A PRELIMINARY ECONOMIC ASSESSMENT ON THE TABUAÇO TUNGSTEN PROJECT, PORTUGAL

Prepared For

Colt Resources Inc.

Report Prepared by

srk consulting

SRK Consulting (UK) Limited
UK5536

COPYRIGHT AND DISCLAIMER

Copyright (and any other applicable intellectual property rights) in this document and any accompanying data or models which are created by SRK Consulting (UK) Limited ("SRK") is reserved by SRK and is protected by international copyright and other laws. Copyright in any component parts of this document such as images is owned and reserved by the copyright owner so noted within the document.

This document may not be utilised or relied upon for any purpose other than that for which it is stated within and SRK shall not be liable for any loss or damage caused by such use or reliance. In the event that the recipient of this document wishes to use the content of this document in support of any purpose beyond or outside that which it is expressly stated or for the raising of any finance from a third party where the document is not being utilised in its full form for this purpose, the recipient shall, prior to such use, present a draft of any report or document produced by it that may incorporate any of the content of this document to SRK for review so that SRK may ensure that this is presented in a manner which accurately and reasonably reflects any results or conclusions produced by SRK.

The use of this document is strictly subject to terms licensed by SRK to its Client as the recipient of this document and unless otherwise agreed by SRK, this does not grant rights to any third party. This document shall only be distributed to any third party in full as provided by SRK and may not be reproduced or circulated in the public domain (in whole or in part) or in any edited, abridged or otherwise amended form unless expressly agreed in writing by SRK. Any other copyright owner's work may not be separated from this document, used or reproduced for any other purpose other than with the document in full as licensed by SRK. In the event that this document is disclosed or distributed to any third party, no such third party shall be entitled to place reliance upon any information, warranties or representations which may be contained within this document and the recipient of this document shall indemnify SRK against all and any claims, losses and costs which may be incurred by SRK relating to such third parties.

© SRK Consulting (UK) Limited 2013

SRK Legal Entity:		SRK Consulting (UK) Limited
SRK Address:		5th Floor Churchill House 17 Churchill Way City and County of Cardiff, CF10 2HH Wales, United Kingdom.
Date:		October, 2013
Project Number:		UK5536
SRK Project Director:	Mike Beare	Corporate Consultant (Mining Engineering)
SRK Project Manager:	Colleen MacDougall	Senior Consultant (Mining Engineering)
Client Legal Entity:		Colt Resources Inc.
Client Address:		2000 McGill College Avenue, Suite 2010, Montreal, Quebec, H3A 3H3. Canada.

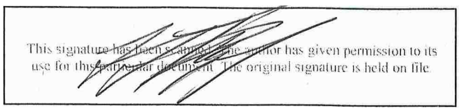
Report Title	A PRELIMINARY ECONOMIC ASSESSMENT ON THE TABUAÇO TUNGSTEN PROJECT, PORTUGAL
Effective Date:	1 October 2013
Signature Date	1 October 2013
Project Number:	UK5536
Contributors:	Gareth O'Donovan MSc CEng FIMMM FGS, Corporate Exploration Geologist Martin Pittuck MSc CEng MIMMM, Corporate Mining Geologist Jurgen Fuykschot MSc MAusIMM(CP), Principal Mining Engineer Kristina Huss, Eng, Consultant Mining Engineer John Willis BEng MAusIMM, Principal Mineral Processing Engineer Neil Marshall CEng MSc MIMMM, Corporate Geotechnical Engineer Kris Czajewski BSc, Principal Tailings Engineer Mark Raynor MSc, Principal Hydrogeologist Christopher Olofsson MSc, Consultant Hydrogeologist Emily Robinson MSc, Senior Environment Consultant
Reviewed by:	Mike Beare CEng MIMMM ACSM BEng
Qualified Person:	 Jurgen Fuykschot MSc MBA MAusIMM(CP) Principal Consultant (Mining Engineering)

Table of Contents: Executive Summary

List of Tables: Executive Summary

SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
City and County of Cardiff
CF10 2HH, Wales
United Kingdom
E-mail: enquiries@srk.co.uk
URL: www.srk.co.uk
Tel: + 44 (0) 2920 348 150
Fax: + 44 (0) 2920 348 199

A PRELIMINARY ECONOMIC ASSESSMENT ON THE TABUAÇO TUNGSTEN PROJECT, PORTUGAL – EXECUTIVE SUMMARY

1 INTRODUCTION

SRK Consulting (UK) Limited ("SRK") is an associate company of the international group holding company, SRK Consulting (Global) Limited (the "SRK Group"). SRK has been requested by Colt Resources ("Colt", hereinafter also referred to as the "Company" or the "Client") to undertake a Preliminary Economic Assessment ("PEA") on the Mineral Assets of the Company comprising the Tabuaço Tungsten Project ("Tabuaço") located in Portugal.

An economic analysis has been completed for six options:

- Option A1 (Base case): Production of a scheelite concentrate by flotation with dry tailings;
- Option A2: Production of a scheelite concentrate by flotation with wet tailings;
- Option A3: Use of ore sorting to reduce the plant feed tonnage, followed by flotation to produce a scheelite concentrate;
- Option B1: Production of a scheelite concentrate by flotation and leaching of concentrate using acid to produce WO_3 with dry tailings;
- Option B2: Production of a scheelite concentrate by flotation and leaching of concentrate using acid to produce WO_3 with wet tailings; and
- Option B3: Use of ore sorting to reduce the plant feed tonnage, followed by flotation to produce a scheelite concentrate and leaching of concentrate using acid to produce WO_3.

2 PROPERTY DESCRIPTION AND OWNERSHIP

The Tabuaço Tungsten Project (100% Colt Resources) is located within the Tabuaço Experimetnal Mining Licence ("EML") formerly known as the Armamar block of the Armamar-Meda exploration concession. The area is situated in the North Central region of Portugal, approximately 300 km north-north east of Lisbon, 100 km to the east-south east of Porto.

The Tabuaço project is located in a port wine growing area of varying topography, overlooking the banks of the Távora River. The Project area is divided into three deposit areas the São Pedro das Águias-Herédias resource area ("QSPA") and the Aveleira and Quintã exploration areas.

Colt has 100% mineral rights on the concession covering the Tabuaço project. The Tabuaço EML was been granted to Colt (February 20, 2013) covering the Sao Pedro Águias block from the former Armamar-Meda exploration license.

3 GEOLOGICAL SETTING AND MINERALISATION

The Tabuaço EML is situated at the border zone of two major geologic units in the Central Iberian Zone (“CIZ”) of northern Portugal. The region is noted for tungsten and tin occurrences and has seen a number of past artisanal workings.

Mineral deposits within the Tabuaço EML and surrounding area include skarn tungsten, tin & tungsten bearing veins and orogenic gold. The Tabuaço project is considered a skarn Tungsten deposit.

The Tabuaço geological model used for exploration is best described as a contact metamorphosed tungsten skarn model. An aplite/skarn interface is present with the mineralisation proximal to the granite boundary.

There are two main skarn horizons: upper and lower skarn, separated by schist. There are also numerous lenses or pods of tungsten bearing skarn material above and below the main horizons along with tungsten-bearing silicified lenses with carbonate.

4 EXPLORATION

A Mineral Resource estimate for the Tabuaço deposit was completed by Colt and reviewed by SRK in the previous NI 34-101 report from October 2012. SRK would anticipate that further exploration would both extend and upgrade the Mineral Resource estimate.

The previous NI 43-101 report recommended further drilling on the Aveleira and Quintã deposits to outline and upgrade the resources in these areas.

5 MINERAL PROCESSING AND METALLURGICAL TESTING

Three metallurgical programs have been conducted on samples from the Tabuaço deposit. The first two studies were conducted by Inspectorate Exploration and Mining Services Ltd (“Inspectorate”) in 2011 and 2012, the first on an outcrop sample and the second on drill core reject material from the Tabuaço property. The third study was undertaken by Guangzhou Research Institute of Non-Ferrous Metals (“GRINFM”) in 2012 under the auspices of Colt, using samples of diamond drill core.

The results of this series of tests demonstrated the following:

- Gravity separation only produced high concentrate grades at particle sizes of the order of 105 µm or less, however recoveries were low, with the best results being concentrates of the order of 50% WO_3 at WO_3 recoveries of the order of 50%;
- Flotation testwork conducted by Inspectorate resulted in relatively high recoveries, however final concentrate grades were quite low; and
- Flotation testwork conducted by GRINFM, who have particular expertise in tungsten processing, produced a relatively high grade final flotation concentrate (55.9% WO_3) at a relatively high WO_3 recovery. As this concentrate was high in P (3.4%), a final acid leach stage was employed, which resulted in a final concentrate grade of 70.6% WO_3, with a negligible loss of W in the acid leach stage.

On the basis of the GRINFM results, an overall flotation with concentrate leaching circuit was estimated to be capable of producing a concentrate assaying 70.6% WO_3 at an overall WO_3 recovery of 87.2%.

Ore sorting testwork has recently been commenced, with the aim of reducing the mass of material that needs to be ground ahead of flotation at a minimal tungsten loss. Colt has also commissioned an investigation into a hydrometallurgical process route using acid to produce tungsten oxide (WO_3) from the Tabuaço run of mine (“RoM”) material, or ore-sorting concentrates produced from the RoM material.

6 MINERAL RESOURCE ESTIMATE

A Mineral Resource estimate for the Tabuaço deposit was completed by Colt and reviewed by Martin Pittuck, Director and Corporate Consultant for Mining Geology with SRK, with input provided by Filipa Matias, Colt’s Geological Engineer. Mr Pittuck is considered a Qualified Person as defined in National Instrument 43-101. The effective date of this resource estimate was 3rd October, 2012.

The 3D geological model demonstrates that there are substantial parts of the deposit which present a continuous body of mineralisation above a cut-off grade (“CoG”) of 0.3% WO_3 and which have sufficient dimensions and grade to have reasonable prospects for eventual economic extraction according to the mining and processing studies carried out to date.

The Mineral Resource estimate for the Tabuaço deposit is shown in Table ES 1.

Table ES 1: Mineral Resource Estimation for the Tabuaço Project effective as the 3 October 2012 using a cut-off of 0.3%WO_3

Classification	Tonnage	Grade	Contained Metal	Contained Metal	Contained Metal
	kt	% WO_3	t WO_3	M lb WO_3	k MTU WO_3
Indicated Mineral Resources	1,495	0.55	8,150	18	815
Inferred Mineral Resources	1,230	0.59	7,200	16	720

1. Mineral Resources are not Mineral Reserves and there is no assurance that any, or all of the Mineral Resources will be converted to Mineral Reserves.
2. The tonnage, grade and contained metal values have been rounded to reflect the accuracy of the Mineral Resource Estimate. Numbers may not add due to rounding.
3. The Mineral Resources are stated above a cut-off grade of 0.3%WO_3, based on realistic mining and processing cost and recovery assumptions.

7 MINERAL RESERVE ESTIMATE

There are currently no Mineral Reserve estimates on the Tabuaço EML.

8 MINING METHOD

8.1 MINING

An underground mining method has been selected for the Tabuaço project due to significant surface constraints formed by environmental and cultural heritage restrictions. Given the depth, poor ground conditions and geometry of the deposits, a drift and fill mining method has been selected for the purposes of this PEA. Drift and fill has the added advantage of providing a high degree of ore selectivity, maximising ore recovery and grade compared to bulk mining methods. This method is also compatible with the variable ground conditions expected during production.

Based on a scoping level of study the following conclusions can be made:

- With the current resource, the mine life at a 0.22% WO_3 CoG is one year of pre-production followed by ten years of production;
- The average RoM grade over the mine life is 0.39% WO_3;

- The average daily ore production rate for the life of mine is 1,415 tpd; and
- The peak ore production rate is in year 9, at 1,492 tpd.

