine grade of only 10 kg/ m^2.

Observations from these results include:

- The consistently high MCF factor indicates that current methodology appears to be conservative in terms of assessing grade. This is why the MEF factor (used in the application of the Pintas formula for the face assay calculations) has been reduced from 1.5 to 1.0 from April 2016 onwards.

- The block factor (BF) factors are mostly below 100, indicating that grades from probable (virgin) reserves tend to be lower when developed into proven (pillar) reserves.

- The combined effect of MCF x BF do help to balance out the overall differences.

These results support the reserve estimation results presented, but also indicate a need for on-going analysis and improvement of the various factors employed in the reserve estimation process.

Table 15-4. Reconciliation Summary – July 2014 – September 2016

	Mine			Plant					
Month	Trammed	WO3	WO3	ROM	WO3	WO3	MCF	BF	MCF X BF
	Kt	%	t	Kt	%	t	%	%	
Jul-14	75.1	0.08	69	76.9	0.14	110	159	83	133
Aug-14	23.1	0.09	16	20.7	0.14	28	180	103	185
Sep-14	80.2	0.09	74	76.4	0.13	102	138	100	138
Oct-14	79.1	0.08	62	76.0	0.13	99	159	65	103
Nov-14	69.9	0.08	56	70.9	0.12	87	154	81	125
Dec-14	61.1	0.07	38	55.9	0.12	68	179	**62**	111
Jan-15	70.8	0.07	49	65.7	0.12	76	156	91	143
Feb-15	60.0	0.08	49	58.7	0.12	72	148	96	142
Mar-15	60.6	0.08	44	59.0	0.13	74	167	71	119
Apr-15	49.8	0.07	40	48.4	0.13	64	158	69	109
May-15	46.8	0.08	44	45.7	0.13	58	131	91	119
Jun-15	43.5	0.10	53	46.1	0.17	79	148	109	162
Jul-15	49.1	0.10	60	48.7	0.18	86	143	100	143
Aug-15	0.0	0.00	0	0.0	0.00	0	0	0	0
Sep-15	33.8	0.14	49	31.7	0.20	63	129	107	138
Oct-15	40.8	0.12	57	40.8	0.16	65	114	132	150
Nov-15	42.2	0.08	35	43.0	0.14	60	171	96	164
Dec-15	26.0	0.08	23	29.8	0.16	47	204	86	175
Jan-16	40.6	0.09	42	41.2	0.15	60	143	107	153
Feb-16	47.4	0.06	34	45.0	0.15	66	194	66	128
Mar-16	55.6	0.06	40	60.2	0.12	70	175	68	119
Apr-16	50.8	0.09	50	56.6	0.12	67	134	108	145
May-16	53.4	0.09	52	56.3	0.13	72	138	138	190
Jun-16	59.3	0.09	56	60.0	0.13	80	143	58	83
Jul-16	55.8	0.09	50	55.3	0.14	78	156	149	232
Aug-16	49.2	0.13	71	49.7	0.15	74	104	130	136
Sep-16	63.6	0.10	68	64.0	0.13	86	126	92	116
Total	**1400.6**	**0.09**	**1281.3**	**1382.6**	**0.137**	**1889.9**	**146**	**91**	**133**

Notes

. MCF call factor = (Plant Contained WO3t)/(Mine Reported WO3t)

. BF block factor = kg from proven reserves (actual = pillars)/
kg from former probable reserves (virgin)

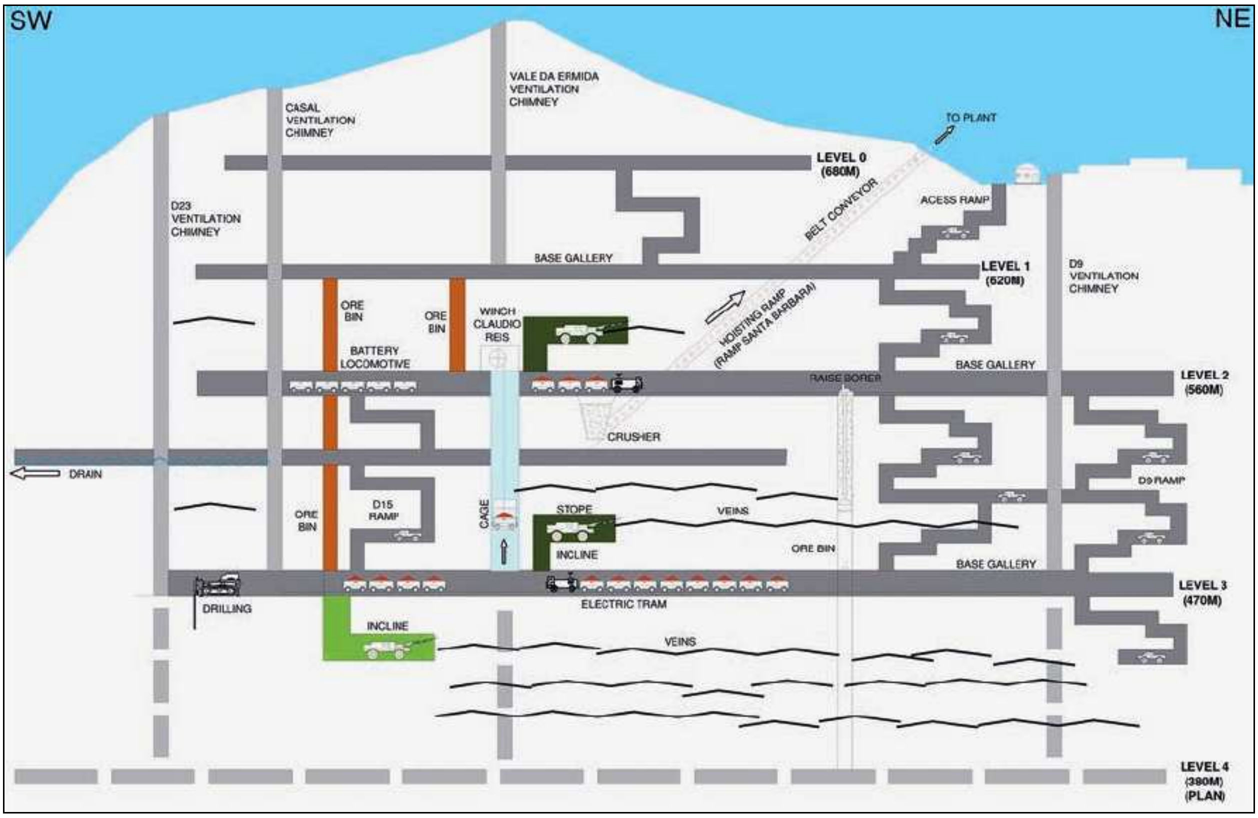
16 MINING METHODS

16.1 Mining Method

The mining method used at Panasqueira is a mechanized room and pillar method that has evolved during the past 50 years. The method is possible due to the sub-horizontal nature of the quartz veins and the very competent host rock. Underground support is relatively rare. When argillaceous faults are encountered, concrete reinforcement is used, with arches either side of such faults in the main drives. Rock bolts, screen and shotcrete are used only when large openings are excavated for underground infrastructure.

The stoping process begins with ramps being driven to access the mineralized veins, in areas where diamond drilling has indicated quartz intersections equal to or above 18cm (indicated resources). When a vein is accessed, it is blocked out on a basic 100m by 100m pattern. Ore passes of 1.8 metre diameter raises are generally bored near drive/panel intersections, and allow ore to be dropped down to main haulage levels. Chutes are installed in the bottom of the orepasses to facilitate the loading of the wagons. A schematic long section of the mine is shown in Figure 16-1, depicting the ore transfer system.

