NOTE

The enclosed technical report, titled "Technical Report on the Mineral Resources and Reserves of the Valtreixal Project, Spain-- NI 43 - 101 Technical Report" and dated as of October 31st, 2015 amends and restates in its entirety the technical report of the same name and date filed by Almonty Industries Inc. on November 19, 2015 (the "Superseded Technical Report"). The enclosed technical report is being filed in order to clarify certain inconsistencies in the Superseded Technical Report regarding the Qualified Person Certificate and reference to a floor price of US$250/MTU of APT (the "Floor Price"). No such Floor Price exists in any contract currently in force to which Almonty is a party.

The following changes were made in the amended Technical Reports to correct the disclosure and remove the inconsistencies:

1. Page 112, Section 19, Market Studies and Contracts – The sentence "There is a floor price of $250/mtu for APT in the contract." has been removed.

2. Page 125, Section 28, Qualified Persons Certificates – Point #4 has been amended to "I am a professional fellow (FIMMM) in good standing of the Institute of Mining, Metallurgy and Materials."

REPORT NI 43-101

TECHNICAL REPORT ON THE

MINERAL RESOURCES AND RESERVES OF THE

VALTREIXAL PROJECT, SPAIN

Prepared for

Almonty Industries

by

Qualified Person:
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31st October 2015

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1 SUMMARY

1.1 Introduction and Overview

This report was prepared to provide a Technical Report compliant with the provisions of National Instrument 43-101 - Standards of Disclosure for Mineral Projects, ("NI 43-101"), and comprises a Resource and Reserve Estimation for the Valtreixal project, as of the end of October 2015. The Valtreixal deposit is a potential open pit operation, and is located in the NW part of the Zamora province, in the Castilla de Leon region of Spain. The principal potential products are tungsten and tin.

This report was prepared by Adam Wheeler, at the request of Mr. N. Alves, Almonty Industries ("Almonty"). Assistance and technical detail were supplied by the technical personnel of Daytal Resources Spain S.L. ("Daytal"), which is a wholly owned Spanish subsidiary of Almonty. Adam Wheeler visited the Valtreixal site most recently on June 15^{th} and 16^{th}, 2015.

The Valtreixal mineralisation has been explored with underground development since the late 1800s, and limited tin exploitation occurred sporadically in the mid-1900s.

1.2 Ownership

Almonty Industries Inc ("Almonty"), is a corporation governed by the Canada Business Corporations Act (the "CBCA"). Almonty trades on the TSX Venture Exchange (TSX-V) under the symbol "AII". In March 2013, Almonty announced the acquiring of an option for 51% interesest in the Valtreixal tin-tungsten project for 1.4M Euros, plus an option to acquire the balance for after 24 months for 2M Euros. Almonty have also created a wholly owned Spanish subsidiary Valtreixal Resources Spain S.L. ("Valtreixal Resources").

Valtreixal Resources have obtained investigation and exploitation permits for the area called C.E. (Concesion de Explotacion) No. 1352, Alto de Repilados, which is an old but valid exploitation licence. Valtreixal Resources have also obtained an exploration licence for P.I. (Permiso de Investigacion) No.1906 Valtreixal. These two licence areas cover the whole project area and known resources. Ongoing studies of the Valtreixal deposit have now been presented to the authorities (el Director Facultativo), in order for both areas to get C.E. (Concesion de Explotacion) status.

1.3 Geology and Mineralisation

Tangential movement along the regional Bragança fault may have assisted in creating dilation zones in the Ordovician lithology. The Calabor River now follows the general direction of this fault. These dilation zones, along a north-east trend, appear to be associated with the development of quartz veins and later tin/tungsten mineralisation in shale. It is considered that a mineralising hydrothermal system of Hercynian age (330 Ma to 280 Ma) was powered by a hypothetical underlying cooling granite in the Valtreixal area.

It is generally considered that the northward movement of the ancient continent Gondwana, and its collision with Laurentia to form the super-continent Pangea, resulted in the Hercynian orogeny. This orogeny was pivotal in the formation of tin/tungsten deposits in this type of setting. As Gondwana advanced, overriding and pushing down, subducting the thin oceanic basaltic crust, there developed a geo-shear zone which dipped back under the continental frontal mountains. This geo-shear would have penetrated through the oceanic crust into the upper mantle where serpentinisation takes place. Serpentinite development is highly exothermic and may circulate accessory calcium and additional heat into the hydrothermal system. At greater depths the geo-shear may penetrate continental, denser cratonic rocks with entrained primordial undifferentiated crust having higher amounts of heavy elements.

The mineralisation at Valtreixal can be classified as a complex vein deposit. Much of the mineralisation, especially scheelite, is situated away from the quartz veins and appears to be stratabound in origin. Tin, in the form of cassiterite, occurs in and around the quartz veins. The linear mineralised zones appear, in a general sense, to be confined to specific stratigraphic intervals and there appears to be a degree of separation into tin and tungsten zones. Although a sedimentary, syngenitic origin for the tungsten mineralisation has been considered, it is unlikely to have eventuated at Valtreixal, because the scheelite hosting shale is of Ordovician age, 488 Ma to 444 Ma, and thus predates by a considerable margin the Hercynian, at 330 Ma to 230 Ma, tin/tungsten mineralisation episodes, with hydrothermal remobilisation and alteration of the mineralised schists.

The local Valtreixal stratigraphy in the Valtreixal area is dominated by 3 main formations, all of which broadly strike SW-NE, and dip at approximately 80^{o} to the south-east:

a) Schists - Capas de los Montes. Cambrian/ordovician. Very stratified and transformed by regional metamorphism, with intercalated quartzites, and marked at the base by conglomerates. Thickness approximately 1000m.
b) Quartzites - Peña Goda/Culebra. Ordovician. Alternating with a variety of types and colours of intercalated schists. Thickness approximately 50-70m.
c) Slates – Pizarras de Luarca. Ordovician. Pelitic series of siliceous slates, phyllites and schists. This formation hosts most of the mineralisation at Valtreixal. High frequency of segregated quartz veins and schist bands sometimes rich in sulphur. Overall thickness approximately 300-600m.

1.4 Database and Resource Estimation

Three types of samples are available for resource estimation: underground channel samples, surface trench samples and diamond drillhole samples. Underground channel samples have been taken in old underground galleries, either by ENADIMSA (pre-1986) or Siemcalsa in the period from 2008-2011. For the current work, only samples from two galleries have been included, owing to the status of survey data. ENADIMSA also completed 10 trench lines over 850m, producing 170 samples. They also drilled 3 diamond drillholes.

Data from 26 surface trenches, covering 3.7 km, have been included from Siemcalsa's 2008-2011 exploration campaigns. Data from 18 surface trenches, covering 2.7 km, have been generated during Daytal's 2013 exploration campaign. One additional surface trench was also taken in an old surface stockpile.

Siemcalsa 2008-11 exploration campaigns included 6 diamond drillholes, with a total length of 1,227m. Daytal's 2013-2015 exploration campaign completed 59 diamond drillholes, with a total length of 10,716 m.

All of the data described above were collated by Daytal in an Excel database, and from there were imported into the CAE Datamine mining software system, for subsequent use in resource estimation. This resource estimation work stemmed from updated interpretation of mineralised structures by Daytal geologists. As well as logged lithological differences, cut-off grades of 0.07% Sn and 0.07% WO_3 were used in the interpretation process. There are 4 main mineralised structures, extending over a strike length of 1.5 km.

These interpretations were used to create a 3D block model, based on a parent block size of 10m x 10m x 10m, with sub-blocks generated down to a resolution of 1m. In addition sub-blocks were extrapolated a maximum distance of 50m from all selected samples, from mineralized intersections, so vein material could also be modelled outside the structurally modelled zones. Dynamic anisotropy was also applied, to allow for varying dip and strike orientations.

The samples selected inside the interpretations were converted into 2m composites, to which top-cut levels of 1.27% Sn and 1.1% WO_3 were applied. These composited grades were used to estimate Sn and WO_3 grades into the volumetric block model, primarily using an ordinary kriging (OK) method of interpolation. For validation purposes, alternative grades were also estimated using a nearest neighbour method. Density values were estimated from core density measurements.

The western part of the deposit , which has now been drilled off with a 30m drilling grid, has generally classified as indicated resources; the remainder of the deposit being classified as inferred.

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1.5 Mine Planning

The current study is at a pre-feasibility (PFS) level. The resource block model has been used as the basis for an open pit optimisation. Optimisation parameters were derived by reference to the Los Santos open pit operating parameters, which is also owned by Almonty, and operating with mining contractors. The parameters were modified to reflect that mining at Valtreixal will not require drilling and blasting. Processing parameters were derived from metallurgical testwork on Valtreixal material. No physical constraints were applied during the optimisation process. Slope angles applied were derived from measured face angles measured in cuttings in and around the deposit area. Following on from the base case optimisation, additional optimisation runs with inferred material enabled demonstrate that additional exploration work will justify a much bigger open pit, advanced over a much longer strike direction.

The pit shell produced by the base case optimisation was used as a reference for the generation of a detailed pit design, which is cut into the west sloping existing hillside. A 10m wide haul road was put into the design, with the exit point at the extreme west end, at an elevation of approximately 870m. Access to the eastern, and higher, part of the pit will be gained from temporary access roads from the existing surface on higher benches. Berms of 4m have been incorporated into the design every 20m vertically. For the extended highwall of the pit up to 1015mRL on the southern and eastern sides of the pit, additional 14m safety berms were put in every 60m vertically.

The overall pit design is approximately 700m in length along strike, and 300m wide at its widest point. Grades of WO_3 and Sn were used to create an WO_3-equivalent grade, which was referenced against the breakeven cut-off grade of 0.08% WO_3 to indicate ore or waste. For the pit design this gave approximately 2.5 Mt of probable ore, with an overall strip ratio of 8.3 : 1. This pit envelope also contained 2.2Mt of inferred resources at economic grades.

Based on the reserves defined within the pit design, a life-of-mine plan was developed, aimed at producing 500 Ktpa of ore, thus producing approximately a 5 year mine life. For scheduling purposes, the pit was divided into two principal pushbacks, approximately dividing the pit into western and eastern halves. Mining will start in the western (lower) pushback, and then as mining progresses deeper in this pushback, mining will also start on the upper benches of the eastern pushback. The general sequencing strategy is to excavate the pit areas from west to east, with dumping of mine waste from the active east advancing benches into the previously excavated western pit areas.

1.6 Mineral Processing

A review and conceptual study, for the Valtreixal deposit, was completed by Saint Barbara LLP (StB), in May 2014. This study included a review of mineralogical studies by Siemcalsa, petrological studies from samples taken from 2013 diamond drill intersections and trenches, heavy liquid separation and QEMSCAN testing completed during 2012 by Wardell-Armstrong, as well as scheelite flotation testing by AGQ Labs during 2013. The AGQ testing was done on a sample of schist taken for the ENADIMSA gallery.

Based on the mineralogical and metallurgical information reviewed by StB, a conceptual plant design was developed to encompass crushing, grinding and gravity separation of scheelite and cassiterite into a bulk concentrate; removal of sulphides from the bulk concentrate by flotation; and drying and electrostatic separation of the bulk concentrate into separate scheelite and cassiterite concentrates. A metallurgical performance was estimated of 65% tin recovery, allowing a 50% Sn concentrate, and a 55% tungsten recovery, allowing a 65% WO_3 concentrate,

Pilot plant studies have also been completed by the company ADVANCED MINERAL PROCESSING, SL (AMP) and concentration tests were performed by the technical personnel working in mine-Fuenterroble Los Santos (Salamanca) laboratory. All of this testwork has been used by Daytal in the design of an ore beneficiation process for Valtreixal.

StB considered that the likely pit geometries, along with the natural topography, lend to an eventual dry disposal of tailings in initial mined out pits. A dry tailings treatment plant has therefore been incorporated into the overall mill design. Initial tailings disposal and waste rock dumps would take place, subject to negotiation, in the government owned forestry area immediately to the south of the open pits. Thereafter, StB propose the backfilling of worked out sections of the open pits. Future mining schedules will take this pit-backfilling requirement into account.

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 indicated resources is shown in Table 1-1. There are no measured resources. Inferred resources are shown in Table 1-2.

Table 1-1. Valtreixal – Indicated Mineral Resources
As of 31st October, 2015
In-Situ Resource Estimation

CLASS	Tonnes Kt	Sn %	WO_3 %	WO_3_Eq %
Indicated	2,828	0.13	0.25	0.34

Notes

. Cut-off applied of 0.05% WO_3_Eq

. WO_3_Eq = WO_3 + (Sn x 0.74), based on:

	Price	Recovery
WO_3	$37,000/t	55%
Sn	$23,150/t	65%

. Maximum extrapolation = 50m

. Density values estimated from measurements

. Resources shown are inclusive of reserves

Table 1-2. Valtreixal – Inferred Mineral Resources
As of 31st October, 2015

CLASS	Tonnes Kt	Sn %	WO_3 %	WO_3_Eq %
Inferred	15,419	0.12	0.08	0.17

Notes

. Cut-off applied of 0.05% WO_3_Eq

Mineral Reserves have been determined, as part of the PFS study described in this report. These reserves are those indicated resources which are inside the current final pit design. These reserves are summarised in Table 1-3.

Table 1-3. Valtreixal –Mineral Reserves
As of 31st October, 2015

Reserve Category	Tonnes Kt	Sn %	WO_3 %	WO_3_Equiv %
Probable Reserves	**2,549**	**0.12**	**0.25**	**0.34**

Notes

. WO_3_Eq = WO_3 + (Sn x 0.74), based on:

	Price	Recovery
WO_3	$37,000/t	55%
Sn	$23,150/t	65%

. Cut-off applied to WO3_Equiv

Breakeven Cut-off = 0.08 WO_3

. Mining factors applied:

Dilution = 5%

Losses = 5%

. Pit design also contains 2.2 Mt of inferred resources at economic grades

1.8 Conclusions

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

a) The Valtreixal project is a viable open project. An open pit has been designed with 2.5 Mt of ore, which suggest a 5 year mine life, based on a mill throughput of 500 Ktpa. An economic analysis indicates an NPV (at a 10% discount rate) of $16.1M, and an internal rate of return of 24%.

b) There are significant amounts of inferred resources, which suggest significant pit expansion both with depth and laterally along strike. Pit optimisations, with inferred resources also activated, suggest over 10 Mt of potential ore.

c) Exploration drilling completed by Daytal over the last 3 years have confirmed and extended the originally previously delineated resource base. In particular, the occurrence of scheelite mineralisation outside of quartz veins, has provided much wider mineralised zones than were previously interpreted.

d) The current open pit design is one coherent excavation. It appears that with more drilling to enhance the resource category of current inferred resources, the resultant pit elongation along strike will offer a very good opportunity for sequential pit extraction from west to east, with concurrent backfilling of excavated volumes with mined waste.

2 INTRODUCTION

2.1 Introduction

This Technical report was prepared in compliance with the provisions of National Instrument 43-101 - Standards of Disclosure for Mineral Projects, (“NI 43-101”), and comprises a Resource estimate for the Valtreixal project, as of the end of October 2015. It is also contains a preliminary y economic assessment (PEA) with respect to a potential open pit operation. The project is considered as a potential open pit operation, which would produce tin and tungsten concentrates.

This report was prepared by Adam Wheeler, at the request of Mr. N. Alves, of Almonty Industries. Assistance and technical detail were supplied by the technical personnel from Daytal, which is a wholly owned Spanish subsidiary of Almonty. Adam Wheeler visited the Valtreixal site most recently on June 15th and 16th, 2015.

2.2 Terms of Reference

The resource estimation work was commissioned by Almonty Industries, and completed by Adam Wheeler, an independent mining consultant.

Adam Wheeler was retained by Almonty to provide an independent Technical Report on the Mineral Resources and Reserves at Valtreixal, as at October 31st, 2015. 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 site visits, Adam Wheeler has carried out studies of all relevant parts of the available literature and documented results concerning the project and held discussions with technical personnel of Daytal who have been doing exploration work at Valtreixal from 2013 – 2015.

The purpose of the current report is to provide an independent Technical Report and update of the resources and reserves of the Valtreixal project, 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.

2.3 Sources of Information

In conducting this study, Adam Wheeler has relied on reports and information connected with the Valtreixal project. 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, 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$1.12 to 1.00 Euros (€), unless otherwise stated.

3 RELIANCE ON OTHER EXPERTS

Adam Wheeler has reviewed and analysed data provided by Almonty 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.

While exercising all reasonable diligence in checking and confirmation, Adam Wheeler has relied upon the data presented by Almonty, and previous reports on the property in formulating his opinions.

For the prefeasibility study (PFS) presented in this document, Adam Wheeler has relied on work completed by Saint Barbara LLP (StB) with respect to all mineral processing aspects of the project.

4 PROPERTY DESCRIPTION AND LOCATION

The Valtreixal tin-tungsten project is located in the NW part of the Zamora province, in the Castilla de Leon region of Spain. It is positioned approximately 5 km north of the Portuguese border and former Calabor tin mine. The nearest town is Puebla de Sanabria, located approximately 10 km to the north-east. Valtreixal is located approximately 100km north-west of the Spanish city Zamora. Its central position is at approximately a latitude of 41° 59' 00''N and a longitude of 62° 42' 30''W. The position of Valtreixal with respect to Spain is shown in Figure 4-1, and more locally in Figure 4-2.

The extent of the Valtreixal exploration licences are shown in Figure 4-3. Valtreixal Resources have obtained investigation and exploitation permits for the area called C.E. (Concesion de Explotacion) No. 1352, Alto de Repilados, which is an old but valid exploitation licence. Valtreixal Resources have also obtained an exploration licence for P.I. (Permiso de Investigacion) No.1906 Valtreixal. The current status of these two licence areas is shown in Table 4-1.

Table 4-1. Current Licence Status

Permission Area Name	P.I. nº 1906 VALTREIXAL	C.E. nº 1352 ALTO DE REPILADOS
Original perimission award date	09 July 2007	15 January 2009
Duration	3 years extendable	30 years
Exploration Status	Active	Active
Current expiration date	22 October 2016	27 May 2017
Area (hectares)	2158.4	85.2

These two licence areas cover the whole project area and known resources. These licences can be extended very 3 years. Ongoing studies of the Valtreixal deposit have been presented to the authorities (el Director Facultativo), in order for both areas to get C.E. (Concesion de Explotacion) status. Valtreixal Resources can then submit a mining plan (Proyecto de Explotacion) and an environmental impact study D.I.A. (Declaración de Impacto) in order to get mining activities approved.

All work that was done by Siemcalsa utilised the UTM coordinate system ED50 in time zone 29. In the current work, all coordinate data has been updated to the ETRS89 coordinate system.

Figure 4-1. Map of Spain Showing Valtreixal Location

Figure 4-2. Map of NW Spain Showing Valtreixal Location

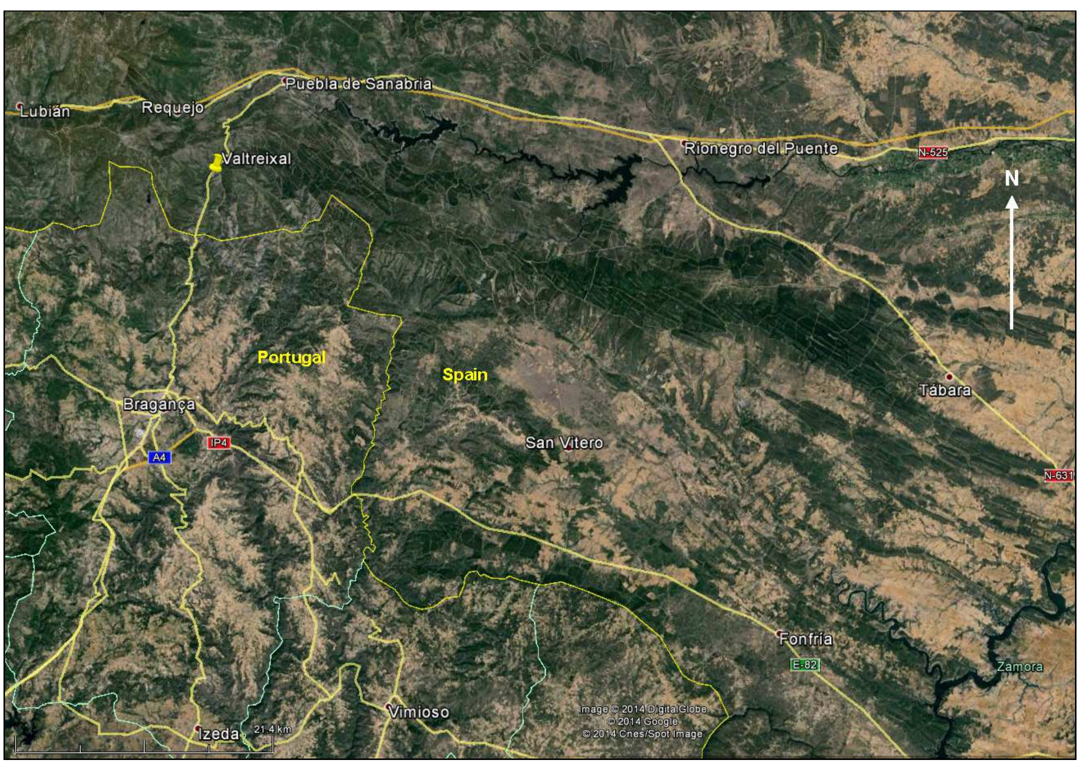
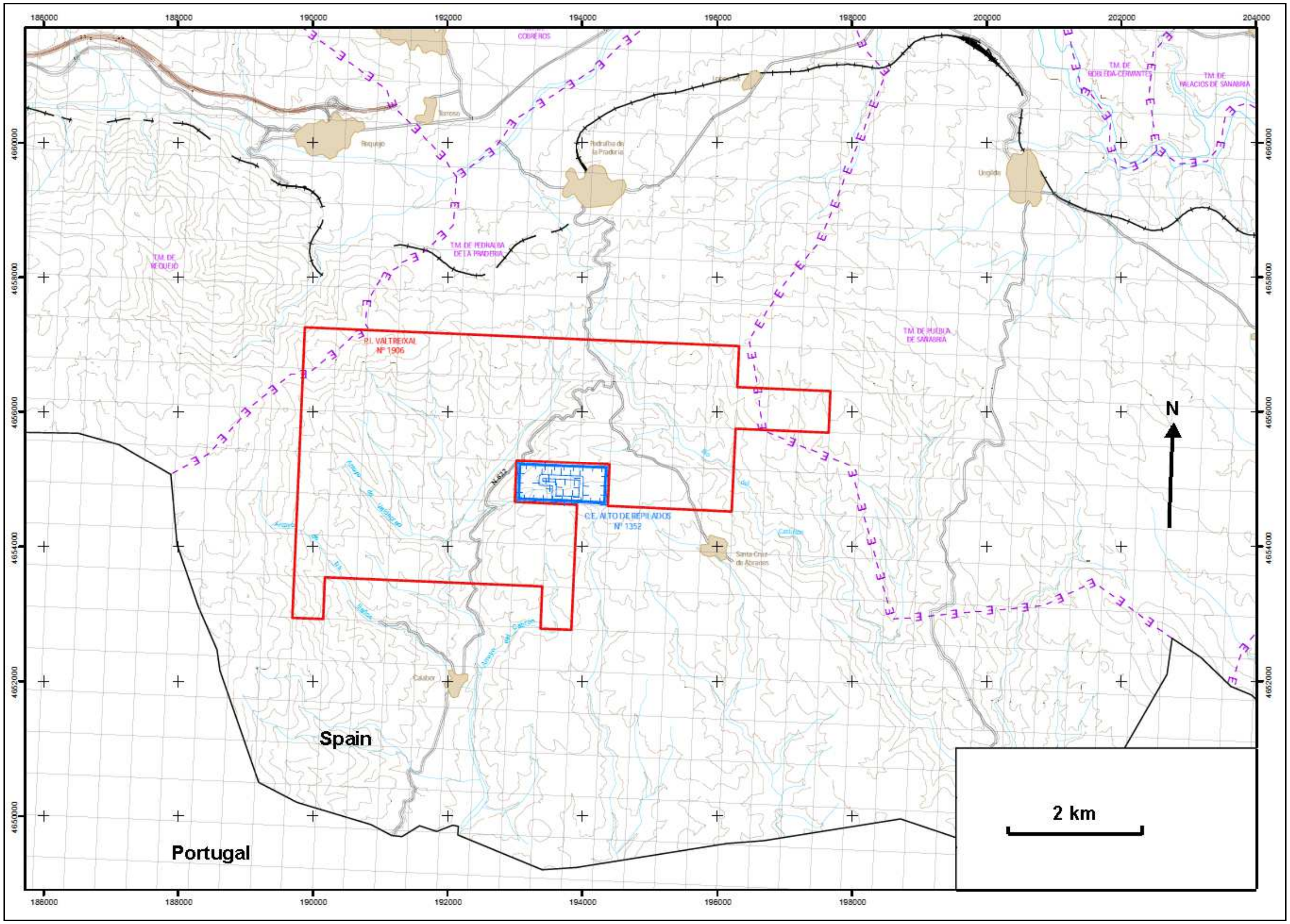


Figure 4-3. Valtreixal Exploration Licenses



5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, PHYSIOGRAPHY

The Valtreixal deposit occurs on the south-eastern side of a valley, in which runs the Calabor River. The Valtreixal site is accessed from the north by means of the main ZA-925 road from Puebla de Sanabria, which cuts across the south-west corner of the deposit, some parts of which outcrop along the road cuttings. The journey time by car from Salamanca is approximately 3 hours, followed by approximately 1 km of dirt road to get the deposit area. A plan of main roads, and with respect to other producing tungsten mines, is shown in Figure 5-1.

The deposit area is located in an area of undulating hills, split up by local small rivers. Among the arboreal vegetation include oak, chestnut, walnut, birch, holly, ash, alder, apple, cherry and pine trees. To a lesser extent there are also poplar trees, made scarce by a tree disease in the late 1980s. Among the shrubs include several species of broom. There are also a variety of herbaceous plants such as foxglove, oregano, nettle, blackberry and mint. Mushrooms are also common, which include the cucurril and several species of boletus. A yew forest stands at Requejo, approximately 5km outside of the deposit area to the north-west, and is said to be the oldest in the Iberian peninsula.

An important part of the Sanabria region is a natural park, the 'Parque Natural Lago de Sanabria y Alrededores', which lies approximately 15km north of Valtreixal and well outside of the deposit area. The protection of this area goes back to 1946, in which Sanabria Lake was declared a "Natural Site of National Interest." This park now covers 22,365 ha. The local fauna is rich, and apart from being one of the last strongholds of the Iberian wolf, contains wild boar, deer and roe deer. Local birds include magpies, hawks, kites, owls and migrating storks. Livestock includes cows, sheep and goats. The nearby Tera River hosts trout and chub.

The elevation of Valtreixal is in the range of 805m – 1050mRL, which places at the threshold between a continental Mediterranean climate and a much colder and damper mountain climate. A weather station located in Robleda-Cervantes, approximately 15 km north-east of Valtreixal, often records the lowest temperatures in Spain, with winter temperatures below $-15^{o}C$. According to the Köppen climate classification, Puebla de Sanabria has a Csb[2] climate (mild with dry warm summer). Monthly average temperatures and rainfalls for Puebla de Sanabria are shown in Table 5-1.

Table 5-1. Monthly Average Climatic Figures – Puebla de Sanabria

Month	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Average Temperature (^{o}C)	1.0	2.1	4.2	6.0	9.7	13.4	16.8	16.4	13.7	8.3	4.2	1.4	8.1
Total Precipitation (mm)	139.0	110.9	79.8	83.2	81.4	40.8	19.6	19.4	62.3	120.2	126.1	143.5	1021.7
Notes													
	. Sources by Ministry of Agriculture, Supply and Envirnment												
	. Temperatures figures derived from 1961 - 2003												
	. Precipitation figures derived from 1961 - 1979												

Figure 5-1. Location of Valtreixal With Respect to Other Tungsten Mines

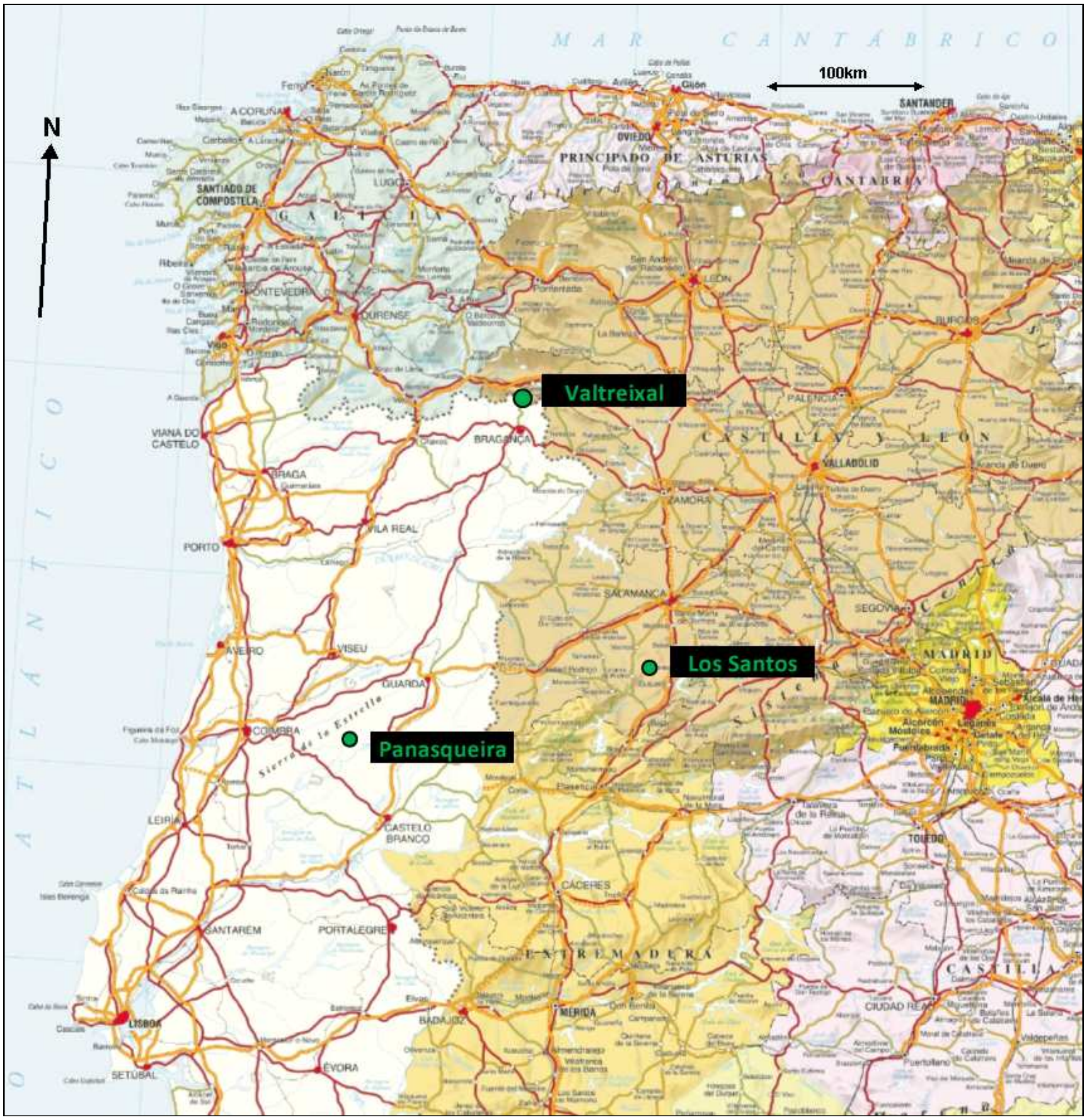


Figure 5-2. Local Contours and River System

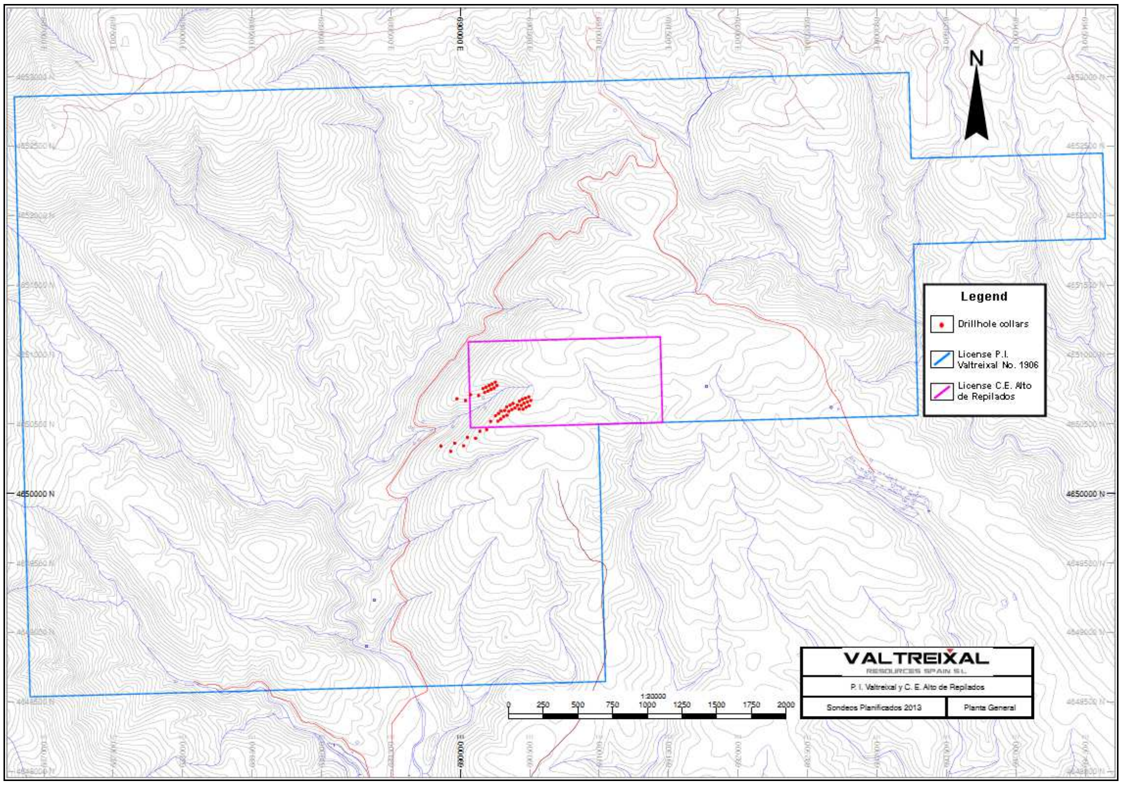


Figure 5-3. View of Valtreixal from North to South

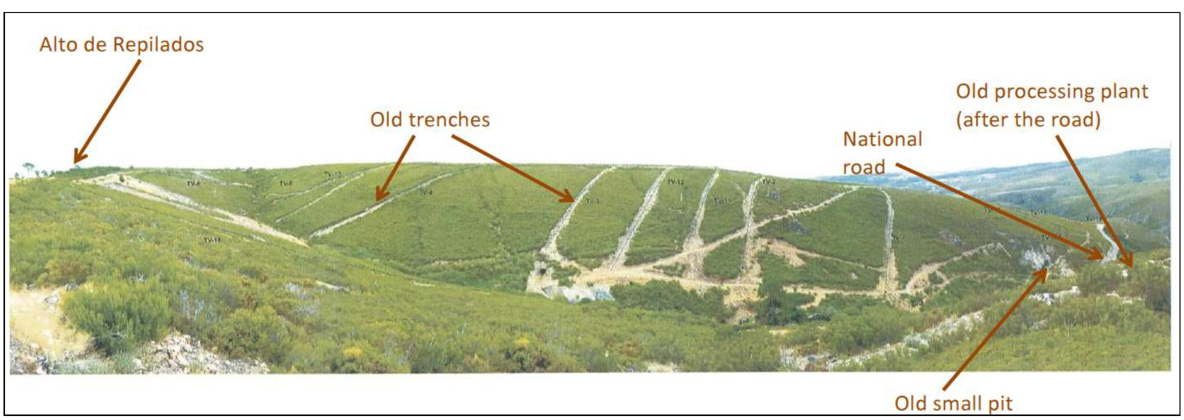


Figure 5-4. View of Valtreixal from North-East to South-West

6 PROJECT HISTORY

6.1 Historical Tin Mining, Up to 1969

The deposit was allegedly exploited by the Romans, although there is no archaeological evidence for this. Mining works had certainly started by the 19th century, with documentation dating back to 1883. By the 1920s, underground mining operations were taking place on several levels.