The following recommendations should be undertaken in the next phase of the project:

- Review mining operating costs, especially the development costs
- Review mine infrastructure designs in order to optimise access to material above the CoG;
- Review equipment productivities to ensure that the mining rate is achievable given the access and deposit geometry
- Review the ventilation concept in order to ensure adequate air flow throughout the mine; and
- Evaluate various backfill systems to optimise the proposed mining method.

8.2 GEOTECHNICAL ASSESSMENT

The engineering geological assessment is based on a review of core photographs and basic geotechnical logs provided by the Client, which contains Rock Quality Designation ("RQD") and core recovery information. The rock mass is dominated by four main lithologies: carbonates, skarn, schists and granites.

The following conclusions can be made from the work carried out for the PEA:

- Materials are within the very poor to poor range of the Q graph. These represent the average value and some areas will be significantly lower;
- Shotcrete and bolting should be in accordance with Barton's Category 4 - "Bolting + shotcrete", and should be considered in all capital development areas as a minimum, until greater confidence in the material can be determined;
- The oredrives should be shotcreted with spot bolting where required; and
- Changing from temporary to permanent environments increases the support requirements.

The following recommendations should be undertaken in the next phase of the project:

- It is recommended that specific triple tube geotechnical boreholes are drilled and logged to ascertain the true rock mass properties opposed to doubling resource drilling where core management was secondary to meterage. Logging Q parameters would enhance the confidence in the database leading to a greater understand of the rock mass stability. This will provide greater confidence in the material properties and has the potential to improve the Q support requirements;
- Relocating the underground mine infrastructure in the granite hanging wall could enhance the excavations stability through improved rock mass properties and increased RQD. This has not been proven in this study as there are insufficient boreholes that intercept the granite at depth in the hanging wall; and
- Increased analysis needs to be undertaken into the material properties to zone the underground workings spatially.

8.3 HYDROLOGICAL AND HYDROGEOLOGICAL RELATED ISSUES

Based on a scoping level of study the following recommendations should be undertaken. A hydrogeological field program is required to:

- Investigate surface water diversions depending on mine water balance and choice of Tailings Management Facility ("TMF");
- Set up a permanent baseline monitoring program;

- Provide data to support the dewatering system design;
- Characterise groundwater conditions at the tailings management facility and waste rock dump locations; and
- Provide data to support rock mechanical assessment.

It is understood that this work is part of the existing hydrogeological study but this should be confirmed, and the existing study extended as necessary.

Following completion of the field campaigns it will be necessary to:

- Design closure plans for the Environmental Impact Assessment ("EIA");
- Review the conceptual design of the TMF and Waste Rock Dump ("WRD") taking account of seepage and runoff management requirements;
- Construct a Site Wide Water Balance Model to evaluate the need for makeup water and mine water discharges;
- Develop a site wide water management plan;
- Construct numerical groundwater and geochemical models to assess inflow rates to the underground mine and mine site seepage post closure, if geochemical testwork indicates the risk to be elevated; and
- Confirm environmental impacts and permitting limits for river abstraction and mine water discharge.

9 RECOVERY METHODS

Four processing options have been presented by Colt for consideration for the PEA:

- Option A1/A2: Production of a scheelite concentrate by flotation;
- Option A3: Use of ore sorting to reduce the plant feed tonnage, followed by flotation to produce a scheelite concentrate;
- Option B1/B2: Production of a scheelite concentrate by flotation and leaching of the concentrate using acid to produce WO_3; and
- Option B3: Use of ore sorting to reduce the plant feed tonnage, followed by production of a scheelite concentrate by flotation and leaching of the concentrate using acid to produce WO_3.

For Options B1, B2 and B3 Colt proposes to use an acid based processing path to produce high purity tungsten oxide end product. The process path is estimated to be less capital and operating cost intensive than the traditional ammonium paratungstate ("APT") processing path.

SRK believes that a significant program of further testwork is required in order to verify the technical feasibility of the proposed process options, and then to confirm the recovery figures assumed for this PEA.

10 PROJECT INFRASTRUCTURE

10.1 TAILINGS

The results of the previous work completed by the Client indicating that the best area for the Tailings Storage Facility ("TSF") would be in the valley of Passa Frio (river) east of the proposed plant location. SRK followed this selection as most appropriate at this stage, however because of difficult terrain and high volumes of dam construction SRK also evaluated a dry stacking concept as viable alternative. The TSF has been based on the base

Case processing option. The TSF will be a valley type of facility enclosed by the main embankment.

For wet tailings it has been assumed that the tailings will be stored sub-aerially behind the rock fill dam with an upstream liner.

The concept of dry disposal has been developed with mine waste rock to be co-disposed with the filter pressed tailings. The starter wall embankment is essentially reduced in size at the front of the disposal area and the mine waste rock can be sequentially dumped with the tailings.

Additional design and investigative work will be required for the next stage of the project. The design of the TSF or the co-storage impoundment will require various studies for the following fields:

- Tailings alternative assessment to include the dry tailings disposal option;
- Groundwater condition at and near the tailings impoundment and dam;
- Geochemical properties of the tailings and the leachate originating from the tailings;
- Geotechnical conditions of the dam foundations and properties of the tailings;
- Borrow source investigation; and
- Engineering.

11 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

A review of current environmental and social documents was undertaken to determine the level of studies completed and provide recommendations for future work.

The following conclusions can be made from the review carried out for the PEA:

- Colt has an experimental mining licence for the Tabuaço project, which covers operations up to five ha in size and/or less than 150,000 ktpa of RoM;
- In accordance with Mining Law 88/90 of 16 March 1990, an application for a full mining licence requires the submission of a Mine Feasibility Study that includes an Environmental Impact Assessment (“EIA”);
- An environmental scoping report was issued to Agencia Portuguesa de Ambiente (Portuguese Environmental Agency) (“APA”) in August 2012. Colt intends to initiate the baseline phase of the EIA in January 2014, although a hydrogeological baseline study has already commenced; and
- Colt intends to obtain the required permits by 2015.

The following recommendations can be made from the review carried out for the PEA:

- The project should plan for appropriate characterisation of waste rock, tailings material and seepage quality in accordance with requirements of Decree Law 10/2010;
- As the project is located within a buffer zone of a United Nations Education, Scientific and Cultural Organisation (“UNESCO”) world heritage site the APA has requested that impacts on cultural heritage should be addressed following the methodology provided in the Guidelines of the International Council on Monuments and Sites;
- Water quality prediction via geochemical modelling should be conducted to confirm the potential impacts of mine water discharge;

- Proactive stakeholder engagement should be planned and implemented in a structured way to ensure that key stakeholder issues are identified and are addressed by the EIA and project design; and
- The EIA schedule should allow for sufficient baseline data to be included in the EIA report and time to respond to any concerns raised by stakeholders during the EIA process. The scope of the baseline studies should also include the recommendations provided by the APA to avoid delays to the permitting process.

12 CAPITAL AND OPERATING COSTS

A summary of the total capital and operating costs for the four options is shown in Table ES 2. The main findings from the capital and operating costs are summarised below:

- The processing costs show the most variation between the options (Options A3 and B3 have the lowest costs);
- The tailings costs show that the wet tailings options (A2 and B2) are higher than the dry tailings options (A1 and B1);
- Option B3 has the lowest operating costs, while Option A1 has the highest; and
- Option A3 has the lowest capital requirements, while Option B2 has the highest.

Table ES 2: Summary of Capital and Operating Costs

	Units	A1	A2	A3	B1	B2	B3
Operating Expenditure							
Mining	USDm	150.4	150.4	150.4	150.4	150.4	150.4
Mine Services	USDm	3.8	3.8	3.8	3.8	3.8	3.8
Ore Sorting	USDm	-	-	7.1	-	-	7.1
Reject handling	USDm	-	-	4.0	-	-	4.0
Crushing	USDm	5.3	5.3	5.3	5.3	5.3	5.3
Processing	USDm	83.3	83.3	40.5	66.7	66.7	32.4
Ore Transport	USDm	13.9	13.9	13.9	13.9	13.9	13.9
Backfill	USDm	19.6	19.6	19.6	19.6	19.6	19.6
Tailings	USDm	11.1	1.4	0.3	11.1	1.4	0.3
Dewatering	USDm	1.1	1.1	1.1	1.1	1.1	1.1
G&A	USDm	13.9	13.9	13.9	13.9	13.9	13.9
Total Operating Expenditure	**USDm**	**302.3**	**292.6**	**259.8**	**285.7**	**276.0**	**251.7**
Capital Expenditure							
Mining	USDm	23.2	23.2	23.2	23.2	23.2	23.2
Plant	USDm	44.2	44.2	26.5	45.0	45.0	27.0
Paste Backfill Plant	USDm	7.0	7.0	7.0	7.0	7.0	7.0
Tailings - Construction	USDm	12.2	47.2	0.3	12.2	47.2	0.3
Tailings - Closure	USDm	1.0	1.3	-	1.0	1.3	-
Waste Management	USDm	0.5	0.5	1.5	0.5	0.5	1.5
Ore Sorter	USDm	-	-	2.0	-	-	2.0
Acid Plant	USDm	-	-	-	8.0	8.0	8.0
Mine Closure	USDm	5.0	5.0	5.0	5.0	5.0	5.0
Drilling/Eng. Studies	USDm	7.2	7.2	7.2	7.2	7.2	7.2
Total Capital Expenditure	**USDm**	**100.3**	**135.6**	**72.7**	**109.1**	**144.3**	**81.2**

13 ECONOMIC ANALYSIS

A discounted cashflow ("DCF") model has been developed based on the production schedules and costs discussed. A base price of USD 400/MTU WO_3 is applied, however a 9% price deduction has been applied to options A1, A2, and A3, as the product from these Options ($CaWO_4$) is different from the product from Options B1, B2 and B3 (WO_3) and will hence not realise as high a price on a WO_3 basis as the product from these latter options. A 1% royalty rate on revenue has been applied in SRK's economic model.

The corporate income tax has been assessed using a 25% state and 2.5% municipal tax rate. No contingencies have been applied in the financial model over the estimated operating and capital expenditures.

The summary results from each production scenario are shown in Table ES 3. The start date of the model is 1 September 2013, and it is in real 2013 money terms.

Table ES 3: Project Scenario Comparison

Economic Model Summary	Units	A1	A2	A3	B1	B2	B3
Production							
Ore Processed	(kt)	3,557	3,557	3,557	3,557	3,557	3,557
	(% WO_3)	0.39	0.39	0.39	0.39	0.39	0.39
	(kt WO_3)	13.7	13.7	13.7	13.7	13.7	13.7
Recovered Metal	(MTU WO_3)	1,195,807	1,195,807	1,136,016	1,305,766	1,305,766	1,240,478
Financial							
Metal Price	(USD/MTU WO_3)	364.0	364.0	364.0	400.0	400.0	400.0
Revenue	(USDm)	435.3	435.3	413.5	522.3	522.3	496.2
Operating Expenditure	(USDm)	302.3	292.6	259.8	285.7	276.0	251.7
Royalty	(USDm)	4.4	4.4	4.1	5.2	5.2	5.0
Operating Profit	(USDm)	128.6	138.3	149.6	231.4	241.1	239.6
Net Profit	(USDm)	119.4	136.1	127.2	196.4	213.1	194.7
Capital Expenditure	(USDm)	100.3	135.6	72.7	109.1	144.3	81.2
Cashflow	(USDm)	19.1	0.6	54.5	87.3	68.8	113.6
Pre-Tax Reporting							
NPV @ 5%	(USDm)	7.1	-12.8	45.3	71.1	51.1	100.6
NPV @ 10%	(USDm)	-5.5	-21.4	25.6	39.2	23.3	64.1
NPV @ 15%	(USDm)	-13.1	-25.9	13.0	19.0	6.1	40.5
IRR	(%)	7.5	0.7	24.3	23.3	17.5	37.5
Post-Tax Reporting							
NPV @ 5%	(USDm)	1.6	-14.0	30.5	48.0	33.1	70.3
NPV @ 10%	(USDm)	-8.9	-22.1	15.4	23.6	11.3	43.0
NPV @ 15%	(USDm)	-15.2	-26.4	5.8	8.1	-2.1	25.3
IRR	(%)	5.6	0.1	19.6	18.9	14.1	30.7
Cash Cost							
Cash Cost	(USD/t ore)	86.2	83.5	74.2	81.8	79.1	72.2
	(USD/MTU WO_3)	256.5	248.3	232.3	222.8	215.3	206.9

SRK notes that the economic assessment is preliminary in nature and the production schedules are inclusive of material classified as Inferred Mineral Resources. Inferred Mineral Resources are considered too geologically speculative to have economic considerations applied to them that would enable them to be classified as Mineral Reserves. There is no certainty that the preliminary economic assessment will be realised.