Figure 16-1. Schematic Long Section of Ore Transfer System

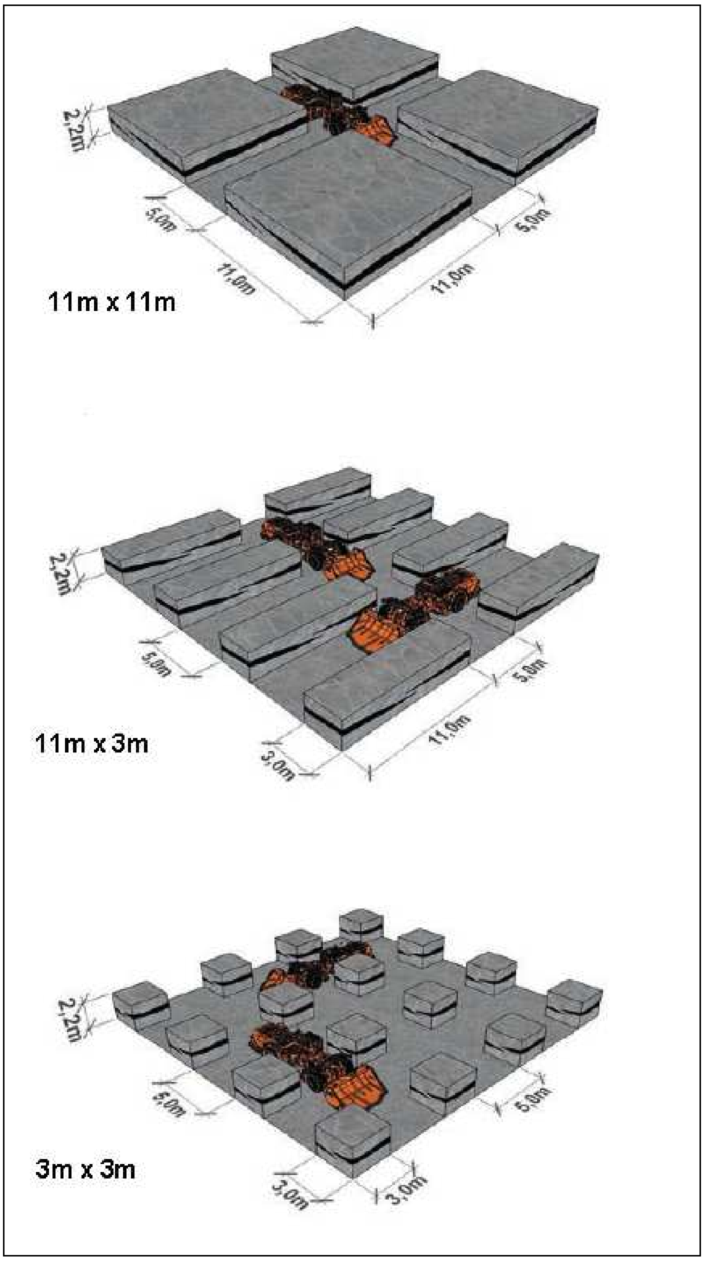


Equipment that is used for mine development and production includes electric hydraulic single boom jumbo drills, 3.5 cubic yard diesel LHDs, low profile 6 and 6.8 tonne LHDs and a raise borer. The target height of the stopes is nominally 2.1 m but increased to 2.3 m in areas where ore bearing veins are more variable in their dip, strike or thickness. The average height achieved is generally 2.2m. With an average in-situ mineralized zone thickness of 25cm, the applied planned dilution 880% i.e. 8 x times as much waste has to be mined within the 2.2m minimum height. No additional dilution factors are added.

As the veins are intersected with development, face sampling is used to verify mineralisation and indicate how the stopes can be developed and expanded so as to best follow the ore. When sufficient ore has been delineated, additional 5m wide openings are excavated between and parallel to the drives and panels to create a series of regular series of 11m by 11m pillars. The pillars are then reduced in size in two passes of 5 m excavations; firstly to 11m by 3m, and then to 3m by 3m to complete the extraction sequence, as depicted in Figure 16-2. This process results in an overall extraction rate of 84%, as the 3m by 3m pillars are not recovered.

Precise survey control is maintained so that all final pillars are aligned vertically on the different levels and experience has shown that the stopes will usually begin to collapse about 4 or 5 months after completion (as shown in Figure 16-6), which gives plenty of time to “sweep” the fines from the floor. A small track mounted excavator is used for a final stope clean-up (sweeping) to recover the wolframite rich material that accumulates on the floor during normal mucking operations. This excavator can be operated by remote control, and is also used to help remove mined pillar material, when the final 3m x 3m pillars are being created, as shown in Figure 16-5.

Figure 16-2. Ore Extraction Sequence – Pillar Sizes



Stope drilling is carried out by electric hydraulic single boom jumbos, as shown in Figure 16-3. Blast holes are drilled 2.4 metres in length and are 43 mm in diameter. Fan cuts or V-cuts are drilled, with two horizontal rows generally just above and below the vein, so as to minimize shattering the brittle wolframite. Regular back holes and lifters are then drilled above or below the cut slice, with variations up or down so as to follow the vertical variation in the vein.

ANFO is loaded pneumatically into the blast holes and delayed non-electric detonators along with small primers are used for blasting. Blasting occurs at midnight using a central blast cable, the mine then ventilates for all of the third shift. Each blasted face produces about 55 tonnes of rock and good drillers can complete more than five faces in a shift. After the blast the muck pile is washed down and the back is scaled, ore is loaded and hauled by 6 or 6.8 tonne diesel LHDs from the heading to the orepasses, as shown in Figure 16-4.

Once the blasted rock has been removed, then the vein can be evaluated, using face sampling as described in Section 11.2. These sample results, together with geological mapping of the veins' variations, and problems that can arise with local faulting, are all taken into account for expansion of the stopes on the 11m x 11m pattern. When the final limits of the stopes are established then the final extraction takes place first with 11m by 3m and after that with 3m by 3m pillars created from the perimeter retreating to the access ramp. Occasionally some pillars cannot be totally extracted due to hazardous conditions created by mining, or for protection purposes, such infrastructure (shafts, conveyor belt, and drives and panels). Subsidence measurements are also used in this assessment, with approximately 6 points per stope, which are measured once a month, In these instances some of the reserve is lost. In the past there have been attempts at recovering pillars using either jacks or timber packs for support, but these exercises have proved expensive, labour intensive and hazardous. For this reason there are no pillar recovery operations currently in use at the mine.

Figure 16-3. Jumbo Stope Production Drilling

Figure 16-4. Low Profile LHD

Figure 16-5. Small Excavator For Final Clean-Up

Figure 16-6. Ultimate Collapse of Final 3m x 3m Pillar

Within any large 100m x 100m mining block, veins are generally stoped out from the top-down. A minimum 3m pillar is always left between one stope and the stope below, when veins converge close together.

The mine currently has four main levels, which are all connected by ramps. The main water drainage level is the 530m, which is between level 2 (560m) and level 3 (470m). The underground crusher is below level 2 (530mRL). Ore from above Level 2 is fed from ore passes into 4t self-tipping rail cars and hauled by electric and diesel trolley locomotives to the ore pass on Level 2 that feeds the underground crusher. Ore from below Level 2 is similarly fed to and trammed on Level 3 to the winze where the rail cars are individually hoisted to Level 2. From this shaft the cars are trammed to where they self-dump into the crusher ore pass. The underground crusher feeds ore on the conveyor belt in the Santa Barbara hoisting ramp.

16.2 Mine Production

Over 2016, the mine has on average been producing at approximately 54,000 tpm being mined, employing 159 people for direct mining operations. The current mining plan of BTW is to sustain this mining and processing rate for 2017.

The mine operates on two shifts (7am to 3pm and 3pm to 11pm), five days per week. Due to the extensive workings in the mine the effective time of each shift is reduced to about six hours due to the time that it takes for workers to reach their workplaces, as well as the lunch time period. There are several four wheel drive pickup trucks that are used to transport men and materials to the underground headings, these vehicles are also used by supervisors to visit the working places during the operating shifts. One vehicle is on permanent standby on surface for use during emergencies.

Drilling is performed primarily on the first shift and then completed on the afternoon shift prior to loading and blasting at midnight. There is a specific loading and blasting crew. Each face generates about 55 tonnes of rock. There have to be enough faces in close enough to facilitate one jumbo drilling and blasting as many faces as possible without long moves.

The mine is ventilated from midnight to the beginning of day shift at 7 am.

A summary of the key parameters associated the mining method at Panasqueira includes:

- **Mining Method.** Mechanised room-and-pillar.

- **Production Rate.** 656 Kt of ore planned for 2017, which gives a mine life of approximately 3 years based on current reserve levels. However, in the opinion of the author, with conversion of inferred resources and on-going exploration, this mine life estimate is very conservative. The mine has historically always operated with a reserve level representing only a few years of production, despite having operated for over a hundred years.

- **Mining Unit Dimensions.** All galleries in ore are generally 5m wide x 2.2m high. A regular grid system of 11m x 11m pillars is used, which subsequent mining reduces to 3m x 3m unrecoverable pillars.

- **Dilution.** With the average in-situ mineralised bed thickness of 25cm, the application of a 2.2m minimum mining height gives a 880% dilution factor.

- **Losses.** Based on the room and pillar dimensions, a mining recovery of 84% is applied to overall large virgin blocks. A mining recovery of 67.3% dilution is applied to the evaluation of 11m x 11m pillars, and a mining recovery of 45% is applied to the evaluation of 11m x 3m pillars.

17 RECOVERY METHODS

17.1 Introduction

Panasqueira produces very high grade concentrates, which are almost pure wolframite, facilitated by the coarse crystalline character of the ore. Most of the sulphide minerals occur as macroscopic crystals within the quartz gangue of the vein. In addition to the sulphides, quartz exists as coarse crystals within vugs in the veins and muscovite in 2cm halos above and below the veins. Wolframite, cassiterite and chalcopyrite are the economic minerals to be recovered as concentrates. Pyrite, arsenopyrite and siderite are the heavy minerals that are removed from concentrates.

A general view of the plant is shown in Figure 17-1. A simplified mill flow sheet is shown in Figure 17-2, and more detailed flowsheet is shown in Figure 17-3.