The first modern mining concession goes back to 1932. At this time the area was called Alto (high) Calabor, as opposed to those closer to the Portuguese border – Bajo (low) Calabor. In 1940, the mine was operated by a German company, and then shut down at the end of World War Two.

In 1948 the property was acquired by Estanifera de S. Barbara, that continued exploitation up until closure in 1969. In parallel, the concession “Alto de Repilados” was granted in 1969, in the eastern end of the deposit, and there is no evidence of works after 1969.

All of this historical exploitation was just focussed on tin inside quartz veins – tungsten was ignored. This is supported by historical records, statements from surviving old workers and preliminary sampling of old waste dumps and plant tailings, returning grades of approximately 0.2% WO_3.

Figure 6-1. Construction of Historical Ore Washing Plant and Gallery Entrance

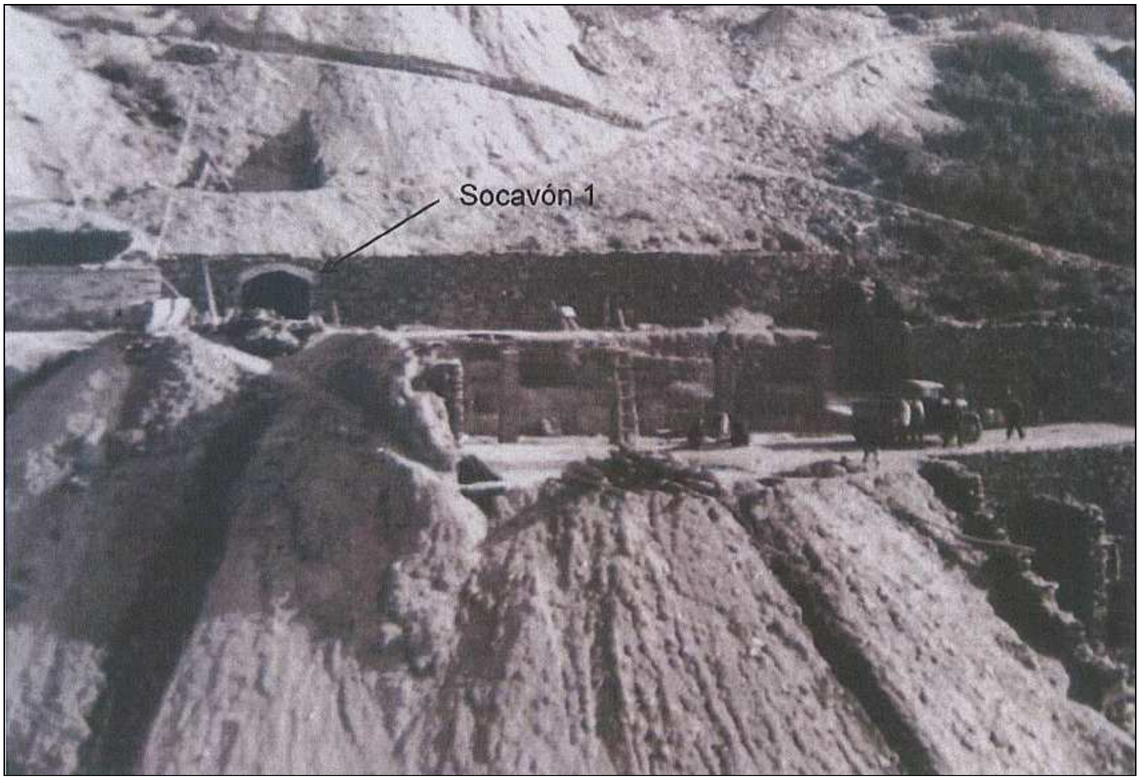


Figure 6-2. Historic Ore Processing Plant

6.2 ENADIMSA Exploration, 1974-2005

In 1974, the state-owned company ENADIMSA started their interest in the area, and as a consequence, exploration work took place from 1979 to 1986. With the collapse of the tin price in 1985, the interest in the property also ended and in 1994 the mining rights (with the exception of "Alto de Repilados") returned to public domain.

Initially in the period up to 1981, the work ENADIMSA completed included:

- Making a photogrammetric map of the whole district.
- A more detailed photogrammetric of the zones around the old mine workings.
- Seven surface trenches, and taking of corresponding samples, covering 660m.

In 1983, ENADIMSA signed a new agreement with Estanifera de S. Barbara, and in the next 30 months completed work which included:

- Development of an underground exploration gallery of 175m in length, allowing 35 x 30 kg samples to be taken at 5m intervals.
- The recuperation of old underground galleries, totalling 140m in length and taking 28 samples.
- Three surface exploration trenches.
- Various preliminary mineralogical tests.

From 1984-1985, ENADIMSA continued with:

- Three drillholes, 273m total in length.
- More mineralogical tests.
- Ten more surface trenches.

6.3 SIEMCALSA Exploration, 2006-2012

In 2006, the state-owned SIEMCALSA (through the regional government) was granted the exploration permit for P.I. Valtreixal No. 1906, which also covered 83 areas of mining rights. This was further augmented in 2007 with the granting of permissions for C.E. Alto de Repilados No. 1352, which included a further 5 areas of mining rights.

A summary of the work completed by SIEMCALSA from 2008-2011 includes:

- Recuperation of more underground galleries, so twenty galleries made accessible, with a total length of 2,500m, and allowing the taking of 260 samples.
- Twenty six surface trenches, for a total of 2,836m.
- Drilling of six surface diamond drillholes, for a total of 1,227m.
- Mineralogical tests, done in the laboratories of Wardell-Armstrong (WAI) in England.

Photographs depicting this work, for gallery re-access, underground sampling and surface trench sampling are shown in Figure 6-3, Figure 6-4 and Figure 6-5, respectively. The extent of all the mapped underground galleries are shown in Figure 6-7.

The mineralogical samples were taken on the 2nd level of the underground gallery called "El Trincheron" in the central area of the deposit, and were considered representative of the deposit overall with fairly continuous grades. The channels taken were approximately 30 cm in height and 10 cm in depth, and covered a transverse length of 24.7m. The average sample grades of this intersection were 0.126% Sn and 0.128% WO_3.

This gave a total sample of approximately 3,000kg, which after quartering yielded a sample of 410 kg, which was sent to WAI for metallurgical testing. At WAI the sample was coned and quartered to produce a representative 15kg sub-sample, which was used for the test programme. The WAI testwork included gravimetric tests using denser media separation, over different size fractions, as well as QEMSCAN analysis.

Figure 6-3. Re-establishing Access to Underground Gallery

Figure 6-4. Underground Channel Sampling – SIEMCALSA

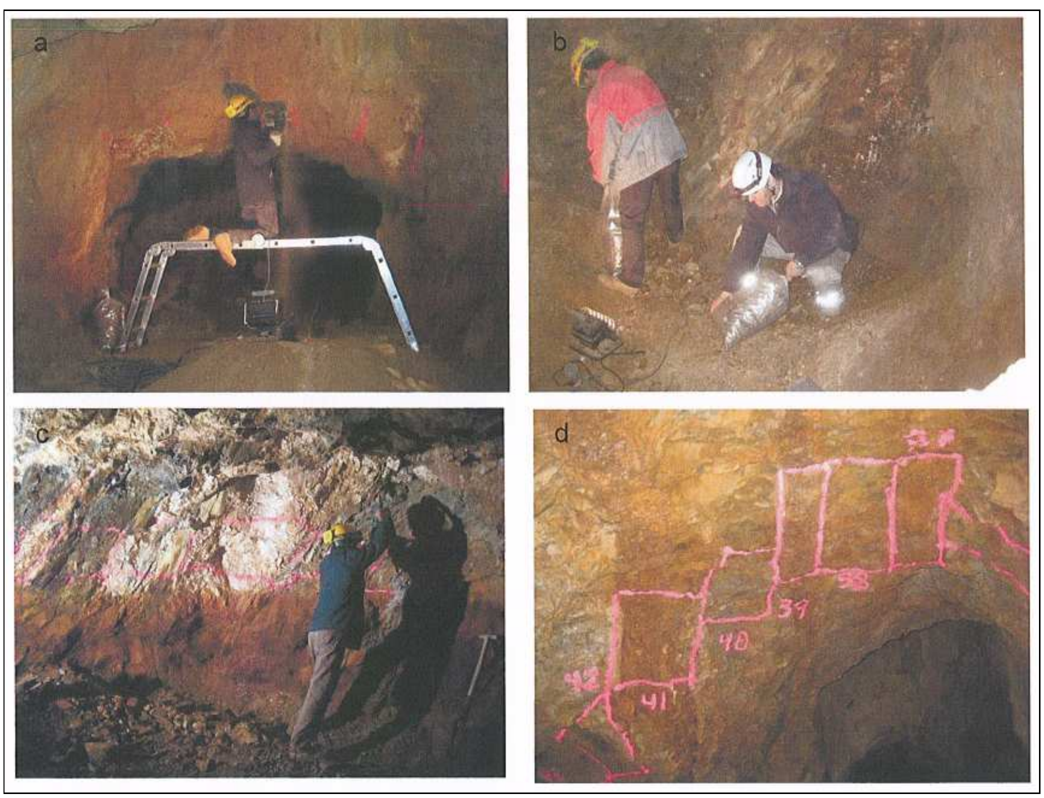


Figure 6-5. Trench Sampling

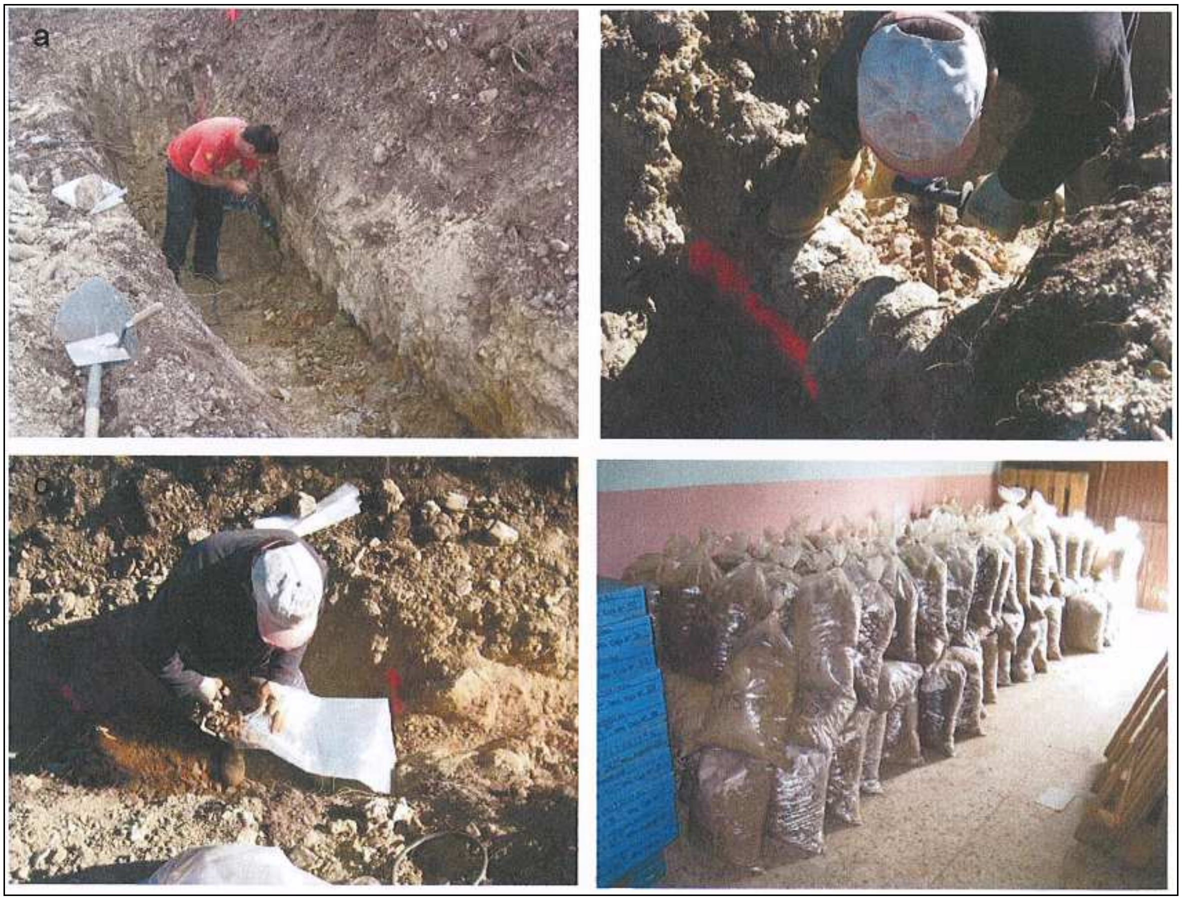
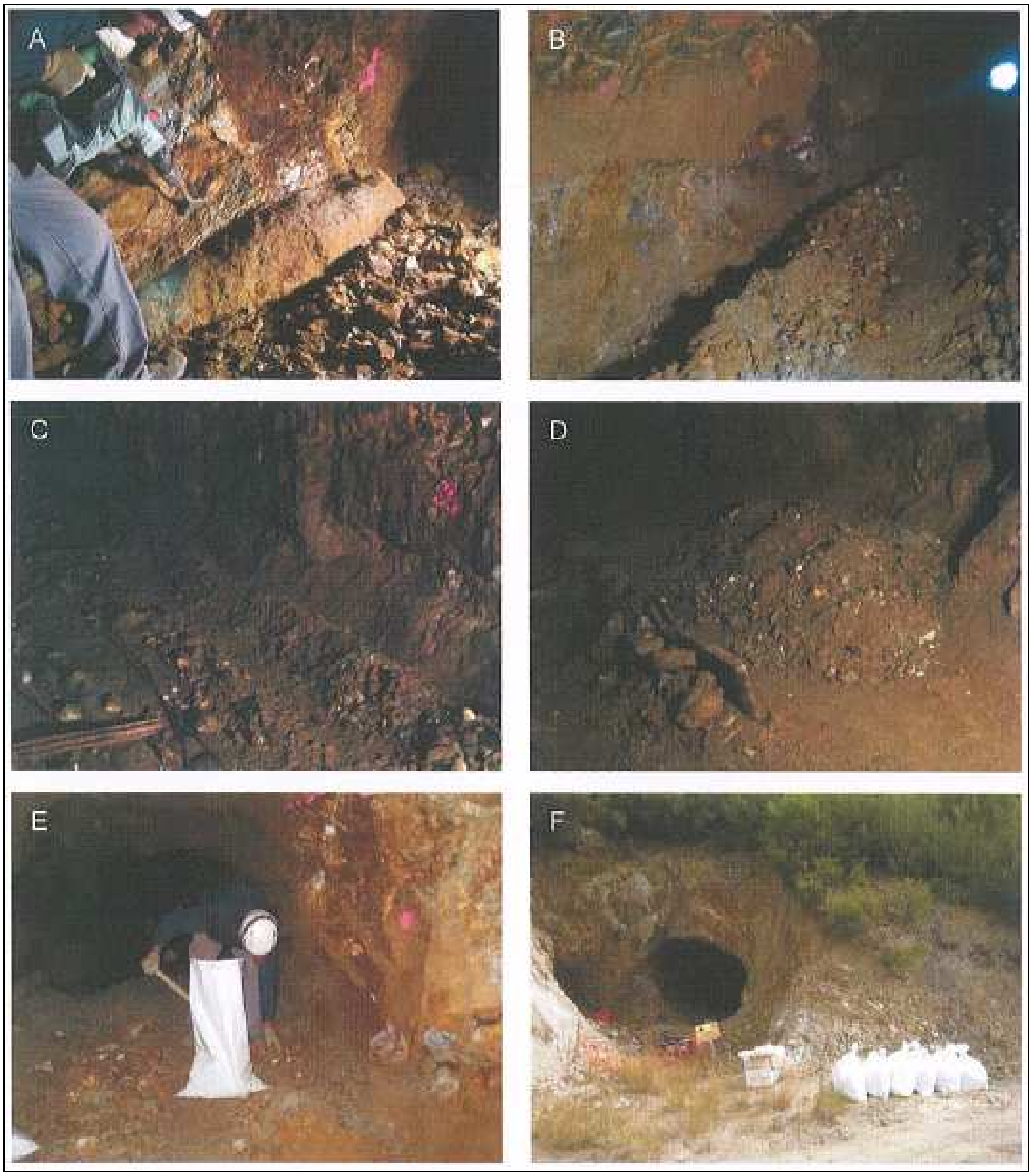


Figure 6-6. Mineralogical Sampling



A Cutting of chanells with pneumatic hammer.

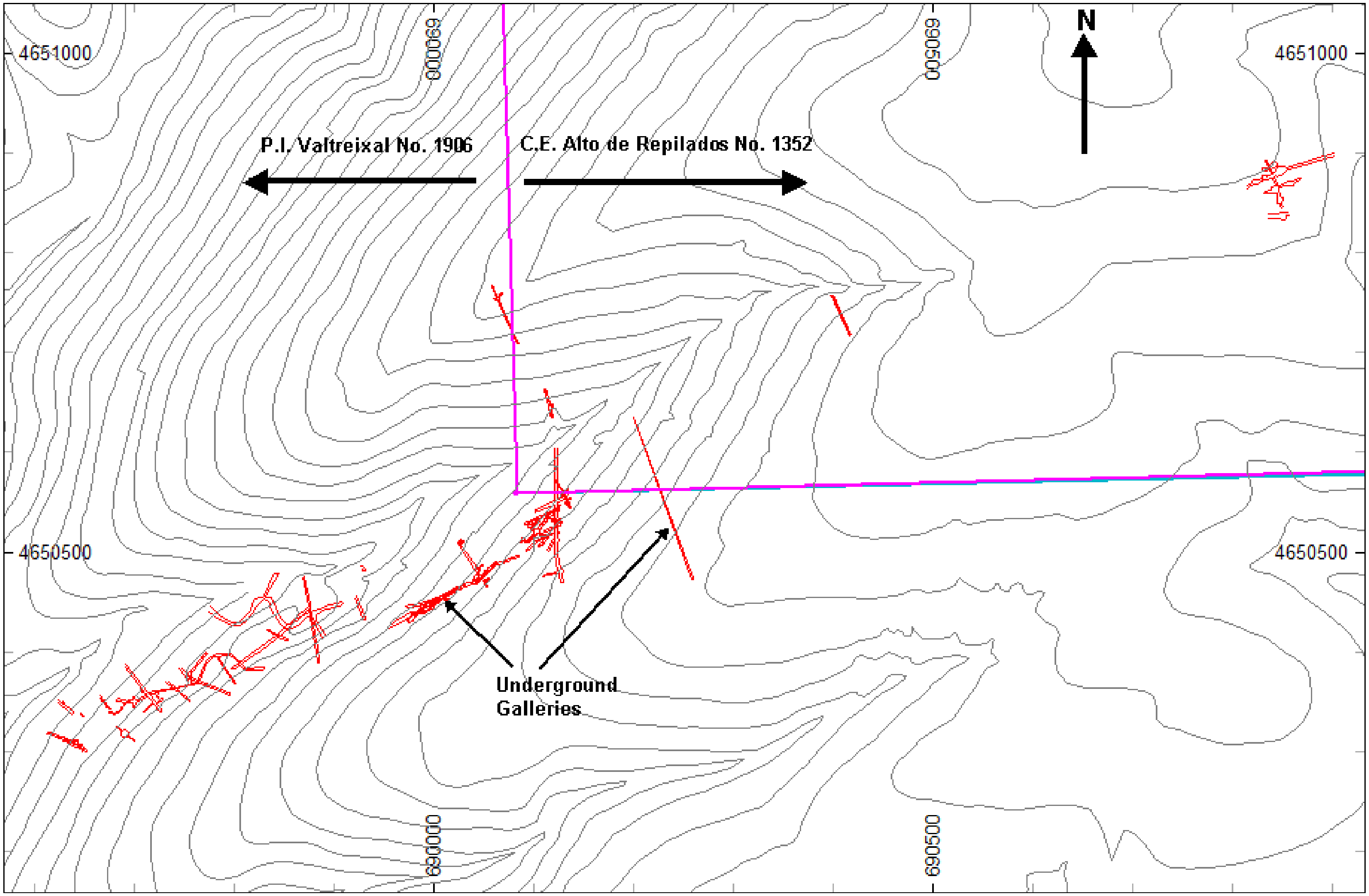
B/C Piles resulting from channel sampling.

D Final pile from one complete channel.

E Collection of sample.

F Sample bags ready for transfer – 475kg in total – 12 sacks of 40kg each.

Figure 6-7. Plan of Underground Galleries



7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The Valtreixal area is located on the northern flank of the Alcañices syncline, in a folded zone referred to as the CentroIiberian Zone of the Hercynnian massif. A plan of the region geology is shown in Figure 7-2.

Tangential movement along the regional Bragança fault may have assisted in creating dilation zones in the Ordovician lithology. The Calabor River now follows the general direction of the fault. These dilation zones then permitted, along a north-east trend, the development of quartz veins and later tin/tungsten mineralisation in shale. It is considered that a mineralising hydrothermal system of Hercynian age (330 Ma to 280 Ma) was powered by a hypothetical underlying cooling granite in the Valtreixal area.

It is generally considered that the northward movement of the ancient continent, Gondwana and its collision with Laurentia to form the super-continent Pangea, resulted in the Hercynian orogeny. This orogeny was pivotal in the formation of tin/tungsten deposits in this type of setting. As Gondwana advanced, overriding and pushing down, subducting the thin oceanic basaltic crust, there developed a geo-shear zone which dipped back under the continental frontal mountains. This geo-shear would have penetrated through the oceanic crust into the upper mantle where serpentinisation takes place. Serpentinite development is highly exothermic and may circulate accessory calcium and additional heat into the hydrothermal system. At greater depths the geo-shear may penetrate continental, denser cratonic rocks with entrained primordial, undifferentiated crust having higher amounts of heavy elements.

Figure 7-1. Plan of Geology – NW Spain and Portugal

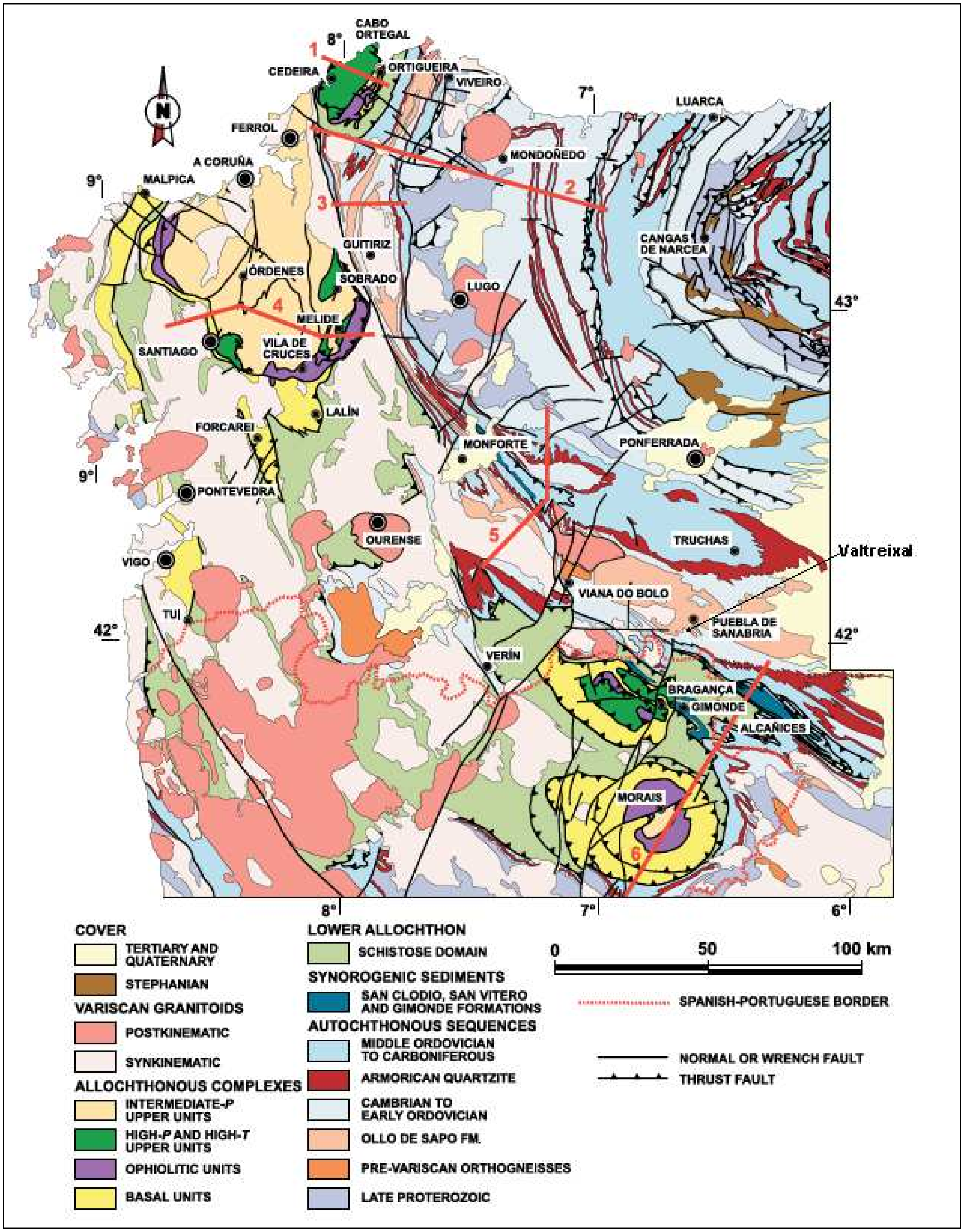
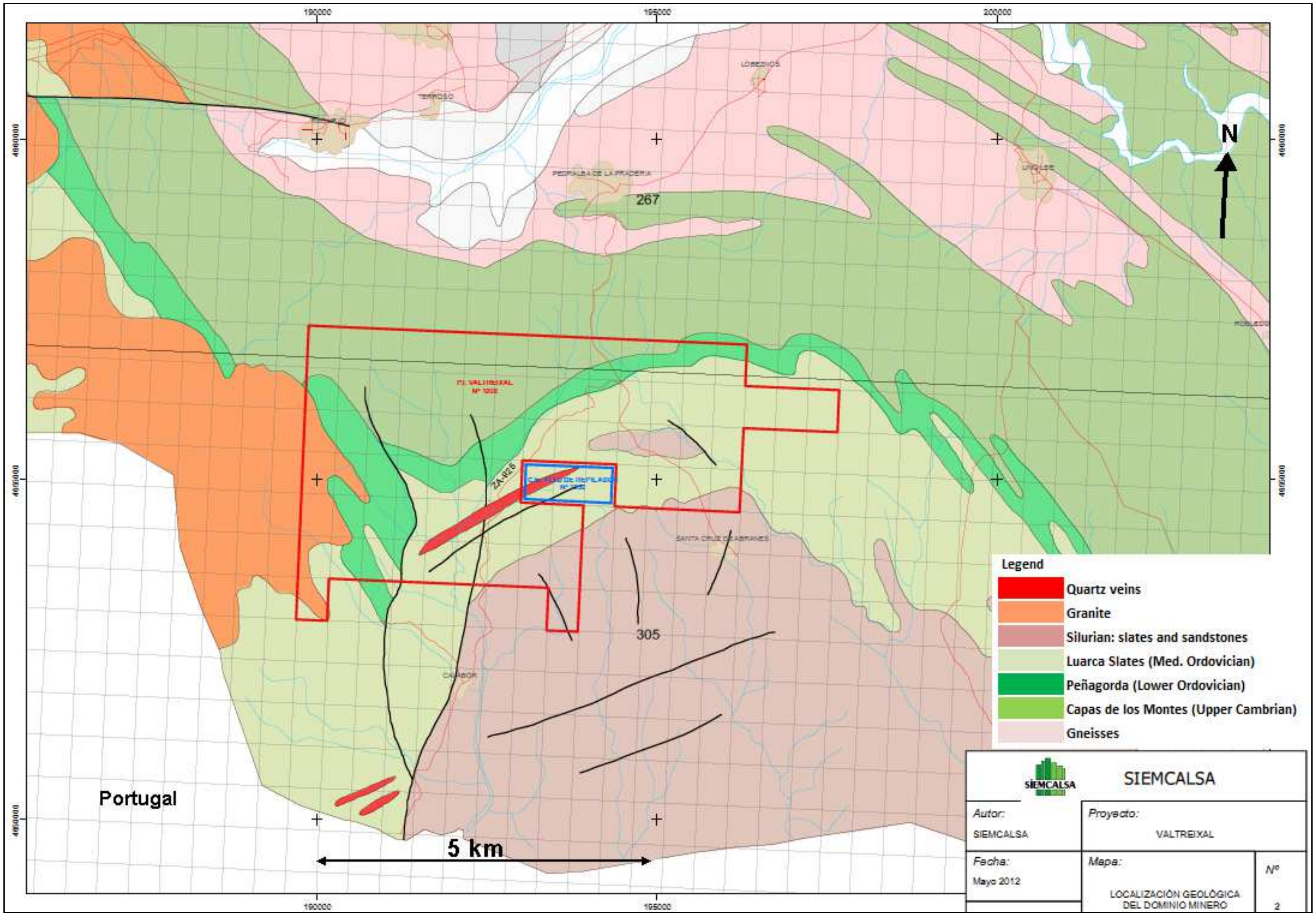


Figure 7-2. Plan of Regional Geology



7.2 Local Geology

The local Valtreixal stratigraphy in the Valtreixal area is dominated by 3 main formations, all of which broadly strike SW-NE, and dip at approximately 80º to the south-east.

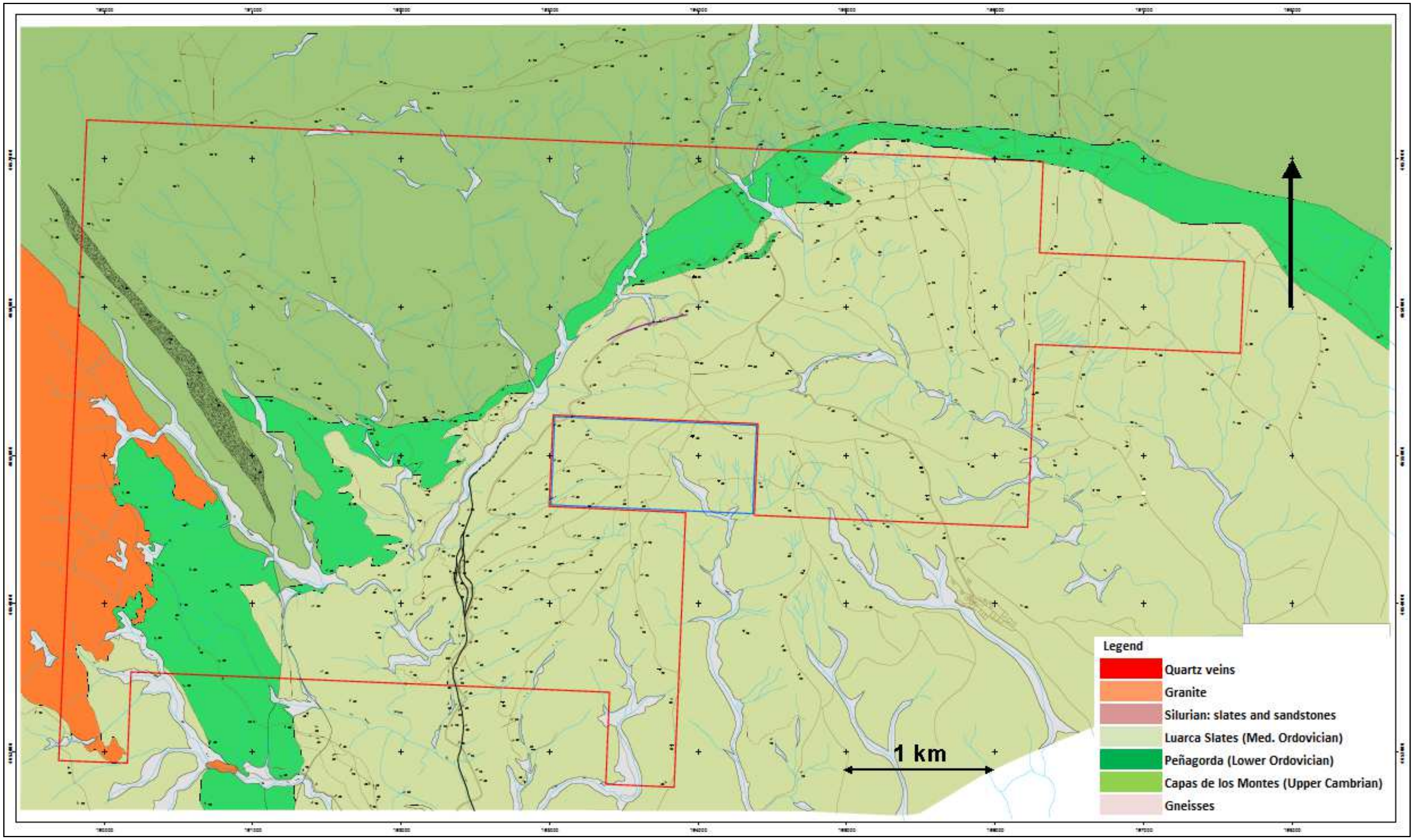
a) Schists - Capas de los Montes. Cambrian/ordovician. Very stratified and transformed by regional metamorphism, with intercalated quartzites, and marked at the base by conglomerates. Thickness approximately 1000m.

b) Quartzites - Peña Goda/Culebra. Ordovician. Alternating with a variety of types and colours of intercalated schists. Thickness approximately 50-70m.

c) Slates – Pizarras de Luarca. Ordovician. Pelitic series of siliceous slates, phyllites and schists. This formation hosts most of the mineralisation at Valtreixal. High frequency of segregated quartz veins and schist bands sometimes rich in sulphur. Overall thickness approximately 300-600m.

A plan of the local geology is shown in Fig 7.3.

Figure 7-3. Plan of Local Geology



7.3 Mineralisation

Much of the mineralisation, especially scheelite, is situated away from the quartz veins and appears to be stratabound. Tin, in the form of cassiterite, occurs in and around the quartz veins. The Valtreixal linear mineralised zones appear, in a general sense, to be confined to specific stratigraphic intervals and there appears to be a degree of separation into tin and tungsten zones. Because of the stratabound nature of the mineralised zones within a shale basin some may consider a sedimentary, syngenitic origin for the tungsten mineralisation to be plausible. However, this is unlikely to have eventuated at Valtreixal, because the scheelite hosting shale is of Ordovician age, 488 Ma to 444 Ma, and thus predates by a considerable margin the Hercynian, at 330 Ma to 230 Ma, tin/tungsten mineralisation episodes.