Table of Contents

List of Tables

List of Figures

List of Technical Appendices

SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
City and County of Cardiff
CF10 2HH, Wales
United Kingdom
E-mail: enquiries@srk.co.uk
URL: www.srk.co.uk
Tel: + 44 (0) 2920 348 150
Fax: + 44 (0) 2920 348 199

A PRELIMINARY ECONOMIC ASSESSMENT ON THE TABUAÇO TUNGSTEN PROJECT, PORTUGAL

1 INTRODUCTION

1.1 Background

SRK Consulting (UK) Limited ("SRK") is an associate company of the international group holding company, SRK Consulting (Global) Limited (the "SRK Group"). SRK has been requested by Colt Resources ("Colt", hereinafter also referred to as the "Company" or the "Client") to undertake a Preliminary Economic Assessment ("PEA") on the Mineral Assets of the Company comprising the Tabuaço Tungsten Project ("Tabuaço") located in Portugal.

1.2 Terms of Reference

The terms of reference for the proposed work were discussed during a series of telephone conversations between the company representative Declan Costelloe and Mike Beare of SRK. SRK understand the primary requirement is to provide a PEA based on the resource 43-101 prepared by SRK Exploration Services ("SRK ES") in November 2012.

1.3 Scope of Work

The scope of work has taken the form of a review of the current Colt technical work available on the project. In addition SRK has augmented this by authoring, as required, a scoping study in accordance with the definition established under the 43-101 guidelines for a PEA.

An economic analysis has been completed for six options:

- Option A1 (Base case): Production of a scheelite concentrate by flotation with dry tailings;
- Option A2: Production of a scheelite concentrate by flotation with wet tailings;
- Option A3: Use of ore sorting to reduce the plant feed tonnage, followed by flotation to produce a scheelite concentrate;
- Option B1: Production of a scheelite concentrate by flotation and leaching of concentrate using acid to produce WO_3 with dry tailings;
- Option B2: Production of a scheelite concentrate by flotation and leaching of concentrate using acid to produce WO_3 with wet tailings; and
- Option B3: Use of ore sorting to reduce the plant feed tonnage, followed by flotation to produce a scheelite concentrate and leaching of concentrate using acid to produce WO_3.

The PEA has been prepared to cover the following technical disciplines to a scoping level:

- Geotechnical: SRK has reviewed the current status of the project and made appropriate recommendations for the PEA geotechnical parameters;
- Hydrogeological: SRK has reviewed the current status of the project and made appropriate recommendations for the PEA mine design, water supply and site wide water balance;
-

- Mining: SRK has received and adjusted a preliminary underground mine design from Colt and created a high level schedule to provide input to the economic analysis. Capital and operating costs have been estimated to a scoping level based on data provided by Colt and adjusted using SRK's knowledge of similar projects;
- Processing: SRK has reviewed the process testwork undertaken to date on the project and the scoping level process flowsheet. Capital and operating costs have been estimated to a scoping level based on data provided by Colt and adjusted using SRK's knowledge of similar projects;
- Geochemistry: SRK has reviewed the project at a high level and made recommendations for addressing any expected geochemical issues;
- Tailings: SRK has reviewed the options for tailings storage and made appropriate recommendations. Capital and operating costs have been estimated to a scoping level from SRK's knowledge of similar projects;
- Environmental: SRK has reviewed the project at a high level and made recommendations for addressing any expected environmental and social issues; and
- Financial Evaluation: SRK has prepared a scoping level financial model in real terms based the mine plan developed for the PEA and using all the inputs from the various technical disciplines involved in the study.

1.4 Qualifications of Consultants

The Consultants preparing this technical report are specialists in the fields of geology, exploration, mineral resource and mineral reserve estimation and classification, mining, geotechnical, environmental, permitting, metallurgical testing, mineral processing, processing design, capital and operating cost estimation, and mineral economics.

None of the Consultants or any associates employed in the preparation of this report has any beneficial interest in Colt. The Consultants are not insiders, associates, or affiliates of Colt. The results of this Technical Report are not dependent upon any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings between Colt and the Consultants. The Consultants are being paid a fee for their work in accordance with normal professional consulting practice.

1.5 Details of Inspection

Jurgen Fuykschot from SRK conducted a site visit to Tabuaço during the period 6 to 8 May 2013. During the site visit, the SRK Qualified Person ("QP") conducted a tour of the concession area and existing infrastructure.

The QP visited the proposed locations for the portals, ventilation raises, mine surface infrastructure, processing plant, tailings and waste rock containment facilities. The QP also visited the 12th century convent which poses a surface constraint to mining.

2 RELIANCE ON OTHER EXPERTS

2.1 Mineral Resources

A Mineral Resource estimate for the Tabuaço deposit was completed by Colt and reviewed by Martin Pittuck, Director and Corporate Consultant for Mining Geology with SRK, with input provided by Filipa Matias, Colt's Geological Engineer. Mr Pittuck is considered a Qualified Person as defined in National Instrument 43-101. The effective date of this resource estimate was 3rd October, 2012.

2.2 Mine Development

At the request of Colt, an underground development design based on the mineralisation wireframes was created by Fernando Real and Diogo Caupers, independent engineering consultants. This design was used by SRK as basis for an updated development design accessing the mining areas above the cut-off grade and associated schedule. This redesign has resulted in a reduction of capital requirements.

3 PROPERTY DESCRIPTION AND LOCATION

3.1 Property Description and Location

The Tabuaço EML is situated in the North Central region of Portugal, approximately 300 km north north-east of Lisbon, 100 km to the east south-east of Porto and only 25 km to the south-east from Peso da Régua, a local regional centre. There are other small towns in the vicinity, including Armamar, Moimenta da Beira, Penedono, S. João da Pesqueira and Tabuaço.

Viseu is the main population and district administrative centre, with several smaller towns and villages in the vicinity of the property.

The Tabuaço EML license comprises the previous São Pedro das Aguias block (Latitude: 41° 5'5.27"N, Longitude: 7°30'55.69"W), which contains the Tabuaço Project.

The Tabuaço project is located approximately 3 km south of the village of Távora within the district of Viseu, and the Tabuaço municipality. It is located along the western flank of the Távora River, a tributary of the Douro River, the main river in the North of Portugal (Figure 3-1). The project has also historically been referred to as the "S. Pedro das Águias" skarn deposit.

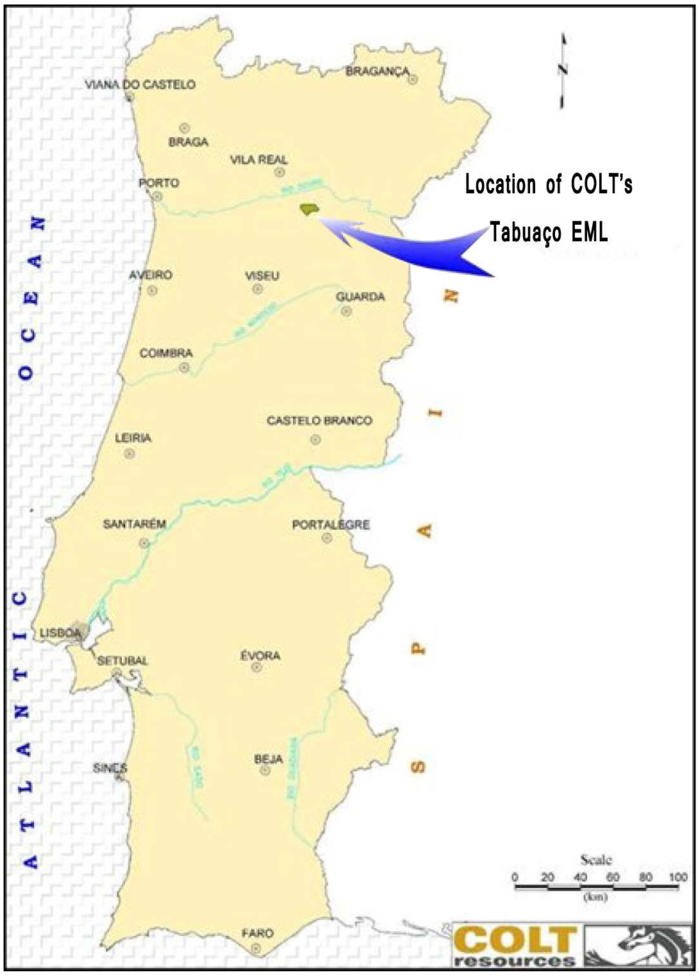


Figure 3-1: Location of the Tabuaço EML in Portugal

3.2 Mineral Titles

The Tabuaço EML has been granted to Colt (February 20, 2013) covering the former Sao Pedro Águias block which has a total area of 45.13 km^2. The EML follows on from the exploration work undertaken by Colt at the Armamar Meda exploration concession between 2007 and 2012. The Armamar Meda exploration licence comprised two distinct areas: Sao Pedro das Águias block (4,513 hectares) and the Meda block (6,407 hectares).

The coordinates of the licence areas are stated in Table 3-1; the coordinates have been converted from Hayford-Gauss (Datum 73) to UTM29T (Datum ED50). Figure 3.2 shows the coordinates for the corner points in ED50 projection.