Figure 17-1. General View of the Panasqueira Mine Plant Installation

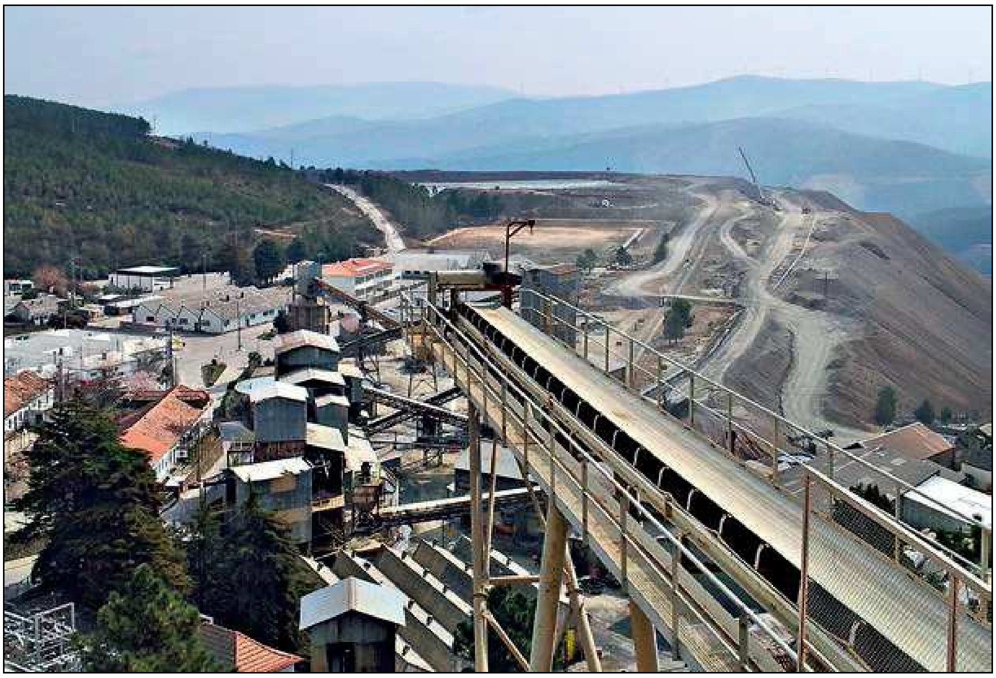


Figure 17-2. Simplified Mill Flowsheet

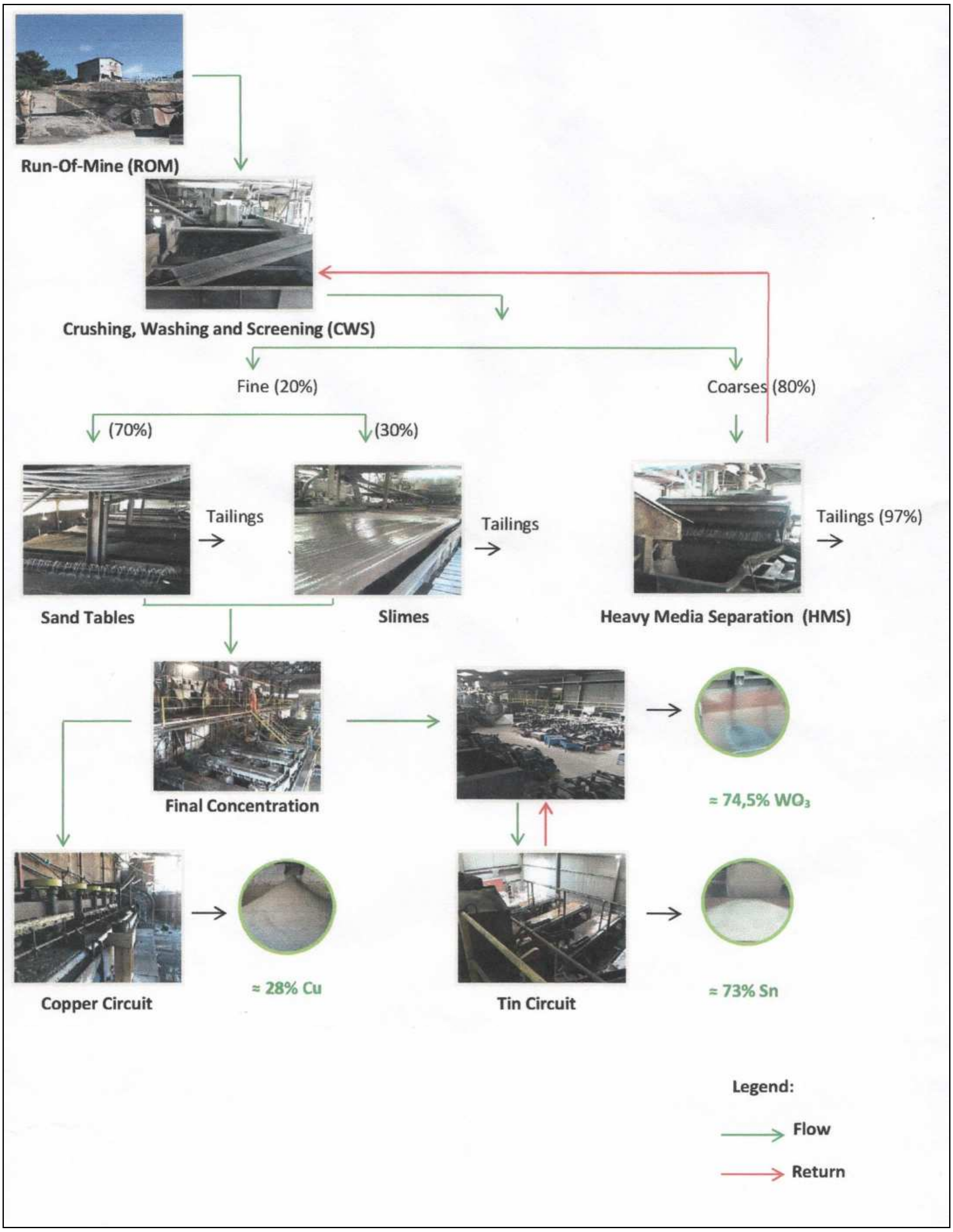
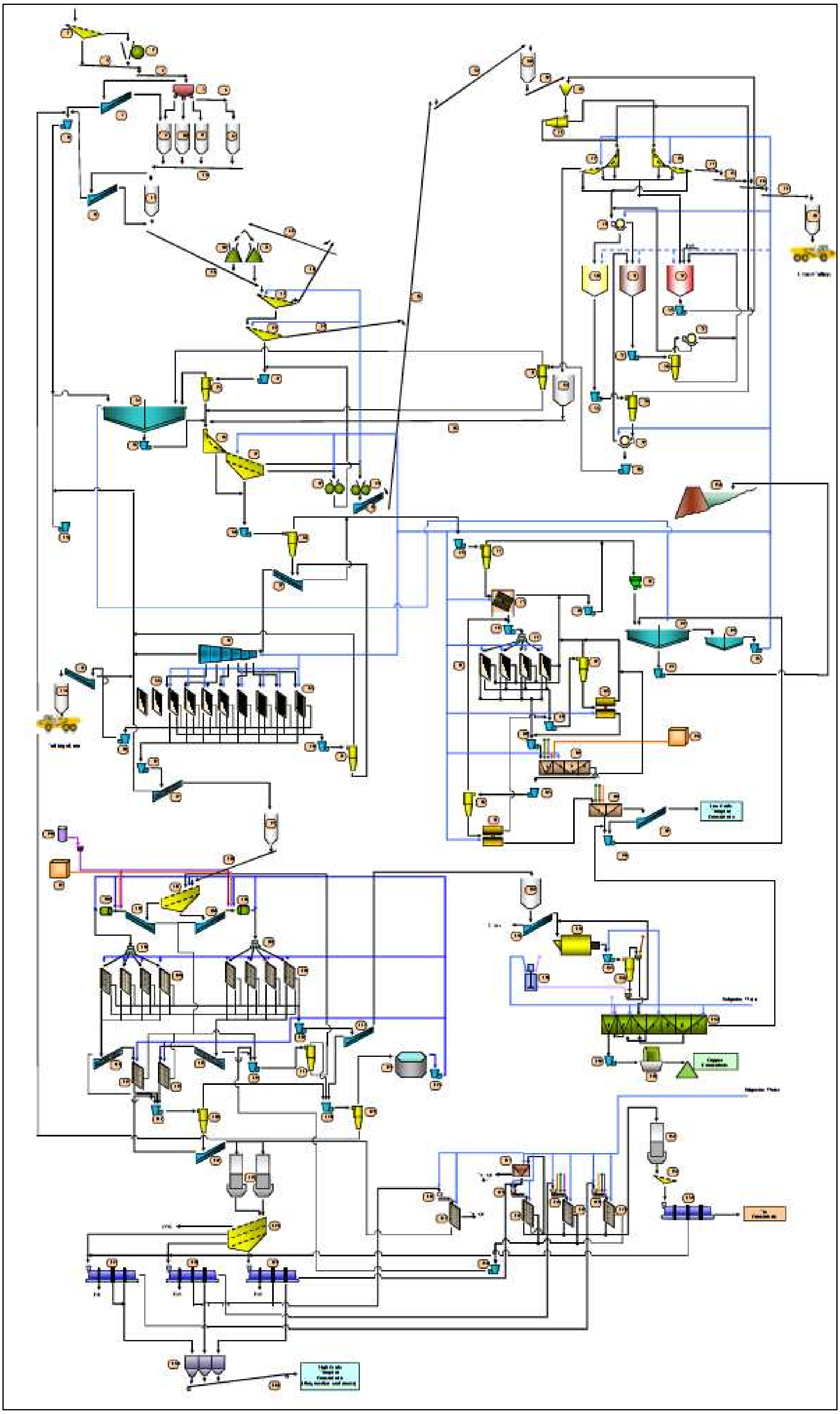


Figure 17-3. Mill Flowsheet



17.2 Wolframite Recovery

The brittle characteristic of wolframite makes recovery difficult from the mine, but assists the plant, as most of the wolframite is easily liberated with only two stages of crushing. Being very brittle, it goes preferentially into the fines making it essential to clean the stoped area of all possible fines upon completion of mining."