The Valtreixal tungsten (scheelite) and tin (cassiterite) mineralisation exhibits two principal modes of occurrence within southeast dipping, weakly schistose Ordovician shale in the Hercynian (Variscan) tectonic belt. Irregular quartz veins cut the moderately steeply southeast dipping shale and may have associated tin and tungsten mineralisation especially where the veins are brecciated and the adjacent wall rock has been altered to sericite schist, as shown in Figure 7-4.

Figure 7-4. Brecciated Quartz Veins Exposed In Adit

In places it was observed that minor amounts of white kaolin clay had developed by schist alteration. It is notable that mica, probably muscovite coats some surfaces of the glassy quartz fragments and minor open spaces may indicate incipient, vuggy greisen development. It is postulated that originally the quartz veins developed along fractures in the country rocks.

Subsequently, the possible intrusion of a hypothetical granite body at depth provided a heat source to drive a hydrothermal system, brecciating the quartz veins. Any such granitic or other heat source was sufficiently deep so that the shale, seen at Valtreixal, is beyond the granitic metamorphic aureole. Strong hydro-fracturing and concomitant seismic activity may have created temporary weakness along the laminated shale allowing lateral migration of mineralising solutions away from the brecciated veins or other sources for a considerable distance.

The shale is relatively impervious to solution penetration across the bedding by absorbing stress and by yielding more plastically. High pressure fluids may have ruptured many of the rigid quartz veins. The country wall rock adjacent to the brecciated quartz vein is metamorphosed to sericite schist. In general the shale, based on visual and textural features, is considered to be in the low, chloritic metamorphic range.

Daytal drill core in the mineralised shale, viewed under UV light, as shown in Figure 7-5, and typically demonstrates scattered scheelite throughout and on a lesser scale scheelite deposited along the foliation.

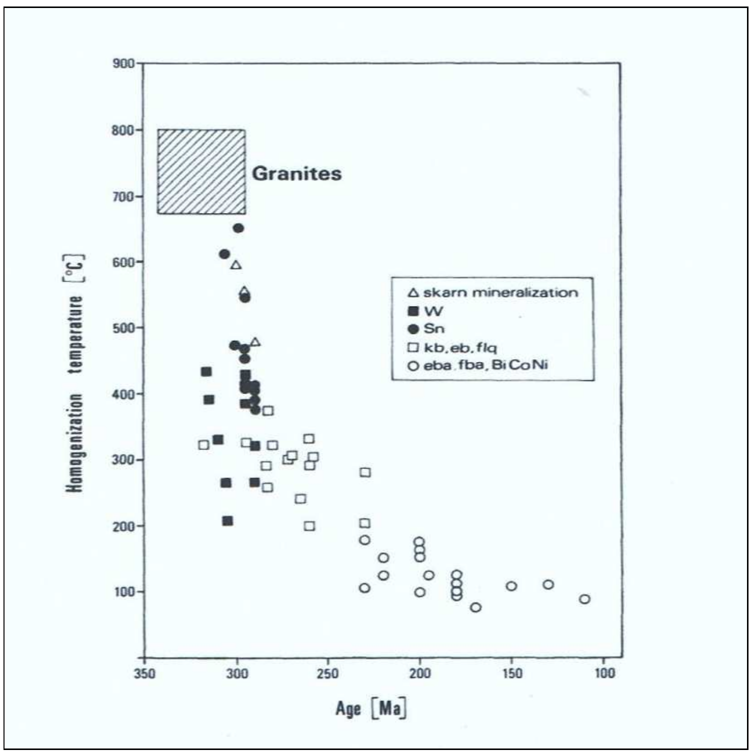
Figure 7-5. Example of Scheelite Under UV Light in Drill Core

In relation to the above it may be appropriate to mention a less likely genetic process which may have had a role in relation to shale mineralisation. There could be a possibility that regional thrusting from the southeast may have produced delamination and dilatant effects on the shale, permitting mineralisation along the foliation followed by metasomatic dissemination of scheelite throughout the shale. This hypothesis would also help to explain the apparent strata bound nature of the mineralisation.

The cooling history of the Hercynian age, 280 Ma to 330 Ma, Erzgebirge Granite in Germany (Figure 7-6) indicates tin being deposited at temperatures between 375° and 650°C which may be similar to the tin/tungsten depositional regime at Valtreixal. Tungsten overlaps with tin being deposited at up to 440°C and as low as 200°C. These depositional temperature ranges may explain in part why at Valtreixal tin may be more abundant in the geothermally hydrofractured quartz vein vicinity, whereas tungsten minerals with lower depositional temperature may extend laterally out into the shale.

Figure 7-6. Tin/Tungsten Deposition Temperatures - Erzgebirge Granite, Eastern Germany

(After Thomas and Tischendorf 1987)



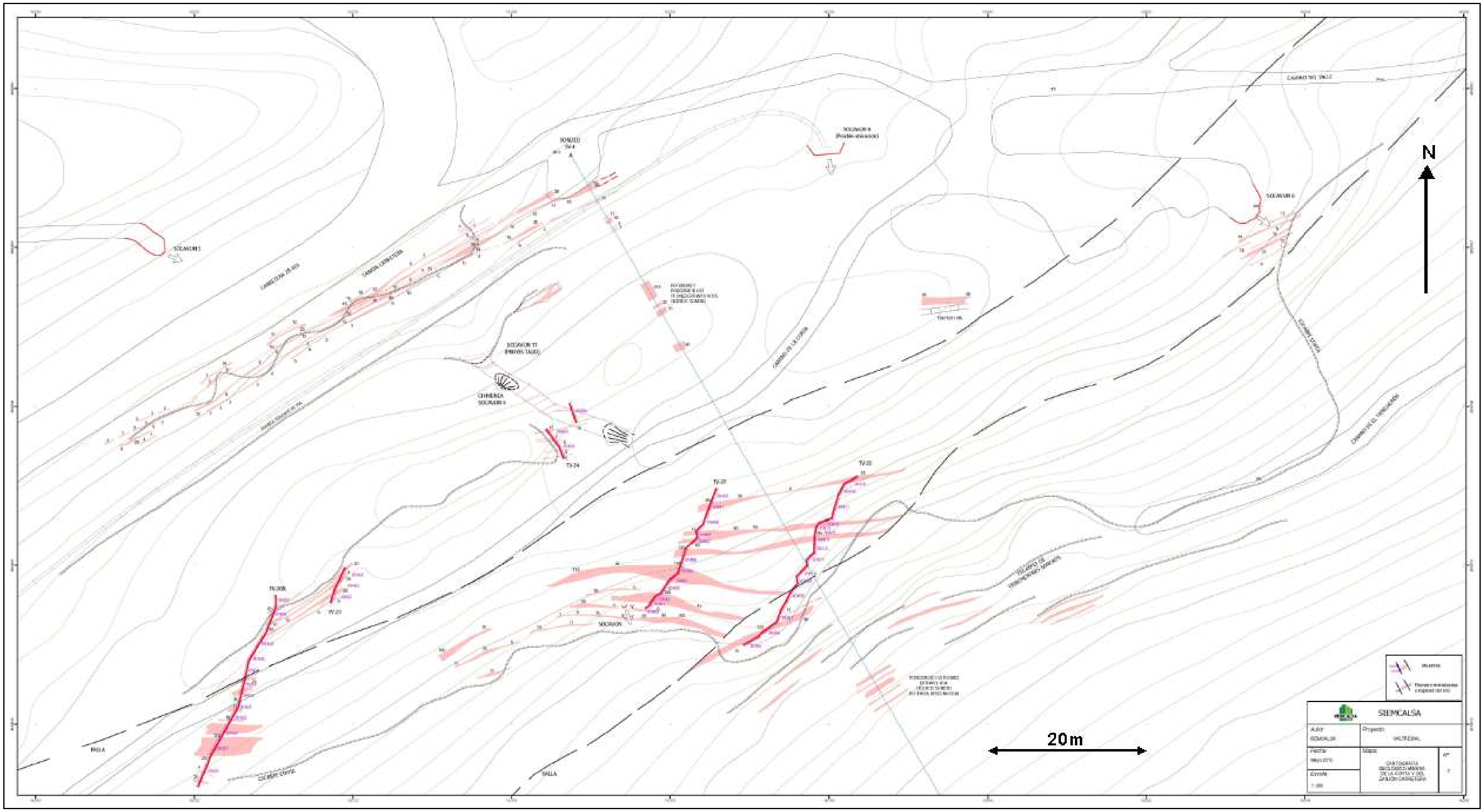
8 DEPOSIT TYPES

The mineralisation at Valtreixal can be classified as a complex vein deposit. The tin mineralisation, in the form of cassiterite, is hosted by a mixture of individual veins, veinlets and vein swarms, generally with the same overall dip and dip direction. The distribution and width of veins can be quite erratic within localised areas.

The tungsten mineralisation, in the form of scheelite, is also associated with the quartz veins, but as discussed in Section 7.3, also often occurs laterally outside of, and sub-parallel to, the quartz veins.

An example of the interpreted quartz veining from trenches and surface outcrops, in the south-western part of the deposit, is shown in the plan in Figure 8-1.

Figure 8-1. Plan of Interpreted Quartz Veins – SW Part of Deposit



9 EXPLORATION

Daytal surface trenching exploration started in November 2013. The trenches were cut with an excavator, so as to produce trenches approximately 0.50m wide and generally 1m deep, or deeper so as to reach bedrock. The trenches have been oriented at right angles to the general direction of mineralisation.

Channel samples were then taken in the walls of the cut trenches, generally 2m in length, as shown in the example in Figure 9 1. This cutting was done manually as well as with an electric hammer. Photographs of trenching operations are shown in Figure 9 2 to Figure 9 4. Typically the sample weight per sample was 8-10 kg. A summary of the trench sampling done by Daytal from 2013-2015 is shown in Table 9 1.

Figure 9-1. Example of Channel In a Surface Trench

Table 9-1. Summary of Daytal Trenching

Trenches	Length *m*	Average Trench Length *m*	Sn Samples	WO3 Samples
18	2,654	147	806	1,131

Figure 9-2. Use of Electric Hammer

Figure 9-3. Samples For Collection

Figure 9-4. Cut Trench

10 DRILLING

Daytal drilling exploration started in July 2013, using drilling contractors ATSG and GEONOR. Over the next year and a half, 11 drillholes were drilled, with an average length of 160m. Holes were generally collared as PQ (core 122.6mm) and then changed to HQ (core 63.5mm) once the alteration zone was passed. Some of the holes were changed to NQ (core 47.6mm) after approximately 100m. Examples of the drilling operations are shown in Figure 10-1 - Figure 10-4. A summary of the Daytal drilling from 2013-2015 is shown in Table 10-1.

The drill equipment used for the Daytal drilling were Longyear DB540, LF90D and LM90 rigs, as well as Christensen 3000, Rolatec 600 and PDB 1500 drill rigs. Downhole survey measurements, of inclination and orientation, were made using Reflex Gyrosmart equipment.

All core was transferred directly from the core barrel to correctly sized wooden core trays at the rig site. Wooden core blocks were placed in the trays to record downhole depths at the end of each drill run. At intervals the core trays were transported to a centralised core handling facility in Puebla de Sanabria, as shown in Figure 10-5.

Here the core was geologically logged by Daytal geological staff. Logging information recorded included:

- Lithology
- Weathering
- Mineralisation
- Vein directions/widths
- Schistosity directions

Logging of the core enabled mineralised portions of the holes to be selected for assay. These selected samples were sawn (Figure 10-6) such that one half core was sent to the laboratory. Sample intervals were selected on geological criteria, with a maximum sample length two (2) metres. All the remaining core, as well as coarse and pulp rejects, are stored at Daytal's facility in Puebla de Sanabria.

Density measurements were made in most of the major lithologies, by weighing the sample in air and in water. Plastic film was used to cover the samples for the water measurements.

Table 10-1. Summary of Daytal Diamond Drilling – 2012-2015

Holes	Length (m)	Average Hole Length (m)	Sn Samples	WO_3 Samples
59	10,716	182	3,478	4,496

Figure 10-1. Drill Set-Up – Hole VS002

Figure 10-2. Drill Set-Up – VS008

Figure 10-3. Drill Set-Up – VS004

Figure 10-4. Drill Set-Up – VS006

Figure 10-5. Core Logging Facility

Figure 10-6. Core Sawing Facility

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Sample Preparation

All of the sample preparation steps take place at the Los Santos laboratory facility. The preparation steps are depicted in Figure 11-1, with the key pieces of equipment in Figure 11-2 - Figure 11-5. The same steps are used for samples both exploration drillholes as well as for trench samples.

The same preparation steps are used for both diamond drillhole and trench samples. All rejects from all stages are stored and archived.

Figure 11-1. Sample Preparation Steps

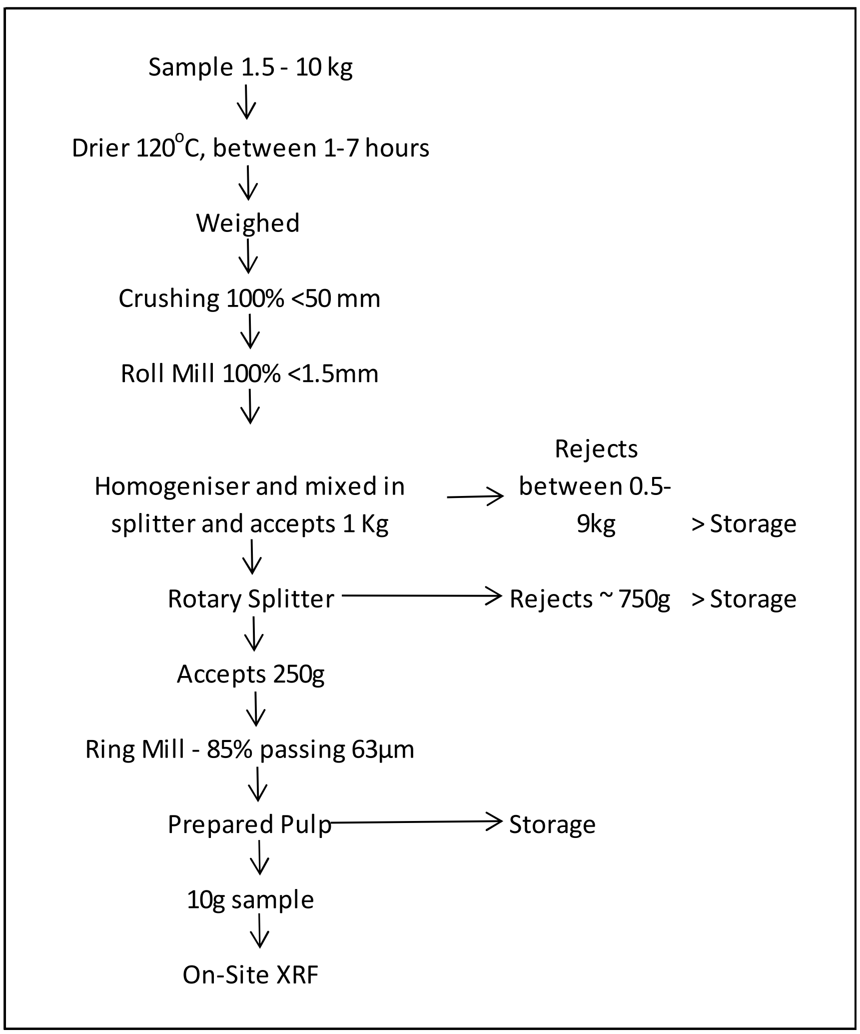


Figure 11-2. Sample Drier

Figure 11-3. Jaw Crusher

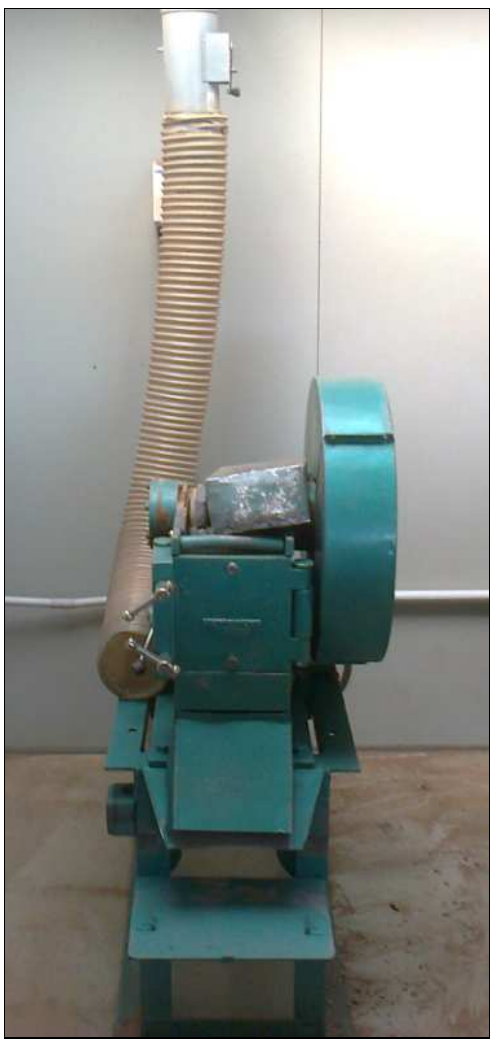


Figure 11-4. Roll Mill

Figure 11-5. Ring Mill Containers

11.2 Laboratory Sample Analysis

Daytal have their own on-site laboratory at Los Santos, equipped with a AXIOS XRF Spectrometer. This spectrometer is calibrated daily against 3 standard samples for different grade ranges. These sets of standard samples were certified by ALS Seville.

For the Energy Dispersive-type XRF analyzer at Los Santos, it is important that the reference standards have a similar matrix composition as the unknown samples to be determined. This is because the unit's read-out is sensitive to X-Ray back-scatter, which in turn is a function of the "background" composition of the sample (principally, its Fe, Al, Ca, Na and Mg contents). This is why Daytal uses Los Santos materials as standards. The ALS Seville results of these samples were used as the reference grades.

Blank reference samples are also measured for approximately every 50 samples.

The on-site laboratory facilities at Los Santos are shown in Figure 11-6 - Figure 11-7.

Figure 11-6. On-Site Laboratory Facilities at Los Santos

Figure 11-7. Los Santos AXIOS XRF Assaying Equipment

11.3 Quality Control

No QA/QC information was available with respect to the historical Siemcalsa data.

QA/QC data obtained for the Daytal assay data are summarised below, derived from the laboratory at Los Santos.

- Internal pulp duplicates, 248 out of 4496, approximately 6%
- Field duplicates, 30 out of 1737, approximately 2%
- Fine Blanks – 25 measurements
- Coarse Blanks – 34 measurements
- Granulometric analysis – 81 measurements

A summary of the results for the duplicates are shown in Table 11-1, with a plot of the WO_3 HARD (half-absolute-relative difference) results are shown in Figure 11-8. For the WO_3 pulp duplicates, 86% of the duplicates have a HARD value less 10%, as shown in Figure 11-8.

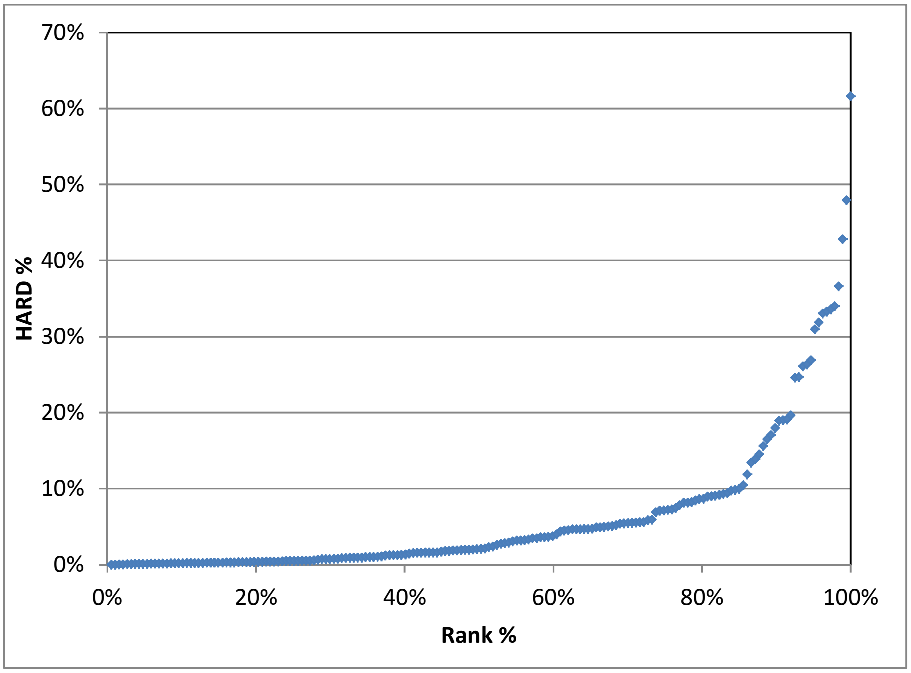
Table 11-1. Summary of QAQC Results – Duplicates

Type	Field	Number	Frequency of Duplicates	HARD @ 90%	Correlation Coefficient	Slope of Regression Line	Prop'n Misclassified
Field Duplicates	Sn	22	2%	42%	0.654	0.42	3.2%
	WO_3	30	2%	30%	0.961	0.934	3.2%
Pulp Duplicates	Sn	248	6%	6%	0.9998	1.0057	0.0%
	WO_3	248	6%	18%	0.9997	0.9957	0.4%

Notes

. 0.07% WO_3 used as cut-off in mis-classification test
. 0.07% Sn used as cut-off in mis-classification test
. HARD = half absolute relative difference

Figure 11-8. HARD Plot for WO_3 Pulp Duplicates



The blanks' duplicates results are summarised below in Table 11-2. Together with no relationship between blanks and previous assays, these blanks results are acceptable.

Table 11-2. Summary of Duplicates' Results

Fine Blanks	Number of Samples			75
	Proportion Above	0.01	$\%WO_3$	7%
	Proportion Above	0.005	%Sn	0%
Coarse Blanks	Number of Samples			34
	Proportion Above	0.01	$\%WO_3$	3%
	Proportion Above	0.005	%Sn	3%

The granulometric results were also acceptable, with less than 10% of measurements passing less than 95% at 75 µm.

In the opinion of the QP, these QA/QC results support the resource estimation results that have been derived in the current study.

12 DATA VERIFICATION

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

- Inspection of drillhole collars, trench positions, surface outcrops and accessible underground galleries on-site at Valtreixal.

- Inspection of core storage and handling facility at Puebla de Sanabria.

- Check measurement of drillhole collars and gallery portals using handheld GPS. This was done at 12 different positions over the whole deposit strike length, and produced acceptable results.

- Analysis of duplicates for Siemcalsa samples that were re-assayed by Daytal. The original Siemcalsa sample had been analysed at ALS laboratories in Spain. The Daytal samples were analysed at the Los Santos laboratory.

Table 12-1. Summary of Check Assay Results

Original assays – Siemcalsa > ALS

Check assays – Daytal Los Santos laboratory

Field	Number	HARD @ 90%	Correlation Coefficient	Slope of Regression Line	Prop'n Misclassified
WO_3	29	12%	0.957	0.908	6.9%
Sn	29	7	0.991	0.917	0.0%

Notes

. 0.07% WO_3 used as cut-off in mis-classification test

. 0.07% Sn used as cut-off in mis-classification test

. HARD = half absolute relative difference

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

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Mineralogical Analysis

In the 2014 conceptual study completed by Saint Barbara (StB), all historical and recent mineralogical reports were examined in order to establish the likely liberation size of the tin and tungsten bearing minerals.

The Siemcalsa 2011 study notes the lack of potential for conventional gravity concentration at sizes above 1mm. Liberation of cassiterite below 1mm was moderate to good indicating a primary grind size of less than 1mm. StB agrees that sequential flotation of cassiterite and scheelite is complex and would require intensive testing at laboratory and pilot scale.

The Siemcalsa 2009 study refers to cassiterite crystals up to 30mm, but not usually greater than 5mm. An example of such crystal sizes was found on the footpath in the vicinity of the ENADIMSA Galeria. Wolframite was reported as rarely up to 25mm with scheelite finely disseminated up to 1-2mm. Sulphides were reported as occurring in two modes: finely disseminated and with large crystals in hydrothermally altered areas. Sulphides were postulated as being re-mobilised by the same fluids that carried the tin and tungsten. Cassiterite and wolframite occurs in the quartz veins

The SGS 2013 study reports a very different picture of economic mineral grain sizes in schist. Average cassiterite and scheelite grain sizes were reported as 16µm and 95µm respectively. Furthermore, SGS tentatively report that, while 98% of tin occurs as cassiterite, 14% of tungsten was observed in Titanite, a calcium titanium silicate mineral of low s.g. (3.5) with chemical formula $CaTiSiO_5$. The tungsten presumably substitutes for titanium in the Titanite matrix.

A petrologist, A. Pinto, completed an initial mineralogical study in May 2014. Based on a sample from drillhole VS005, the study reported scheelite grains with a very wide size range from 10-1000µm. Fluorite was noted to occur together with scheelite. Scheelite and cassiterite can have similar flotation characteristics to each other, and to fluorite. The presence of fluorite in tin concentrates is highly deleterious. While information in StB's possession does not list fluorite as a deleterious impurity in scheelite concentrate, StB recommends checking this with any potential future customers.

For a sample from drillhole VS003, fluorite was also noted in this schist sample in which the scheelite occurs in a banded form in grey schists. Scheelite grain size was not stated.

For a sample from trench VT011, the assays shown were assayed as having tungsten values, but no visible fluorescence, and no scheelite or wolframite was detected under visible light. The presence of altered schist was noted, which perhaps supports the observation by SGS of the presence of Titanite.

13.2 Process Testing

Scheelite flotation testing was undertaken in 2013 on a sample of schist from the ENADIMSA Gallery by AGQ. This showed that high (>80%) rougher recoveries were possible to a concentrate of just 3.6%W. However, concentrate cleaning produced an average grade of 16%W with recovery of only 45% on average.

For an initial test run, StB commented that this is a reasonable start at this stage of investigation, but it cannot be assumed that scheelite flotation would form a part of the processing route.

Heavy liquid separation tests, again on a sample from a ENADIMSA Gallery, by Wardell Armstrong International (WAI) indicated that the pre-concentration of the economic minerals was not possible at particle sizes above 1 mm. However, below 1mm, reasonable liberation to produce a rough concentrate by gravity was indicated. No conclusions could be drawn on the tungsten mineralisation from the QEMSCAN analysis, but cassiterite liberation was considered adequate below 1mm to make a rough concentrate for subsequent regrinding and final upgrading.

At the current state of knowledge, StB consider that a potential future milling operation could consist of:

- Crushing, grinding and gravity separation of scheelite and cassiterite into a bulk concentrate;
- Removal of sulphides from the bulk concentrate by flotation; and
- Drying and electrostatic separation of the bulk concentrate into scheelite and cassiterite concentrates.

Based on the limited mineralogical and metallurgical information currently available and making comparison with the Los Santos plant, StB anticipate tin and tungsten recoveries of approximately 70% and 60%, respectively. Allowing for inefficiencies in electrostatic separation due to the likely fine nature of the bulk concentrate, StB proposed the following overall metallurgical performances:

- 65% tin recovery to a 50%Sn concentrate
- 55% tungsten recovery to a 65%WO_3 concentrate

Pilot plant studies have also been completed by the company ADVANCED MINERAL PROCESSING, SL (AMP) and concentration tests have been performed by the technical personnel working in mine-Fuenterroble Los Santos (Salamanca) laboratory. All of this testwork has allowed the design of an ore beneficiation process, as described in section 17.

13.3 Tailings Disposal

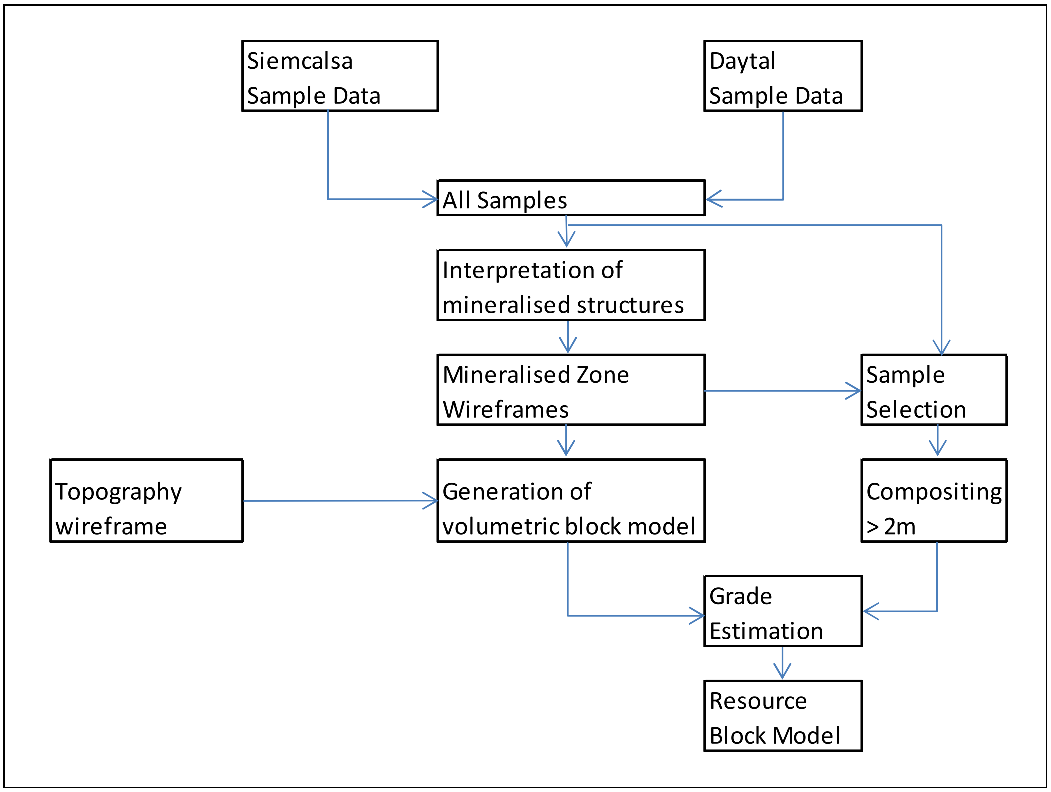
StB considered that the likely pit geometries, along with the natural topography, lend to an eventual dry disposal of tailings in initial mined out pits. Initial tailings disposal and waste rock dumps would take place, subject to negotiation, in the government owned forestry area immediately to the south of the open pits. Thereafter, StB propose the backfilling of worked out sections of the open pits. Future mining schedules will have to take this pit-backfilling requirement into account.

14 MINERAL RESOURCE ESTIMATES

14.1 General Methodology

An updated mineral resource estimation was completed, during September 2014, by the Qualified Person. This estimation employed a three-dimensional block modelling approach, using CAE Datamine software. The general methodology for the current update is described in the flowsheet in Figure 14-1.

Figure 14-1. Block Modelling Methodology



The interpretation was started by Daytal geologists, and this work was continued by the qualified person. This interpretation was broadly based on cut-offs of 0.07%WO_3 and 0.07% Sn. Within the overall limits of the mineralised zone structures, further internal waste zones were extrapolated semi-automatically, directly from composites and based on cut-offs of 0.07%WO_3 and 0.07% Sn. The orientation of this extrapolation was controlled by user-defined orientation control strings.

14.2 Sample Database

A summary of the available sample data is shown in Table 14 1. Plans of all these sample data, overlain on the topographic contours, are shown in Figure 14 2, along with a long section in Figure 14 3. As can be seen from these plots, the resultant spacing of samples with these different historical campaigns has ended up being fairly sporadic, with sections spaced at distances from 30m to 100m. Most of the trench and drillhole samples have been aligned at right angles to the overall deposit orientation of approximately 065°. The lithology codes assigned during logging are summarised in Table 14 2.

Table 14-1. Summary of Sample Database

COMPANY	SAMPLE TYPE	Holes / Lines	Length m	Average Hole Length m	Sn Samples	WO3 Samples
ADARO	TRENCH	10	850	85	165	158
SIEMCALSA	DRILLHOLE	6	1,227	204	235	234
SIEMCALSA	GALLERY	15	113	8	102	102
SIEMCALSA	TRENCH	26	2,836	109	925	929
DAYTAL	DRILLHOLE	59	10,716	182	3,478	4,496
DAYTAL	CHANNELS	5	268	98	119	121
DAYTAL	TRENCH	18	2,654	147	806	1,131

Notes

. Siemcalsa channels all from underground galleries

. The samples categorised as 'Siemcalsa' also include earlier ENADIMSA samples

. Daytal channels from a mixture of galleries and surface outcrops

. Samples counted above are those >0.001%Sn/WO_3

Table 14-2. Exploration Drillhole Principal Lithology Codes

	CODE	
Description	Without Faulting	With Faulting
Grey schists	EG	EGZF
Mauve schists	EM	EMZF
Grey-green schists	EG	EGZF
Green schists	EV	EVZF
Black schists	EN	ENZF
Quartz Vein	F	FZF

The principal assays made were for Sn and WO_3. Drillhole recoveries were not consistently good, with the Siemcalsa drilling have approximately 60% of samples achieving 90%+ recoveries and 80% of samples achieving 50%+ recoveries. The Daytal drilling results were better, with 76% of samples achieving 90%+ recoveries and 97% of samples achieving 50%+ recoveries.