Table 3-1: Co-ordinates of Licence Area Vertices Tabuaço EML

Point	ED 50		HG73	
	North	East	North	East
1	4,551,909.8	619,025.9	160,000	46,135
2	4,551,993.2	627,398.5	160,000	54,510
3	4,549,203.4	629,873.8	157,185	56,958
4	4,548,008.4	628,886.0	156,000	55,959
5	4,547,969.4	624,929.0	156,000	52,000
6	4,545,439.2	622,448.7	153,494	49,494
7	4,549,954.4	618,440.5	158,050	45,530

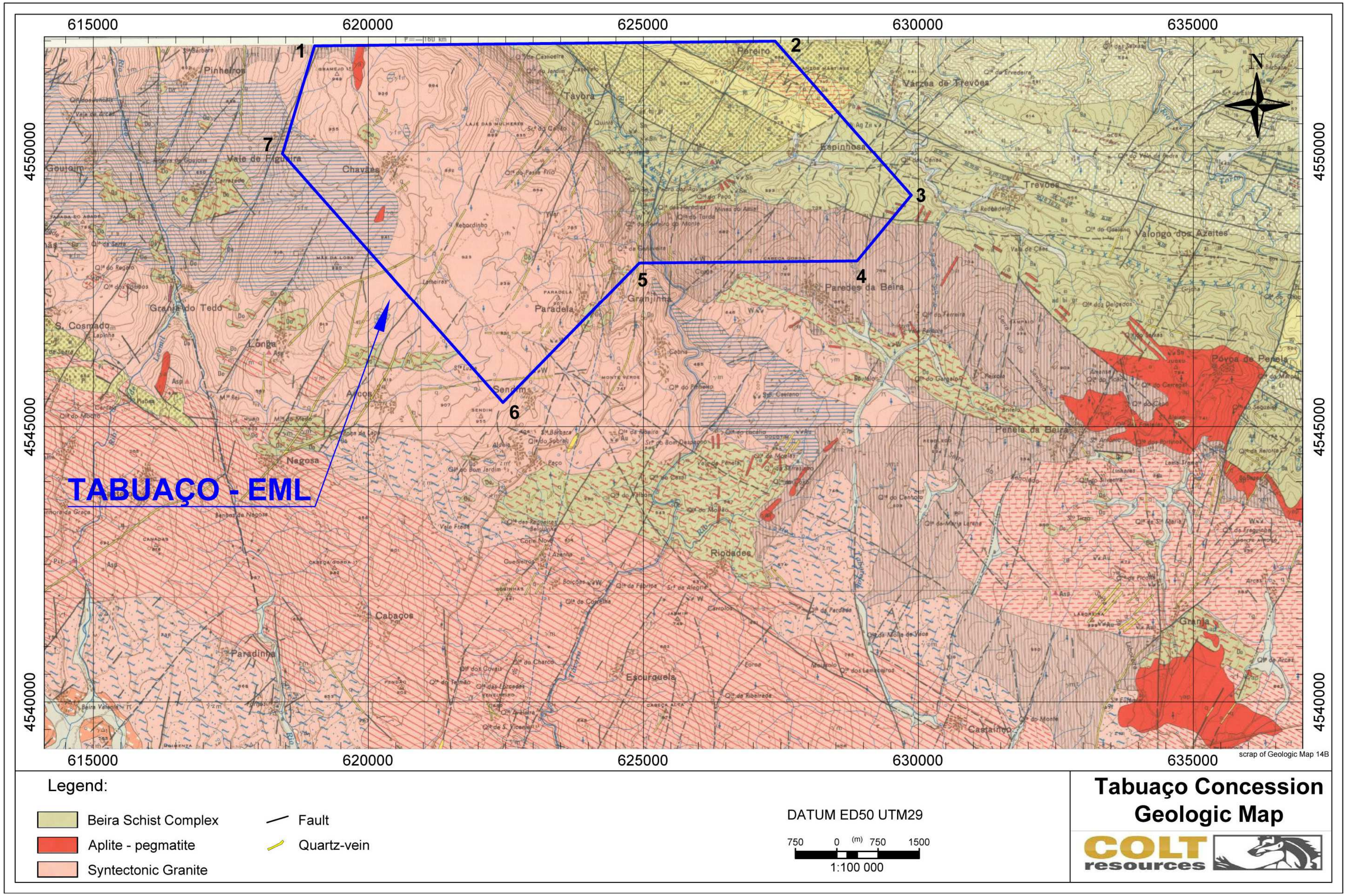


Figure 3-2: Tabuaço EML Boundary with Coordinates in ED50 Datum

3.2.1 Nature and Extent of Issuer's Interest

The Tabuaco EML contract, covering 45.23 km^2, was signed on the 20th February 2013 and is valid for a period of four years, and can be extended for a single one year period without any surface area relinquishment. Therefore the EML can remain valid up to the 19th February 2018.

The expenditure commitment required the whole four year period is EUR 4.5m.

3.3 Royalties, Agreements and Encumbrances

The standard corporate tax rate in Portugal is 25%, and an additional municipality tax is also imposed. This municipality tax is variable amongst municipalities and for Lisbon the total corporate tax rate is currently 26.5%.

Upon the granting of a Mining License, and in the event that mining activities are to take place, then the Company shall be obligated, at the Portuguese Government's sole discretion, either to pay:

- 10% of the net profit derived from its mining activities;

or, alternatively, pay a:

- Net Smelter Return up to 4% depending on the price per mtu:
  - 0% up to USD 300/mtu,
  - 1% from USD 301 to 400/mtu,
  - 2% from USD 401 to 450/mtu,
  - 3% from USD 451 to 500/mtu,
  - 4% above USD500/mtu,
- The Direcção Geral de Energia a Geologia (General Directorate for Energy and Geology "DGEG") may request a payment in kind up to 5% of the total payment due, which part of could be spent on local programmes.

On granting of an exploitation license the Company will be obligated to pay EUR 50,000 as a commercial discovery bonus to the municipalities of Tabuaço and S João da Pesqueira, on a pro rata basis, depending on the surface area of the mining license included on each municipality.

3.4 Environmental Liabilities and Permitting

Colt has been awarded the Tabuaço EML (February 20, 2013), which permits mining operations up to five hectares and/or less than 150,000 tonnes (RoM) per annum.

In accordance with Artº 14º-2-f Mining Law 88/90 of 16 March 1990, an application for a full mining licence requires the submission of a Feasibility Study that includes an EIA and a mine plan. The EIA which details the mining operations requires approval from the APA.

3.5 Mining License

Application for a full mining license may be made at any time during the 4 year term of the license or during the 1 year extension period at the end of the 4 year EML by February, 2018. The following comments describe the criteria required to achieve a full mining license in Portugal.

The Mining License ("ML") is awarded upon approval by both the Direcção-Geral de Energia e Geologia (DGEG) and the APA of a detailed Mine Feasibility Study ("FS") that includes an EIA.

Once the ML is granted, a project plan will be prepared for the main physical structures (mine, plant, Tailings Management Facility ("TMF"), support buildings etc.) and subsequently approved by the relevant authorities. The EIA also has to be approved as part of the mine plan, which requires the following:

- Introduction: Company ID, general description of the project, main objectives, production capacity of the mine, short description of infrastructure i.e. access roads, railways, rivers, power lines, etc.;
- "Ore" deposit: To include detailed topographic and geological maps, cross and longitudinal sections of the deposit clearly showing the main physical characteristics including the grade variations;
- Mining method: To include the relevant details of the mining method, mine schedule and the general blasting diagram;
- Other facilities: Description of all facilities including the offices, warehouses, material storage sites, loading areas, waste dumps, tailings dam and fresh water dams, drainage, power and water supply systems including a specific closure and contingency plan;
- Concentration process: Location, description of the mineralisation, flow sheet, water and energy circuits, metallurgic balance, equipment, etc.
- Health and safety and closure plan: Details the evolution of the mining activities and landscape reclamation from the pre mining natural state until the closure of the mine. This plan will be used to calculate the bank guarantee needed;
- Environment Impact Study: To include a complete and comprehensive review of how all aspects of the mining activity will impact on the environment; and
- Land access: To include documentation which proves that the company has access (acquisition/lease or rental) to the land included in the mining area.

3.6 Other Significant Factors and Risks

3.6.1 Compliance Evaluation

The Tabuaço Project is located within an area of extensive port wine vineyards. Port wine making is an important economic activity for the local communities and they are sensitive to environmental issues. Colt is currently the land owner over the majority of the QSPA deposit area and thus far all activities have been conducted in an environmentally conscientious manner.

Issues regarding land ownership on the area designated for the process plant and tailings management facility at Passa Frio are being addressed by Colt. There is a risk that consensus will not be reached with some landowners.

While the local communities have not as yet shown any resistance to the possibility of a mine at Tabuaço, Colt will need to exercise all due care to ensure that this remains the case and that the communities remain supportive.

3.6.2 Surface Constraints

A 12th century convent is located near the outcrops of the mineralisation zones, near the winery. According to Colt, a minimum offset of 60 m is legally required to preserve the building. SRK recommends that vibration monitoring is undertaken when experimental mining is taking place to determine the appropriateness of the 60 m offset and if a greater distance is required to prevent damage to the building.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

4.1 Topography, Elevation and Vegetation

The Tabuaço project area overlooks the banks of the Távora River and the topography of the licence area ranges from undulating rolling hills in the south-west to steep peaks and corresponding deep v-shaped valleys in the central, eastern and north eastern areas surrounding the project (Figure 4-1); elevations range from +500 m down to +225 m above mean sea level. The vegetation of the project area is a mix of terraced grape vines and orchards, small farming areas, wooded areas and fallow ground.

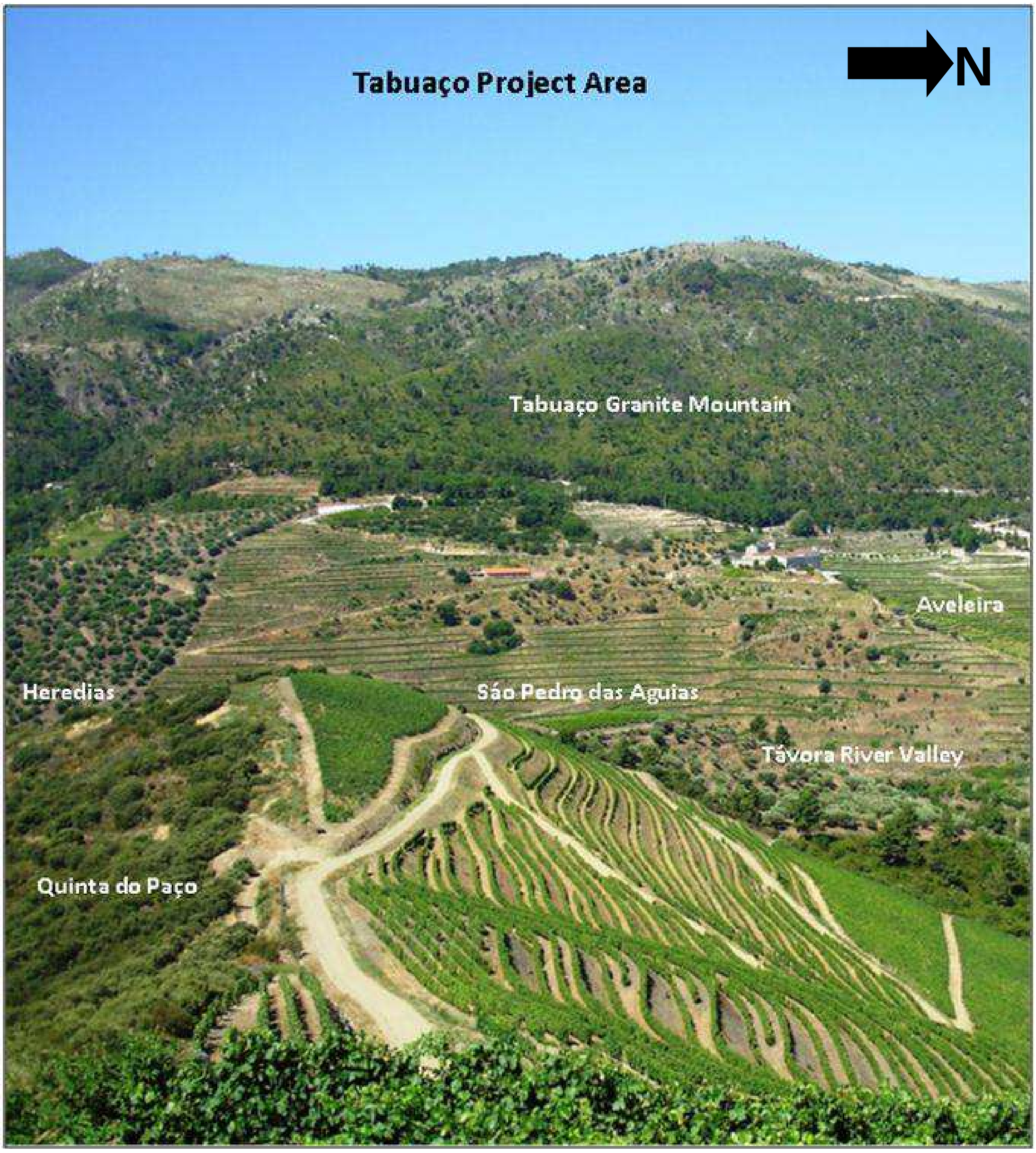


Figure 4-1: View of the Tabuaço Project Looking West (circa 2009)

4.2 Accessibility

The Tabuaco EML area is immediately accessible through a good network of local roads and is bisected north to south by the national highway N323. The east-west highway N222 is also located 12 km to the north of the project area. The project is located at a distance from the modernised major national highway system but it does have good access to the main north-south road networks, in particular the IP2/N102 highway to the east and A24/IP3 highway to the west. The nearest population centre is the village of Quintã, approximately 1 km north with a population of approximately 100 people.