The underground jaw crusher delivers <100mm ROM, containing about 11% of vein material rock to four storage bins via the 1,203m long 17% Santa Barbara conveyor belt (Figure 17-4). Vibrating feeders discharge ore to a conveyor which feeds the crushing plant at a rate of about 160tph. Plus 25mm material is produced by primary screens and crushed in closed circuit by one of two short head cone crushers (Figure 17-5).

Figure 17-4. ROM Ore Entering the Plant by Conveyor Belt

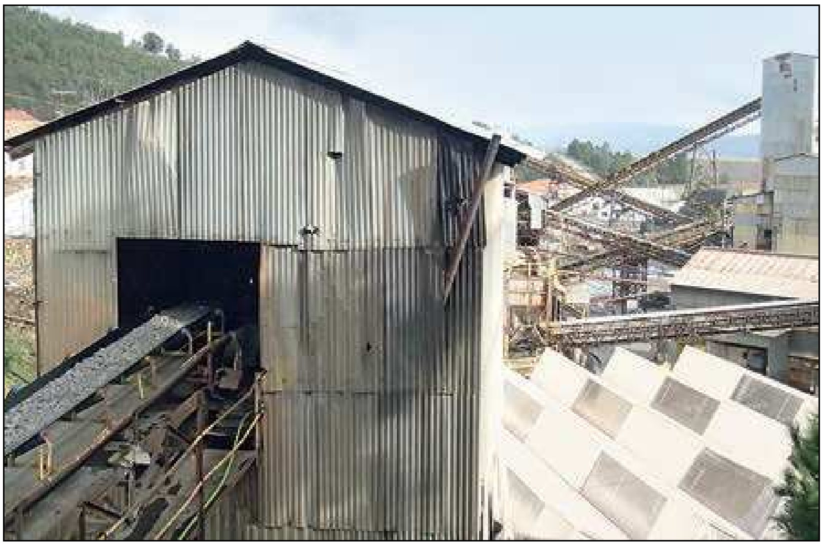


Figure 17-5. Symons Short Head Cone Crushers

All screening done in the plant is aided by copious amounts of spray water. Secondary screens, remove minus 0.8mm material from the minus 25mm Heavy Media Separation (HMS) feed (Figure 17-6).

Figure 17-6. Screen Discharge for Concentrate and Tailings in HMS

The HMS process uses a hydrocyclone for the separation, with a dense slurry media maintained at an SG of 2.7 using very fine magnetic ferrosilicon (FeSi); the characteristics of the Panasqueira ore lend themselves perfectly to the use of HMS to remove approximately 80% of the ore which has no tungsten content.

The tailings from the HMS circuit consists of +0.5 -25 mm material which is washed and screened from the cyclone overflow by sieve bends and shaking screens; this material represents about 80% of the feed by weight containing about 10% of the WO3 feed content.

One of the roll crushers is dedicated to +3 –5 mm material from the HMS concentrate and this material is re-circulated to the HMS plant. The minus 3mm material is treated in an hydraulic classifier prior to concentration by gravity shaking tables (Figure 17-7).

Figure 17-7. Coarse Gravity Shaking Table

The table concentration eliminates all the gangue minerals, namely quartz and silicates, whilst the concentrates contain the dense minerals (pre-concentrate) which include, as well as wolframite, all the sulphides (pyrite, chalcopyrite, arsenopyrite, sphalerite) plus cassiterite and siderite.

In the slimes treatment section, tables are used to concentrate the possible finest particles and have produced satisfactory results, as almost all the wolframite in the slimes tailings is finer than 150µ.

The pre-concentrate is conditioned with sulphuric acid, Nafta and Gasoil prior to passing over the table where the sulphide minerals float to the tailings, even 2mm sulphide grains will float down the table. These table tailings then become feed for the copper circuit (Figure 17-8).

Figure 17-8. Final Concentration on James Tables, to Remove Sulphides

This concentrate is recovered using differential flotation in Denver type cells. The tailings are ground in a ball mill to 60% minus 200 mesh. Lime is fed to this mill to increase the feed pulp pH to 11.5. It has been possible to produce concentrates with +28% Cu limiting the content of As to less than 2%. The main sulphide present in the vein material being arsenopyrite, it is eliminated in the tailings, which present arsenic contents up to 30%, from the copper circuit.

The table concentrates without sulphides are dried and screened to prepare three sized lots (+20, -20+60, and –60 mesh); each size fraction is treated separately using dry high-intensity cross-belt magnetic separators to produce the high grade wolframite concentrate, non-magnetic cassiterite and magnetic siderite (with a small amount of ferrosilicon) , as shown in Figure 17-9.

Plant recoveries over 2016 has averaged 80.3%. Over 89% of the recovered mtus are typically in the high grade concentrate averaging over 75% WO_3 with the remainder from the fine lower grade concentrate, which averages above 74% WO_3. The bulk of the losses of WO_3 occur in the coarse tails.

The Wolframite concentrates are packed in one ton bags which are transported in containers by road from the mine to the port at Lisbon, whence they are shipped to their destination. Water from the thickener overflow is retreated in the mill.

The water treatment unit is located downhill at Salgueira, as described in Section 18.

Figure 17-9. Dry High Intensity Magnetic Separation

(Magnetic Siderite and Wolframite/Non-magnetic Cassiterite and Pyrite)

17.3 Copper Circuit

The copper circuit typically treats about 5,000 tonnes per year. It incorporates a small ball mill and 7 flotation cells.

From January-December 2016, approximately 380t of copper concentrate were produced, with a grade of approximately 27% Cu. The average arsenic grade in the copper tailings is approximately 20%, with a copper circuit recovery of 77%. The copper plant tailings contain much of the arsenopyrite present in the ore, and it might be possible to contain this material by itself thus reducing the discharge of arsenic to the larger tailings pond. The copper circuit is one of several modules within the plant that operates when there is sufficient material to feed it for a prolonged period of time. The copper plant usually starts late at night when reduced power rates permit more efficient ball mill start-up.

17.4 Tin Circuit

In the tin circuit, the siderite is discarded whilst the cassiterite goes to further treatment using tables to eliminate the finer gangue particles, and flotation in a mechanical cell to eliminate the remaining finer sulphide particles. Approximately 70t of tin concentrate are produced per year, with a grade of about 74% Sn.

18 PROJECT INFRASTRUCTURE

18.1 Summary

The infrastructure for the mine is very well established, mining having been undertaken at Panasqueira and Barroca Grande for over 100 years. A summary of the main infrastructure requirements for the mine includes:

- **Underground Infrastructure.** This includes the underground jaw crusher below level 2 (560mRL), the belt conveyor in the Rampa Santa Barbara which takes ore from the crusher up to surface, the main access ramp to surface for mechanised equipment and personnel, four main ventilation raises, the internal hoist and shaft which connects level 2 and level 3 (470mRL), and the train level haulage systems on levels 2 and 3.

- **Coarse Waste Dump.** As described in Section 20.6, the Barroca Grande waste dump lies adjacent to and above Bodelhão Creek that flows south-easterly to the Rio Zêzere and comprises primarily HMS tailings (gravel), waste rock and sand tails, all of which are very benign and possibly marketable as aggregate. This waste dump is approximately 1km to the east of the mill. A belt conveyor takes waste from the mill out to a discharge point just to the west of the waste dump area. The waste material is then transported and discharged by truck onto the waste dump, or use for the building up of embankments in the construction of the new TMF 2B fine tailings disposal area.

- **Fine Tailings Disposal areas (TMFs).** There are two historic disposal areas – TMF 1, which is completely finished and TMF 2A, into which fine tailings is currently being discharged. TMF 2A has approximately 4 years' capacity, and so a new disposal area, TMF 2B, is under construction. A pipeline takes fine tailings hydraulically from the mill out to the disposal area, along the same access roadway as the coarse waste conveyor.

- **Electrical Power Distribution System**. The electrical power is supplied by EDP at 60kV, into a sub-station at the Rio site, from where it is transported to Barroca Grande sub-station by the company 40kV line.

- **Water Systems**. There are two main water pipe systems: one for process/mine water and one for potable water. The process/mine water pipeline feeds the Salgueira water treatment plant.

18.2 Electrical Supply

The electrical power is supplied by EDP at 60kV, into a sub-station at the Rio site, from where it is transported to Barroca Grande sub-station by the company 40kV line. There are 6 main transformers at this sub-station which transform all incoming power to 3kV which is the feed voltage up to the consuming centres. The majority of motors are of 400V. Contracted power is 3750 kW (2006 NI 43-101 report).

Electricity typically accounts for just over 9.3% of operating cost. Power price per kWh increased substantially since 2007 to €2.12/t ore, making the monitoring and control of large consumers even more important.