Figure 14-2. Plan of Exploration Sample Data

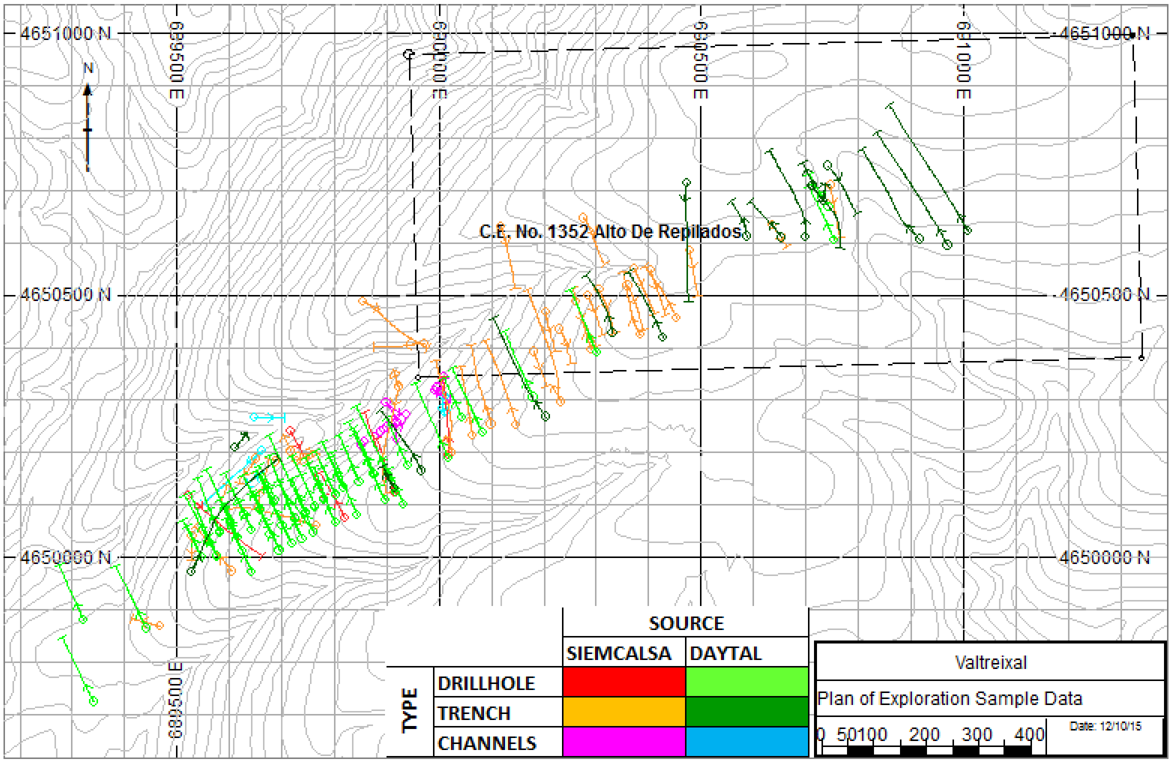
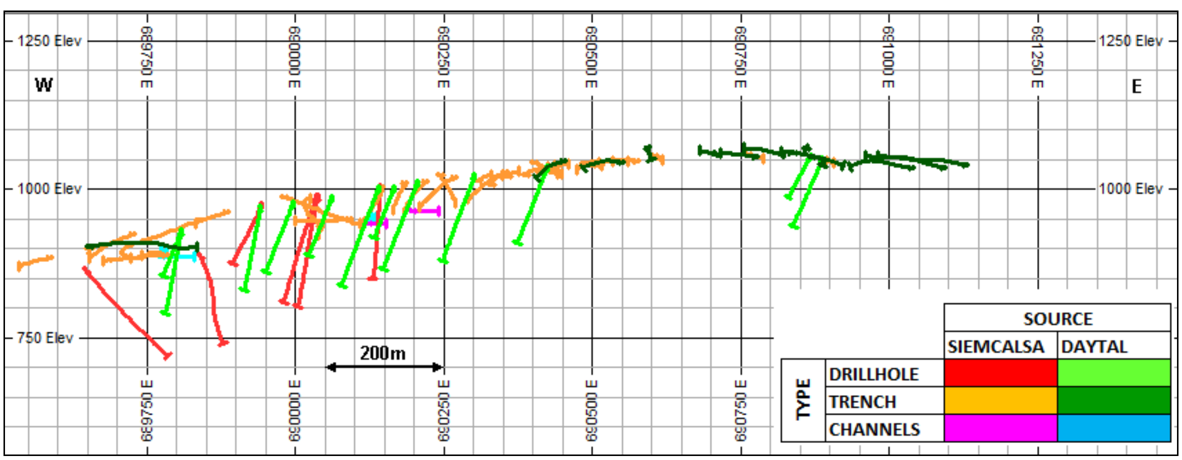


Figure 14-3. W-E Long Section of Exploration Sample Data



14.3 Interpretation

Previous interpretations had been made of up to 15 different individual vein structures (bandas) – predominantly from surface outcrops and trench intersections. With the new Daytal sampling and the combined sample set, a single set of mineralised zones for both Sn and WO3 were created. Regions of internal waste with respect to Sn or WO3, within these overall zone models, were subsequently modelled semi-automatically using dynamic anisotropy, to achieve assignment of mineralised zones for cut-offs of both 0.02% WO3 and 0.05% Sn. These cut-off levels were selected from examination of different populations from log-probability plots. In general, the interpreted vein structures have a consistent strike direction of around 65°, and dip at approximately 45-60° in the direction 155°. These structures are shown in plan and long section in Figure 14-6 and Figure 14-7. The overall extent of mineralisation is summarised in Table 14-3.

Table 14-3. Overall Mineralisation Dimensions

	Vertical Limits			Horizontal Width: Individual Structures		Horizontal Width: Overall Mineralisation		
Strike Length *m*	Minimum Base Elevation *m RL*	Maximum Outcrop Elevation *m RL*	Max. depth *m*	Max *m*	Average *m*	Max *m*	Average *m*	Dip Range *(°)*
1,800	750	1080	150	40	20	270	190	40 - 90

An example interpreted cross-section is shown in Figure 14-4. In this section a particular mineralised intersection has been highlighted in hole VS002. The data for this intersection are summarised in Table 14-4, and the same intersection is identified in core photographs in Figure 14-5.

Figure 14-4. Interpreted Section 18

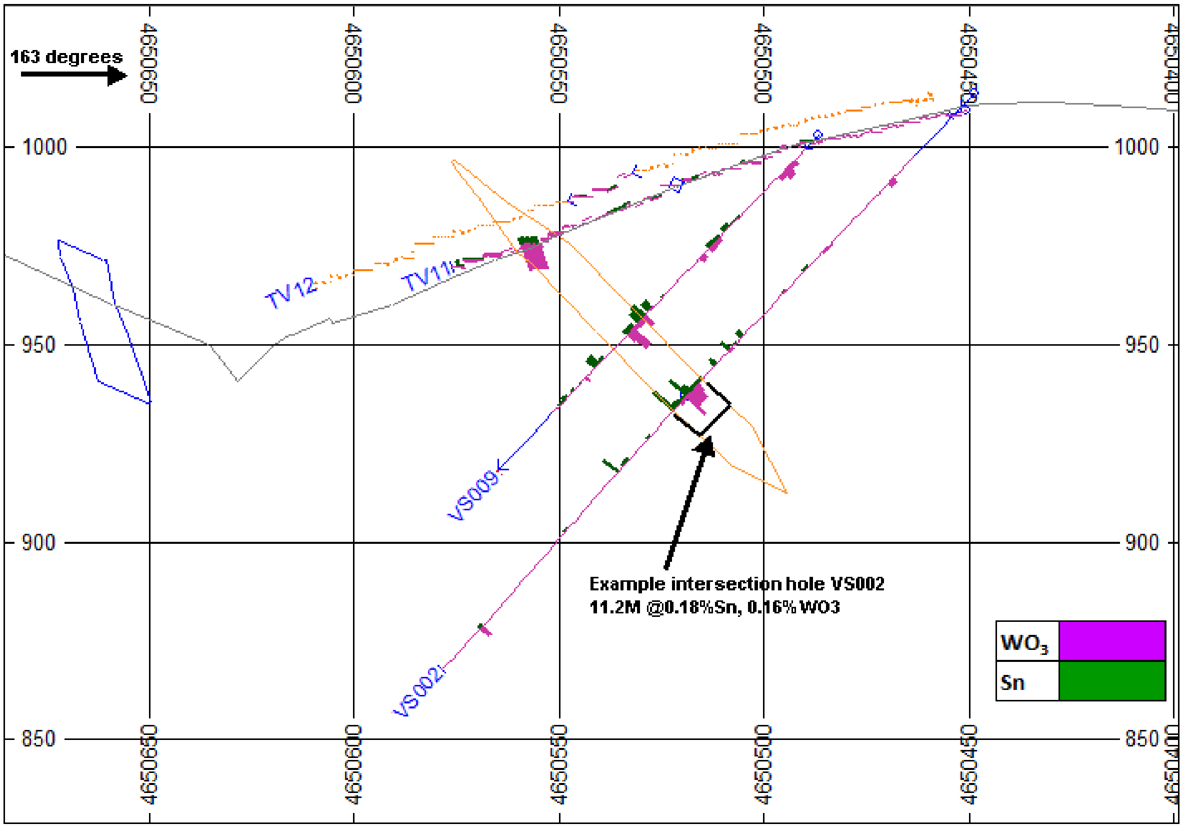


Table 14-4. Hole VS002 Data – 97 to 114m

BHID	FROM	TO	LENGTH	ROCKCODE	SAMPLE	SN	WO3
VS002	97.15	99.05	1.9	F	VS00209905	0.00	0.00
VS002	99.05	100.5	1.45	EGO	VS00210050	0.02	0.02
VS002	100.5	101.95	1.45	EGO	VS00210195	0.10	0.03
VS002	101.95	102.95	1	EGO	VS00210295	0.07	0.21
VS002	102.95	104.55	1.6	EGO	VS00210455	0.09	0.35
VS002	104.55	106.2	1.65	EGO	VS00210620	0.22	0.34
VS002	106.2	106.75	0.55	EG	VS00210675	0.52	0.55
VS002	106.75	108.75	2	EG	VS00210875	0.08	0.05
VS002	108.75	110.75	2	EG	VS00211075	0.12	0.02
VS002	110.75	111.7	0.95	F	VS00211170	0.59	0.01
VS002	111.7	112.9	1.2	EG	VS00211290	0.02	0.01
VS002	112.9	114.1	1.2	EG	VS00211410	0.02	0.02

= 11.2m
@0.18%Sn,
@0.16%WO_3

Notes:

- **Intersection also marked in Figure 14-5**

Figure 14-5. Core Photographs of Hole VS002 – 96 to 113m

Figure 14-6. Plan of Mineralised Zones

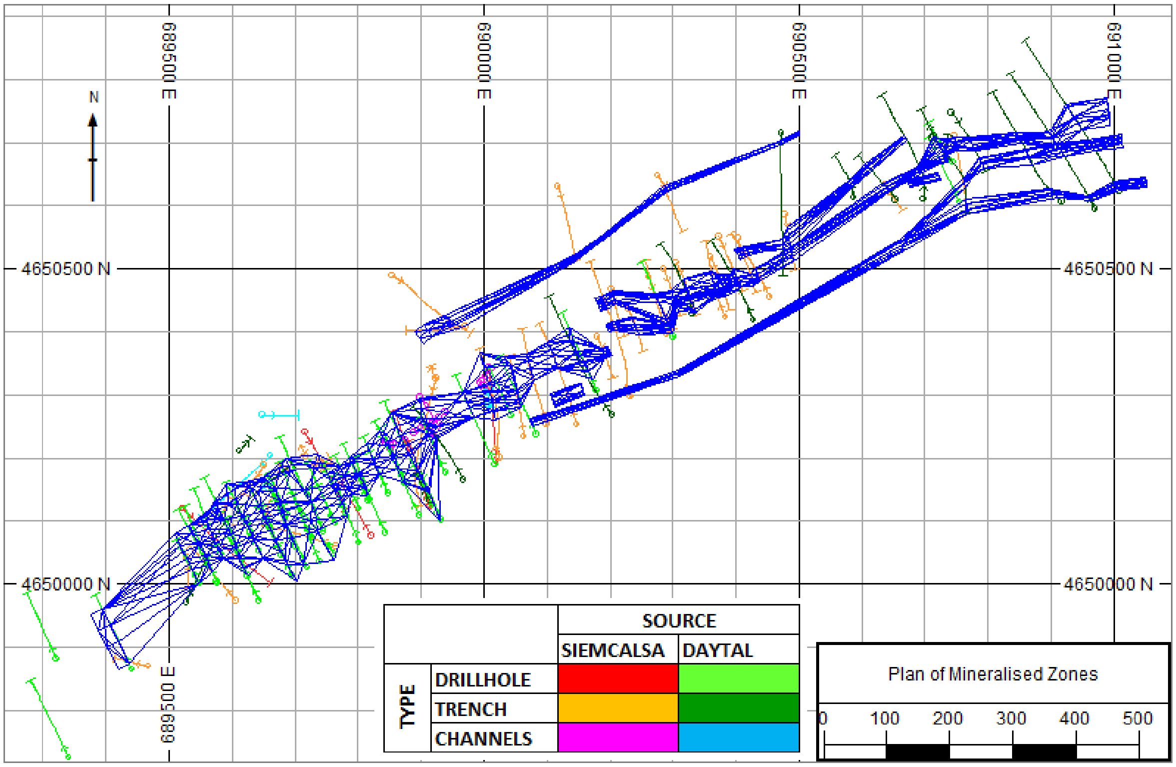
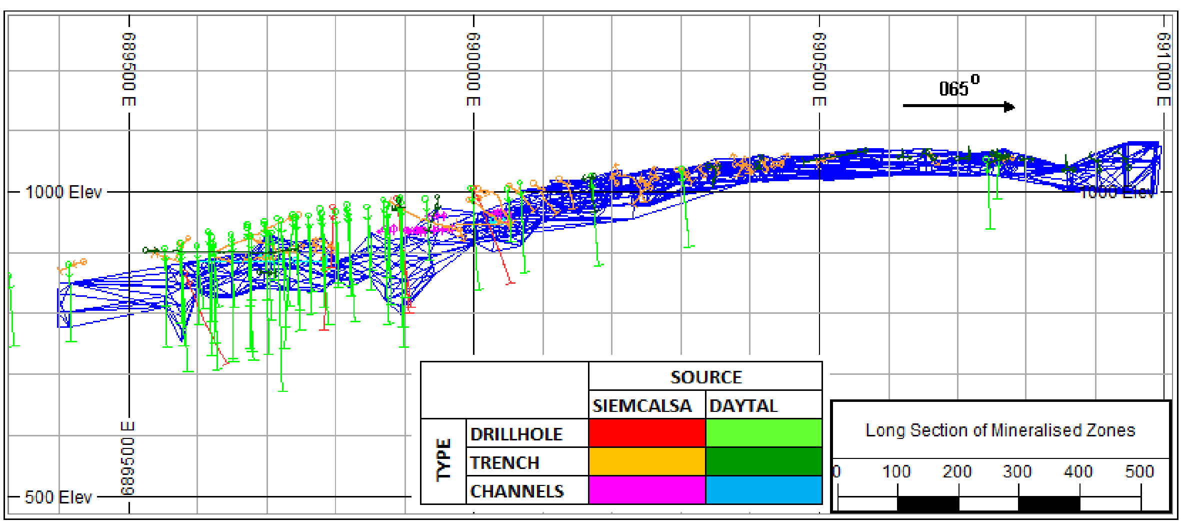


Figure 14-7. Long Section of Mineralised Zones



14.4 Sample Selection and Compositing

All available diamond drillhole, trench and underground channel samples were selected inside the supplied mineralised wireframe envelope. A summary of the selected data is shown in Table 14-5. The separate zones were assigned IDs 1-4, as shown in Figure 14-9. Sporadic samples with Sn or WO_3 grades greater than 0.07% are assigned with ZONE=5.

Table 14-5. Selected Sample Summary

Structure	Description	SOURCE	Holes/ Trenches/ Galleries	Length *m*	Length / Intersection *m*	Samples
Sn	Inside Wireframe	ADARO	11	186	16.9	39
		SIEMCALSA	31	532	17.2	277
		DAYTAL	80	1,928	24.1	1,278
	External	ADARO	1	0	0.4	1
		SIEMCALSA	15	56	3.7	25
		DAYTAL	81	242	3.0	182
WO_3	Inside Wireframe	ADARO	13	186	14.3	40
		SIEMCALSA	34	532	15.6	278
		DAYTAL	81	1,928	23.8	1,182
	External	ADARO	6	30	5.1	7
		SIEMCALSA	36	125	3.5	91
		DAYTAL	78	321	4.1	222

Note:

. External samples based on being +0.07% Sn/WO_3

The externally selected samples are those relatively erratic above-cut-off samples which are outside the principal wireframe zones (ZONE=5).

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

1. Composite length 2m. This compositing length was applied a slightly variable, such that an equal composite length of 2m was applied across each intersection. This length was chosen as it generally corresponds to the largest sample size and it reflects a possible selection unit size within potential open pit operations.

2. Minimum composite length = 0.5m.

3. Minimum/maximum gap length = 0.05m / 0m.

4. Retaining of ZONE identifiers on the composites, so five principal separate mineralised zones are preserved.

5. Top-cut values were assigned to Sn and WO_3 composited grades, as follows:
   Sn % - top-cut = 1.27%
   WO_3 % - top-cut = 1.1%

These top-cut values stem from:

- Decile analyses.
- Log-probability plots.
- Coefficient of variance (cv) analysis plots.

Figure 14-8. Coefficient of Variance Analysis Plots

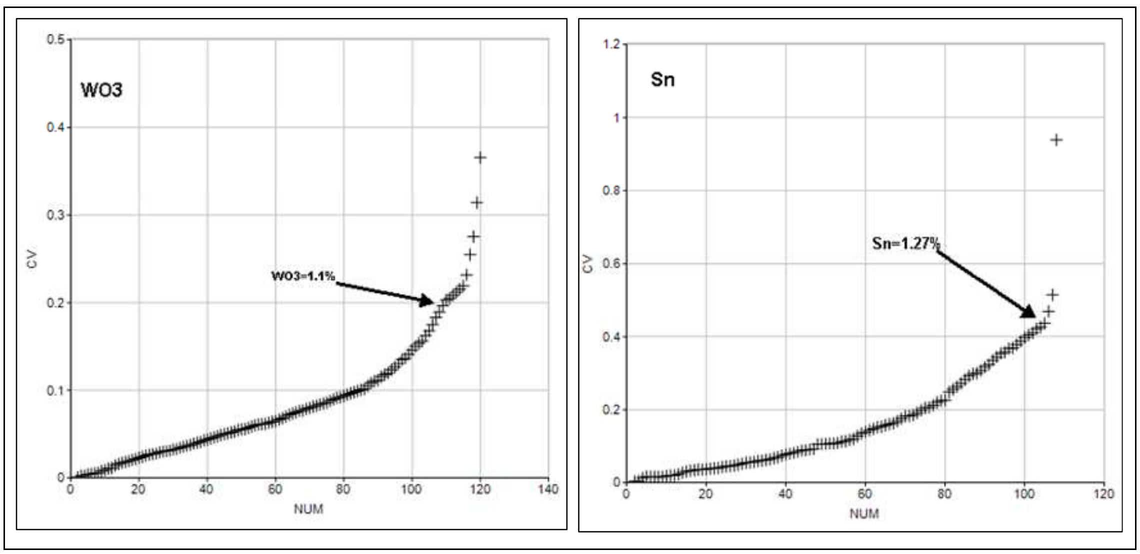


Figure 14-9. Plan of Mineralised Zone IDs

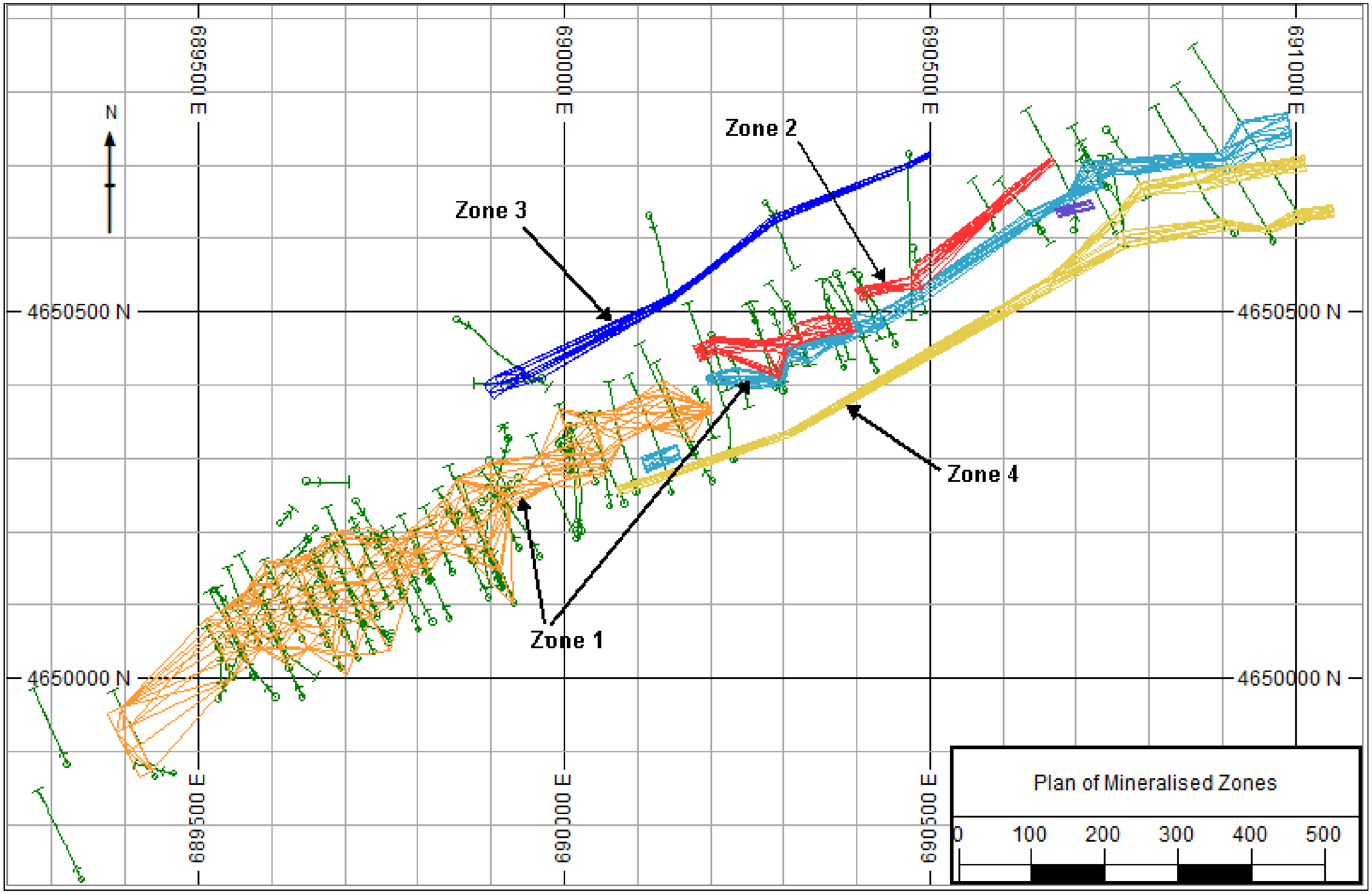


Table 14-7 depicts a summary of the decile analysis results for the selected sample grades. It can be seen from these results that the chosen top-cut levels broadly identify those top1-2% of samples, which contain more than approximately 10% of the total contained metal in the selected samples. Log histograms and log-probability plots of the selected samples are shown in

Figure 14-10 to Figure 14-13. A summary of the effect of the applied top-cut levels is shown in Table 14-6.

Table 14-6. Summary of Top-Cut Effect

						Mean Value	
FLDNAM	ZONE	Top-Cut Level %	No. of Composites	Number Cut	Proportion % of Composites with TC Applied	Un-Cut	With TC
SN	1	1.27	1,143	11	0.96	0.11	0.11
SN	2	1.27	78	0	-	0.12	0.12
SN	3	1.27	24	0	-	0.12	0.12
SN	4	1.27	28	0	-	0.02	0.02
SN	5	1.27	306	6	1.96	0.24	0.22
WO3	1	1.1	1,166	13	1.11	0.20	0.20
WO3	2	1.1	78	0	-	0.11	0.11
WO3	3	1.1	24	0	-	0.03	0.03
WO3	4	1.1	44	0	-	0.12	0.12
WO3	5	1.1	197	7	3.55	0.26	0.25

Table 14-7. Decile Analyses – Selected Samples

	Q%_FROM	Q%_TO	NUMBER	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
Sn	0	10	239	0.01	0.01	0.02	3.55	0.94
	10	20	240	0.03	0.02	0.03	6.32	1.68
	20	30	268	0.04	0.03	0.06	10.81	2.87
	30	40	231	0.07	0.06	0.08	17.78	4.72
	40	50	280	0.09	0.08	0.10	22.78	6.04
	50	60	277	0.11	0.10	0.13	28.14	7.46
	60	70	245	0.14	0.13	0.16	35.58	9.44
	70	80	246	0.18	0.16	0.21	45.01	11.94
	80	90	357	0.25	0.21	0.30	61.83	16.40
	90	100	270	0.58	0.31	7.35	145.25	38.52
	90	91	30	0.31	0.31	0.32	7.74	2.05
	91	92	27	0.33	0.32	0.34	8.11	2.15
	92	93	22	0.35	0.34	0.36	8.31	2.20
	93	94	24	0.38	0.36	0.40	10.08	2.67
	94	95	23	0.43	0.40	0.45	10.34	2.74
	95	96	18	0.46	0.45	0.49	11.53	3.06
	96	97	39	0.55	0.49	0.60	13.31	3.53
	97	98	18	0.65	0.60	0.72	17.69	4.69
	98	99	33	0.81	0.72	0.92	20.45	5.42
	99	100	36	1.49	0.92	7.35	37.70	10.00
	0	100	2653	0.15	0.01	7.35	377.06	100.00
WO_3	Q%_FROM	Q%_TO	NUMBER	MEAN	MINIMUM	MAXIMUM	METAL	METAL%
	0	10	274	0.01	0.01	0.02	3.15	0.56
	10	20	224	0.02	0.02	0.03	5.35	0.95
	20	30	250	0.04	0.03	0.05	9.62	1.70
	30	40	225	0.07	0.05	0.08	17.25	3.05
	40	50	194	0.10	0.08	0.12	24.10	4.26
	50	60	236	0.15	0.12	0.19	36.37	6.43
	60	70	216	0.25	0.19	0.31	62.11	10.98
	70	80	265	0.37	0.31	0.44	91.17	16.12
	80	90	268	0.50	0.44	0.58	125.47	22.18
	90	100	304	0.77	0.58	3.53	191.07	33.78
	90	91	33	0.59	0.58	0.61	14.11	2.49
	91	92	23	0.62	0.61	0.63	15.80	2.79
	92	93	24	0.64	0.63	0.65	15.59	2.76
	93	94	34	0.66	0.65	0.68	16.52	2.92
	94	95	42	0.69	0.68	0.70	16.47	2.91
	95	96	25	0.71	0.70	0.72	17.82	3.15
	96	97	29	0.74	0.73	0.76	17.97	3.18
	97	98	33	0.80	0.76	0.84	19.16	3.39
	98	99	32	0.91	0.85	1.02	23.33	4.12
	99	100	29	1.33	1.04	3.53	34.32	6.07
	0	100	2456	0.23	0.01	3.53	565.66	100.00

Figure 14-10. Log Histogram of Selected Sn Samples

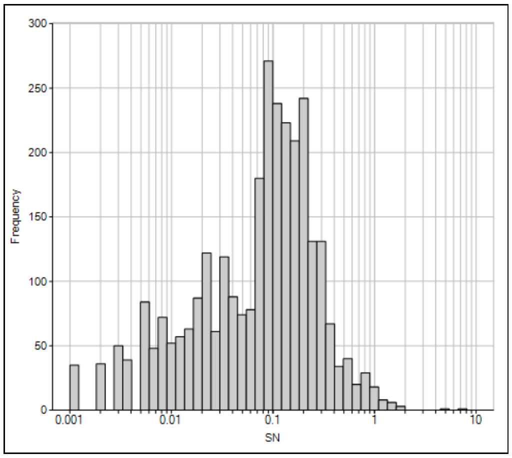


Figure 14-12. Log Histogram of Selected WO_3 Samples

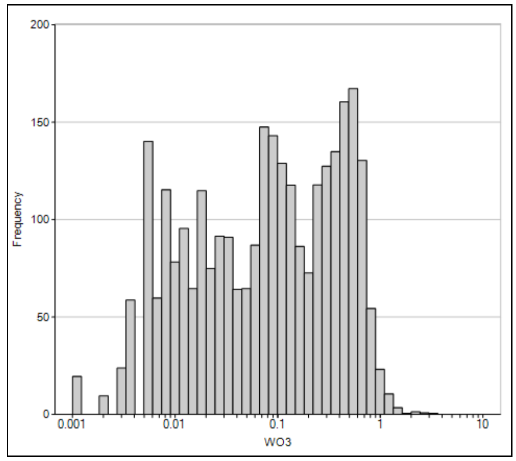


Figure 14-11. Log-Probability Plot of Selected Sn Samples

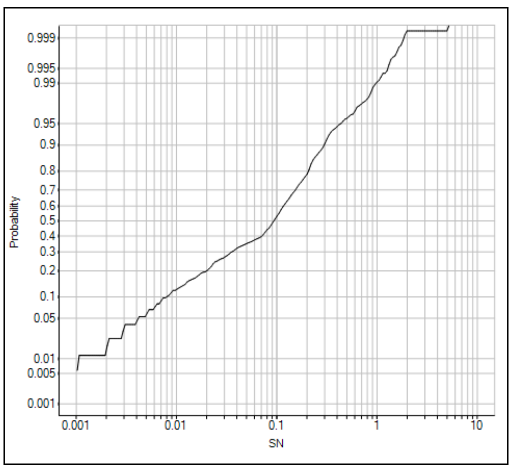
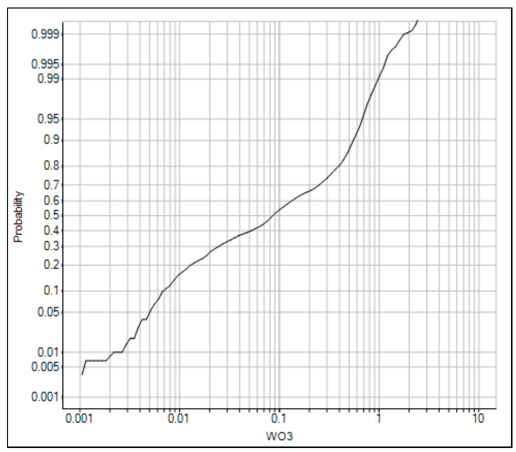


Figure 14-13. Log-Probability Plot of Selected WO_3 Samples



14.5 Geostatistics

A statistical summary of the selected samples is shown in Table 14-8. These statistics are divided by zone assignment, as well as shown overall. It can be seen that all of the coefficient of variation (CV) values are generally just over 1.

Table 14-8. Summary Statistics of Selected Samples

FIELD	ZONE	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	COEFFVAR
SN	1	2172	0	7.35	0.11	0.04	0.19	0.15	1.8
SN	2	195	0.001	1.27	0.12	0.03	0.19	0.13	1.6
SN	3	86	0.001	0.51	0.12	0.02	0.13	0.19	1.1
SN	4	29	0	0.12	0.02	0.00	0.03	0.05	1.5
SN	5	535	0.07	5.14	0.21	0.07	0.27	0.19	1.3
SN	All	3017	0	7.35	0.12	0.04	0.21	0.17	1.7
	1	2271	0	3.53	0.21	0.07	0.26	0.29	1.2
WO3	2	195	0.004	0.44	0.11	0.02	0.13	0.12	1.2
WO3	3	86	0.001	0.18	0.03	0.00	0.04	0.03	1.6
WO3	4	48	0	0.62	0.11	0.02	0.14	0.22	1.2
WO3	5	278	0.07	2.28	0.22	0.06	0.24	0.21	1.1
WO3	All	2878	0	3.53	0.20	0.06	0.25	0.28	1.3

A statistical summary of the generated composites is shown in Table 14-9, with corresponding log-probability plots from Figure 14-14 to Figure 14-19.

Table 14-9. Summary Statistics of 2m Composites

FIELD	ZONE	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV	LOGESTMN	COEFFVAR
SN	1	1,143	0.000	0.99	0.11	0.02	0.13	0.17	1.2
SN	2	78	0.006	0.88	0.12	0.03	0.16	0.12	1.3
SN	3	24	0.001	0.46	0.12	0.01	0.11	0.19	1.0
SN	4	28	0.000	0.12	0.02	0.00	0.03	0.05	1.5
SN	5	306	0.070	1.27	0.20	0.04	0.19	0.19	0.9
SN	All	1,579	0.000	1.27	0.12	0.02	0.14	0.18	1.2
WO3	1	1,166	0.002	1.10	0.20	0.04	0.21	0.29	1.0
WO3	2	78	0.004	0.44	0.11	0.02	0.13	0.12	1.2
WO3	3	24	0.001	0.18	0.03	0.00	0.04	0.03	1.6
WO3	4	44	0.000	0.62	0.11	0.02	0.13	0.25	1.1
WO3	5	197	0.070	1.10	0.22	0.04	0.20	0.21	0.9
WO3	All	1,509	0.000	1.10	0.19	0.04	0.20	0.28	1.0

All plots for 2m Composites

Figure 14-14. Log Histogram - Sn

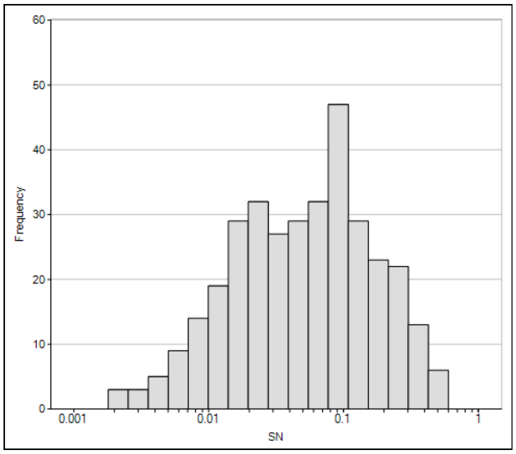


Figure 14-15. Log-Probability Plot – Sn

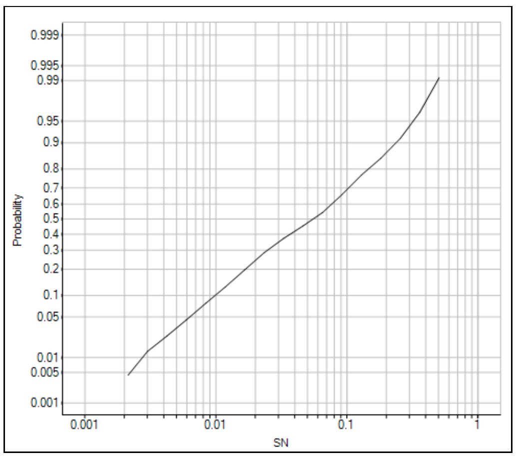


Figure 14-16. LP Plot Sn by Zone

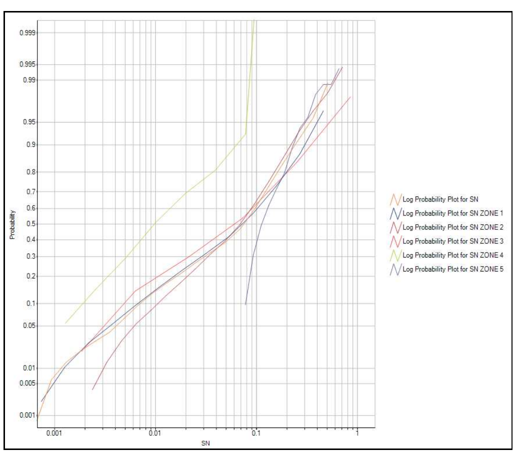


Figure 14-17. Log Histogram – WO_3

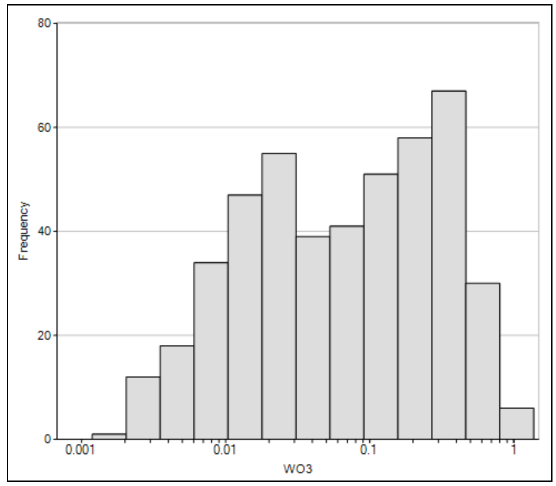


Figure 14-18. Log-Probability Plot - WO_3

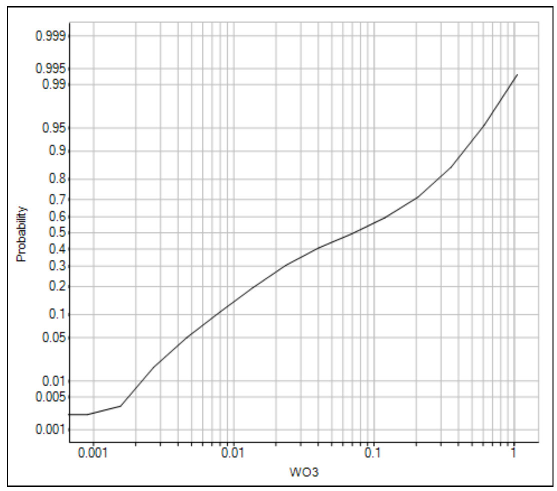
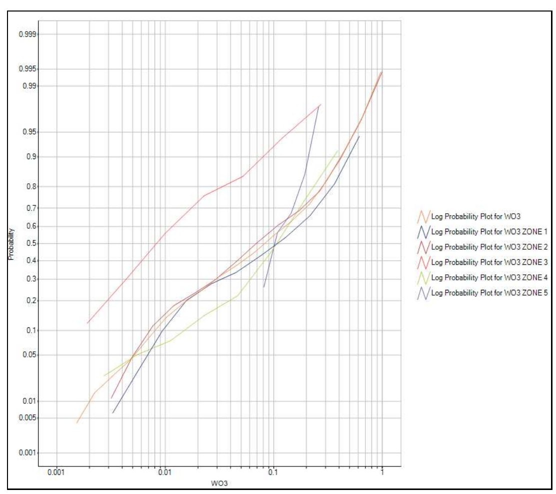


Figure 14-19. LP Plot $WO_3$by Zone



It can be seen from Table 14-9 that the coefficient of variation values have been reduced to near 1.0, by the effect of compositing and top-cut application. Individually the zones' grade population tend to shown long-normal populations.