4.3 Climate

The climate of Portugal is described as typically Mediterranean, consisting of dry hot summers and temperate wet winters. Additional climate detail is available in the November 2012 NI 43-101 Report.

The climate of the Tabuaço project area is distinct from its immediate surrounds, due to the steep valley setting producing a microclimate effect. It can be described best as temperate-humid with a mean temperature of +15℃ and annual rainfall of approximately 20 cm; the majority of which usually falls during November through to March. Summer temperatures can reach up to 45℃ and are usually above 30℃ between June and September, with brisk winds to the elevated project areas. Winters in the project area are relatively mild, with infrequent snowfalls on the higher peaks of the region which usually dissipate within a few days.

The lack of any distinct period of high rain or snowfall, together with good site access, means the operating season for exploration and development can therefore suitably be described as year-round.

4.4 Sufficiency of Surface Rights

Colt is the owner of 100% of the surface rights over the Tabuaço project area and is on good terms with the majority of the local landowners. At this stage the surface rights are deemed sufficient for the project's immediate requirements (experimental mining). Colt is currently in negotiations with other land owners regarding acquisition of further surface rights in areas that may be required for development purposes namely at the site for the proposed main portal, process, tailings and waste storage facilities.

4.5 Infrastructure

The project area is well serviced by the national grid, with a high-voltage 60 kVa power line passing immediately adjacent to the project along the N323 highway between Granjinha and Quintã. The entity responsible for the power infrastructure and main supplier of both base load and renewable power in Portugal is, respectively Rede Eléctrica Nacional ("REN") and Energias de Portugal ("EDP"), formally known as Electricidade de Portugal.

Supply and regulation of the drinking water and sewerage is controlled by the Water and Sanitation Regulation Agency Entidade Reguladora de Águas e Resíduos ("ERSAR"). Service provision is shared between the local municipalities and the national water company, Águas de Portugal ("AdP").

Potable water is available from the local municipal system and the Távora river passes through the project licence area, making it an option for sourcing larger volumes of water subject to permitting. A further option is pumping water from a number of existing reservoirs that are located close to the project area.

There are several populated towns in close proximity to the operation. While most of the manual labour force will come from these near-by locations the majority of the higher skilled labour force will need to be sourced elsewhere in Portugal.

Colt has identified several areas in close proximity to Tabuaço that would be potentially suitable for locating process facilities, tailings and waste disposal. The Client is involved in discussions designed to acquire one of these areas in which all this specified infrastructure could be located.

5 HISTORY

5.1 Mining History

During the 1970's, geologists from the Serviço de Fomento Mineiro ("SFM") Mines Department discovered scheelite bearing skarns along the flanks of the Távora River, through geological mapping and mineral-light exploration, using a short-wave ultra violet ("UV") lamp. This work was first documented in 1980 by Sousa, Ramos & Viegas. From 1980 to 1982, a joint venture between the Portuguese Sociedade Portugesa de Empreedimentos ("SPE") and the French Société d'Études de Recherches et d'Exploitations Minières ("SEREM"), the 100% owned exploration arm of the Bureau de Recherches Géologiques et Minières ("BRGM") explored the area which now includes the Tabuaço Project.

There is no record or evidence of any past mining activity to exploit scheelite mineralisation at Tabuaço.

5.2 Prior Ownership and Ownership Changes

The entire Armamar-Meda Licence was once included in a larger exploration licence, "Alto Douro" exploration concession (Sn-W, Au, etc.), held by the SPE-BRGM consortium in period of 1979-1984.

Part of the Armamar-Meda Licence was once included in Greystar Resources Inc.'s Penedono exploration concession (Au) during the period of 1995-1997 and part of the Armamar-Meda Licence was included in Rio Narcea Gold Mines' first Penedono exploration concession (Au) during the period of 1999-2004.

5.3 Previous Exploration and Development Results

Tungsten skarns were first discovered at the Tabuaço project area by Government geologists in the 1970's during geological fieldwork and UV light prospecting. Further exploration of the Tabuaço project area was completed in the 1980's by a consortium of the Portuguese companies (SPE) and the BRGM (through subsidiary SEREM) with work focused on São Pedro das Águias, Quintã das Herédias, Quintã and Quintã do Paço.

A copy of the detailed exploration data was obtained directly from SPE's project geologist Paulo Alves. This was compiled into a report by Filipe Faria of GeoLog on behalf of Colt.

The initial phase of the SPE-SEREM exploration work conducted during 1980 to 1981 focused on regional mapping, UV lamp prospecting, grab sampling, channel rock sampling and stream sediment and soil sampling programmes.

The bulk of the work was conducted on the São Pedro das Águias prospect where vegetation was removed to allow for the extensive sampling of the exposed skarn outcrop in the northern and southern boundaries. The northern skarn was sampled through two channels, 16.70 m long and 15.50 m long, with weighted averages of 0.67% and 0.56% W and 0.086% Sn. The southern skarn samples returned assay results of 0.10% to 0.55% W, with roughly 0.05% Sn.

At Quintã exploration was limited due to a tin mining concession covering the area at the time, which has now expired. Exploration results located a number of skarn outcrops, with initial results reported to range from 0.17% to 0.76% WO_3.

Initial drilling at the Tabuaço project comprised a total of six diamond drill holes to determine the geometry, structure and grade of these scheelite rich skarns and were completed during the 1981-1982 seasons by a local contractor (Teixeira Duarte), Table 5-1 and Table 5-2. This contractor was not a mineral exploration driller, and core recoveries were poor, reportedly averaging 76%.

Holes S1 and S2 indicated the presence of an upper carbonate and a lower skarn horizon dipping sub-horizontally, with mineralisation concentrated within the skarn horizon. The northernmost drilled hole, S9, was thought to represent the central area of the deposit, intersecting 19.35m of mineralization reported at 1.18% WO_3, although no further drilling was carried out at the time to confirm this theory.

Table 5-1: List of Historic Diamond Drilling Conducted by SPE-SREM (UTM ED50 Datum)

Hole ID	Easting	Northing	Elevation	EOH_m	Collar_Dip	Collar_Azi
S1	624726	4549382	380.82	101.05	-90	0
S2	624726	4549382	380.82	105.25	-55	250
S3	624835	4549225	392.39	108	-90	0
S4	624835	4549225	392.39	84.8	-40	250
S6	624914	4549241	358.93	81.05	-90	0
S9	624662	4549465	386.63	63.8	-90	0

Table 5-2: Best Intercepts Reported from the SPE-SEREM Drilling

DDH	From	To	Interval	WO_3
S1	50.00	52.10	2.10	0.93%
S1	63.55	68.72	5.17	0.48%
S1	76.03	78.14	2.11	0.68%
S2	80.93	85.60	4.67	0.73%
S2	87.40	95.00	7.60	0.52%
S2	96.75	98.40	1.65	0.38%
S9	25.00	44.35	19.35	1.18%

Laboratory analysis on the samples from the SPE-SEREM exploration programme were conducted at SFM's, which is now the Portuguese Geological Survey department ("LNEG"), laboratory at S. Mamede de Infesta, near Porto. Analysis was completed using X-ray fluorescence spectrometry ("XRF"), reporting results in ppm W and ppm Sn with manual conversion of these results to % WO_3 and % Sn. Neither of the conversion formulae used to calculate the % WO_3 or % Sn is known. The suitability of the laboratory, equipment or sample preparation methods used in the SPE-SEREM exploration is also unverifiable; however XRF is a currently accepted analytical method for such mineralisation.

Check analyses were conducted by the BRGM laboratory in Orleans, France, reportedly with positive matches.

A systematic effort was carried out by Colt in the early stages of its exploration work at Tabuaço with a view to locate the historical drill core from the SPE-SEREM programme The historical drill core could not be found at any of the Government core storage facilities and therefore it has not been possible to confirm the WO_3 grades reported by means of core re-sampling.

Colt has referred to this historical data when positioning drillholes etc., but has not relied upon or included these historical results in any Mineral Resource estimation work.

5.4 Historic Mineral Resource and Reserve Estimates

The reported historical "resource" and "reserve" inventories cannot be considered a mineral resource or a mineral reserve under the Canadian Institute of Mining ("CIM") guidelines as economic parameters used to derive the estimates do not reflect accurately the current economics of exploiting this deposit. Furthermore, procedures and data used have not been reviewed and verified by a Qualified Person and therefore cannot be classified as a Mineral Resource under Canadian Securities Administrators NI 43-101 guidelines. In all cases, insufficient documentation exists that would allow SRK to classify historic reserve and resource estimates into the categories as currently defined by CIM guidelines. These historic estimates should be considered unclassified mineralised material.

Previous work by SPE-SEREM estimated, a non-compliant, geological resource of approximately 1 Mt of scheelite skarn mineralisation in stratiform horizons, grading 0.87% WO_3.

6 GEOLOGICAL SETTING AND MINERALISATION

6.1 Regional Geology

The geology of Portugal is subdivided into two large domains: the Hesperian Massif and the Epi-Hercynian (Variscan) cover rocks. The Hesperian Massif is itself subdivided into four main tectonic domains, which date from the Pre-Cambrian through the Paleozoic:

- Galicia – Trás-os-Montes Zone ("GTMZ");
- Central Iberian Zone ("CIZ");
- Ossa-Morena Zone ("OMZ"); and
- South Portuguese Zone ("SPZ").

The GTMZ occurs in the north-east corner of the country and is characterized by the mafic and ultramafic Bragança and Morais massifs. The rocks surrounding the massifs are mainly Silurian and represented by acid and basic volcanic rocks, as well as pelitic meta-sediments, which are thrust against the massifs. Alkali and porphyritic granites are also present.

The CIZ is characterized by the predominance of schist and greywacke with minor carbonates, representing metamorphosed flysch-type rocks dating from the Late Precambrian and Lower Cambrian. There are also large areas of alkali and calc-alkali granite and granodiorite.

The OMZ is a complex and diverse domain, with a stratigraphic sequence that goes from the Precambrian through Cambrian and Silurian and ends with flysch units in the Devonian. The contact with the CIZ is a regional tectonic feature known as the Tomar-Cordoba Shear Zone.

The SPZ is characterized by a Late Devonian to Early Carboniferous volcano-sedimentary complex ("VS"), which is overlain by the Mid-Late Carboniferous flysch sequence ("CULM"). These rocks are all underlain by the Devonian Phyllite-Quartzite Group ("PQ") and the Pulo do Lobo Formation, comprising phyllite, quartzite and occasional acid and basic volcanics. The rich polymetallic massive sulphide deposits of the Iberian Pyrite Belt, which includes the large Aljustrel and Neves-Corvo mines in southern Portugal, are associated with the acid volcanics of the VS.

Finally, the Epi-Variscan ("Variscan") cover rocks include the Mesozoic-Cenozoic sedimentary units (limestone, clay and sandstone) of the south and west of Portugal, and the Tertiary detrital basins of the Tagus (Tejo) and Sado Rivers.

The contact of the SPZ with the OMZ is the Ferreira-Ficalho thrust, which is the remnant of the geo-suture resulting of the late Carboniferous subduction or collision of the SPZ into the OMZ.

The Tabuaço Project is located within a segment of the CIZ in northern Portugal as shown in Figure 6-1.