The low electricity consumption is mainly the result of not having to grind the coarsely liberated ore. A large degree of natural ventilation also occurs in the mine. Power is adequately reported on and controlled by monthly reporting.

The mine has modern telecommunication, with touch-tone dialling, internet connected data communication, international cellular coverage and a mine radio communication system.

18.3 Transportation

Easy all-weather road access is provided by a combination of two-lane and four-lane divided highways, for approximately 260km from the capital Lisbon to the town of Fundão. The mine is located approximately 35km from Fundão along a two-lane windy, paved road to Barroca Grande, the community that supports the mine and contains the Barroca Grande plant and offices. Government maintained paved roads or dirt roads maintained by the company can access most areas of the concession.

All essential services such as food and lodging are available from the numerous nearby towns and villages including all heavy-duty equipment. Alternative access to the mine is via two and four lane highways to Porto, a port city located approximately 200km to the north-west. The railway from Lisbon reaches the town of Fundão.

The wolframite concentrates are transported in containers by road from the mine to the port at Lisbon whence they are shipped to their destination

18.4 Water

The company holds the licence for using water resources.

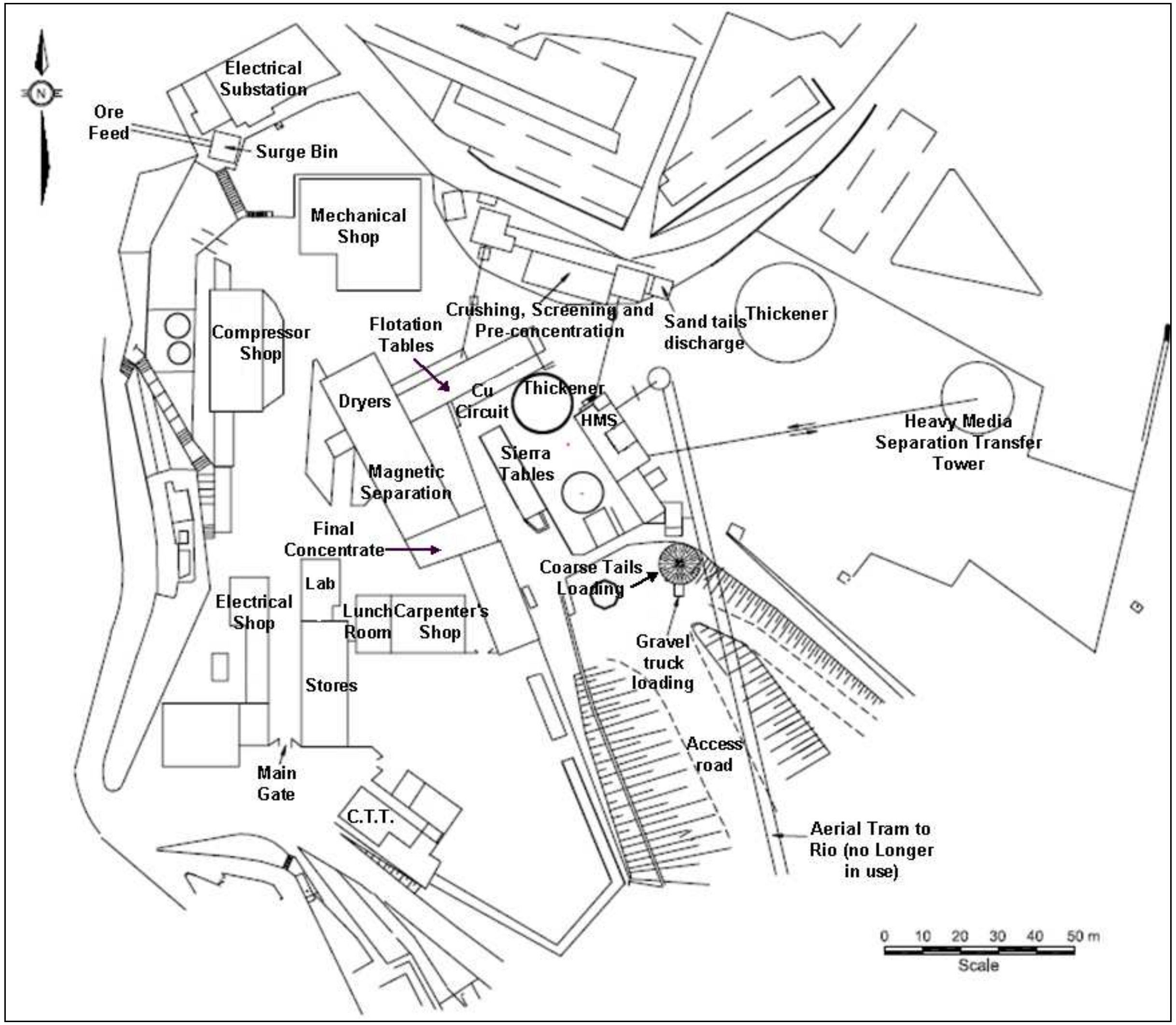
The concession contains abundant water sufficient to continue to support future mining operations. The mine is situated in a natural cirque at the top of a valley which ends at the Rio Zêzere (Zêzere River). Numerous unnamed creeks and underground water sources feed the mine and drain into the small Bodelhão Creek that flows south-easterly to the Rio Zêzere. The Rio Zêzere is the major source of water for the city of Lisbon, so the mine conducts constant water monitoring.

18.5 Metallurgical Plant and Workshops

The Metallurgical Plant site plan is shown in Figure 18-1.

The maintenance workshops are quite spread out geographically.

Figure 18-1. Plant Site Plan



18.6 On-site laboratory

Beralt has an on-site laboratory to test tungsten concentrate grades as well as contaminant levels in the water quality samples. The laboratory uses a gravimetric analysis to assess tungsten oxide (WO3) and total sulphur content. It uses a titration analysis to assess the grade of by-products (tin, copper, arsenic, and phosphorus). The analyses performed by BTW on site are comparable with the analysis performed by BTW main customer's laboratory.

18.7 Ancillary Activities

In previous times the mine has also sold some of their waste coarse material for local construction purposes.

The quantity of copper and tin concentrate streams is so small that any change in commodity price has minimal effect on overall profitability. The principle reason to operate these circuits is to keep tin and copper levels lower in the tailings pond.

The mine operates a crystal shop at the main office installations. The shop is operated by one man and supervised by a Geologist. All the Staff team who work underground have the responsibility to bring crystals to the shop. Any proceeds from the shop are split between the company and the operator. The operator distributes the benefits of the sales to help support the local miners' families.

BTW owns many bunkhouses, shops and houses around the plant site of Barocca Grande. They attempted to sell some of the multi-family dwellings but had difficulty in finding a way to transfer the title of individual units within the multi-unit buildings.

19 MARKET STUDIES AND CONTRACTS

As described in Section 17.2, over 89% of the recovered wolframite is typically in the high grade concentrate averaging over 75% WO_3, with the remainder being sold in the form of a fine lower grade concentrate, which averages above 74% WO_3.

The QP can confirm that the rates and conditions of Panasqueira contracts are within industry norms. Two direct contracts are already in place for 2017. Further information on these contracts cannot be stated here, as this would be in breach of the terms of each contract and could be detrimental to the future of the mine's business when contract renewal is being negotiated.

The long-term price forecast of Almonty supports a received price assumption of $280/mtu, which supports the cut-off grade of 0.12% WO_3 used in the current reserve estimation.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT

20.1 Environmental Audit

In December 2005 a comprehensive Environmental Audit was undertaken by EnviEstudos, the conclusions of which formed the basis for the Environmental Licence Application submitted in June 2007. The main environmental issues related to ongoing operations were listed as:

- Treatment of acid mine drainage;
- Noise (audiometry testing commenced in 2007);
- Continued disposal of coarse and fine tailings (completed during 2014).

Summaries of the current environmental, industrial and other licences for Panasqueira are shown in Table 20-1, Table 20-2 and Table 20-3.