Experimental variograms were generated for the generated composite data sets. Model variograms were fitted in each case, as depicted in Figure 14-20 and Figure 14-21. These variograms are poor in quality, which was not surprising given the relatively large area being covered by relatively very low number of samples. However, indicative ranges of influence of approximately 40-50m are suggested for both Sn and WO_3.

Figure 14-20. Experimental and Model Variograms – Sn

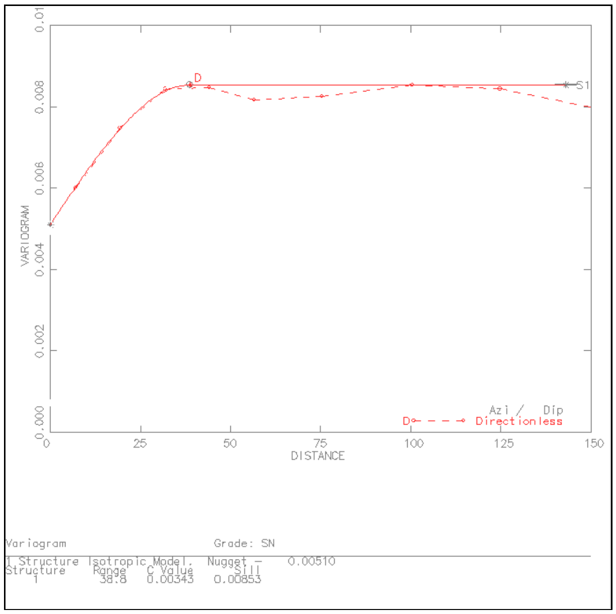
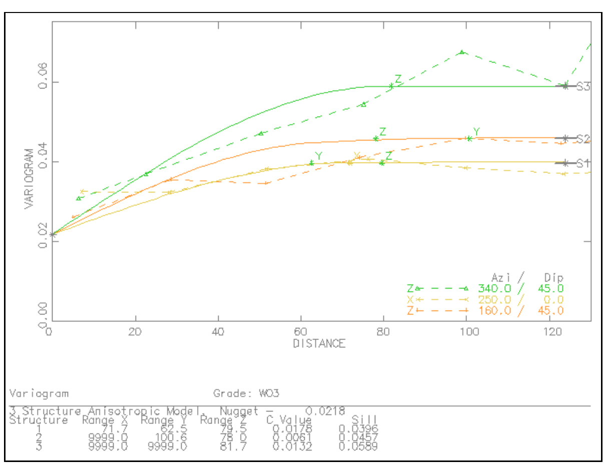


Figure 14-21. Experimental and Model Variograms – WO_3



14.6 Volumetric Modelling

An overall block model prototype was set up using the parameters summarised in Table 14-10. A parent block size of 10m x 10m x 10m was selected. Laterally, 10m is approximately half to one third of the general trench spacing. It is also considered that 10m vertically is a potential mining bench height.

Table 14-10. Resource Model Prototype

	Min *m*	Max *m*	Range *m*	Size *m*	Number
X	689,450	691,300	1,850	10	185
Y	4,650,050	4,651,110	1,060	10	106
Z	750	1,130	380	10	38

Physical controls used, during the generation of the volumetric block model, include:

- Natural topography wireframe model.
- Mineralised zone envelope wireframe model.
- Dip and strike orientation control strings.

Sub-cells were created at the edge of these structures, down to a resolution of 0.5m.

A ZONE identifier field was also set into the model, so as to be able to demarcate individual mineralised zones.

Field set into the volumetric block model include:

ZONE	Mineralised zone identifier
FLAG_SN	Mineralised with respect to Sn grade > 0.07%
FLAG_WO3	Mineralised with respect to WO3 grade > 0.07%
TRDIPDIR	Dip direction of mineralisation
TRDIP	Dip of mineralisation
DENSITY	In-situ density value.

14.7 Densities

Density measurements have been made from core samples since October 2104, using water immersion. Up until June 2015, the measurements were based on uncovered core. Subsequent measurements were made using plastic film. These results with plastic film show that uncovered density measurements were overstating density values by approximately 7%. These former density values were then adjusted by a factor of 93%. By the end of September 2015, 1280 density measurements were available. A summary of these density values by rocktypes are summarised in Table 14-11. These measurements were subsequently estimated into the resource block model.

Table 14-11. Summary of Density Measurements

ROCKCODE	Description	NUMBER	MINIMUM	MAXIMUM	MEAN	VARIANCE	STANDDEV
EG	Grey Schist	232	2.15	2.91	2.55	0.020	0.14
EGZF	Grey Schist+Faulting	248	1.54	2.77	2.35	0.027	0.16
EM	Mauve Schist	12	2.19	2.40	2.32	0.005	0.07
EMZF	Mauve Schist+Faulting	2	2.23	2.29	2.26	0.001	0.03
EN	Black Schists	318	2.07	3.61	2.53	0.015	0.12
ENZF	Black Schists+Faulting	188	1.96	2.66	2.32	0.015	0.12
EV	Green Schists	100	2.29	3.09	2.81	0.024	0.16
EVZF	Green Schists+Faulting	100	2.02	2.91	2.45	0.031	0.18
F	Quart vein	75	2.08	2.82	2.43	0.010	0.10
FZF	Quart vein+Faulting	4	2.40	2.48	2.44	0.001	0.03

14.8 Grade Estimation

Sn and WO_3 grades were estimated into the volumetric block model, primarily using an ordinary kriging method of estimation. For validation purposes, alternative grades were also estimated using an inverse-distance and a nearest neighbour method. The parameters used for grade estimation are summarised in Table 14-12.

Table 14-12. Grade Estimation Parameters

Fields	Search No.	Distance XYZ m	Min. No. of Composites	Min. No. of Drillholes
Sn and	1	15 x 15 x 5	6	2
WO_3	2	30 x 30 x 10	6	2
	3	50 x 50 x 20	1	1

Notes

. Sn and WO3 interpolated with ordinary kriging (OK)
. Alternative grades also estimated using nearest neighbour (NN) and inverse-distance (ID^2) for validation purposes
. Density estimated using inverse-distance weighting (^2)
. Dynamic anisotropy applied, such that:

x	**Along-strike**
y	**Down-dip**
z	**Cross-strike**

14.9 Mineral Resource Classification

In order to test resource classification criteria, a conditional simulation exercise was completed, which focussed on the precision of evaluation that may be obtained with different drillhole spacings, related to mining blocks containing ore broadly equivalent to 3 months of production and 1 year of production. This analysis was completed with the following stages:

1. A panel was defined, in the central part of the Valtreixal deposit, with an assumed average thickness of 20m, and a length and width of 52.5. These dimensions were selected, so that this block contains approximately 130 Kt of material, which is roughly equivalent to 3 months of production (at a assumed production rate of 500 Ktpa). This was used to create a volumetric test model.

2. Based on all available drillholes in the same area, a grid of densely spaced pseudo-samples were generated, based on the same statistical parameters as the original distribution of actual samples. Using this data set, samples corresponding to any different theoretical drilling grids could be selected. In this way, different composite groups were created for drilling grids spaced at 10m, 15m, 20m, etc up to 50m.

3. The mineralised composite set for this central part of the Valtreixal deposit was converted into normal score form, and used to provide experimental variograms, from which model variograms were determined, for WO_3.

4. A conditional simulation was then run using each of the different pseudo-drilling grid sets. The parameters used for these simulation runs included:

   a) Sequential gaussian simulation.
   b) An internal point density of 2m x 2m was used inside the test area for simulated points.
   c) 50 simulation runs were completed for each test.
   d) Normal transformed model variograms used.
   e) Horizontal search distances of 80m were used.
   f) Minimum/Maximum no. of composites = 2 / 20

5. For each conditional simulation run, the distribution of overall average values were approximately normally distributed, as shown in the example in Figure 14-23. The standard deviation of these results was then used to calculate the relative error of the overall average grade, at the 90% probability level.

6. From these results, the relative errors at the 90% probability level were also determined for a block corresponding to approximately one year's production.

An overall summary of these results is shown in Table 14-13.

Figure 14-22. Normal Score Variogram for WO3

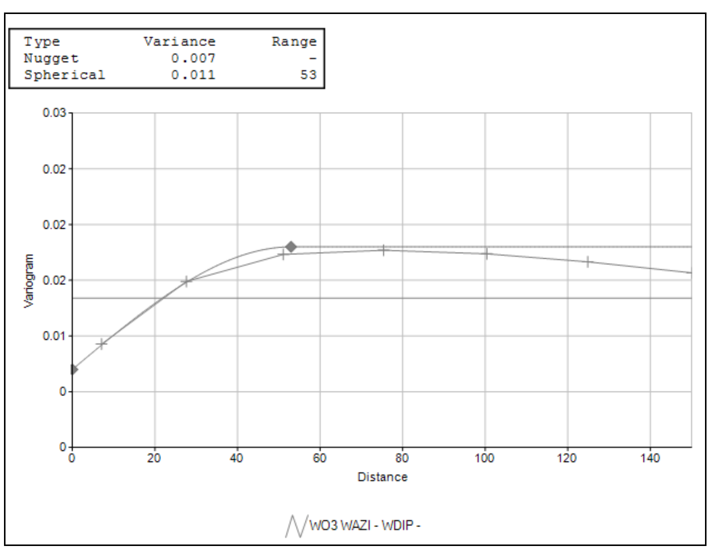


Table 14-13. Resource Classification Testwork

		Results for Quarterly Mining Block (138Kt)						
FIELD	Drilling Grid Spacing m	MEAN	STANDDEV	MINIMUM	MAXIMUM	+/- Tolerance at 90% Probability Level	Relative Error %	Relative Error for Annual Mining Block (550Kt) %
WO3	10	0.275	0.016	0.241	0.320	2.6%	9.6	4.8
WO3	15	0.250	0.022	0.202	0.312	3.6%	14.5	7.2
WO3	20	0.304	0.033	0.239	0.392	5.4%	17.7	8.9
WO3	25	0.153	0.030	0.102	0.235	4.9%	32.2	16.1
WO3	30	0.257	0.047	0.175	0.382	7.7%	29.8	14.9
WO3	35	0.198	0.049	0.121	0.337	8.1%	40.8	20.4
WO3	40	0.129	0.049	0.058	0.262	8.0%	61.6	30.8
WO3	45	0.080	0.037	0.031	0.195	6.1%	76.4	38.2
WO3	50	0.084	0.047	0.028	0.225	7.7%	91.6	45.8

Notes

. 50 simulation runs completed for each test drilling grid

. Conclusions:

Measured Resources

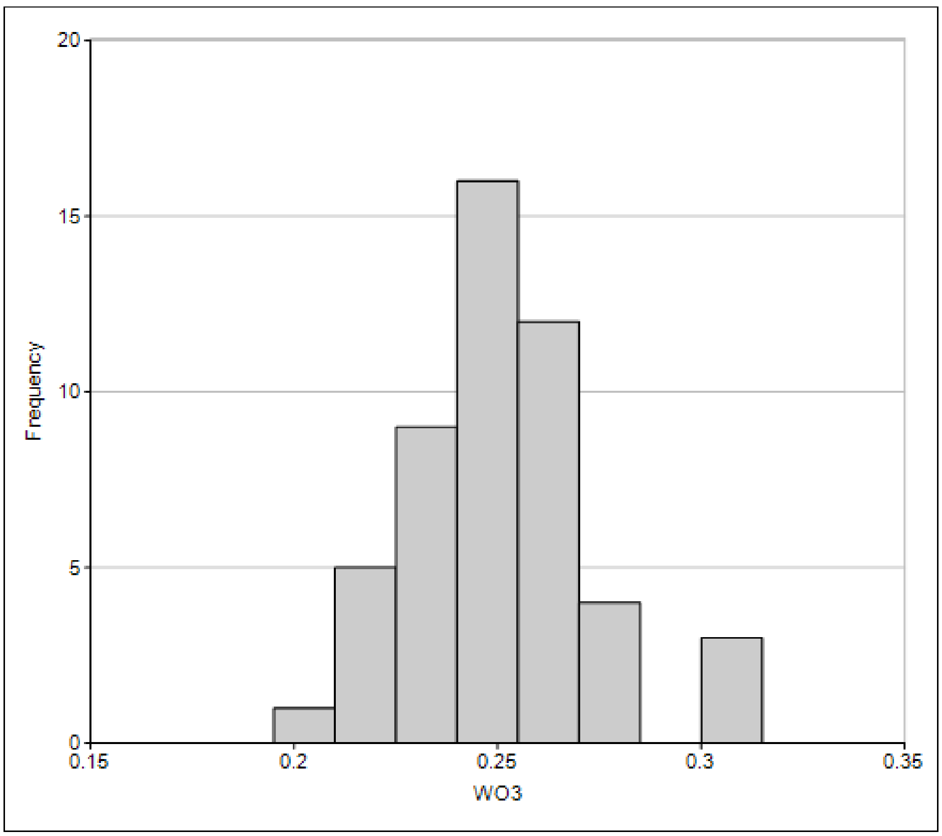
A drill grid spacing of 15m gives quarterly 90% confidence levels of +/-14.5% for WO3 grade

Indicated Resources

A drill grid spacing of 30m gives annual 90% confidence levels of +/-14.9% for WO3 grade

Figure 14-23. Example Histogram of Simulated Average WO_3 Grades

(For 130Kt Block, 15m Grid)



For the assessment of resource classification, it has been assumed that Measured Resources should be known within ±15%, with 90% confidence for a production quarter (3 months). Similarly, it has been assumed that Indicated Resources should be known within ±15%, with 90% confidence on an annual basis. This method of resource classification has wide acceptance.

From the results produced, the following criteria were used for resource classification.

Table 14-14. Resource Classification Criteria

Category	Criteria
Measured	Drill grid spacing of at least 15m (None currently present at Valtreixal)
Indicated	Drill grid spacing of at least 30m Within main interpreted zone
Inferred	Drill grid spacing of +30m Maximum extrapolation of 50m from drillholes or trenchs

These criteria were applied at Valtreixal. There are currently no measured resources at Valtreixal, as no areas have yet been systematically been drilled off with 15m x 15m grid. The areas classified as indicated resources, as covered by at least a drilling grid of 30m, are shown in plan in Figure 14-25m and in an example section in Figure 14-24.

Figure 14-24. Example Section – Resource Classes

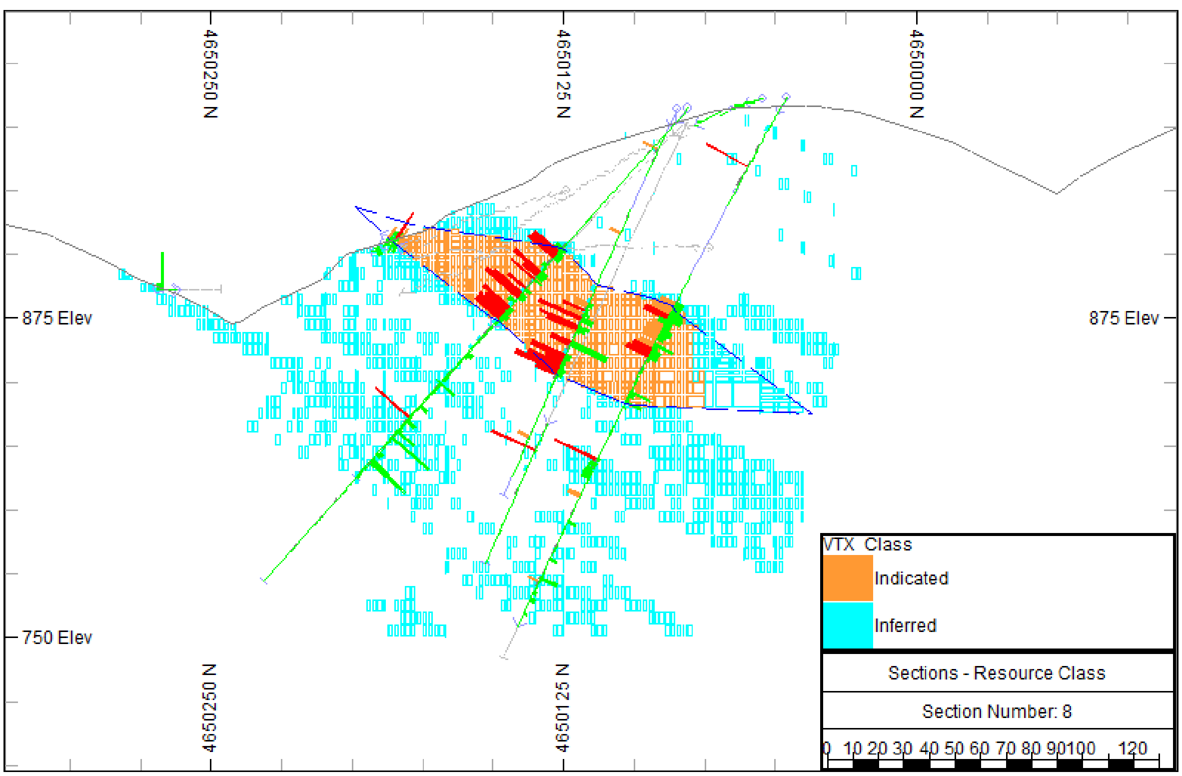
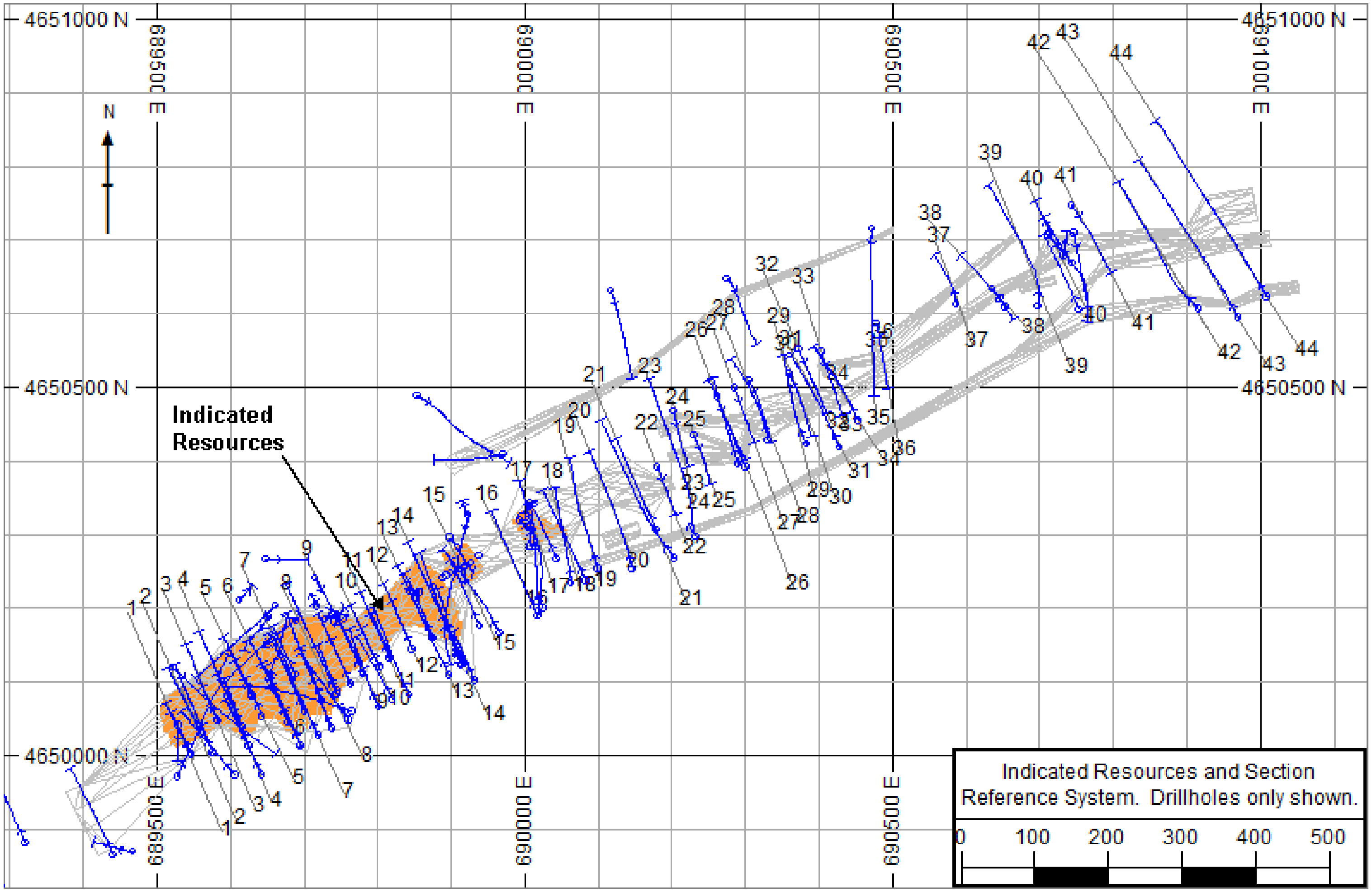


Figure 14-25. Plan View of Indicated Resources



14.10 Model Validation

14.10.1 Visual Comparisons

Sections were created through the resource block model, and compared with the sample composites used in for the grade estimation. A reference plan for these sections is shown in Figure 14-6, and example sections at the west end of the deposit (section 2) are shown in Figure 14-26 and Figure 14-27, for Sn and WO_3 respectively. All of the sections are shown in Appendix B. In general, these sections show a fairly good correspondence between estimated and composited sample grades.

14.10.2 Comparison of Global Average Grades

A comparison was made of the average Sn and WO_3 model grades, with the corresponding average sample and composite grades for the different modelled zones. These results are summarised in Table 14-15.

Table 14-15. Comparison of Global Average Grades

				Block Model		
ZONE	**FIELD**	**Samples**	**Composites**	**OK**	**ID**	**NN**
1	**Sn**	0.11	0.11	0.10	0.10	0.10
2	**Sn**	0.12	0.12	0.10	0.10	0.12
3	**Sn**	0.12	0.12	0.11	0.12	0.12
4	**Sn**	0.02	0.02	0.04	0.04	0.03
5	**Sn**	0.21	0.20	0.11	0.11	0.11
1	**WO_3**	0.21	0.20	0.19	0.19	0.19
2	**WO_3**	0.11	0.11	0.10	0.10	0.10
3	**WO_3**	0.03	0.03	0.04	0.04	0.04
4	**WO_3**	0.11	0.11	0.10	0.10	0.10
5	**WO_3**	0.22	0.22	0.22	0.23	0.22

Notes

. ID - grades from inverse distance weighting
. NN - grades from nearest neighbour estimation
. No cut-offs applied
. All grades shown are global average values
. All resource classes shown

These results compare fairly well, particularly given the uneven spread of sample data over the deposit.

Figure 14-26. Section 2 – Sn

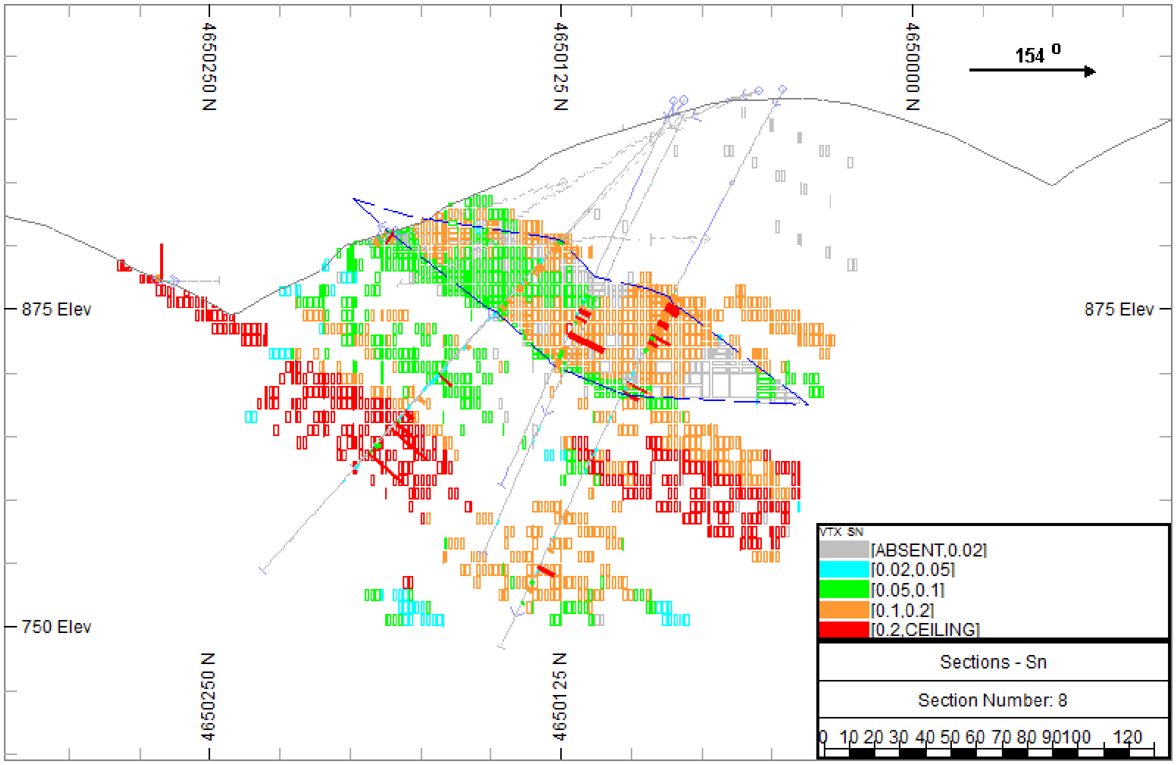
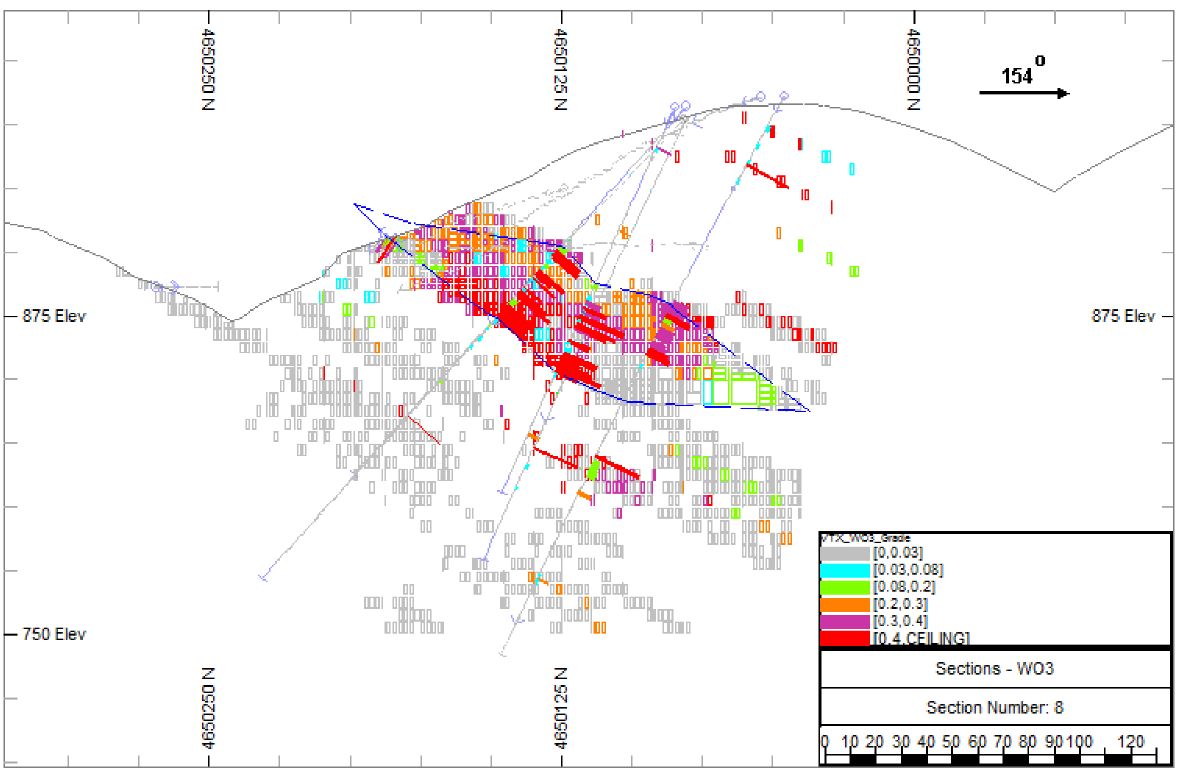


Figure 14-27. Section 2 – WO_3



14.10.3 Comparison of Local Average Grades

As part of the model validation process, grade profiles (swath plots) were also produced on 50m north-south slices, and the average grades (derived from different estimation methods) per slice, compared with the composites on the same slices. Examples of a grade profile plot are shown in Figure 14-28 and Figure 14-29 for Sn and WO_3, respectively, within ZONE=1 material. These shows a favourable comparison between composite grades and model grades, derived from inverse distance weighting and nearest neighbour estimation. All of the grade profiles produced are shown in Appendix A.

Figure 14-28. Example Grade Profile Plot – Sn– ZONE=1

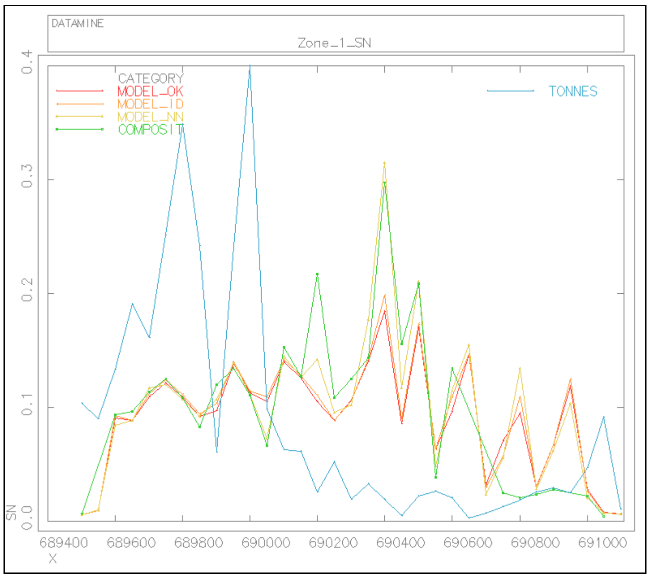
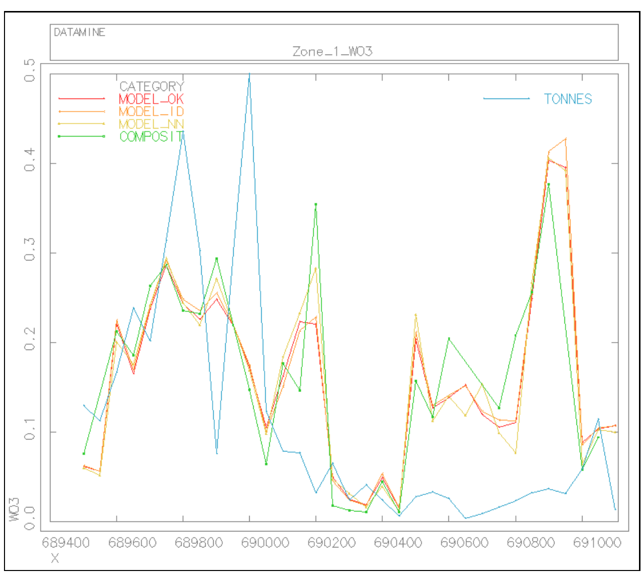


Figure 14-29. Example Grade Profile Plot – WO_3 – ZONE=1



14.10.4 Historical Comparison

A historical comparison of the current and previous resource estimates is summarised in Table 14-16. This reflects the influence of the extensive exploration drilling completed over the past two years.

Table 14-16. Historical Comparison

		Tonnes	Sn	WO_3
		Mt	*%*	*%*
2012 Siemcalsa		10.34	0.10	0.11
Mar-14	**AW Resource**	8.88	0.11	0.17
Oct-15	**AW Resource**	17.65	0.12	0.11

Notes

. AW = Adam Wheeler

. Cut-off applied 0.07% ($Sn+WO_3$)

. The Siemcalsa had no resource classification system

. The current evaluation has all inferred resources

The Siemcalsa evaluation was based on a polygonal methodology with much more detailed interpretation of individual mineralised bands. It also includes 20 intersections derived from underground gallery samples, whereas the current evaluation has only used 4 intersections derived from underground gallery samples.

No resource classification has been applied in this comparison – and the cut-off of 0.07 $Sn+WO_3$ has been applied purely for comparative purposes.

14.11 Mineral Resource Reporting

A grade-tonnage table of the in situ contents is shown in Table 14-18. This is based on the selectivity in the inherent parent block size of the resource block model: 10m x 10m x 10m, although also influenced by the zonal extrapolation applied, which broke blocks down to 5m x 2m x 5m sub-blocks.

The overall in-situ resource evaluation results' breakdown is shown in Table 14-17 at cut-off grades of 0.05% and 0.10% WO_3_Eq. The WO_3-equivalent grade has been calculated based on the assumed Sn and WO_3 metal prices and recoveries. These cut-off grades have been applied as they are close to the breakeven cut-off grade derived from currently assumed base case parameters.