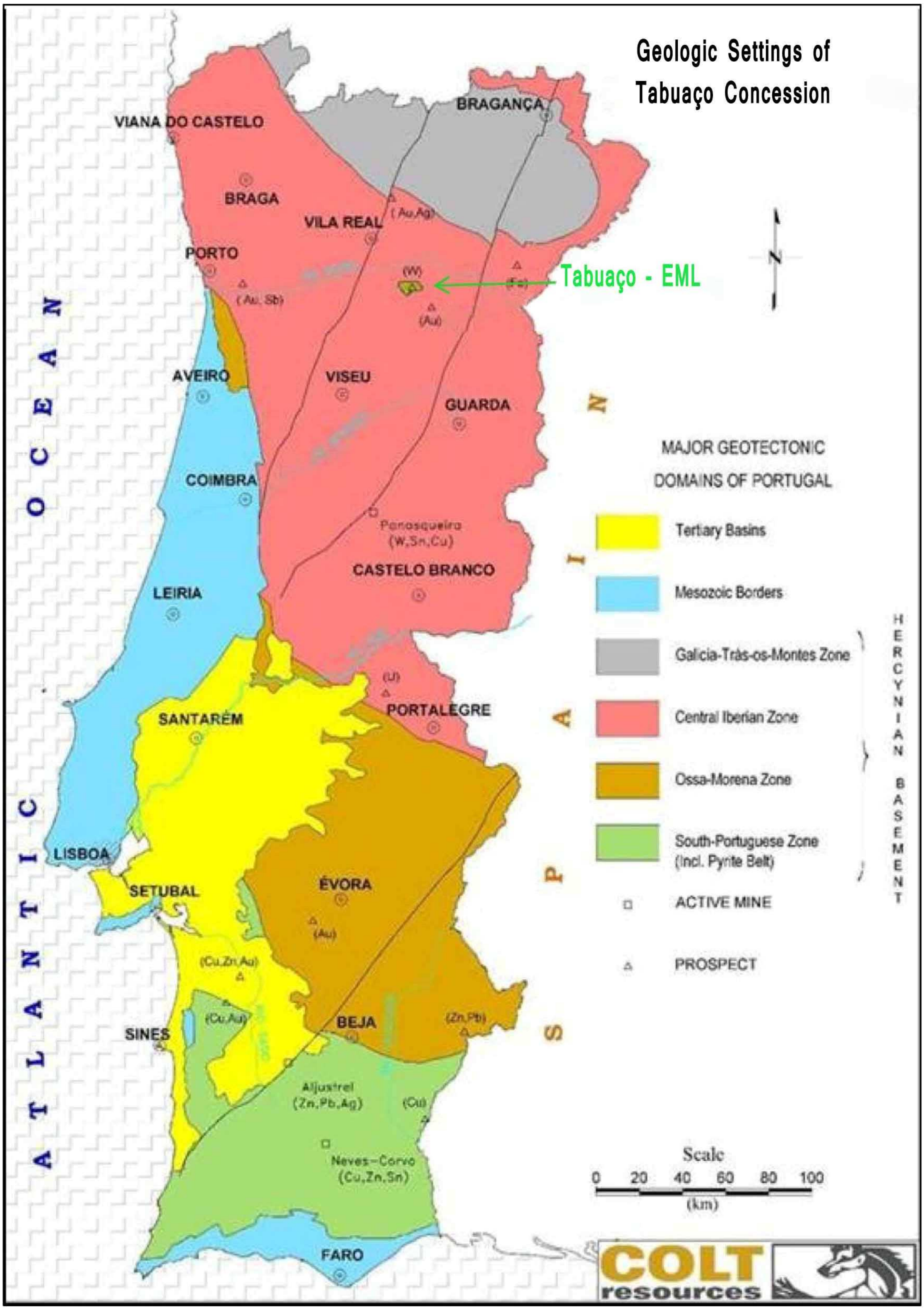


Figure 6-1: Simplified Geological Map of Portugal

6.2 Local Geology

The Tabuaço EML is situated on the border zone of two major geologic units in the CIZ of northern Portugal, namely the Variscan age Beiras granite batholiths and the Douro Valley Schist-Greywacke Complex ("SGC") (Faria, 2008), also referred to as the "Beira Schist Complex" (Figure 6-2). The latter, of lower Cambrian age, comprises several formations that include shale, schist, siltstone, sandstone and greywacke, with subordinate conglomerate and carbonate beds. These rocks were folded and undergone low-grade regional metamorphism during the Variscan orogenic cycle (Upper Paleozoic). Intrusion of several successive granite plutons also took place during the Variscan cycle, since pre-tectonic till late-tectonic times, originating significant contact metamorphic haloes in the enclosing metasediments.

The lower part of the SGC, the Bateiras Formation, is characterized by a basal unit of black graphitic schist overlain by grey biotite schist with some carbonate beds intercalated. These lithologies normally outcrop only at the lowest portions of the river valleys, where the core zones of anticlines are exposed by erosion. Outcropping rocks in the region comprise as follows;

- Schistose Unit: black and grey schist and phyllite, locally containing minor intercalated calc-silicate;
- Limestone Facies: series of banded, grey-green to bluish crystalline limestone and calc schist consisting of alternating thin layers of crystalline carbonate and meta-pelite that have undergone greenschist-facies metamorphism. Also includes local calc-silicate and skarn horizons. Mineralogy consists of plagioclase feldspar, quartz, calcite, garnet and vesuvianite;
- Skarn: a much more developed metamorphic/metasomatic formation than the above unit with amphibolite-facies metamorphism. Represents a series between 'dry' hornfels with spotty amphibole at granite contacts to retrograde garnet-diopside skarn in proximity to limestone. Whitish to dark greenish colour and consisting of plagioclase, quartz, garnet, hornblende, epidote-group minerals, vesuvianite, calcite and fluorite;
- Granite: medium-grained "two-mica" and "muscovite" granites; the latter forms a rim around the largest intrusion of the "two-mica granite", and is often tourmalinized and with feldspars often transformed to damourite, a greenish variety of muscovite; and
- Aplite: aplitic-pegmatitic-sills and rare quartz veins. Alteration of the aplite results in a greenish, finer-grained muscovite bearing (± damourite) often tourmalinized rock.

The Schist-Greywacke Complex lithlogies are in contact with the Armamar-Tabuaço granite batholith within the Tabuaço project area. This contact is associated with the Távora anticline, which hosts graphitic schist at its core, overlain by biotite schist with skarn and carbonate horizons. Metasomatism of the carbonates is thought to have produced the quartz-garnet-amphibole-pyroxene skarns that host the target tungsten mineralisation.

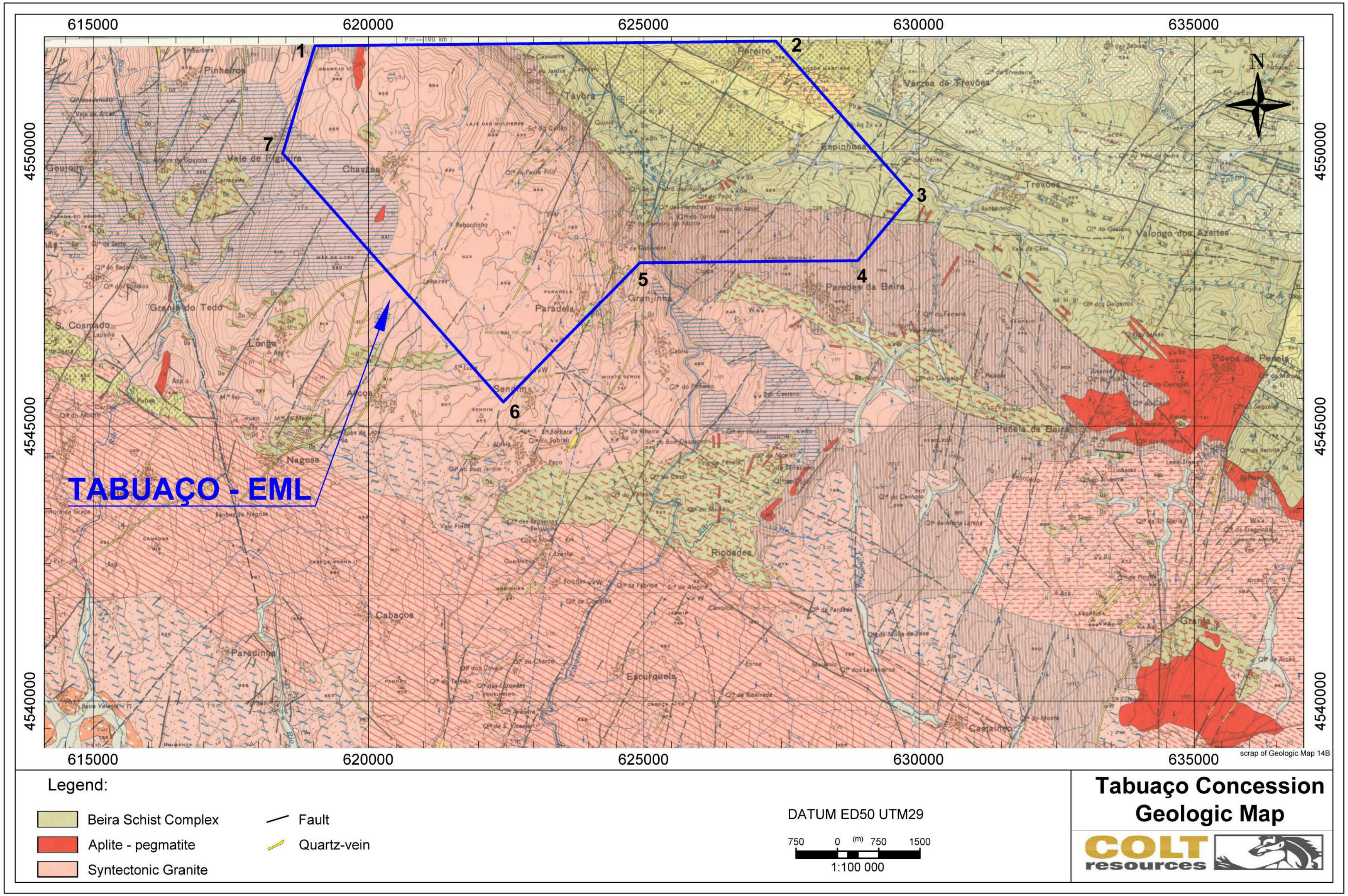


Figure 6-2: Geological Map of the Tabuaço Project Area

6.3 Property Geology

The landscape on the property shows a striking contrast between pine forestry developed on granite upper slopes of the valley to the vineyards and olive trees that are exclusively developed on schist.

6.3.1 Sao Pedro das Águias

The Sao Pedro das Águias deposit is located at the contact of the Armamar-Tabuaço granitic intrusion and schist of the 'SGC' on the western limb of the Távora anticline. Figure 6-3 shows the Tabuaço area geology, the green boundary demarcates the approximate limit to the resource area at Tabuaço (SPA) and Aveleira.

The granite is typical late-stage fractionated S-Type with coarse equigranular texture. The mineralogy consists of feldspar, quartz, muscovite and biotite. Tourmaline is observable in hand specimen as black acicular laths. Finer grained 'sericitization' of plagioclase is damourite rather than sericite – potassium-enriched muscovite which gives a green to lilac hue. The body of the granite is homogeneous but becomes more complicated near the contact with the schist. In places there is some evidence for chilling with 1 m to 3 m of speckled rhyolitic rock. In other parts of the intrusion the contact is tectonic. Aplite dykes and sills also stream off from the granite. The contact zone can be considered as asymmetrical with a relatively narrow unmineralised endocontact and a wider exocontact zone over several, up to tens of metres depending on the steepness of the contact. Drilling in the northern and western part of the Tabuaço deposit where several holes have intersected schist below granite reveal a shallow dip of 25° to 30° south-we st in line with the dip of schistosity in the country rock.