Table 20-1. Summary of Environmental Licences

License	License Nº.	Location	Authority	Issue date	Valid until	Date request for renewal
Environmental License LA nº. 347/2009	347/2009	Couto Mineiro da Panasqueira	APA - Portuguese Environmental Agency	18/12/2009	18/12/2014	Extention of Environmental License until issue of final decision of the ongoing process. 16-11-2016 Enviado Formulário PCIP Revisto a DGEG
License for the use of Water Resources - Wastewater rejection/discharge (ETAM)	L000548.2016.RH5	Water treatment station - Barroca Grande	APA - Portuguese Environmental Agency	13/01/2016	13-01-2017. Waiting for the issue of Environmental License (APA 18-1-2017)	13-07-2016. Request for extension made on 13-07-2016
License for use of Water Resources for captation of Surface water	2011.001281.000.T.L.CA.SUP	Bodelhão stream - Aldeia de S. Francisco de Assis	ARH - Tejo - Administration of Tagus Basin Region	12/05/2011	12-05-2016. Waiting for the issue of the Environmental Licence (APA 18-1-2017)	(12-12-2015) Written application addressed to APA - ARH TEJO for renewal of licence was made on the 4-12-2015
License for the use of Water Resources for rejection of domestic residual water on soil	2011.001280.000 T.LRJ.DAS	Septic tank at Rebordões - complex Couto Mineiro Panasqueira	ARH - Tejo - Administration of Tagus Basin Region	12/05/2011	12/05/2021	12/11/2020
Use of water resources - groundwater capture with a power extraction means at or below 5 hp for Industrial Activity (Mina - AC1)	Ref. Oficio DRHI-01409-OFI-2011	Alvoroso - Aldeia de S. Francisco de Assis	ARH - Tejo - Administration of Tagus Basin Region	13/05/2011	No permit is issued because the extraction equipment installed in catchment does not exceed 5hp.	
Use of water resources - groundwater capture with a power extraction means at or below 5 hp for Industrial Activity (Mina - AC2)	Ref. Oficio DRHI-01410-OFI-2011	Alvoroso - Aldeia de S. Francisco de Assis	ARH - Tejo - Administration of Tagus Basin Region	13/05/2011		
Use of water resources - groundwater capture with a power extraction means at or below 5 hp for Industrial Activity (Mina - AC3)	Ref. Oficio DRHI-01411-OFI-2014	Alvoroso - Aldeia de S. Francisco de Assis	ARH - Tejo - Administration of Tagus Basin Region	13/05/2011		

Table 20-2. Summary of Concession and Industrial Licences

License	License Nº.	Location	Authority	Issue date	Valid until	Date request for renewal
Concession contract and prospection and exploration contracts						
Contract for the allocation of an area to Beralt Tin and Wolfram (Portugal) S.A. for exploitation of mineral deposits of Wolfram, Tin, Copper, Silver, Zinc and Arsenic	C-18	Couto Mineiro da Panasqueira	Ministry of Industry and Energy - Secretary of State for Industry	16/12/1992	16-12-2052, renewable for one or two further periods, in a total of 30 years	Request must be made up to 2 years before the end of the previous period.
Addendum to Concession Contract for exploitation of mineral deposits of Wolfram, Copper, Zinc, Tin, Silver and Arsenic, known as Panasqueira, dated 16-12-1992	C-18	Couto Mineiro da Panasqueira	Ministry of Economy and Innovation - DGGE	17/11/2005		
Addendum to Concession Contract C-18, dated 16-12-1992 (Royalties)	C-18	Couto Mineiro da Panasqueira	Ministry of Economy and Employment - DGEG	23/03/2012	Term of 10 years, after this period it may be reviewed - valid until 23/03/2022	
Technical Director of Concession Contract C-18	C-18 - Letter from DGEG ref. 456 dated 14-1-2016	Couto Mineiro da Panasqueira	Direcção Geral de Energia e Geologia (Directorate-General for Energy and Geology)	14/01/2016		
Contract for prospection and exploration (Unhais-o-Velho area)	MN/PP/008/12	Unhais-o-Velho	Ministry of Economy and Employment - DGGE	30/03/2012	30-03-2014, renewable for a year, in maximum 3 times. 3rd renewal approved on 20-04-2016. Valid until 30-3-2017	
Person Responsible for the Prospection and Exploration Project (Unhais-o-Velho)		Unhais-o-Velho	Ministry of Economy and Employment - DGGE		While Exploration Project valid	
Industrial License						
Permit License nº. 7080/1997 (CMCovilhã) (Industrial Area)	7080-1997	Barroca Grande - Aldeia de S. Francisco de Assis	Municipality of Covilhã	29/07/1997	It has no expiration date.	
Change authorisation license of industrial establishment Type «2» of REAI	Proc. Nº. 2014268/02	Barroca Grande - Aldeia de S. Francisco de Assis	Ministry of Economy and Employment - Regional Delegation for the Center	13/04/2010		
Endorsement of Industrial Establishment Licence (Change of Company name to Beralt Tin and Wolfram)	Proc. Nº. 2014268		Ministry of Economy and Employment - Regional Delegation for the Center			22/01/2016
Fuel						
Licence to use Petrol Station / Repsol		Barroca Grande - Aldeia de S. Francisco de Assis	Covilha Municipality			
Electricity						
Endorsement of Licences SE of 7000 KVA; PT 1 type ALV de 615 KVA; Aerial Cable 60 KV with 4581 m from Barroca Grande SE to Rio; Instalation of low voltage at Cabeço do Pião and Barroca Grande	Proc. Nº. 0261/5/3/2		Direcção Geral de Energia e Geologia (Directorate-General for Energy and Geology)	09/05/2016		
Licences SE of 7000 KVA; PT 1 type ALV de 615 KVA; Aerial Cable 60 KV with 4581 m from Barroca Grande SE to Rio; Instalation of low voltage at Cabeço do Pião and Barroca Grande	Proc. Nº. 0261/5/3/2		Direcção Geral de Energia e Geologia (Directorate-General for Energy and Geology)	26/01/2016		
Compressores						
Certification of Compressors	Certificates 001 a 009/2017	Mine and Plant	IPQ	2016 and 2017	01/01/2022	
Pressure gauges	IPQ Registration	Barroca Grande - Aldeia de S. Francisco de Assis	IPQ		31/12/2017	

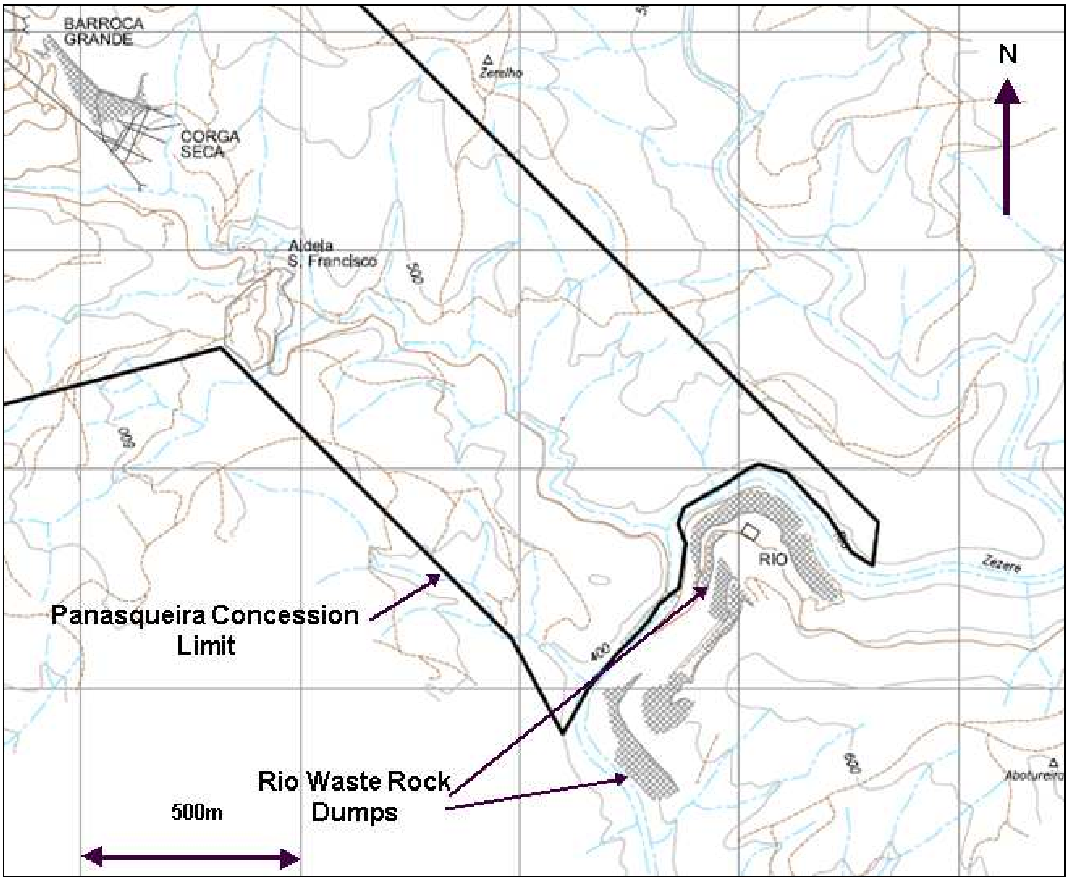
Table 20-3. Summary of Explosives and Other Licences

License	License Nº.	Location	Authority	Issue date	Valid until	Date request for renewal
Licence for explosive products warehouse and authorisation for use of explosives						
Permit nº. 1/2014 - Storage of explosive products (two warehouses)	1/2014	Rebordões - Barroca Grande	Ministry of Internal Administration	14/01/2014	Permanente duration	
Authorisation for purchase of explosive	195	Couto Mineiro da Panasqueira - C18	Ministry of Internal Administration - National Directorate of Public Safety Police	17/01/2017	31/12/2017	Request new licence in September 2017
Authorisation nº. 608 - Explosives - Extension	608	Couto Mineiro da Panasqueira - C18	Ministry of Internal Administration - National Directorate of Public Safety Police	10/08/2016	31/12/2017	
Statement of Responsibility for Exploration - Protection against lighting in explosive warehouse		Rebordões - Barroca Grande		24/02/2016		
Certificate 225/2015 - "Approved" Explosive data collection system	225/2015	Couto Mineiro da Panasqueira - C18	Public Safety Police - Department for firearms and explosives	03/02/2016		
Others						
Authorisation nº. 7045/2016 replaced Authorisation nº. 2363/2007 - Video surveillance	Proc. nº. 10466/2016 replaced Proc. nº. 840/07 (initial). Proc nº. 861/2015 (change to 29 camaras)	Couto Mineiro da Panasqueira	National Data Protection Commission	19-07-2016/26-11-2007		
Authorisation nº. 7044/2016 replaced Authorisation nº. 2926/2013 - Video surveillance	Proc. nº. 10465/2016 replaces Proc. nº. 2926/13	Rebordões - Barroca Grande	National Data Protection Commission	19-07-2016/09-04-2013		
Licence for private radiocommunication network of land mobile service	License nº. 516291		Anacom - Autoridade Nacional de Comunicações	16/02/2015	16/02/2020	
Change of Company name	Minutes nº. 306 and Permanent certificate 6032-4556-7343			January 2016		
Revogation of Power of Attorney and Power of Attorney				05/02/2016		
Constituition of the Company		Couto Mineiro da Panasqueira		26/07/1973		
Articles of Association (Alterations)				July 2016		
Declaration of change of activity			AT Tax Authority and Customs	September 2016		

20.2 Rio Site

The Rio site was part of the company's holdings until March 21, 2006, when the Portuguese government signed an agreement that removed the site, and all associated environmental liabilities, from the Beralt concession (Figure 20-1). The Rio waste rock dumps and tailing piles are therefore not environmental liabilities for BTW.