Table 14-17. Resource Evaluation – In-Situ
As of End-October, 2015

	CLASS	Tonnes Kt	Sn %	WO_3 %	WO_3_Eq %
WO_3_Eq >= 0.05%	*Indicated*	2,828	0.13	0.25	0.34
	Inferred	15,419	0.12	0.08	0.17
	TOTAL	18,247	0.12	0.11	0.20

	CLASS	Tonnes Kt	Sn %	WO_3 %	WO_3_Eq %
WO_3_Eq >= 0.10%	*Indicated*	2,572	0.13	0.27	0.37
	Inferred	9,213	0.14	0.13	0.24
	TOTAL	11,785	0.14	0.16	0.27

Notes

. WO_3_Eq = WO_3 + (Sn x 0.74), based on:

	Price	Recovery
WO_3	$37,000/t	55%
Sn	$23,150/t	65%

. Maximum extrapolation = 50m

. Density values estimated from measurements

. Resources shown are inclusive of reserves

Table 14-18. Grade-Tonnage Table – Indicated Resources

WO_3_Eq Cut-Off	Tonnes	WO_3_Eq	Sn	WO3
%	Kt	%	%	%
0.05	2,828	0.34	0.13	0.25
0.06	2,804	0.34	0.13	0.25
0.07	2,757	0.35	0.13	0.25
0.08	2,710	0.35	0.13	0.26
0.09	2,637	0.36	0.13	0.26
0.10	2,572	0.37	0.13	0.27
0.11	2,513	0.37	0.13	0.28
0.12	2,451	0.38	0.13	0.28
0.13	2,405	0.38	0.13	0.29
0.14	2,365	0.39	0.13	0.29
0.15	2,330	0.39	0.13	0.29
0.16	2,298	0.39	0.13	0.30
0.17	2,263	0.40	0.13	0.30
0.18	2,241	0.40	0.13	0.30
0.19	2,218	0.40	0.13	0.31
0.20	2,184	0.41	0.13	0.31
0.21	2,153	0.41	0.13	0.31
0.22	2,128	0.41	0.13	0.32
0.23	2,106	0.41	0.13	0.32
0.24	2,084	0.41	0.13	0.32
0.25	2,058	0.42	0.13	0.32
0.26	2,032	0.42	0.13	0.32
0.27	2,002	0.42	0.13	0.32
0.28	1,967	0.42	0.13	0.33
0.29	1,931	0.43	0.13	0.33
0.30	1,891	0.43	0.13	0.33

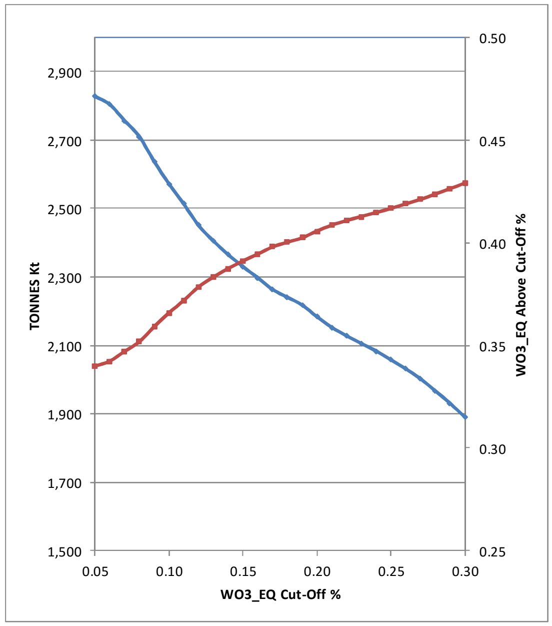
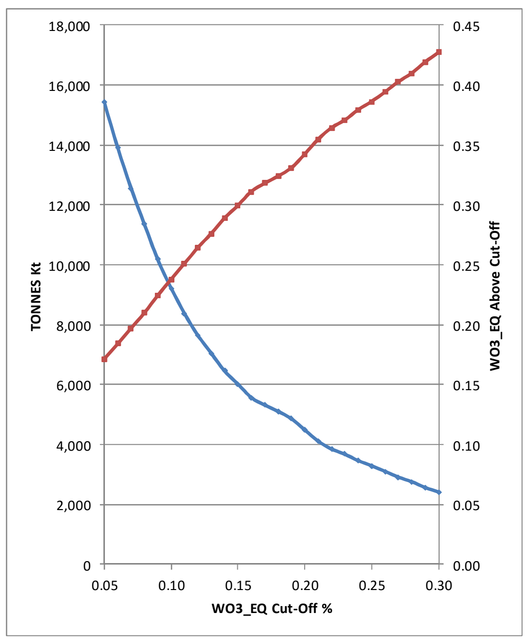


Table 14-19. Grade-Tonnage Table – Inferred Resources

WO_3_Eq Cut-Off	Tonnes	WO_3_Eq	Sn	WO3
%	Kt	%	%	%
0.05	15,419	0.17	0.12	0.08
0.06	13,908	0.18	0.12	0.09
0.07	12,555	0.20	0.13	0.10
0.08	11,352	0.21	0.13	0.11
0.09	10,173	0.22	0.14	0.12
0.10	9,213	0.24	0.14	0.13
0.11	8,373	0.25	0.14	0.15
0.12	7,640	0.26	0.15	0.16
0.13	7,046	0.28	0.15	0.17
0.14	6,453	0.29	0.15	0.18
0.15	6,017	0.30	0.15	0.19
0.16	5,566	0.31	0.14	0.20
0.17	5,319	0.32	0.14	0.21
0.18	5,105	0.32	0.14	0.22
0.19	4,862	0.33	0.14	0.22
0.20	4,484	0.34	0.15	0.23
0.21	4,115	0.35	0.15	0.24
0.22	3,848	0.36	0.15	0.25
0.23	3,678	0.37	0.15	0.26
0.24	3,459	0.38	0.15	0.27
0.25	3,292	0.39	0.15	0.28
0.26	3,104	0.39	0.15	0.28
0.27	2,912	0.40	0.15	0.29
0.28	2,759	0.41	0.15	0.30
0.29	2,568	0.42	0.15	0.31
0.30	2,413	0.43	0.15	0.32



15 MINERAL RESERVE ESTIMATES

15.1 Open Pit Optimisation

The parameters used for pit optimisation are summarised in Table 15-1, along with the parameters varied for sensitivity analyses. These base case set of optimisation parameters were developed with reference to current operating cost levels and parameters at Los Santos, as well as reflecting processing recoveries estimated by Santa Barbara. It should also be noted that these optimisation runs had no physical constraints. All of these parameters are summarised in Table 15-1. Additional mining factors were also applied of 5% dilution and 5% losses (95% mining recovery).

In the absence of geotechnical testwork, actual topographical toes and crest positions have been measured in areas where there are road cuttings or surface excavations for drilling platforms or access purposes, as shown in Figure 15-1. From these measurements, an average face slope angle of 56° was determined. Based on an assumed bench configuration of 10m benches, 4m berms every 2 benches, and 10m wide haul roads (or safety berms), this has given an estimated overall slope angle of approximately 42°, as shown in Figure 15-2. This overall slope angle has therefore been applied in optimisation work.

Figure 15-1. Plan of Face Slope Measurements

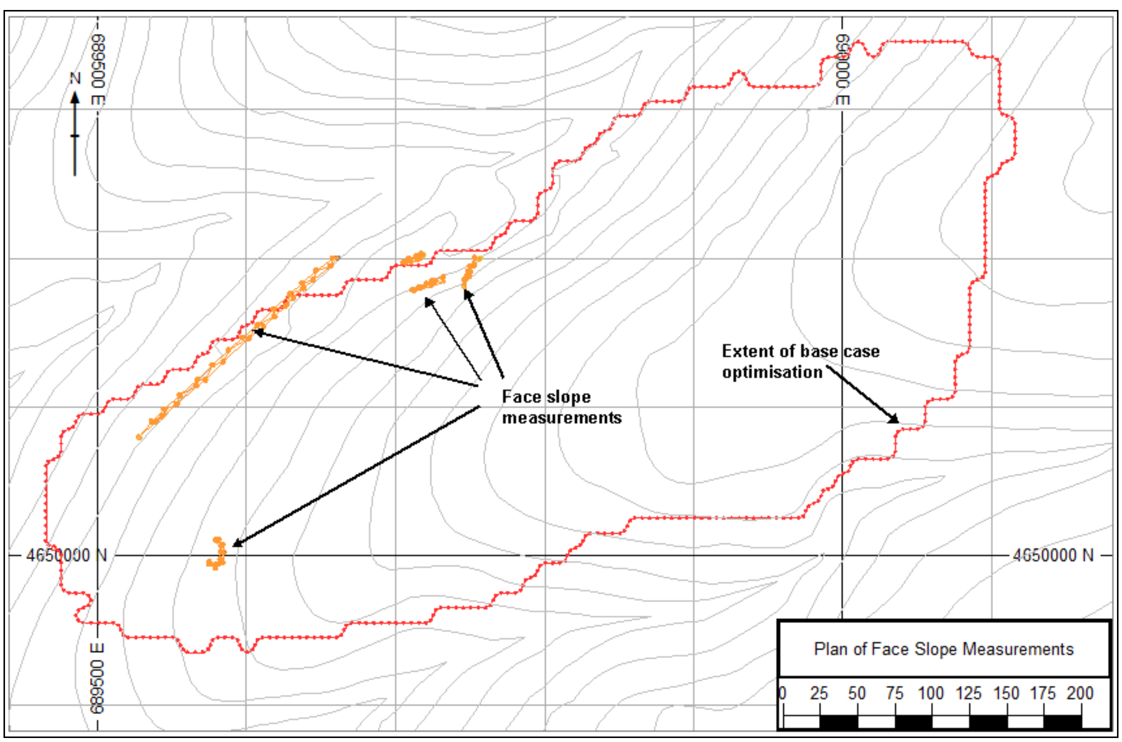
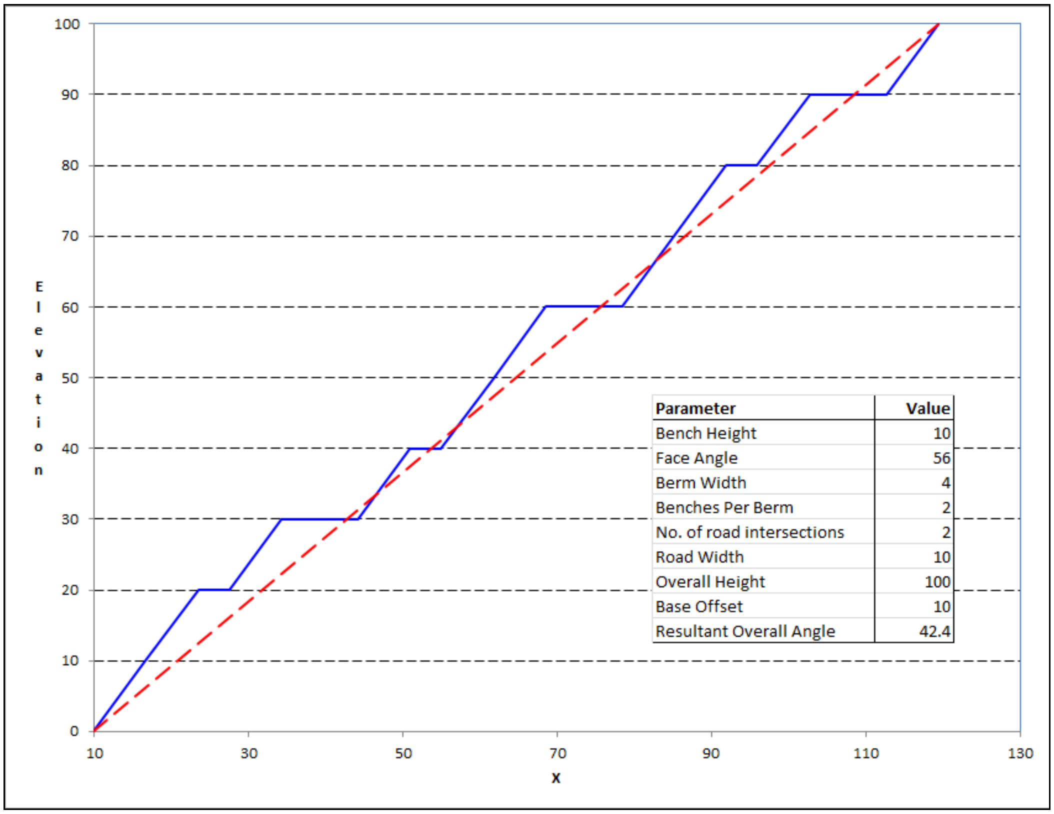


Figure 15-2. Overall Slope Angle Estimation



For optimisation work, sensitivities have also been applied, representing +/-10% variations of the base case parameters for the metal prices, the mining cost and the processing cost. More optimistic metallurgical recoveries were also tested. The results from all these optimisation runs, for the maximum cashflow pit in each case, are summarised in Table 15 2. 3D views of runs 1 and 2 are shown in Figure 15-5 and Figure 15-6. A graph of profit and discounted cashflows against pit shell, for the base case optimisation, is shown in Figure 15-3. A graph of the variation in cost per tonne of recovered metal is shown in Figure 15-4.

A plan depicting the different optimisation runs' extents is shown in Figure 15-7.

Figure 15-3. Graph of Base Case Optimisation Results

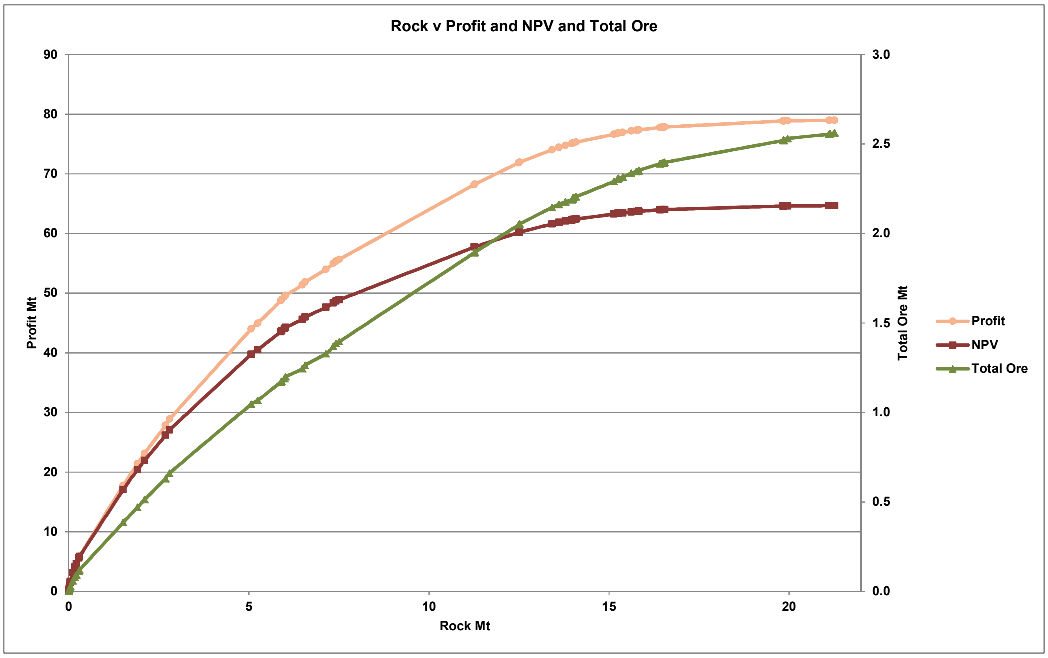


Figure 15-4. Graph of Cost of Recovered Metal – Base Case Optimisation

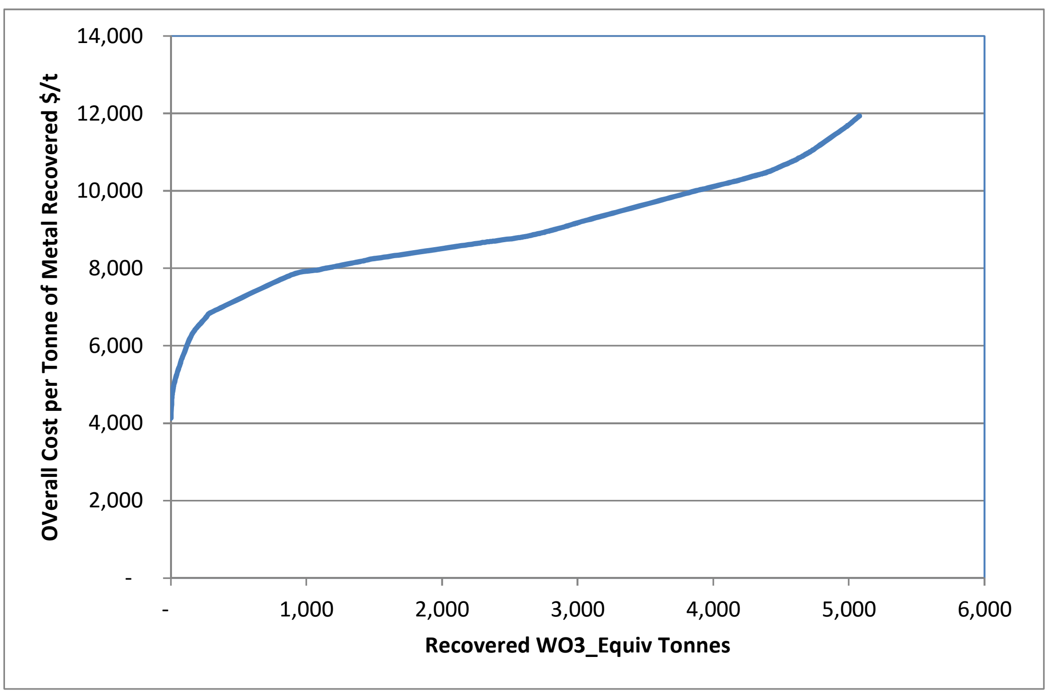


Figure 15-5. 3D View of Optimisation Extent – Base Case

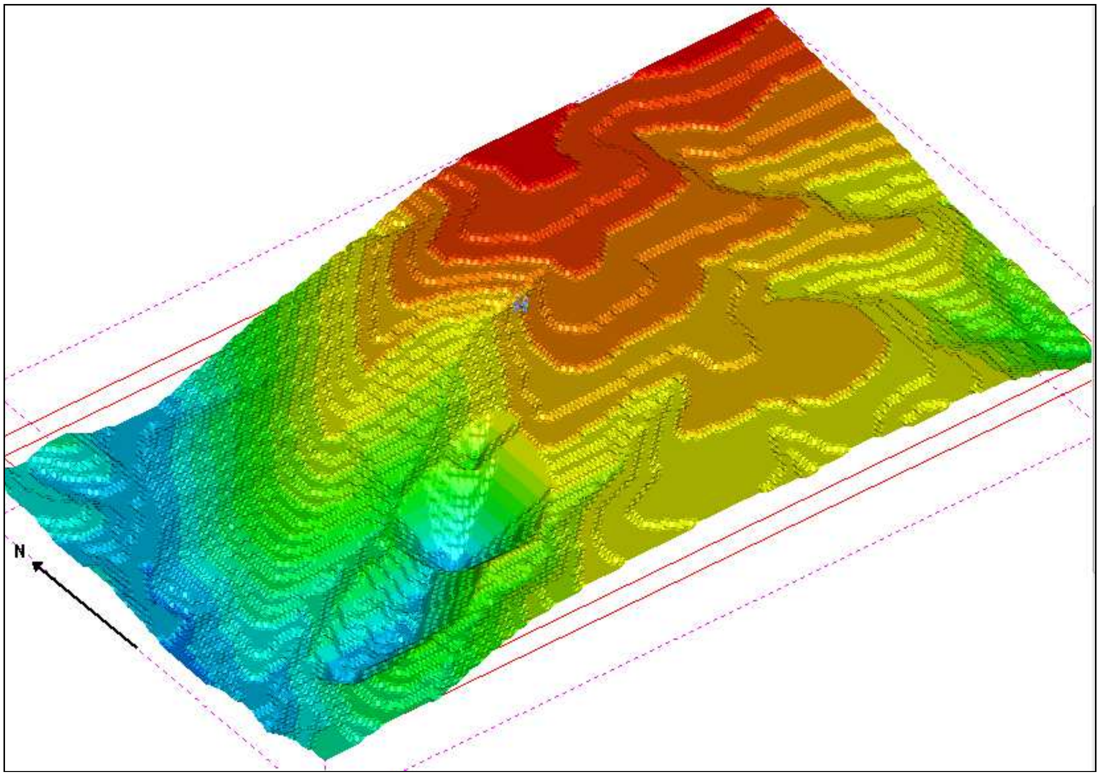


Figure 15-6. 3D View of Optimisation Extent – Indicated and Inferred Enabled

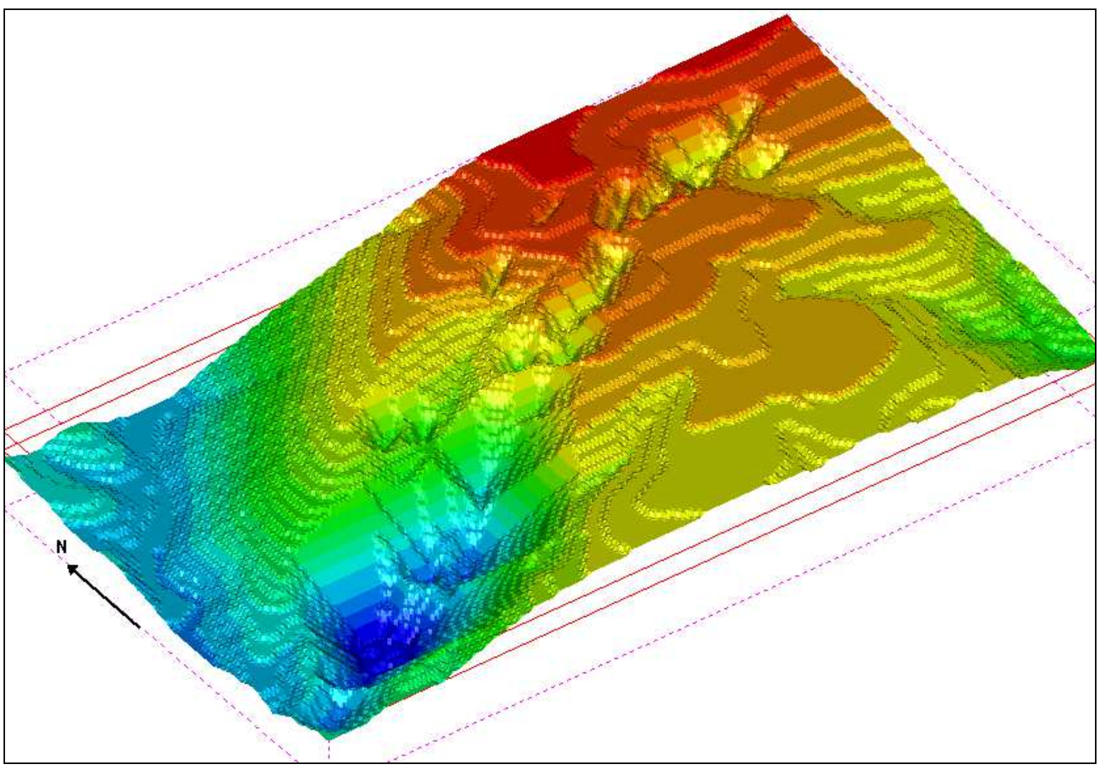


Table 15-1. Pit Optimisation Parameters

Description		Unit	Base Case	With Inferred	Price -10%	Price+10%	MC-10%	MC+10%	PC-10%	PC+10%	Recoveries	Slopes-2°	Slopes+3°
Run			1	2	3	4	5	6	7	8	9	10	11
Resources Enabled			Ind Only	Ind+Inf	Ind+Inf	Ind+Inf	Ind+Inf	Ind+Inf	Ind+Inf	Ind+Inf	Ind+Inf	Ind Only	Ind Only
Processing													
WO3	APT Price	*\$/mtu WO_3*	**370**	**370**	**333**	**407**	**370**	**370**	**370**	**370**	**370**	**370**	**370**
	Metal Price - received,	*\$/mtu WO_3*	**288**	**288**	**259.2**	**316.8**	**288**	**288**	**288**	**288**	**288**	**288**	**288**
	after transport + smelting	*\$/t WO_3*	28,800	28,800	25,920	31,680	28,800	28,800	28,800	28,800	28,800	28,800	28,800
	Recovery	*%*	**55%**	**55%**	**55%**	**55%**	**55%**	**55%**	**55%**	**55%**	**65%**	**55%**	**55%**
Sn	Metal Price	\$/lb	**10.5**	**10.5**	**9.45**	**11.55**	**10.5**	**10.5**	**10.5**	**10.5**	**10.5**	**10.5**	**10.5**
		\$/t	23,149	23,149	20,834	25,463	23,149	23,149	23,149	23,149	23,149	23,149	23,149
	Received, after transport + smelting	*\$/t Sn*	18,036	18,036	16,232	19,839	18,036	18,036	18,036	18,036	18,036	18,036	18,036
	Recovery	*%*	**65.0%**	**65.0%**	**65.0%**	**65.0%**	**65.0%**	**65.0%**	**65.0%**	**65.0%**	**70.0%**	**65.0%**	**65.0%**
	Processing Cost	*\$/t ore*	9.72	9.72	9.72	9.72	9.72	9.72	8.75	10.69	9.72	9.72	9.72
	G & A	*\$/t ore*	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25	3.25
	Total Applied Ore Cost	*\$/t ore*	12.94	12.94	12.94	12.94	12.94	12.93	11.96	13.91	12.94	12.94	12.94
	(Processing+G&A+OreMining-WasteMining)												
Mining (No drill and blast required)													
	Contractor Cost	*\$/bcm rock*	2.55	2.55	2.55	2.55	2.30	2.81	2.55	2.55	2.55	2.55	2.55
	Indirect Cost	*\$/bcm rock*	0.67	0.67	0.67	0.67	0.60	0.73	0.67	0.67	0.67	0.67	0.67
	Total Mining Cost	*\$/bcm rock*	3.22	3.22	3.22	3.22	2.90	3.54	3.22	3.22	3.22	3.22	3.22
	Ore mining	*\$/t ore*	1.26	1.26	1.26	1.26	1.13	1.38	1.26	1.26	1.26	1.26	1.26
	Waste mining	*\$/t waste*	1.29	1.29	1.29	1.29	1.16	1.42	1.29	1.29	1.29	1.29	1.29
Mining Parameters													
	Mining Recovery	*%*	**95**	**95**	**95**	**95**	**95**	**95**	**95**	**95**	**95**	**95**	**95**
	Dilution	*%*	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**	**5**
	Breakeven Economic Cut-Off	*% WO_3*	0.082%	0.082%	0.091%	0.074%	0.082%	0.082%	0.076%	0.088%	0.069%	0.082%	0.082%
		% Sn	0.110%	0.110%	0.123%	0.100%	0.110%	0.110%	0.102%	0.119%	0.102%	0.110%	0.110%
Pit Slope Parameters													
	Overall Slope Angle		42°	42°	42°	42°	42°	42°	42°	42°	42°	40°	45°

Key

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

PC = Processing Cost
MC = Mining Cost

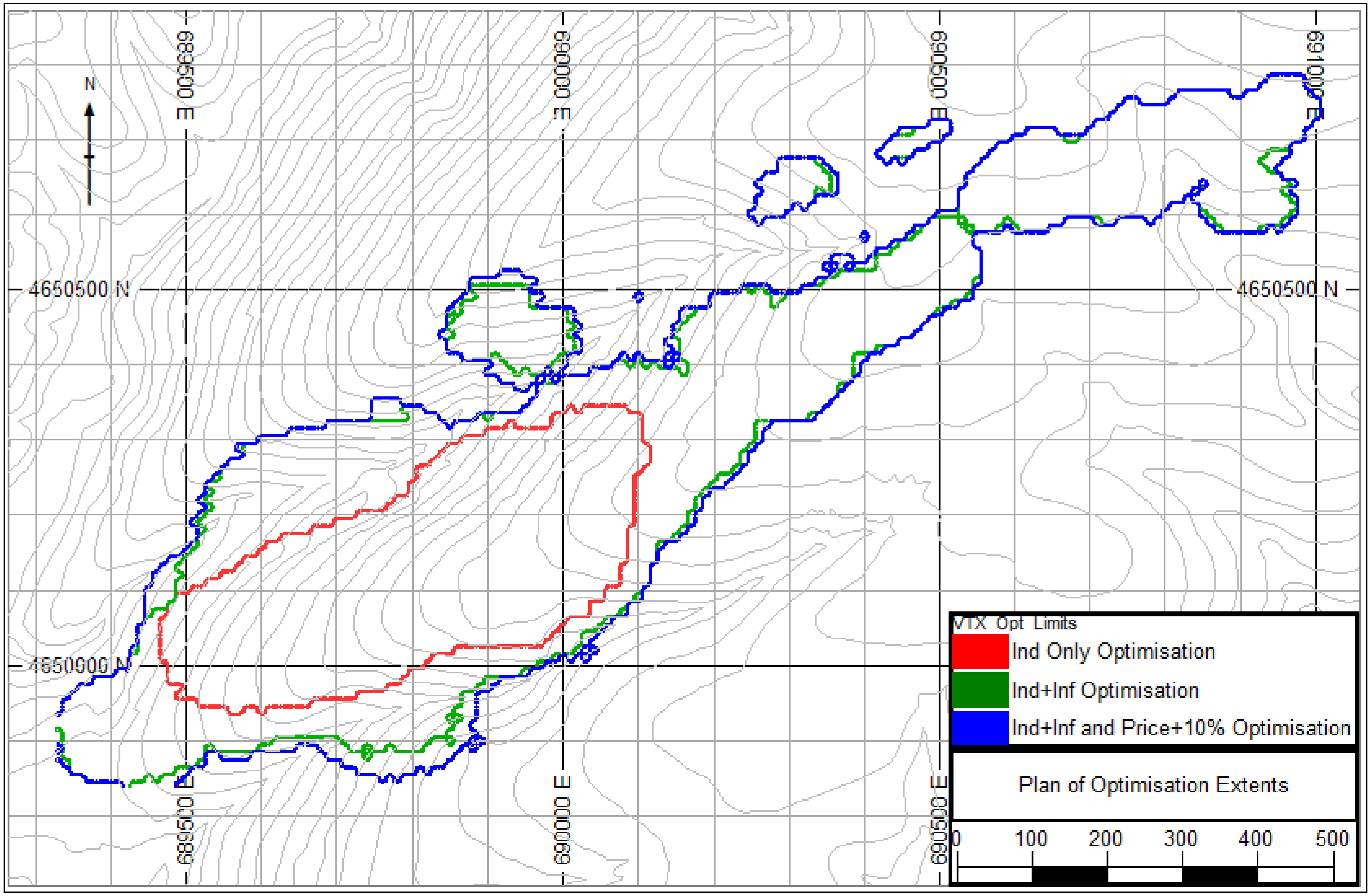
Table 15-2. Optimisation Results' Summary

Run	Description	DCF $M	Profit $M	Revenue $M	Processing Cost $M	Mining Cost $M	Rock Mt	Potential Ore Feed: Ore Mt	Sn %	WO3 %	Total Waste Mt	Strip Ratio	Recovered Products: Sn Tonnes	WO3 Tonnes	Ore Split: Ind Mt	Inf Mt
1	Base Case	64.7	79.0	139.6	33.2	27.4	21.3	2.6	0.12	0.25	18.7	7.3	2,043	3,567	2.56	0.00
2	With Inferred	120.8	228.5	456.2	151.1	76.6	59.4	11.7	0.13	0.15	47.7	4.1	10,019	9,566	2.66	9.01
3	Price -10%	102.4	184.3	385.2	131.6	69.3	53.7	10.2	0.14	0.17	43.6	4.3	8,973	9,241	2.60	7.57
4	Price+10%	152.7	339.6	616.1	190.4	86.1	66.7	14.7	0.13	0.12	52.0	3.5	11,990	9,812	2.78	11.94
5	MC-10%	123.3	236.4	460.1	152.9	70.8	61.0	11.8	0.13	0.15	49.2	4.2	10,121	9,637	2.66	9.15
6	MC+10%	117.7	221.0	448.5	147.2	80.3	56.6	11.4	0.13	0.15	45.2	4.0	9,763	9,459	2.66	8.71
7	PC-10%	123.1	240.4	467.9	149.1	78.5	60.8	12.5	0.13	0.14	48.4	3.9	10,505	9,669	2.70	9.77
8	PC+10%	118.2	217.6	442.1	151.1	73.4	56.9	10.9	0.13	0.16	46.1	4.2	9,414	9,456	2.62	8.24
9	Recoveries	148.4	293.7	541.5	165.6	82.1	63.7	12.8	0.13	0.14	50.9	4.0	11,533	11,578	2.71	10.09
10	Slopes-2^{o}	118.6	224.1	450.8	148.6	78.1	60.5	11.5	0.13	0.15	49.0	4.3	9,809	9,511	2.66	8.82
11	Slopes+3^{o}	122.1	232.0	458.9	152.4	74.5	57.8	11.8	0.13	0.15	46.0	3.9	10,096	9,611	2.66	9.11

Notes

. Max cashflow pit showed in each case
. All calculations based on operating costs only
. PC = processing cost
. MC = mining cost
. Runs 2 - 11 all with inferred resources enabled
. Run 9 - WO_3 recovery = 65% and Sn recovery = 70%, for duration of project

Figure 15-7. Plan of Optimisation Extents



15.2 Pit Design

Daytal management selected the maximum cashflow pit from the base optimisation run as being the most appropriate pit shell for ongoing design purposes. Pit design parameters used are summarized in Table 15-3.

Table 15-3. Pit Design Parameters

	Unit	Value
Bench Configuration		
Face Angle		56°
Berm Width, every 20m	*m*	4
Bench Height	*m*	10
Overall max slope with road intersections		42°
Safety berm width	*m*	14
Safety berm frequency	*m*	60
Haul Road		
Gradient		10%
Width	*m*	10

The resultant pit design is shown in Figure 15-8. Cross-sections are shown in diagrams from Figure 15-9 to Figure 15-12, along with 3D views in Figure 15-13 and Figure 15-14.

The majority of the pit design is cut into the west sloping existing hillside. A 10m wide haul road been put into the design with the exit point at the extreme west end, the lowest of point of the optimal pit rim, at an elevation of approximately 870m. This road is used to access lower parts of the design, down towards the lowest point at 820mRL. Access to the eastern, and higher, part of the pit will be gained from temporary access roads from the existing surface on higher benches.

Berms of 4m have been incorporated into the design every 20m vertically. For the extended highwall of the pit up to 1015mRL on the southern and eastern sides of the pit, additional 14m safety berms were put in every 60m vertically.

The overall pit is approximately 700m in length along strike, and 300m wide at its widest point.