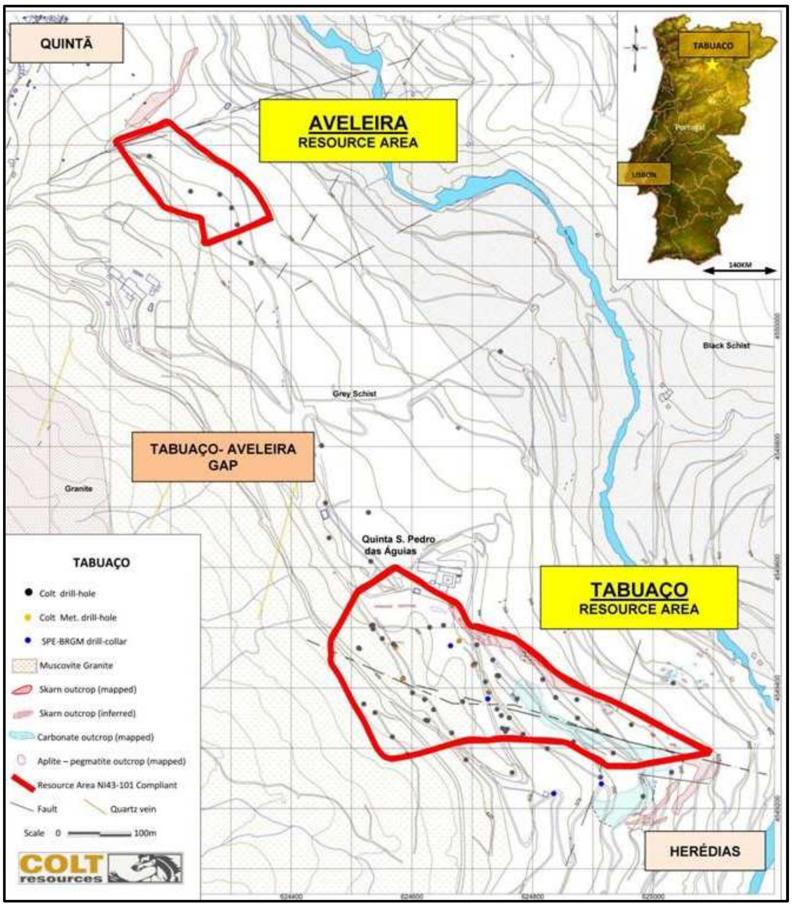


Figure 6-3: Property Geology within the Tabuaço Project Area

The country rock of the SGC as described by Faria (2008) comprises hangingwall and intermediate layers to mineralisation of biotite and chloritic schist and phyllite. Secondary mineralisation includes lenses and bands of silica and calc silicate minerals (commonly epidote) and occasionally calcite. Where calc silicate banding becomes interlayered the lithology is logged as a separate laminated pelitic calc silicate or calc schist type.

The footwall to mineralisation tends to be marked by the presence of fine-grained homogeneous black chloritic schist with graphite, pyrite of diagenetic origin on cleavage planes and is largely devoid of calc silicate minerals, and unaffected by hydrothermal alteration.

The carbonate facies are all affected by alteration to some degree so the original protolith is difficult to ascertain and the carbonate may represent a persistent block of impure reef limestone with cycles of thin limestone and marl underneath. This environment with the presence of silica impurities within makes a fertile environment for skarnification.

There are three main units containing tungsten mineralisation that attain thicknesses of several metres and have a close generic relationship:

- Upper Carbonate Horizon ("UCH");
- Main Skarn Zone ("MZ"); and
- Lower Skarn Zone ("LZ").

The UCH is classified separately from the skarn as tungsten is associated with sub-massive blocky limestone with strong silica banding.

The MZ is characterised by a greyish white rock with a bluish-pink hue that is blocky or with a weakly banded texture with spotty black porphyroblasts (Figure 6-4). The LZ shows classic pink and green banding characteristic of contact metamorphosed rocks and skarn resulting from contrast between pink/red prograde mineral assemblage and the green retrograde mineral assemblage (Figure 6-5). Detail on the mineralogy will be discussed in Section 6.5.

The distribution of MZ to LZ is not always regular, the two units occasionally juxtapose and separate into leaves with several layers particularly evident with the LZ type. There are discontinuous lenses of both types in the intermediate schist. This classification has, however, provided a good framework on which to build the geological model as explained in Section 7.

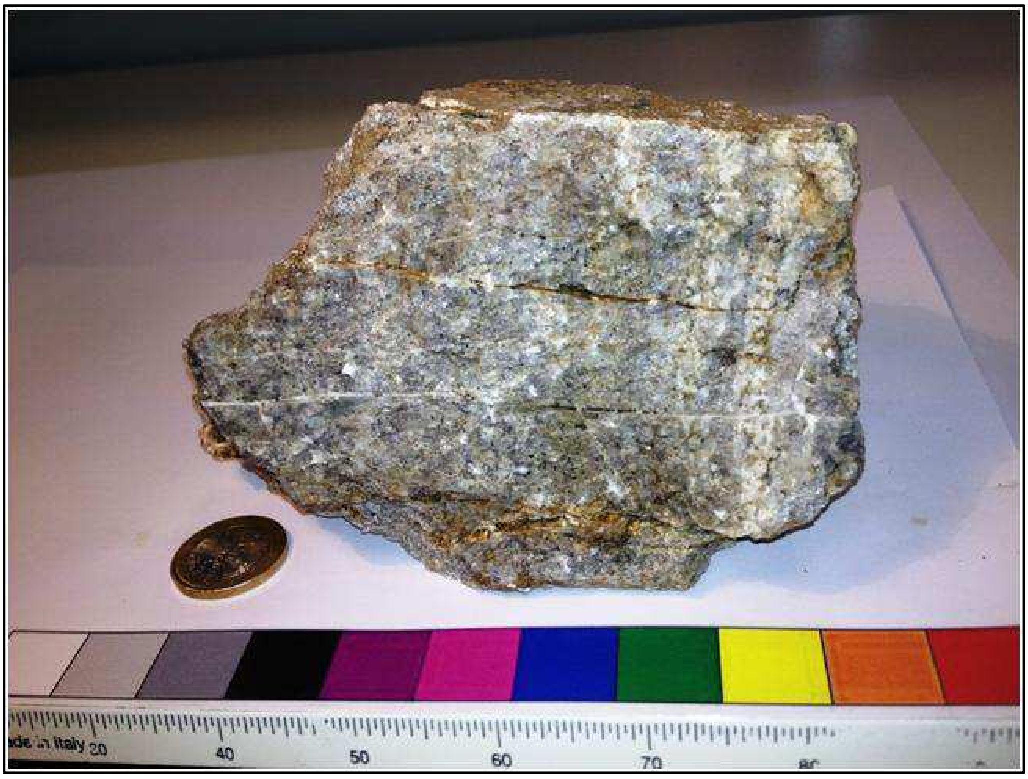


Figure 6-4: Typical Examples of Skarn in Hand Specimen in Main Zone

Figure 6-5: Typical Examples of Skarn in Hand Specimen in Lower Zone

6.3.2 Aveleira

This area is located around 700 m northwest of the Tabuaço deposit (SPA), and hosts significant, non-outcropping tungsten bearing skarn beds which were not discovered until January 2012, when Colt drill tested soil geochemical anomalies.

Although detailed geologic knowledge of this new hidden tungsten deposit requires further drilling, it is already known that it bears striking similarities with the SPA deposit. As in the latter, two main tungsten-bearing skarn horizons of metric thicknesses have already been recognised, enclosed in a biotite schist sequence near to the contact with the Armamar-Tabuaço granitic intrusion. The skarn horizons also appear to be gently dipping to the West-Southwest.

6.3.3 Quintã and Quintã – Távora Zones

The Quintã and Quintã – Távora zones are located 1.1 km northwest of the Tabuaço deposit (SPA). Three holes have been drilled at Quintã to date with inconclusive results.

The skarn bedding in the main part of this outcrop is sub-horizontal and gently dips to the Northwest. It is understood to be located at the very crest zone of the Távora anticline not far from the contact with the Armamar – Tabuaço granite intrusion. Typically, the outcrop is discontinuous making it difficult to see whether there is one or two skarn horizons present.

Taking into consideration the structural knowledge acquired in the Tabuaço and Aveleira areas, the Quintã and the Quintã – Távora zone are interpreted as having the potential to host additional hidden tungsten-bearing skarn deposits.

6.4 Structure

The major structure of the Tabuaço project area is dominated by two deformational events. A D1 event attributable to regional metamorphism is shown by tight chevron and drag folding in schist and also marly limestone of the UCH. The D2 event produces mesoscale folding not observable in core on an amplitude of tens of metres. This event is responsible for the Távora Anticline trending NW-SE on the western side of the Távora River which is exposed near the Ponte de Fumo Bridge and a regional Hercynian trend. Conjugate to this antiform at QSPA-Herédias is an open synform which produces a SW dipping bowl-shaped structure in 3D on the skarn and carbonate horizons.

Two sets of brittle faults are important. A brittle north-west to south-east fault, FLT1, which is broadly parallel to the Távora Anticline axis. The second set which has also been modelled is orientated perpendicular to FLT1 and appears to cross-cut in a NE-SW to NNE-SSW direction. The mapped upper and lower skarn outcrops at QSPA are notably separated from the Quintã das Herédias skarn outcrops to the south by a major strike-slip fault, which has been mapped from surface.

Displacement on these fault sets appears to be in the order of a few metres in terms of offsetting mineralisation but the pattern may be more complicated near the contact of the main granite body where there is less evidence from drilling. Several fault zones may be stacked here with ‘screens’ of granite (allochthonous blocks and lenses detached from the main intrusion) in between the faults.

Originally the north-south fault was considered to limit the mineralisation to the south-west of QSPA but subsequent drilling from the 2012 campaign has shown this not to be the case. The reason here may be due to a steepening of the granite contact and more limited range of hot hydrothermal fluids emanating from the granite rather than this particular structure.

At Aveleira and Quintã the influence of FLT2 set is apparent as it downthrows the lithology several metres to the south between Quintã and Aveleira.

6.5 Mineralisation

The Tabuaço region is noted for tungsten and tin occurrences, and has seen a number of past artisanal workings.

In the Tabuaço EML, there are skarn outcrops that are mapped as being mineralised. For the purpose of this report and resource estimation, only the SPA and Aveleira mineralised zones lie within the immediate resource area with Quinta das Herédias, Quintã, Quintã-Távora and the 'gap' area between Aveleira and SPA lying outside the immediate resource area but considered as exploration targets to the north and north-west but still within the Tabuaço project area (Figure 6-2). The SPA and Aveleira deposits contain the most significant known tungsten mineralisation of the Tabuaço area. Mineralogical studies to date have been concentrated on the SPA deposit area.

The skarn mineral assemblage comprises scheelite with epidote group minerals (epidote, zoisite, clinozoisite), albite plagioclase, grossular garnet and calcite. Also in the gangue vesuvianite, k-feldspar, fluorite, apatite, muscovite and quartz. Sulphides were identified under reflected light in trace amounts, namely pyrite, chalcopyrite and arsenopyrite.

In the Main Zone the granoblastic grey-bluish banded texture is due to the presence of albite, vesuvianite, sericite, fluorite and scheelite. Sericite occurs from the breadown of scapolite and plagioclase. High grades of tungsten occur in this zone as demonstrated under UV light in Figure 6-6 and Figure 6-7.

The Lower Zone is less albitic but still contains some plagioclase; instead K-feldspar is more prominent giving the pinkish banding. Green banding is caused by the presence of cac-silicate minerals principally by epidote group minerals, vesuvianite and Ca-pyroxene (diopside) with minor scapolite and grossular garnet. Black speckling is due to porphyroblasts of amphibole or bioitite.