Figure 20-1. Rio Waste Rock Dumps



The Rio site contained the old mill, a large coarse waste rock dump and an old dried up 25- 70m thick tailings pond situated on top of the waste rock dump. The tailings pile contained: coarse waste rock primarily composed of schists and quartz (which comprised the majority of the material in the dump) with minor amounts of pyrite and arsenopyrite, fine grained mill tailings, and a pile of arsenopyrite flotation tails. Some estimates indicated that the total pile could have exceeded 8Mt of material.

The revised concession block was subdivided into two subsections based on the geographic location of the two tailings/waste-dump sites of Panasqueira (the old mine), and Barroca Grande (site of the active mine/plant and dumps).

20.3 Panasqueira Site

The Panasqueira site contains the old mine, and tailings pile used in the earliest years of production from the late 1890's to 1927 when a new plant was constructed at Rio. The site contains remnants of old buildings as well as a large tailings pile down slope from the old mill buildings. Numerous old mine workings and small dump piles are scattered around the hillside; development in this area was done by small hand-driven mining operations that apparently contain little in the way of environmental concerns as they appear to be naturally re-vegetated. Work by Golder in 1999 summarised the old tailings and dumps as having "little danger of imminent, massive failure".

20.4 Barroca Grande Site

The Barroca Grande site includes the underground mine and portals, processing plant, mine offices and employee housing in addition to the active tailings disposal areas, and Salgueira water treatment plant.

Crosby (SECOR 2001) considered that the primary environmental issue from mining was the discharge of mine water from underground via a drainage gallery which discharged at the Barroca Grande treatment plant at Salgueira.

Mine water flows are seasonal, with minimal base flow conditions during the dry season and high flow during the rainy season. The water quality results were reported to be quite consistent despite the large fluctuations in flow rates. The pH values from March 1999 to December 1999 ranged from a low of 3.49 to a high of 3.73. Although quite consistent, these low pH values are typical of AMD waters and resulted in elevated concentrations of metals which are harmful to the environment.

The concentrations for copper, zinc, manganese and arsenic (June 2000) all exceeded the regulatory limits for water quality; although the water discharges to the Salgueira treatment plant and is effectively treated during the summer months.

20.5 Salgueira Water Treatment Plant

The original Salgueira plant was apparently designed in 1957, whose capacity was 300m^3/hr. This plant was expanded and upgraded in late 2011, so it can now handle 500m^3/hr.

It consists of treatment tanks, a multi-level building, and a lime storage and hopper system. Lime is delivered to a loading area above the plant, fed by gravity to the treatment plant below, mixed with a small volume of water and then fed to the treatment tanks. Flocculant is added to precipitate out manganese and other metals, which will be separated with the sludge.

Figure 20-2. Salgueira Water Treatment Plant

The treatment plant requires four to five operators, and power to operate the plant and pump the sludge up to the pond.

The Salgueira treatment plant collects water from several sources. Water is treated with the addition of lime, precipitate sludge is pumped to the tailings pond, and treated water is then discharged to the creek channel adjacent to the plant or pumped to holding tanks for later use in the mill and mine. The sources include mine water drainage, surface accumulation on the old tailings area (Barragem Velha), water from the new tailings pond (Barragem Nova) and seepage from the base of the tailings.

Sometimes during winter, high discharge periods cannot be treated, resulting in discharges to the creek. These discharges were reported to coincide with high flows in the creek valley and in the Zêzere River, with the result, that the untreated AMD water was diluted with surface water.

The mine has obtained a waiver on the maximum pH of waters to be released in the river (level of 10 instead of 9) and manganese. The output pH is 9.8. Twice a week samples are taken and analysed in the mill laboratory. Once a month samples are taken and sent to an independent laboratory. These results are checked every 6 months by the local authorities.

20.6 Waste and Tailings Disposal

20.6.1 Historical

The Barroca Grande dump lies adjacent to and above Bodelhão Creek that flows south-easterly to the Rio Zêzere and comprises primarily HMS tailings (gravel), waste rock and sand tails, all of which are very benign and possibly marketable as aggregate. Fine tails are the biggest liability as they contain precipitate from Salgueira as well as the copper circuit tailings with up to 25% arsenic.

Golders (1999) were of the opinion that the stability of the coarse tailings deposits was not precarious, was greater than for the Rio deposits, and that stability would not decrease with time. Stability of the low elevation fines (slimes) containment was considered to be less assured than for the coarse tailings upslope of the slimes deposit.

The SECOR report (2001) noted that the tailings disposal areas at Barroca Grande extended from the town and mill site approximately 0.8km to the east. Waste rock was being dumped at the eastern extent of the disposal area, that would eventually result in the tailings filling a small valley which drains just east of the Salgueira treatment plant. It was also mentioned that dumping in the valley would provide additional waste rock disposal capacity, but would require collection and treatment of AMD seepage from the valley.

The report also summarised the existing facilities and environmental problems in both the Panasqueira and Barroca Grande dump areas and recommended:

- Assessment of the stability of the existing tailings piles;
- Development of a Tailings Closure Plan; based on the studies conducted above a tailings closure plan could be developed for each disposal area;
- Water Treatment System Assessment; the treatment of AMD-impacted water would present a significant long-term cost to the facility; a detailed study of AMD generation and treatment was merited to minimise the current and on-going costs.
- Facility Closure/Post-Closure Plan and Cost Estimate; following the completion of the above studies, a detailed closure/post-closure plan and cost estimate would be prepared.
- Monitoring and Compliance Programme; the existing monitoring programme should be modified to comply with the EU compliance requirements and to better assess discharge from the facility.

20.6.2 Recent

On–site disposal installations are shown in Figure 20-4 and Figure 20-5. Sand and slimes tailings are disposed of in a dam built into the eastern section of the heavy media separation reject stockpile (TMF 2A). This tailings pond also receives sludge from the water treatment plant.

In anticipation of the filling of the current tailings pond (TMF 2A) with fine tailings, by the end of 2020, BTW has started the construction of a new pond, as shown in Figure 20-3.

In order to match construction of the required volume for fine tailings storage with the availability of supporting landfill construction coarse waste material, the work is planned to be carried out in 3 phases. A geomembrane will be installed on the pond base.

Solution for tailings storage construction allows a total storage volume of about 1,726,500m3 or ~ 30 years given the current volume:

- 164,500 m^3 during the 1st phase ~ 3 years
- 510,500 m^3 during the 2nd phase ~ 9 years
- 1,051,500 m^3 during the 3rd phase ~ 18 years

Beralt benefits from the EU tailings ponds Directive, in that ponds can be used until the end of their useful life and there is no obligation to immediately cover a pond that has reached the end of its useful life.

Figure 20-3. On-Going Construction of New Tailings Disposal Area (TMF 2B)

Figure 20-4. Plan View of Current Disposal Area

Figure 20-5. View of Tailings and Waste Dump Areas, From South-West

20.7 Soil Contamination

Arsenopyrite is the main mineral present as well as a rejected waste sulphide. The metal assemblage identified in soil (Ag-As-Bi-Cd-Cu-W-Zn; potentially toxic elements (PTEs)) reflects the influence of the tailings, due to several agents including aerial dispersion. PTEs and pH display a positive correlation confirming that heavy metal mobility is directly related to pH and, therefore, affects their availability.

The estimated contamination factor classified 92.6% of soil samples as moderately to ultra-highly polluted.

21 CAPITAL AND OPERATING COSTS

Capital costs planned for the next year are summarised in Table 21-1.

Table 21-1. Summary of Planned Capital Costs, Oct 2016- Sept 2017

	Capital *US$ x 1000*
Mine Development	$832
UG Drilling	$46
New Mine Equipment	$179
Mine Equip Refurbishment	$0
New Plant Equipment	$140
Plant Refurbishment	$0
Environment	$154
Civil Construction+Others	$78
Total	**$1,429**

Notes

. Stope access development excluded, as this is included in the underground operating cost

. Most of the Environment cost is for the development of TMF 2B tailings pond

A summary of planned mine operating costs over the next year are shown in Table 21-2.