Figure 15-8. Pit Design Plan

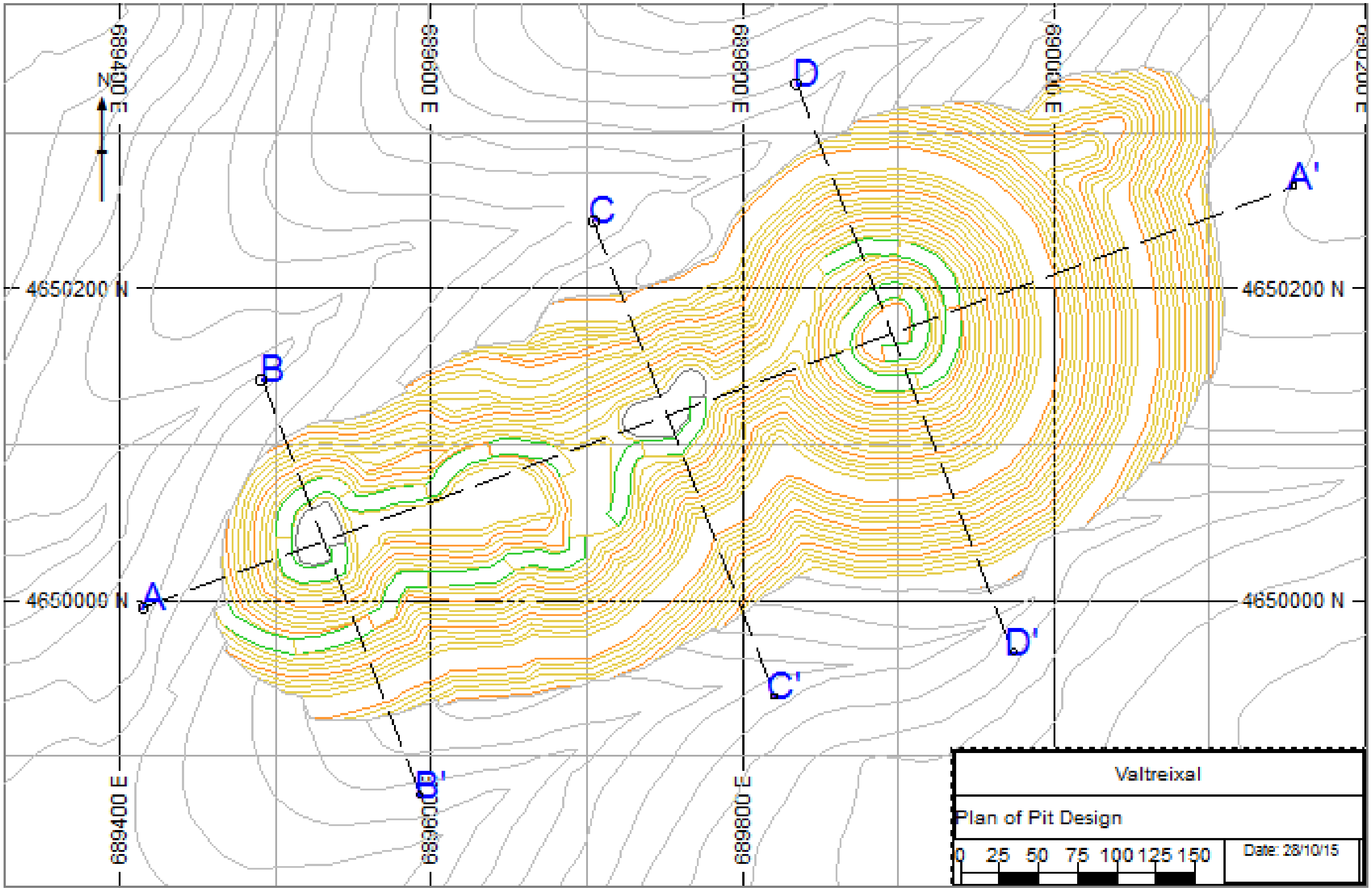


Figure 15-9. Pit Section A-A'

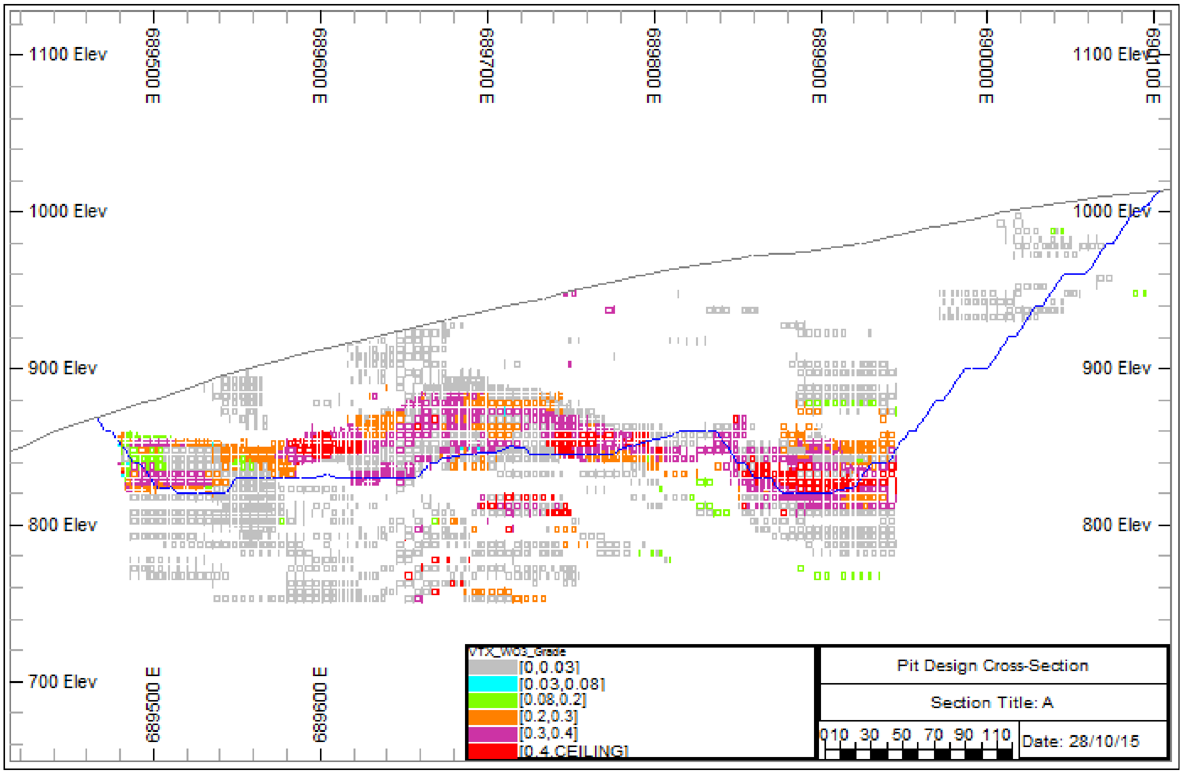


Figure 15-10. Pit Section B-B'

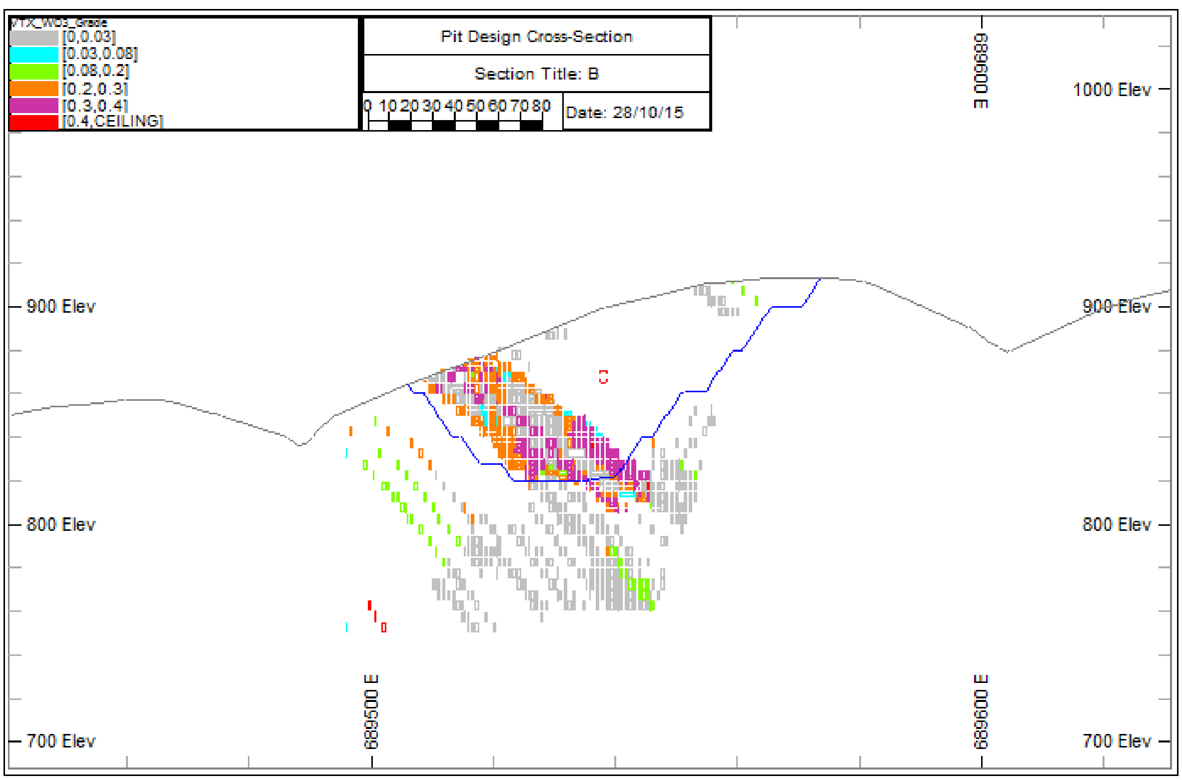


Figure 15-11. Pit Section C-C'

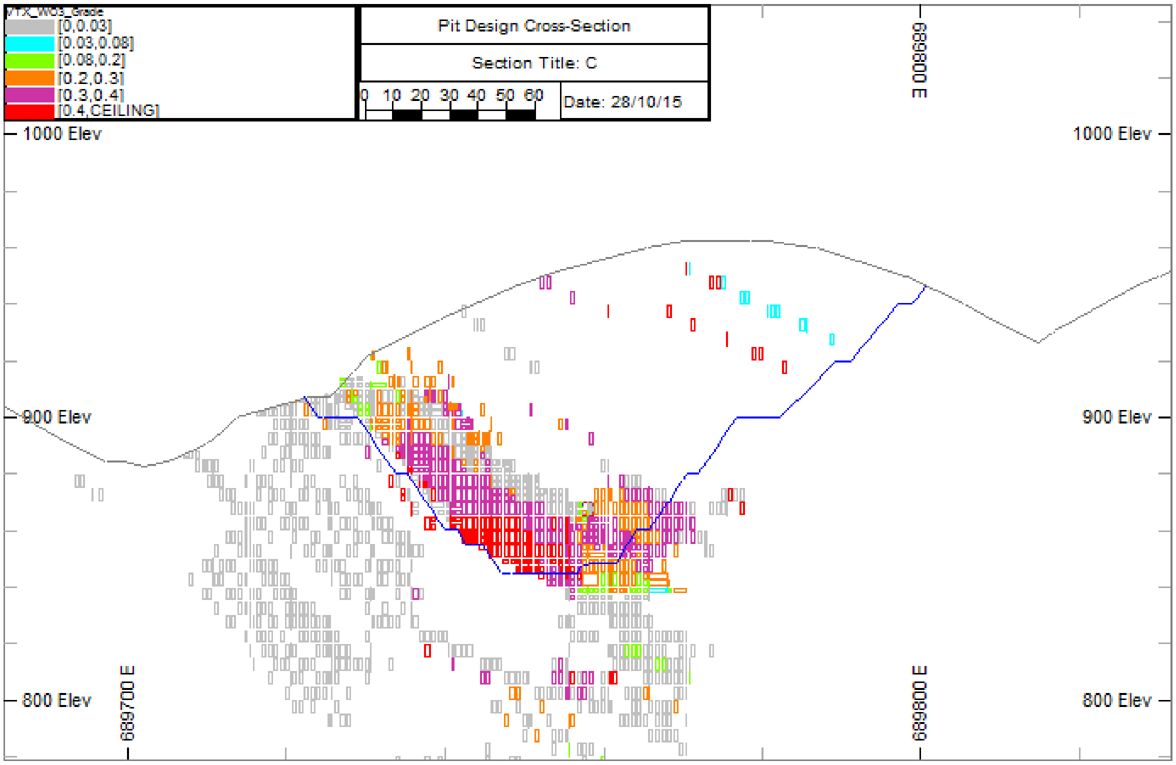


Figure 15-12. Pit Section D-D'

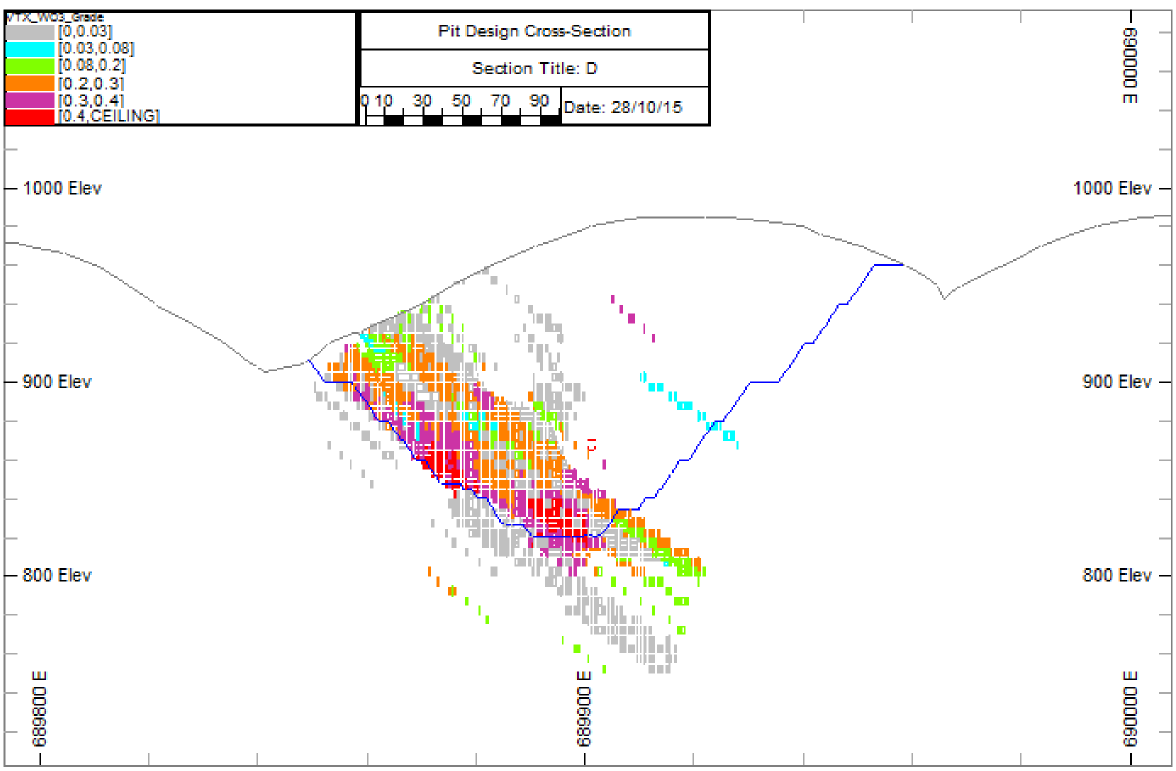


Figure 15-13. 3D View of Pit Design, from SW

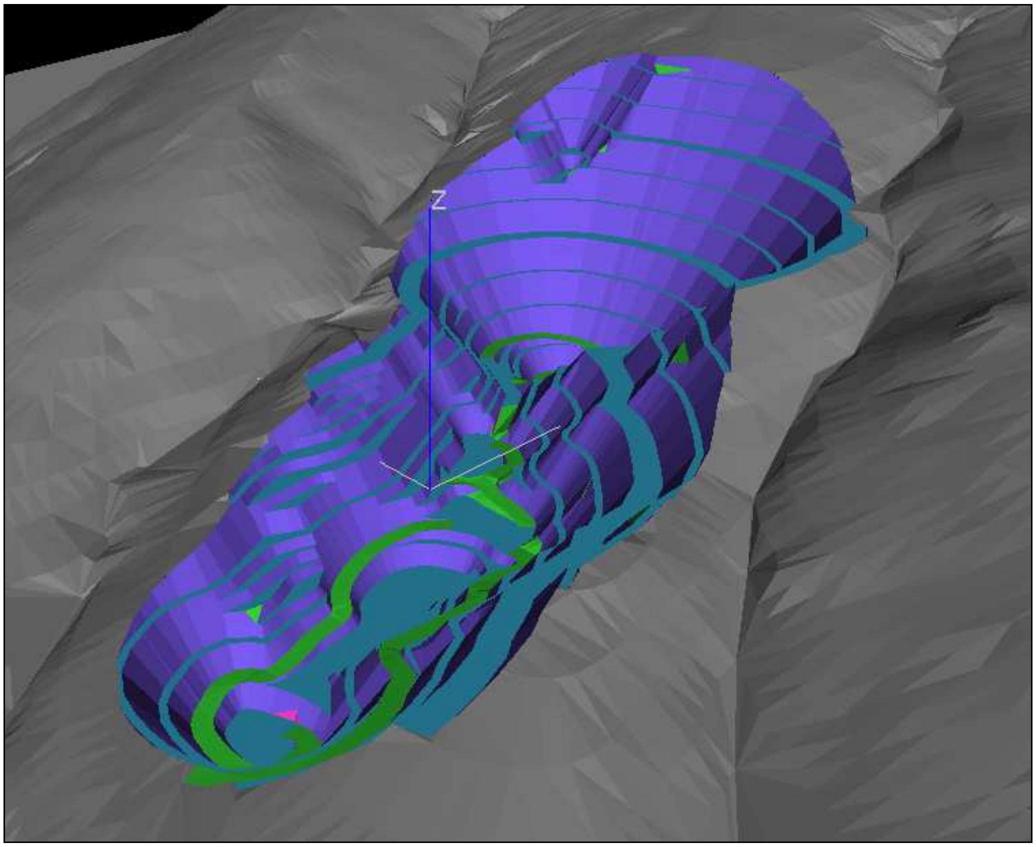
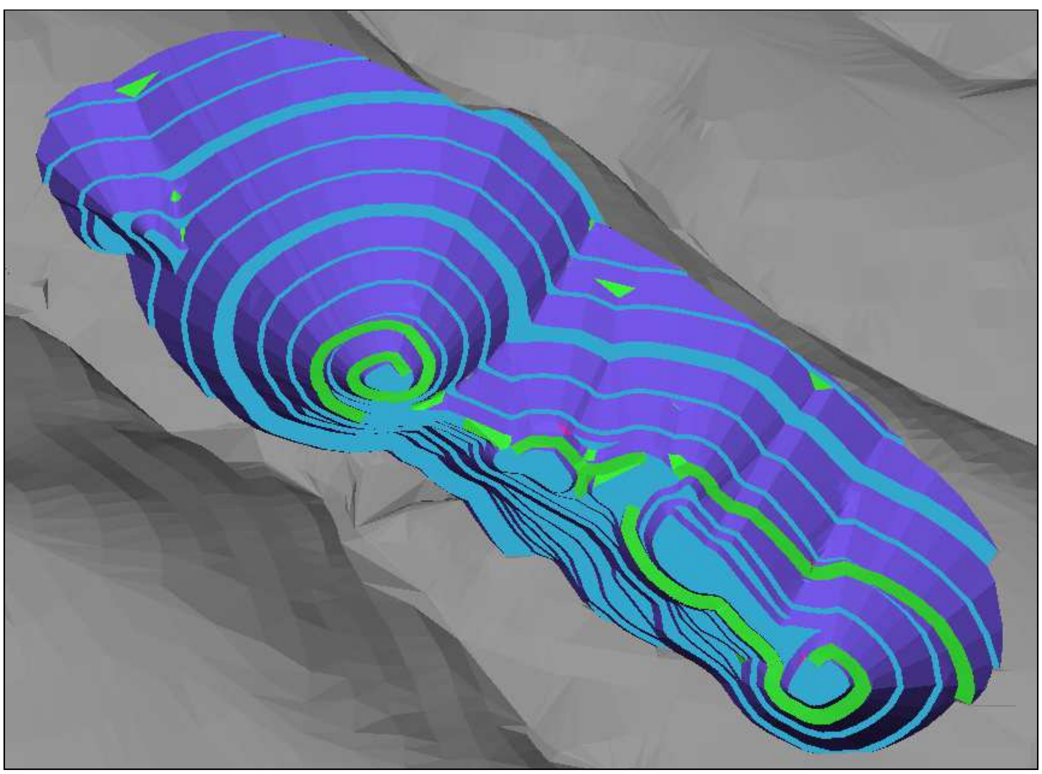


Figure 15-14. 3D View of Pit Design, from NW



15.3 Mineral Reserves

Based on this pit design, reserves have been calculated. Additional mining factors of 5% dilution and 5% mining losses have also been applied. An overall reserve summary is shown in Table 15-4, with a bench breakdown in Table 15-5 and grade-tonnage table in Table 15-6.

In these evaluations an WO_3 equivalent grade has also been determined. The equivalence factor has been determined based on the relative prices and metallurgical recoveries of WO3 and Sn. Both of the contributions of WO_3 and Sn have been used to determine ore blocks, which is equivalent to the application of the WO_3 breakeven cut-off, 0.08%, against the WO_3_Eq grade.

Table 15-4. Reserve Summary

As of October 31st, 2015

Reserve Category	Tonnes	Sn	WO_3	WO_3_Equiv	Waste (including Inferred)	Rock	Strip
	Kt	%	%	%	Kt	Kt	Ratio
Probable Reserves	**2,549**	**0.12**	**0.25**	**0.34**	21,160	23,710	8.3

Notes

. WO_3_Eq = WO_3 + (Sn x 0.74), based on:

	Price	Recovery
WO_3	**$37,000/t**	**55%**
Sn	**$23,150/t**	**65%**

. Cut-off applied to WO3_Equiv

Breakeven Cut-off = 0.08 WO_3

. Mining factors applied:

Dilution = 5%

Losses = 5%

. Pit design also contains 2.2 Mt of inferred resources at economic grades

The WO_3 equivalent grade was determined using the formula below. This equivalence factor was determined from the different relative metal prices and recoveries, such that:

WO3_Eqiv = WO3 + Sn x 0.74

Equivalence factor = (Price_Sn x Rec_Sn) / (Price_WO3 x Rec_WO3)
= (18,036*0.65) / (28,000 x 0.55)
= 0.74

Table 15-5. Reserve Bench Breakdown

	Probable Reserves				Inferred	Waste	Waste+Inf	Total Rock
BENCH	Tonnes	Sn	WO_3	WO_3_Equiv	Tonnes	Tonnes	Tonnes	Tonnes
mRL	Kt	%	%	%	Kt	Kt	Kt	Kt
1010	-				-	10	10	10
1000	-				1	147	149	149
990	-				9	341	350	350
980	-				14	641	655	655
970	0.07	0.17	0.04	0.16	24	1,041	1,065	1,065
960	12	0.16	0.13	0.25	47	1,349	1,396	1,408
950	23	0.14	0.11	0.22	70	1,423	1,493	1,516
940	32	0.13	0.11	0.21	107	1,544	1,651	1,683
930	24	0.13	0.09	0.19	155	1,622	1,776	1,801
920	31	0.13	0.08	0.17	156	1,832	1,987	2,018
910	72	0.12	0.13	0.22	173	1,882	2,055	2,127
900	173	0.11	0.20	0.28	183	1,886	2,069	2,243
890	243	0.11	0.24	0.32	123	1,505	1,628	1,871
880	322	0.11	0.28	0.37	161	1,304	1,465	1,787
870	408	0.12	0.28	0.37	182	957	1,139	1,547
860	423	0.13	0.27	0.37	201	744	946	1,369
850	381	0.13	0.27	0.36	235	404	639	1,020
840	244	0.12	0.26	0.35	187	225	412	656
830	118	0.14	0.29	0.39	108	109	218	336
820	42	0.15	0.34	0.45	30	28	57	99
TOTAL	**2,549**	**0.12**	**0.25**	**0.34**	**2,169**	**18,992**	**21,160**	**23,710**

Notes

. WO_3_Eq = WO_3 + (Sn x 0.74), based on:

	Price	Recovery
WO_3	$37,000/t	55%
Sn	$23,150/t	65%

. Cut-off applied to WO3_Equiv

Breakeven Cut-off = 0.08 WO_3

. Mining factors applied:

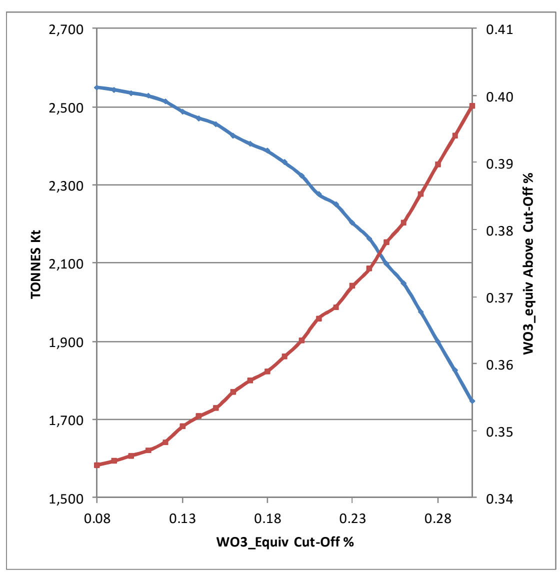
Dilution = 5%

Losses = 5%

. Inferred material reported above is at economic grades

Table 15-6. Grade-Tonnage Table – Pit Reserves

WO3_Equiv Cut-Off	Tonnes	WO_3_Equiv	Sn	WO_3
%	Kt	%	%	%
0.08	2,549	0.34	0.12	0.25
0.09	2,543	0.35	0.12	0.25
0.10	2,535	0.35	0.12	0.26
0.11	2,527	0.35	0.12	0.26
0.12	2,513	0.35	0.12	0.26
0.13	2,488	0.35	0.12	0.26
0.14	2,470	0.35	0.12	0.26
0.15	2,455	0.35	0.12	0.26
0.16	2,426	0.36	0.12	0.26
0.17	2,405	0.36	0.12	0.27
0.18	2,386	0.36	0.12	0.27
0.19	2,357	0.36	0.12	0.27
0.20	2,324	0.36	0.12	0.27
0.21	2,276	0.37	0.12	0.27
0.22	2,250	0.37	0.12	0.28
0.23	2,202	0.37	0.12	0.28
0.24	2,162	0.37	0.12	0.28
0.25	2,097	0.38	0.13	0.29
0.26	2,048	0.38	0.13	0.29
0.27	1,975	0.39	0.13	0.29
0.28	1,900	0.39	0.13	0.30
0.29	1,825	0.39	0.13	0.30
0.30	1,747	0.40	0.13	0.30



16 MINING METHODS

Considerable amounts of earthworks have been completed over the past 2 years at Valtreixal, for preparation of drilling platforms and road access purposes. These cuttings have been made in all of the main rock types that will occur inside the pit envelope. Geologists have also had to take bulk samples from underground galleries for metallurgical testing. This experience has indicated that all of the rock types within the Valtreixal pit can be mined by free digging, so drilling and blasting will not be required.

Main benches will be 10m high, with 5m sub-benches in ore, in order to assist with ore/waste discrimination. Grade control will be assisted by the taking of bench face and trench samples, as well as uv lights to assist with the identification of scheelite.

It is assumed for this study that mining operations will completed by contractors. Unit contract rates have been used from the current mining operations at Los Santos (also owned by Daytal), with adjustments for the absence and drilling and blasting, as well as for the application of the average ore and waste rock densities at Valtreixal.

For scheduling purposes in the current study, the pit has been divided into two principal pushbacks, approximately dividing the pit into a west half and an east half. Mining will start in the western (lower) pushback, and then as mining progresses deeper in this pushback, mining will also start on the upper benches of the eastern pushback. Many of these upper benches will be accessible from temporary roads on south-eastern side of the pit. This pushback arrangement is shown in the long section in Figure 16-1. These pushbacks were used in the development of a mining schedule, for 500 Ktpa ore feed to the plant, as summarised in Table 16-1, with a pre-strip in year 0.

The development of the waste dumping facilities will based on the following guidelines:

- Minimisation of the size of the final open hole from pit excavations.
- Progressive restoration of the exploited areas as the project advances.
- Minimisation of the additional dump of mine waste.
- Reduction of mining costs by reducing haulage distances with in-pit waste dumping.

As demonstrated in the optimisation runs with inferred material, additional exploration work will justify a much bigger open pit, advanced over a much longer strike direction. The general sequencing strategy is to excavate the pit areas from west to east, with dumping of mine waste from the active east advancing benches into the previously excavated western pit areas.

Figure 16-1. Long-Section of Pushbacks

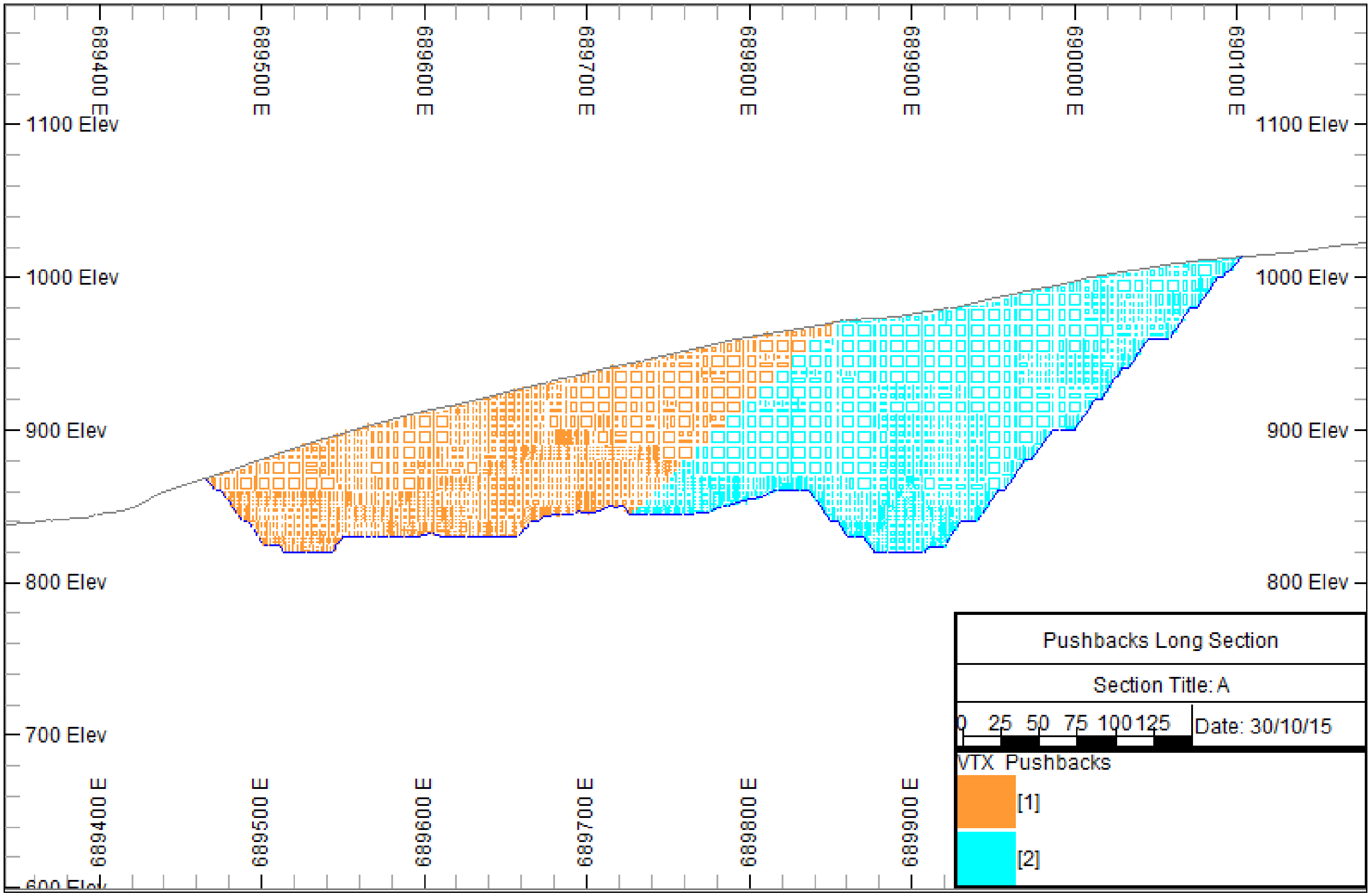


Table 16-1. Mining Schedule

	Units	Total	Year 0	1	2	3	4	5
----- PUSHBACK 1 ----								
PB 1 VOL WASTE	*m3*	3,317,208	1,663,858	949,548	375,184	216,528	112,091	
PB 1 VOL ORE	*m3*	597,913	19,735	187,051	172,970	150,224	67,933	
PB 1 VOL TOTAL	*m3*	**3,915,121**	1,683,594	1,136,599	548,154	366,751	180,023	
PB 1 TON ORE	*t*	1,528,837	52,150	487,462	437,566	379,389	172,270	
PB 1 WO3 %	*%*	0.28	0.25	0.30	0.29	0.26	0.25	
PB 1 Sn %	*%*	0.12	0.10	0.10	0.13	0.13	0.12	
PB 1 %WO3_EQ	*%*	0.37	0.32	0.38	0.38	0.35	0.34	
PB 1 RATIO (m3)		5.55	84.31	5.08	2.17	1.44	1.65	
----- PUSHBACK 2 -----								
PB 2 VOL WASTE	*m3*	5,160,750		1,392,752	1,070,875	1,178,077	975,682	543,365
PB 2 VOL ORE	*m3*	394,064		5,542	24,126	47,937	125,476	190,983
PB 2 VOL TOTAL	*m3*	**5,554,814**		1,398,294	1,095,000	1,226,014	1,101,158	734,348
PB 2 TON ORE	*t*	1,030,622		14,141	62,594	122,343	327,989	503,554
PB 2 WO3 %	*%*	0.21		0.12	0.11	0.11	0.19	0.27
PB 2 Sn %	*%*	0.13		0.16	0.14	0.12	0.13	0.13
PB 2 %WO3_EQ	*%*	0.31		0.24	0.21	0.20	0.28	0.37
PB 2 RATIO (m3)		13.10		251.30	44.39	24.58	7.78	2.85
---- LIFE OF MINE ----								
Vol Waste	*m3*	8,477,958	1,663,858	2,342,299	1,446,059	1,394,605	1,087,772	543,365
Vol Ore	*m3*	991,977	19,735	192,593	197,096	198,160	193,409	190,983
Vol TOTAL	***m3***	**9,469,935**	1,683,594	2,534,893	1,643,154	1,592,765	1,281,181	734,348
Ton ORE	***t***	**2,559,459**	52,150	501,603	500,160	501,732	500,259	503,554
% WO3	*%*	0.25	0.25	0.30	0.26	0.22	0.21	0.27
% Sn	*%*	0.12	0.10	0.10	0.13	0.13	0.13	0.13
% WO3_EQ	***%***	**0.34**	0.32	0.37	0.36	0.32	0.30	0.37
STRIP RATIO	*t/t*	**8.2**	79.4	11.6	7.2	6.9	5.4	2.7
STRIP RATIO	*m3/m3*	8.5	84.3	12.2	7.3	7.0	5.6	2.8

17 RECOVERY METHODS

The beneficiation plant will have a nominal treatment capacity of 500,000 t / year of crude ore. Under normal conditions, the plant will work 24 hours a day, 365 days a year, and so will operate at approximately 58 tph. All of the milling operations will be housed in single steel clad building structure, with an area of approximately 20,000 m^2. It is estimated that the electric power required for the Treatment Plant will be approximately 1,500 kW.

The milling operation is envisaged as:

- Crushing, grinding and gravity separation of scheelite and cassiterite into a bulk concentrate;
- Removal of sulphides from the bulk concentrate by flotation; and
- Drying and electrostatic separation of the bulk concentrate into scheelite and cassiterite concentrates.

Similar to current milling operations at Los Santos, crushing and grinding operations would like comprise a jaw crusher, cone crushers, followed by a rod mill. Following separation into different size fractions through hydrocyclones, gravity separation is done through spirals and shaking tables. An overall flowsheet is shown in Figure 17-1.