Table 21-2. Summary of Planned Operating Costs, Oct 2016- Sept 2017

Description			Unit	Values
General				
		$:Euro conversion		**1.036**
Production Data				
		ROM tons processed	*tonnes*	659,311
		Average %WO_3 Grade Plant	*%WO_3*	0.134
		Average Monthly Ore Treated	*tpm*	54,943
		WO_3 product	*MTUs*	70,898
Operating Costs				
		Admin	*$/t ore*	$1.61
		General	*$/t ore*	$3.28
		Environmental	*$/t ore*	$0.35
		G&A Sub-total	*$/t ore*	$5.24
		Processing Cost	*$/t ore*	$4.04
		Underground Cost - Stoping and Development	*$/t ore*	$12.65
		Total Operating Cost	*$/t ore*	$21.9
		Total Operating Cost	*$/MTU*	$204

Most of the costs are directly incurred in Euros. With actual received tungsten prices between 2014 and 2016, derived breakeven cut-offs range between 0.11 and 0.14% WO_3.

The G&A costs listed include costs for:

- General Administration, including the Lisbon office
- Auditing and Consulting.
- Bonus and Allowances.
- Safety Bonus.
- Energy
- Transport of Concentrates.

22 ECONOMIC ANALYSIS

The data summarised in Table 22-1, summarised the planned cashflow over the next year, based on an assumed received WO_3 price of $280/mtu. This cashflow does not include deductions for capital costs.

Table 22-1. Summary of Planned Cashflow, Oct 2016- Sept 2017

Description		Unit	Values
General			
	$:Euro conversion		**1.036**
Prices			
	Metal Price - received per sold MTU	*$/mtu WO_3*	$280
	Conc grade	*%WO_3*	74.80%
	Price per t of concentrate	*$/t conc*	$20,944
	after transport + smelting	*$/t WO_3*	$28,000
Production and WO_3 Revenue			
	ROM tons processed	*tonnes*	659,311
	Average %WO_3 Grade Plant	*%WO_3*	0.134
	Recovery	*%*	80.1%
	WO_3 production	*MTUs*	70,898
	WO3 sold	*MTUs*	71,520
	WO_3 revenue	*US$ x 1000*	**$20,026**
By-Products			
	Copper Con Revenue	*US$ x 1000*	$325
	Tin Con Revenue	*US$ x 1000*	$1,204
	Crystal sales, plus coarse tails & scrap	*US$ x 1000*	$82
	Sub-total	*US$ x 1000*	$1,611
Total Revenue		*US$ x 1000*	**$21,636**
Costs			
	Mine	*US$ x 1000*	$8,342
	Plant	*US$ x 1000*	$2,664
	G&A	*US$ x 1000*	$3,454
	Total Cash Costs	*US$ x 1000*	**$14,461**
Net Direct Cashflow		*US$ x 1000*	**$5,565**
Net Direct Cashflow including by-products		*US$ x 1000*	**$7,175**

23 ADJACENT PROPERTIES

There are no known other producing tungsten properties in the vicinity of the Panasqueira mine. The area surrounding the Panasqueira mine contains numerous pits and dumps created by hand mining usually by individual miners. None of these old showings are currently active. It was not possible to locate any documentation discussing other tin or tungsten occurrences in the immediate vicinity of the mine.

24 OTHER RELEVANT INFORMATION

BTW has developed a mine closure plan since 2011. The Mine Closure Plan covers all infrastructures and facilities and has involved all authorities (local, regional and national) and all stakeholders (affected groups and local organizations). Calculated costs for the closure plan assume 2038 as a hypothetical closure date. The plans and costs of implementation of this closure plan are to be reviewed at intervals of five years (2018, 2023, 2028, etc).

Connected with this plan, the CENOR consultancy group completed a study (November 2011) for closure of the Barroca Grande waste and coarse tailings dump.

25 INTERPRETATION AND CONCLUSIONS

The Panasqueira mine has been operated successfully for many decades with the current mining method and mill set up. Exploration has consisted principally of surface and underground diamond drilling. The method of focusing on quartz intersections appears to have been successful, even when the quartz has little mineralisation. The very erratic nature of wolframite occurrences mean that it is very easy to intersect veins in barren patches, while the veins overall are still heavily mineralised.

The operation has a low total operating cost per tonne of $US22/t. The use of low profile equipment is successful, and the mechanised room and pillar mining method is systematically applied throughout the mine. The method is also flexible in terms of being able to follow vertical variations in the veins being mined. Although there are few additional support measures used, apart from in faulted areas. The mine seems to be managed well, with subsidence monitoring and experience used to move out of damaged areas when necessary, and leave the final 3m x 3m pillars to collapse.

The plant has been in its current configuration for many years, and consistently achieves recoveries of approximately 80%. A very important feature of the Panasqueira ore and plant set-up is that this recovery level is maintained even with declines in feed grade. The HMS arrangement works well in terms of recovery, and this produces coarse tailings, this greatly assists in minimising the amount of fine tailings which has to be deposited in a tailings pond.

At the current production rate and reserve levels, the mine life of approximately 3 years. However, this mine life estimate is very conservative, as Panasqueira has generally continued to preserve overall resource and reserve levels with ongoing drilling and development. Further development of resources also depends on further exploration of the Panasqueira deep area to the north, as well as Level 4.

The D9 formula is used in resource estimation work, to convert intersected quartz vein thicknesses into equivalent WO_3 grades. Together with the reserve estimation parameters, the use of this formula is supported by many years of reconciliation. However, in some more extreme parts of the ore zones, such as the Panasqueira Deep areas, there is less confidence in the same thickness-grade relationship expressed in the current D9 formula. Future analysis of exposed new areas should investigate if this parameter should be varied locally to some extent. Similarly, the 60% and 40% confidence factors applied to the current resources estimates should be reassessed in on-going work.

It is beyond the scope of the report to value all assets and liabilities. Further studies are required to state categorically the status of all three dumps and their discharges.

In the opinion of the QP, the following conclusions have been reached:

a) The empirical formulae developed at the mine, for evaluation purposes, have been used for decades and are supported by a very large amount of reconciliation data. The QP considers that these formulae, along with the other parameters and guidelines applied, do provide reliable methods of resource and reserve estimation.

b) The current resource and reserve estimations shown in this report have been reviewed by the QP. In the opinion of the QP, this review supports the estimation results presented.

c) The same resource/reserve cut-off grades have been in use since 2011. Since that time, the total reserve quantity has been maintained, although the overall total resource base has generally declined. This means that the mine’s on-going stope development has elevated resource categories as planned, although drilling levels have declined, which has led to a reduction in overall resources.

d) The most important areas of the mine which offer the most scope for overall resource expansion are the Panasqueira deep area and Level 4 (below 470mRL).

26 RECOMMENDATIONS

Ongoing research into possible re-processing of tailings material seems well worthwhile. Similarly, investigation into the use of ore sorters to allow potential processing of HMS reject material is a logical potential development.

27 REFERENCES

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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 Panasqueira Mine, 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 Fellow (FIMMM) in good standing of the Institute of Materials, Minerals and Mining.

5. I have worked as a mining engineer in the minerals industry for over 30 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. 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 19 years as an independent mining consultant, involved with evaluation and planning projects for both open pit and underground mines. I have worked on the resource and reserve evaluation of tungsten deposits for over 10 years. By reason of my education, experience and professional registration, I fulfil the requirements of a "qualified person" as defined by NI 43-101. I am responsible for the entire technical report titled "Technical Report on the Mineral Resource and Reserves of the Panasqueira Mine, Portugal" and dated December 31st, 2016. I visited the mine site most recently from November 23rd - 25th, 2016.

7. 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.

8. I am independent of Almonty Industries Inc. and BTW, pursuant to section 1.5 of the Instrument. I have had no prior involvement with the Panasqueira Mine property.

9. 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.

10. 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 21st of February, 2017

AWheeler

A. Wheeler, C.Eng.

APPENDIX A:

Glossary of Terms

UNITS OF MEASURE AND ABBREVIATIONS

DMT	dry metric tonne
Ktpa	Kilo-tonnes per annum
m	meters
m/h	meters per hour
mtu	metric tonne unit
	1 mtu = 10kg = 0.01t
m^3	cubic meter
m^3/h	cubic meters per hour
t	Tonne (1,000 kg)
km	Kilometers
kt	Tonnes x 1,000
Mt	Tonnes x 1,000,000
NI	National Instrument (43-101)
NSR	Net smelter return
ppb	Parts per billion
ppm	Parts per million
tph	Tonnes per hour
tpa	Tonnes per annum/year
3D	Three-dimensional

QA/QC	Quality assurance/ quality control
ha	hectares
US$	US dollars
UTM	Universal Transverse Mercator
XRF	X-ray fluorescence

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 at the mine for its concentrate sales is 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.

European APT prices are widely used as a reference price and are quoted in the "Metal Bulletin" twice weekly. Since peaking at US$252/257 per mtu in September 2008, prices declined to a low of US$170/200 per mtu in July 2009 as the Global Financial Crisis deepened. Since then, prices have improved and spreads have narrowed to reach US$330/340 per mtu by the end of 2010.