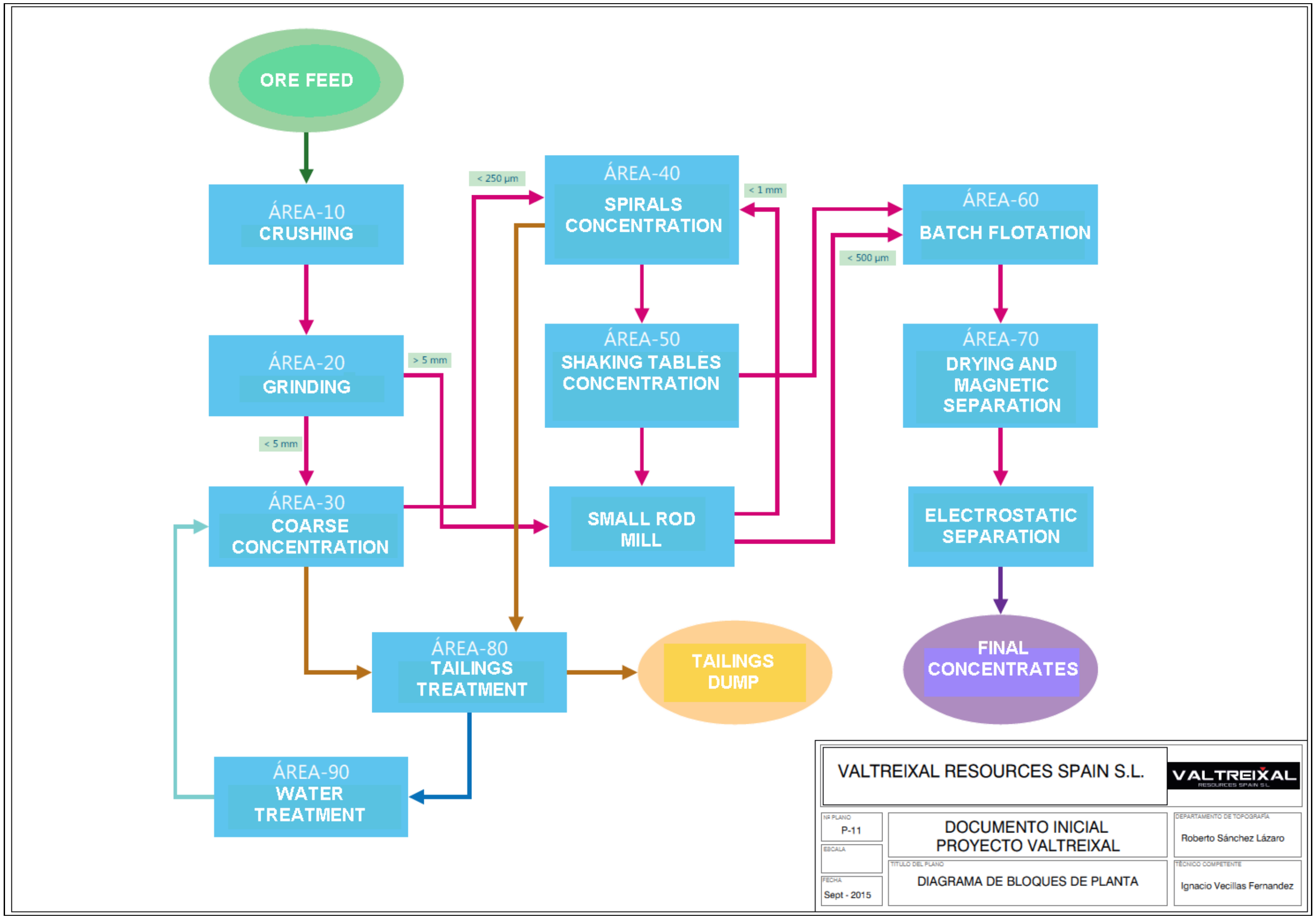
From the recent pilot plant studies, ore beneficiation at Valtreixal is dependent on:

- The fragility of scheelite, which assists in the production of a recoverable amount of ultrafine material, through classical gravity concentration systems.
- The lithology of the mineralized zone, and its geotechnical characteristics, will allow easy release of the mineral without using excessively aggressive media. This should also avoid the production of excessive fines which hamper recovery.
- The existence of two minerals with WO_3 and Sn, but with similar densities, provides different behavior which can assist using different methods for their comminution.

Mined ore will fed to a jaw crusher from a ROM pad. It is envisaged that separate stockpiles, according to WO_3 or Sn content will be developed maintained. The jaw crusher material will produce minus 200mm material.

The material will then be passed through a cylindrical trommel so as to separate clay and very fine material. The next stage will be a one or two stage roller mill, producing minus 5mm material.

Figure 17-1. Overall Mill Flowsheet



The minus 5mm material from the trommel will go through a process of pre-concentration. Test are still being conducted to determine the best dense media or jig arrangement for this. Two process lines will now be created through use of a Reichert spiral classifier, based on a 250 micron split:

- Coarse material for Sn concentration.
- Fine material for WO3 concentration.

The small rod mill will produce material less than 0.5mm, to assist cleaning in the subsequent flotation. The rod mill will not be more than 20 kW, and run at approximately 1 tph.

Two identical circuits will now be used, with successive stages roughing, cleaning and refining. The roughing step will generate a pre-concentrate which will be passed onto the shaking tables). Mixed material will pass through further spirals stages and a hydroclassifier, for more detailed separation.

There will be then three separate lines of Wilfley shaking tables, working on three different granulometric ranges.

A discontinuous flotation process will be used to eliminate sulphides that may accompany ore concentrates. This process will be performed on the bulk concentrate from the first concentration step whose particle size was between 250 microns and 5 mm. The rate of discontinuous flotation will not exceed 2 t / hour.

Flotation concentrate will be dried by a rotary drier with direct flame using diesel fuel. Subsequent to drying, the product will be through magnetic separation, for the removal of metallic or ferromagnetic penalizing particles.

Thereafter, the product will pass through an electrostatic separation process. In this case the mineral particles will pass through a charged by induction metal plate, so that the minerals low conductivity (scheelite, WO3) remain adhered to the plate and high conductivity (cassiterite, Sn) are repelled by the plate.

Tailings will be generated at three different points:

1. Steriles between 250 microns and 5 mm, generated in the initial dense media/Jig preconcentration.

2. Sterile less than 1mm, from the gravity concentration spirals.

3. Sterile less than 38 microns (slimes) from the overflow of the hydrocyclones located at the start of gravity concentration circuit.

In the tailings treatment plant, the first two groups will be drained through vibrating screens, arranging and accumulating material on a draining floor to reduce its humidity below 10% for subsequent handling and final transport.

The third group of tailings will first be subjected to settling in with an anionic flocculant. The decanted solids will then pass through a filter press or vacuum filter, to produce material with a humidity below 25%. This material will then be handled along with the thicker dry tailings material. All reclaimed water from tailings treatment will be reused in the gravimetric sorting circuit.

Water will be used extensively during gravimetric classification. It is expected that 600 m3/hour of water flow will be required in the water treatment plant, for reuse in the processing plant. Suspended fines will be settled out in the treatment, so clean overflow can be recirculated back. On the mine site ditches, will collect all surface run-off, which will be collected in settling ponds for reuse or discharge. Two open water tanks, each of 5,000m^3, will be installed.

18 PROJECT INFRASTRUCTURE

Various aspects of the planned infrastructure have been documented in a 'Documento Inicial', which has been submitted to the local authorities in Sept, 2015.

It is envisaged that the initial waste dumping would be to fill a set of small valleys south of the deposit. After this further mining, from new exploitation zones would fill those pit areas already excavated, progressing from the south-west. A waste dump and a dry tailings dump have been designed, as shown in the plan in Figure 18 1. Design parameters for these dump volumes are summarised in Table 18-1.

Resultant dump volumes are summarised in Table 18-2. These designed volumes, with some in-pit dumping of waste, would provide sufficient capacity for the mining operations outlined for the reserve pit in the current study.

Table 18-1. Dump Design Parameters

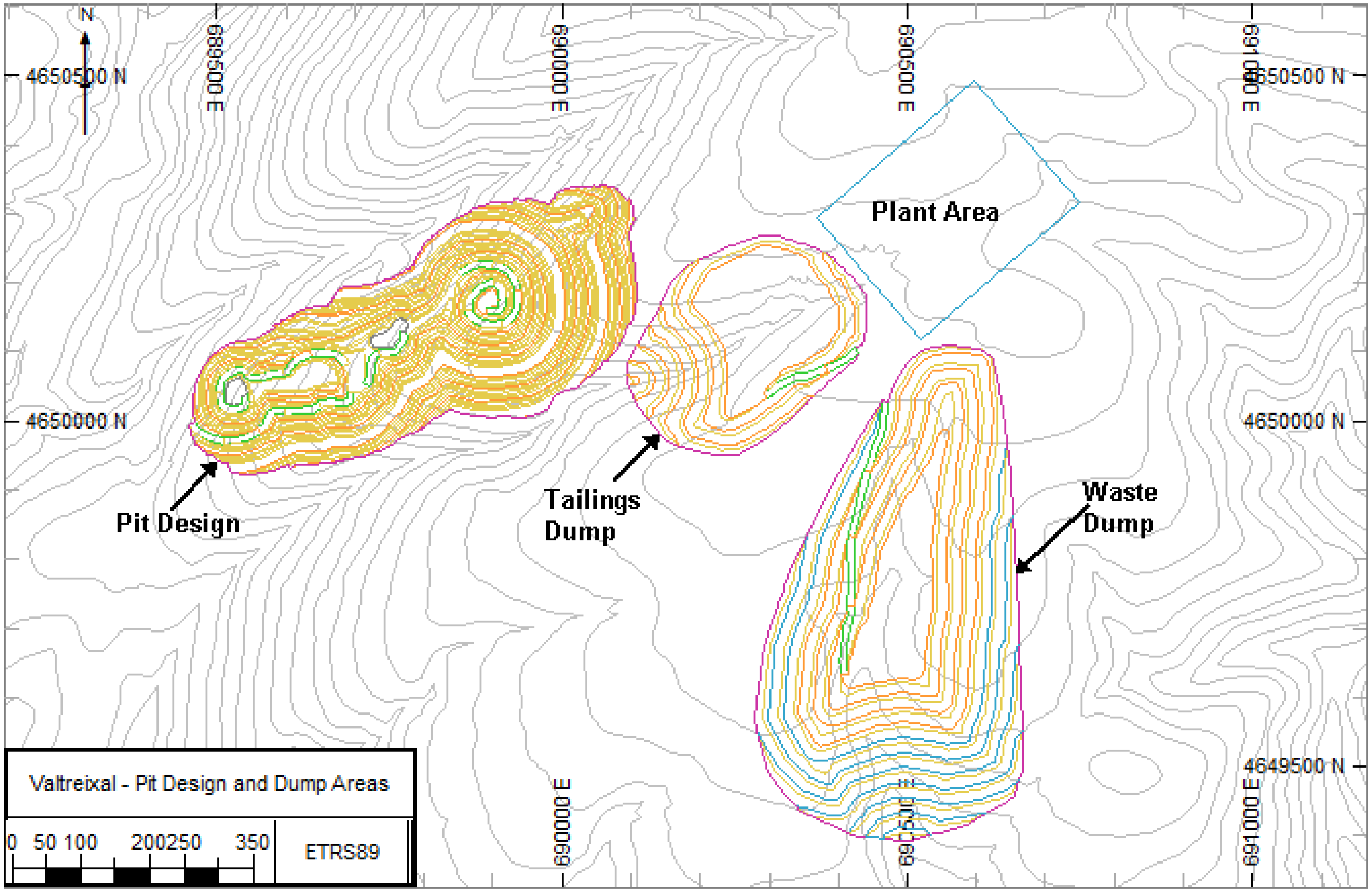
	Unit	Value
Bench Configuration		
Face Angle		35°
Berm Width, every 10m	*m*	7
Bench Height	*m*	10
Overall max slope resulting		24°
Haul Road		
Gradient		10%
Width	*m*	10

Table 18-2. Designed Dump Volumes

Waste Dump	
BENCH mRL	Volume m^3 x 1000
940	7
950	52
960	138
970	290
980	631
990	1,122
1,000	1,279
1,010	1,200
1,020	941
1,030	655
1,040	400
Total	**6,715**

Tailings Dump	
BENCH mRL	Volume m^3 x 1000
960	2
970	20
980	71
990	181
1,000	340
1,010	479
1,020	415
Total	**1,508**

Figure 18-1. Plan of Pit and Dump Areas



A potential area for the plant location would be in a small valley south and parallel to the Valtreixal valley. This would be approximately be in the centre of gravity of the overall deposit with respect to ore transport, and would allow easy access from nearby roads; and electricity and water lines.

Pit and underground drainage water can be collected, will provide a supply of water all year long. In addition to this water can be obtained from a nearby river. This will require authorisation from Spanish public administration, which is an easy process and no problems are foreseen with this.

Other aspects of project infrastructure which have been planned include:

- **Offices**. The office installation will require an area of approximately 350 m^2, and will be in the form of prefabricated models sitting on cemented pads.

- **Laboratory**. This installation will require an area of approximately 400 m^2. As well as the equipment for XRF assaying purposes, this will include cutting and sample preparation facilities.

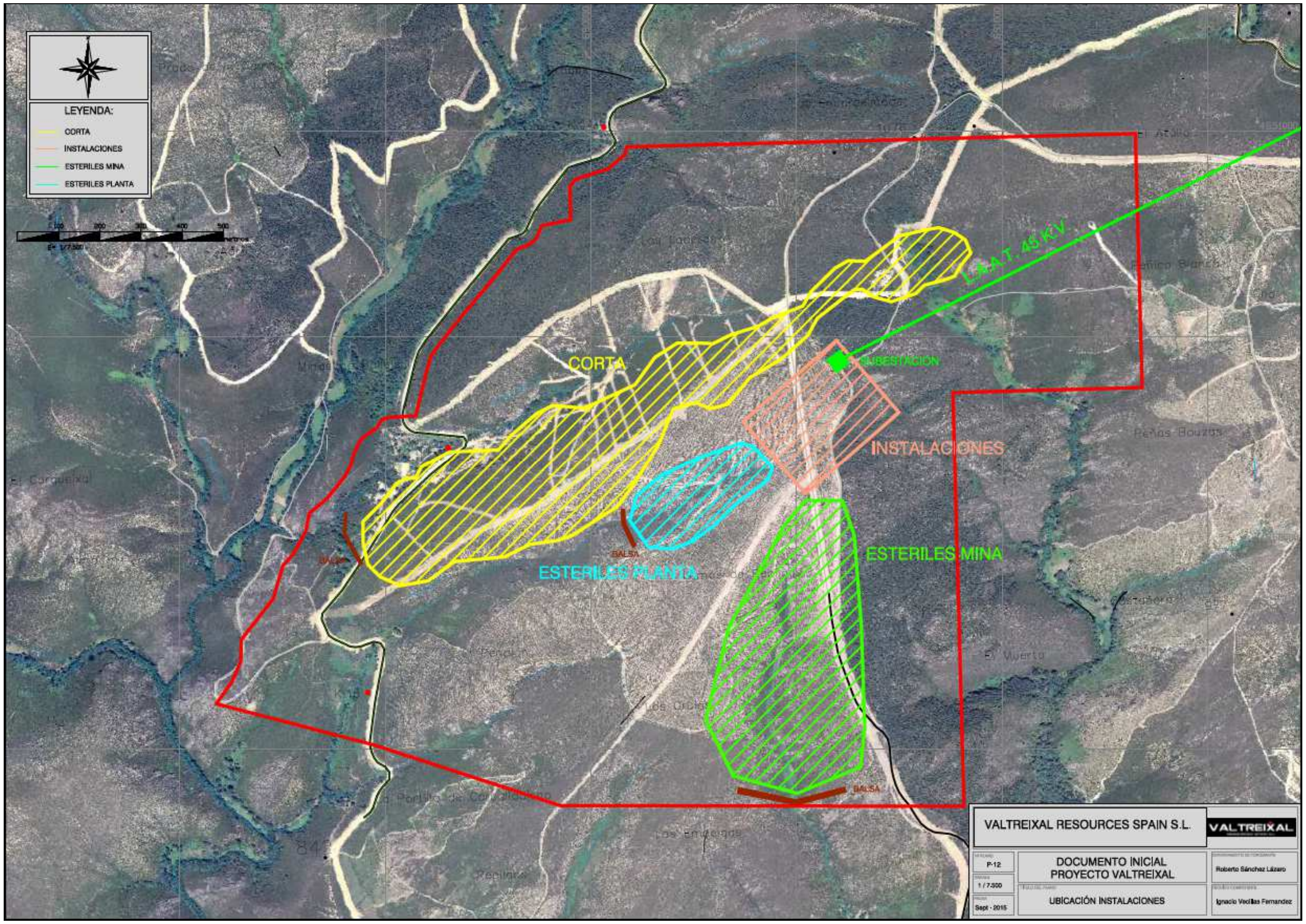
- **Maintenance Workshop**. This will require an area of approximately 600 m^2, for maintenance and repair of mining machinery.

- **ROM Pad**. The ROM pad area of 15,000 m^2 will allow 10m banked-up stockpiles, and will have an ore capacity of approximately of 120,000 t.

- **Hydrocarbon Storage Area**. The operation will have three tanks for storing hydrocarbons, with a capacity of 40,000 l each. The area required will be 350 m^2.

- **Electrical Supply.** Electrical power will be supplied by the company Gas Natural FENOSA. The facilities will include a new 45 kV overhead supply line from Cobreros, a 45/20 kV transformer substation, an underground power line on-site and 20 kv/400v transformer centre.

Figure 18-2. Overall Plan of Mine Infrastructure



19 MARKET STUDIES AND CONTRACTS

The majority of production will be committed under a long-term contract, equivalent to that of Los Santos, Almonty's other producing mine in Spain. This contract calls for delivery, in 1 tonne Bulk Bags, in standard short container lots per consignment, of tungsten concentrate grading plus 65% WO_3.

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

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

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL IMPACT

The western part of the P.I. Valtreixal No. 1906 license area is currently classified as "Reserva de Caza" – hunting reserve, which is relatively low level of environmental protection. All the other areas have no specific environmental protection. Discussions with the municipality of Pedralba and the mining directorship of the Castilla and Leon Region have started and advanced, so as to move the affected part of the hunting reserve out of the Valtreixal license areas.

The C.E. Alto de Repilados No. 1352 license area is already an active mining concession, and so will only need environmental impact assessment (Declaración de Impacto Ambiental, DIA) for open pit mining exploitation. Ongoing studies of the Valtreixal deposit have been presented to the authorities (el Director Facultativo), in order for both areas to get C.E. (Concesion de Explotacion) status. Valtreixal Resources can then submit a mining plan (Proyecto de Explotacion) and an environmental impact study D.I.A. (Declaración de Impacto) in order to get mining activities approved.

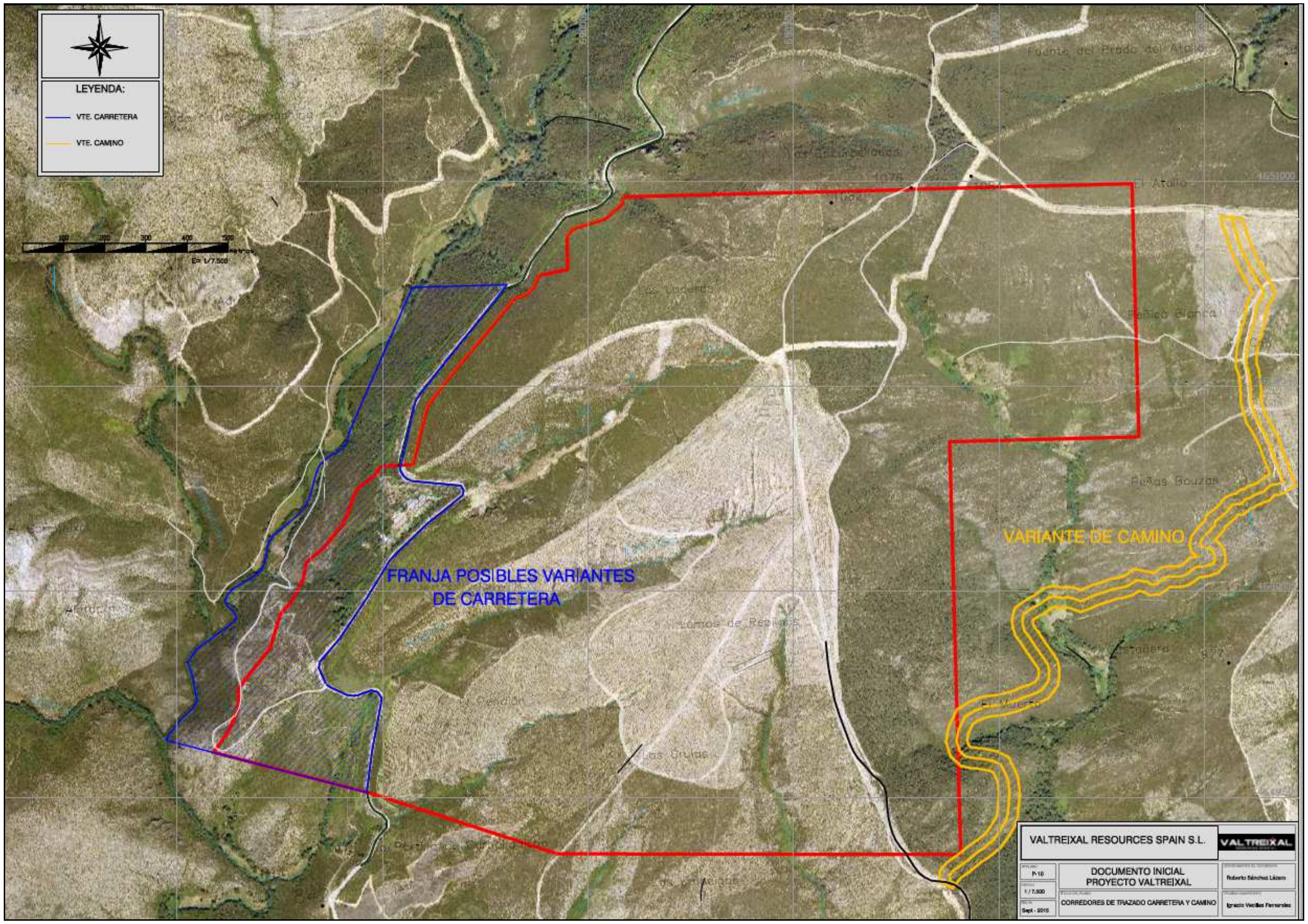
The local municipality, as well as the local government for the province of Zamora, are very supportive of potential mine development at Valtreixal. There is very little economic activity in the region, and both Calabor and Valtreixal are old mining areas. It is therefore expected that full cooperation of the local administrations should help relatively delay-free development of the project.

20.1 Road ZA-925

The south-eastern corner of the planned open pit cuts access the road ZA-995, which connects Puebla de Sanabria with the Calabor at the Portuguese border. It therefore appears that diversion of this road is necessary. Valtreixal Resource will therefore be preparing a corresponding Environmental Impact Study Project connected with the necessary road diversion and will continue in contact with the holder body of the road, Territorial Development Service Zamora. A possible road diversion scheme is shown in Figure 20-1. An amount of 3M Euros has been estimated in the capital costs of the current study, associated with this road diversion.

It is understood that the road diversion will be take into account protection distances regulated by Law 10/2008, of December 9, Highway Castilla and Leon, and its Regulation approved by Decree 45/2011.

Figure 20-1. Possible Road Diversion



20.2 Telephone Line

There is a suspended telephone line that runs substantially parallel to the current road ZA-925, so that the necessary deviation of the telephone line should be in line with the planned diversion of the ZA-925 road. This line provides telecommunications service to the town of Calabor, from Pradería Pedralba, and belongs to the telephone company Telefónica de España, SAU. Ongoing work is required, therefore, for the planned deviation of this line, along whether the new line should be suspended or partly buried.

20.3 Power Line to Calabor Village

As in the case of the road, the suspended electric power line currently supplying to the town of Calabor and the Spa Neighbour will probably need diverting. This will occur at around 17 km of road ZA-925. It is not expected to affect the course of the line that supplies electricity to the town of Santa Cruz de Abranes. Consultation will therefore be required with the owners, GAS NATURAL FENOSA, to organise the possible diversion of the line, along with 45 kV electrical supply line to the mine site.

20.4 Impact on riverbeds and banks

There are two streams that will be potentially affected by the Valtreixal project are:

• Cabuerca de la Mina, 1.39 km in length, and will be affected by a short section along its length.

• Regato of Cuballón , 1.67 km in length, located in the area of the tailings facility and Treatment Plant , parallel to the above but south . It will be affected along approximately 40% of its length, and will therefore require diversion with buried pipes near the mine installations.

20.5 Condition of "Smuggling Route "

As its name suggests, this path was formerly used to move goods (snuff and coffee) from the border undetected at border crossings. At present, it is marked as a tourist route of great landscape value, linking the towns of Pedralba of Pradería and The Abeleda in the Portuguese zone. There is no degree of cultural protection especially this route or path, since the phenomenon of smuggling was common and continuous throughout the Spanish-Portuguese border during the post-war era.

This probable deviation of the route would avoid the foreseeable impact of the operation on the landscape value of the route, and maintain all its qualities.

21 CAPITAL AND OPERATING COSTS

Mine and plant operating costs have been derived from operational experience at. Los Santos, as well as the mineral processing study by Santa Barbara. These parameters have been updated since the preliminary optimisation parameters shown in Table 15-1 were estimated. Combined with the estimated metallurgical parameters from Santa Barbara, this has led to an updated compilation of estimated costs and parameters. Owing to the combined economic contribution of WO_3 and Sn, blocks have been assigned as ore where their combined revenue is greater than the total ore and G&A costs of $12.94/t.

Operating cost levels, used for this preliminary economic analysis, were derived for current cost level at Los Santos, as summarised in Table 21-1. Mining costs were derived from the current contractor costs at Los Santos, and then modified for the absence of drilling and blasting at Valtreixal, as well as for different rock densities.

Start-up capital costs were estimated by Daytal, and summarised in Table 21-2.

Table 21-1. Estimated Operating Costs

Description			Unit	Values
Processing				
	WO3	APT Price	$/mtu WO3	**370**
		Metal Price - received,	$/mtu WO3	**288**
		after transport + smelting	$/t WO3	28,800
		Recovery	%	**55%**
	Sn	Metal Price	$/lb	**10.5**
			$/t	23,149
		Sn % payment		89.21%
		Received, after transport + smelting	$/t Sn	20,651
		Recovery	%	**65.0%**
		Processing Cost	$/t ore	9.72
		G & A	$/t ore	3.25
Mining	**(No drill and blast required)**			
		Contractor Cost	$/bcm rock	2.55
		Indirect Cost	$/bcm rock	0.67
		Total Mining Cost	$/bcm rock	3.22
		Ore mining	$/t ore	1.26
		Waste mining	$/t waste	1.29
Mining Parameters				
		Mining Recovery	%	**95**
		Dilution	%	**5**
		Breakeven Economic Cut-Off	% WO3	0.08%
			% Sn	0.11%

Table 21-2. Estimated Capital Costs

	Category	Amounts US$ x 1000	Sub-total US$ x 1000
Investments Completed	Project purchase	4,088	
	Exploration	3,360	**7,448**
Infrastructure and Mill	Metallurgical testing	224	
	Feasibility Study	1,120	
	Permit application	280	
	Roads and access	280	
	Pit preparation	560	
	Road diversion	2,240	
	Civil works	1,680	
	Mill buildings	840	
	Electric line and transformers	336	
	Process equipment	16,800	
	Equipment installation	3,920	
	Coummunications	56	**28,336**
Mobile equipment	Mixed loader/backhoe	28	
	Telescopic lift	34	
	Fork-lift	28	
	Min scooptram	28	
	2 CAT 962 loaders	224	
	25 t truck	112	
	2 articulated trucks	280	**734**
Buildings	Offices	140	
	Laboratory	336	
	Workshop	224	
	Warehouse	140	
	Changehouse	140	
	Canteen	56	**1,036**
Vehicles	3 all-mine terrain	56	
	1 3, 500 kg truck	28	
	4 transport vehicles	140	**224**
Water handling	Pumps	112	
	Collecting ponds	280	
	Water treatment	168	
	Tanks	168	**728**
	Sub-total		**38,506**
	10% Contingency		**3,851**
	Total		**42,357**

22 ECONOMIC ANALYSIS

A life-of-mine (LOM) schedule was developed based on the pit design and reserves developed in this study. The estimated capital and operating costs summarised in Section 21 were then used to develop an economic cashflow model, as summarised in Table 22-1.

Important aspects of this economic cashflow include:

a) Year 0 is essentially a pre-strip period building up 52,000 tonnes of ore for mill commissioning.

b) The capital costs for mine start-up include costs estimated for the road, telephone and power line diversions.

c) The mine operating costs are based on the current mine contractor costs at Los Santos, but adjusted to reflect that the will be drilling and blasting at Valtreixal.

d) Processing and G&A costs are also similarly based on Los Santos.

e) The designed pit also contains 2.2 Mt of inferred material at ore grades, so the cashflows shown are likely to be conservative.

Table 22-1. Economic Cashflow Model

		Units	Total	Year 0	1	2	3	4	5
---- LIFE OF MINE ----									
Vol Waste		m3	8,477,958	1,663,858	2,342,299	1,446,059	1,394,605	1,087,772	543,365
Vol Ore		m3	991,977	19,735	192,593	197,096	198,160	193,409	190,983
Vol TOTAL		**m3**	**9,469,935**	1,683,594	2,534,893	1,643,154	1,592,765	1,281,181	734,348
Ton ORE		**t**	**2,559,459**	52,150	501,603	500,160	501,732	500,259	503,554
% WO3		%	0.25	0.25	0.30	0.26	0.22	0.21	0.27
% Sn		%	0.12	0.10	0.10	0.13	0.13	0.13	0.13
% WO3_EQ		**%**	**0.34**	0.32	0.37	0.36	0.32	0.30	0.37
STRIP RATIO		t/t	**8.2**	79.4	11.6	7.2	6.9	5.4	2.7
STRIP RATIO		m3/m3	8.5	84.3	12.2	7.3	7.0	5.6	2.8
---- PRODUCTS ----									
WO3	t		3,567	72	826	724	613	574	757
Sn	t		2,035	33	329	420	411	407	435
---- CASHFLOW ----									
Revenue	WO_3 Sales	US$ x 1000	102,720	2,086	23,792	20,854	17,656	16,523	21,808
	Sn Sales	US$ x 1000	42,027	678	6,799	8,679	8,482	8,407	8,983
	Total Revenue	US$ x 1000	144,748	2,765	30,591	29,532	26,138	24,930	30,791
Cash Costs	Mining	US$ x 1000	30,467	5,417	8,155	5,286	5,124	4,122	2,363
	Mill	US$ x 1000	24,870	507	4,874	4,860	4,875	4,861	4,893
	Admin	US$ x 1000	8,328	170	1,632	1,627	1,633	1,628	1,638
	Operating Cash Costs	US$ x 1000	63,665	6,093	14,662	11,774	11,632	10,611	8,894
Operating Margin		US$ x 1000	**81,082**	**-3,328**	**15,930**	**17,759**	**14,506**	**14,320**	**21,897**
Capital Costs	Investments completed	US$ x 1000	7,448	7,448					
	Infrastructure and Mill	US$ x 1000	28,336	28,336					
	Mobile equipment	US$ x 1000	734	734					
	Buildings	US$ x 1000	1,036	1,036					
	Vehicles	US$ x 1000	224	224					
	Water handling	US$ x 1000	728	728					
	Sub-total	US$ x 1000	38,506	38,506					
	Contingency 10%	US$ x 1000	3,851	3,851					
	Total Capital	**US$ x 1000**	**42,357**	42,357					
Cashflow		**US$ x 1000**	**38,726**	**-45,685**	**15,930**	**17,759**	**14,506**	**14,320**	**21,897**

Net Cash Flow		US$ x 1000	**38,726**
NPV	**6%**	US$ x 1000	**23,616**
NPV	**10%**	US$ x 1000	**16,135**
Internal Rate of Return			**24%**

23 ADJACENT PROPERTIES

There are no adjacent properties.

24 OTHER RELEVANT INFORMATION

Daytal plans to progress with further drilling, metallurgical test and other feasibility study work. As described previously, the 'Documento Inicial; has now been submitted to the local authorities, in order to obtain the necessary mining and environmental permits.

25 INTERPRETATION AND CONCLUSIONS

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

Table 25-1. Mineral Resources – In-Situ
As of End October 31st, 2015

CLASS	Tonnes	Sn	WO_3	WO_3_Eq
	Kt	%	%	%
Indicated	2,828	0.13	0.25	0.34
Inferred	15,419	0.12	0.08	0.17

Notes

. Cut-off applied of 0.05% WO_3_Eq

. WO_3_Eq = WO_3 + (Sn x 0.74), based on:

	Price	Recovery
WO_3	$37,000/t	55%
Sn	$23,150/t	65%

. Maximum extrapolation = 50m

. Density values estimated from measurements

. Resources shown are inclusive of reserves

Table 25-2. Mineral Reserves

Reserve Category	Tonnes	Sn	WO_3	WO_3_Equiv	Waste (including Inferred)	Rock	Strip
	Kt	%	%	%	Kt	Kt	Ratio
Probable Reserves	**2,549**	**0.12**	**0.25**	**0.34**	21,160	23,710	8.3

Notes

. WO_3_Eq = WO_3 + (Sn x 0.74), based on:

	Price	Recovery
WO_3	$37,000/t	55%
Sn	$23,150/t	65%

. Cut-off applied to WO3_Equiv

Breakeven Cut-off = 0.08 WO_3

. Mining factors applied:

Dilution = 5%

Losses = 5%

. Pit design also contains 2.2 Mt of inferred resources at economic grades

The following conclusions have been reached:

a) The Valtreixal project is a viable open project. An open pit has been designed with 2.5 Mt of ore, which suggest a 5 year mine life, based on a mill throughput of 500 Ktpa.

b) There are significant amounts of inferred resources, which suggest significant pit expansion both with depth and laterally along strike. Pit optimisations with inferred resources enabled suggest over 10 Mt of potential ore.

c) Exploration drilling completed by Daytal over the last 3 years have confirmed and extended the originally previously delineated resource base. In particular, the occurrence of scheelite mineralisation outside of quartz veins, has provided much wider mineralised zones than were previously interpreted.

d) The current open pit design is one coherent excavation. It appears that with more drilling to enhance the resource category of current inferred resources, the resultant pit elongation along strike will offer a very good opportunity for sequential pit extraction from west to east, with concurrent backfilling of excavated volumes with mined waste.

26 RECOMMENDATIONS

For further development of the project the next logical stage is a feasibility study, which includes:

1. A geotechnical study for analysis of pit slope angles.

2. A more detailed mine plan and production schedule.

3. A more detailed study of operating and capital costs.

4. A more detailed development of the milling operation and site infrastructure.

5. A more detailed plan associated with the road, telephone and power line diversions.

6. More drilling to clarify south-western extension of the deposit.

27 REFERENCES

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"Valtreixal S-W Mineralisation Heavy Liquid Tests and QEMSCAN Anaysis", by Wardell-Armstrong International, March 2012.

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"An Investigation by High Definition Mineralogy Into The Mineralogical Characteristics Of A W-Sn Composite Sample, Los Santos Mine Project, Spain", by SGS Canada, November 2013.

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"Documento Inicial – Proyecto Valtreixal", by Valtriexal Resources Spain S.L., September 2015.

28 QUALIFIED PERSONS CERTIFICATES

Certificate Of Author

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

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

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

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

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

5. I have worked as a mining engineer in the minerals industry for over 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. By reason of my education, experience and professional registration, I fulfil the requirements of a "qualified person" as defined by NI 43-101. My work experience includes 5 years at an underground gold mine, 7 years as a mining engineer in the development and application of mining and geological software, and 19 years as an independent mining consultant, involved with evaluation and planning projects for both open pit and underground mines.

7. I am responsible for the preparation of the technical report titled "Technical Report on the Mineral Resource and Reserves of the Valtreixal Project" and dated October 31st, 2015. I visited the mine site most recently on June 15th and 16th, 2015.

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

9. I am independent of Almonty Industries Inc., pursuant to section 1.5 of the Instrument.

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

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

Dated this 31st of October, 2015

AWheeler

A. Wheeler, C.Eng.