Lane (2011) investigated the mineralogy of a sample of concentrate derived by gravity separation using heavy liquids which was originally taken from skarn at outcrop (Section 12). The relative abundance of minerals screened at 425 µmm is shown in Table 6-1.

The skarn mineralised zones host virtually no sulphide mineralisation nor any molybdenum. Powellite, with Mo replacing for W in scheelite, may have been identified under UV light (glows yellow rather than blue) but has not been seen in significant quantities to affect processing. Apart from the presence of fluorite the assemblage is favourable for both mineral processing and having few deleterious elements (e.g. sulphide) to impact on the environment.

Figure 6-6: Scheelite Bearing High Grade Tungsten Mineralisation in MZ shown in Plane Light

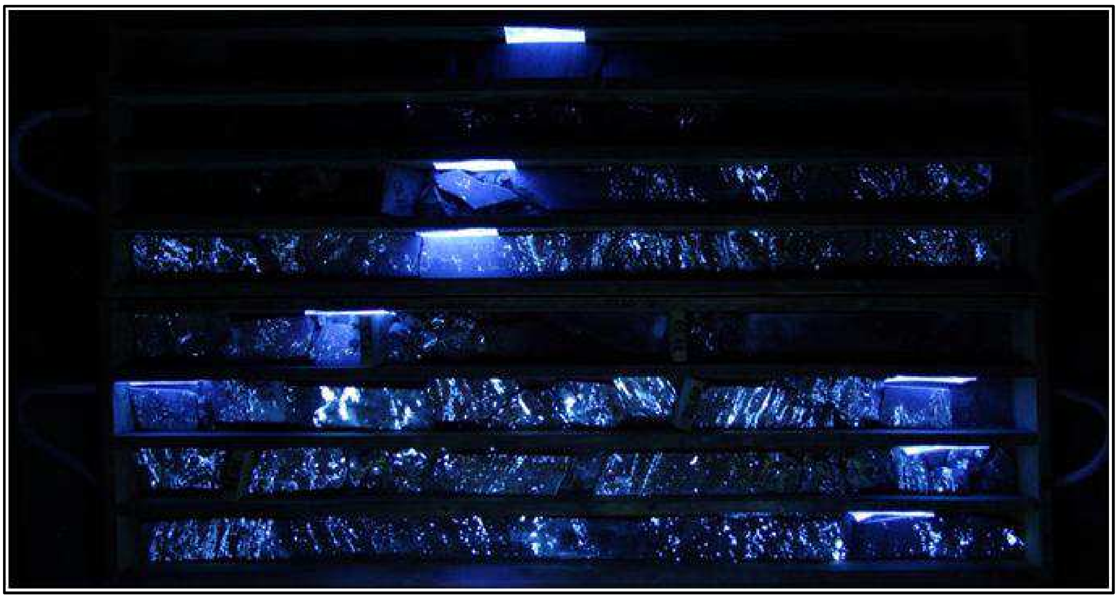


Figure 6-7: Scheelite Bearing High Grade Tungsten Mineralisation in MZ shown under UV Light

Table 6-1: Mineral Abundance by Size Fraction

Model Abundance	Scheelite	Sulphides/ Cassiterite	Apatite/Fluorite /Carbonate	Pyroxene	Feldspars
+425µm	0.44	0.01	3.62	58.96	37.0
-425µm	0.40	0.02	4.48	67.72	27.4
Head (Calc)	**0.42**	**0.01**	**4.13**	**64.16**	**31.3**

7 DEPOSIT TYPE

Mineral deposits within the Armamar Meda licence and surrounding area include skarn tungsten, tin & tungsten bearing veins and orogenic gold. The Tabuaço project is considered a skarn Tungsten deposit.

Tungsten skarns are one of seven types of skarn deposits that comprise an economically significant class of mineral deposit and which account for the majority of tungsten produced worldwide. This classification is documented by Ray and Webster (1991). As with most skarns, deposits form in “reactive” rocks such as calcareous sediments and/or volcanics. Skarns form by regional or contact metamorphism, the latter being the more common geologic model. In these settings intrusive rocks are often nearby and often provide the heat source for the hydrothermal activity that alters the host lithologies and introduces mineralisation.

The term skarn is an old Swedish mining term originally used to describe a type of silicate gangue, however in modern usage the term "skarn" has been expanded to refer to calcium-bearing silicates. In the USA the term "tactite" is often used synonymously with skarn.

Skarns and tactites are most often formed at the contact zone between intrusions of granitic magma bodies in contact with carbonate sedimentary rocks, where hydrothermal fluids derived from the granitic magma are rich in silica, iron, aluminium, and magnesium. These fluids mix in the contact zone, dissolve calcium-rich carbonate rocks, and convert the host carbonate rock to skarn deposits in a metamorphic process known as "metasomatism". The resulting metamorphic rock may consist of a very wide variety of mineral assemblages dependent largely on the original composition of the magmatic fluids and the purity of the carbonate sedimentary rocks.

The actual ore-bearing mineral containing the tungsten at Tabuaço is scheelite, a calcium tungstate mineral ($CaWO_4$). Pure scheelite fluoresces bright blue under shortwave ultra violet light.

7.1 Geological Model

At Tabuaço the geological model used for exploration is best described as a contact metamorphosed tungsten skarn model. The key considerations for paragenesis of scheelite mineralisation are:

- Proximity to granite intrusion and related aplite dykes and sills;
- Circulation of hot fluids about the granite and dykes and interaction with cooler meteoric water in the country rock; and
- Chemistry of the country rock and amenabilty to chemical reaction with hydrothermal fluids and metasomatism.

There is no clear correlation between tungsten mineralisation and proximity to the granite contact – ie higher grades in the exocontact nearest the intrusive body itself. Ordinarily the heat gradient falls away fairly quickly from the contact within 20 m to 30 m. Conversely as contact metamorphism is a relatively localised effect the model needs to address how the skarn mineralisation is extended for several ten of meters and sometimes over 100 m away from the main granite body.

Good skarn development is propagated by successive intrusive events. Pulses of magma and sheeting of dykes recharges the heat flow and creates a reflux effect whereby cooler sinking fluids are replaced by hotter rising ones. The morphology of the granite contact can also have an important bearing on mineralisation. The laccolith shape of the granite in the centre and north of QSPA may have created a barrier trapping fluids enabling enough time for the creation of a thick zone of metasomatism, hence greater skarn development and tungsten mineralisation where the granite contact is shallow. Figure 7-1 shows a very simplified representation of such a paragenesis.

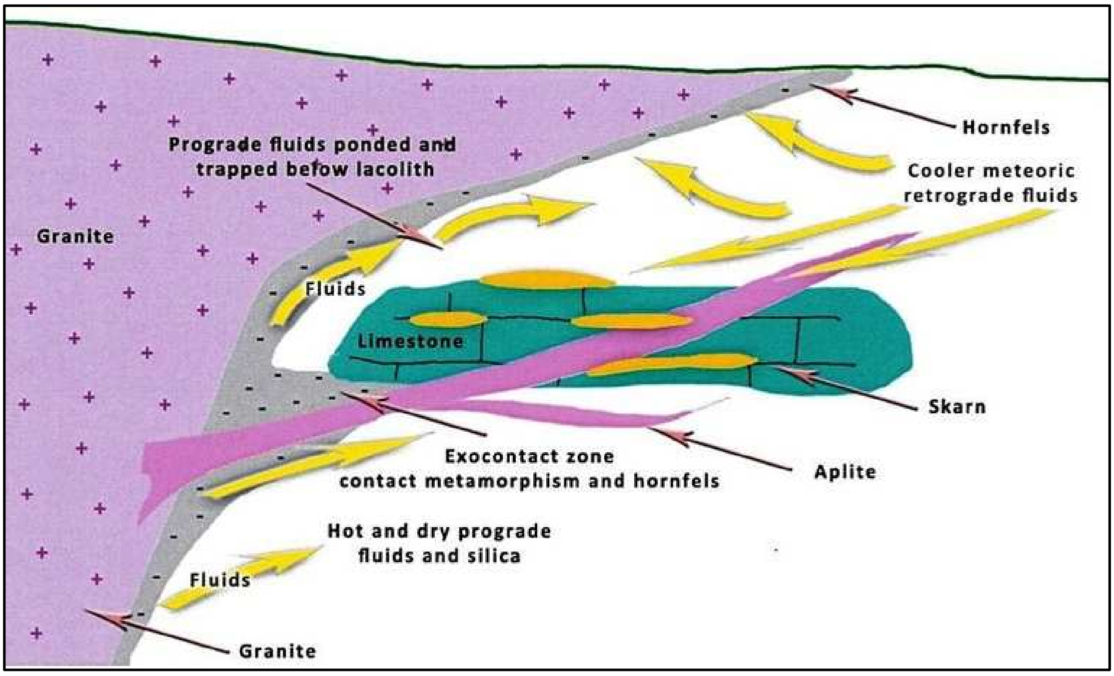


Figure 7-1: Simplified paragenesis to explain the formation of skarn at QSPA

The chemical composition and impurities within limestone and amenability of limestone and other calcium rich strata also has a bearing on skarn development. The impure reefal limestone responsible for the UCH and MZ is persistent and probably stretches for several kilometers but high grade tungsten occurs where there is low magnesium and silica present. Mineralisation in UCH in particular may be due to the local occurrence of silica replacing fossils/stromatolite. The LZ may be formed from metasomatism of several thin band of limestone and calcareous marl below the reefal limestone. Stacking of several layers gives a high surface for fluids to react over.

8 EXPLORATION

After acquisition of the exploration licence and an assessment of the available data, Colt began field exploration on the Tabuaço project in 2008.

Initially these activities focused on prospecting, mapping and collection of rock chip samples, to establish the full surface extent of the mineralised zone. Ultraviolet lamp prospecting has also been conducted on the scheelite during a number of night time field visits. Results of this prospecting continued to be favourable leading to the initiation of diamond drilling in November 2009.

9 DRILLING

The current exploration area is focussed on showings of scheelite mineralisation or hidden deposits outside of the Tabuaço (SPA) deposit resource area which includes Quinta da Aveleira, Quintã, Quintã-Távora and the 'gap' area between Aveleira and SPA.

Prospecting holes were planned to test the newly discovered soil geochemical anomalies resulting from the survey completed in November 2011 (Section 8). The first scout diamond drill hole completed in February 2012 led to the discovery of the Aveleira skarn tungsten deposit. Since then and up to October 5, 2012, a total of 16 diamond drill holes for a total of 1,278.27 m have been completed mostly using mobile APAFOR™ 400 series, but also Mustang A32 and Christensen CS(10) rigs where possible, on the Aveleira deposit as well as the gap between the latter and the SPA deposit. The APAFOR™ rig is a small but powerful conventional coring rig designed for ground investigation work but adapted for negotiating vines and narrow terraces. The majority of these holes intersected tungsten-bearing skarn beds with thicknesses of several metres.

At Aveleira the results of prospect drilling to date confirm the extension of the skarn tungsten deposit for a strike length of at least 200 m in the north-west south-east direction (Figure 6-3 and Figure 9-1). Step-out drilling has already been initiated across strike to determine the lateral extent of the deposit, which is thought to dip gently into the mountain and underneath the granite outcrop, in style similar to Tabuaço (São Pedro das Águias).

In order to fast track towards the delineation of an inferred resource for the Aveleira deposit, a 75 m x 75 m drill grid was planned comprising an additional 13 diamond drillholes adding up to an estimated total of 1,620 m.

A drilling programme of 9 short vertical holes was also planned to follow up on additional soil geochemical anomalies in the Tabuaço-Aveleira gap; 6 of these holes have already been completed, of which 4 intersected skarn tungsten mineralisation whereas 2 were barren. Additional drilling is currently being planned for this gap which seems to be prospective for blind tungsten mineralisation.

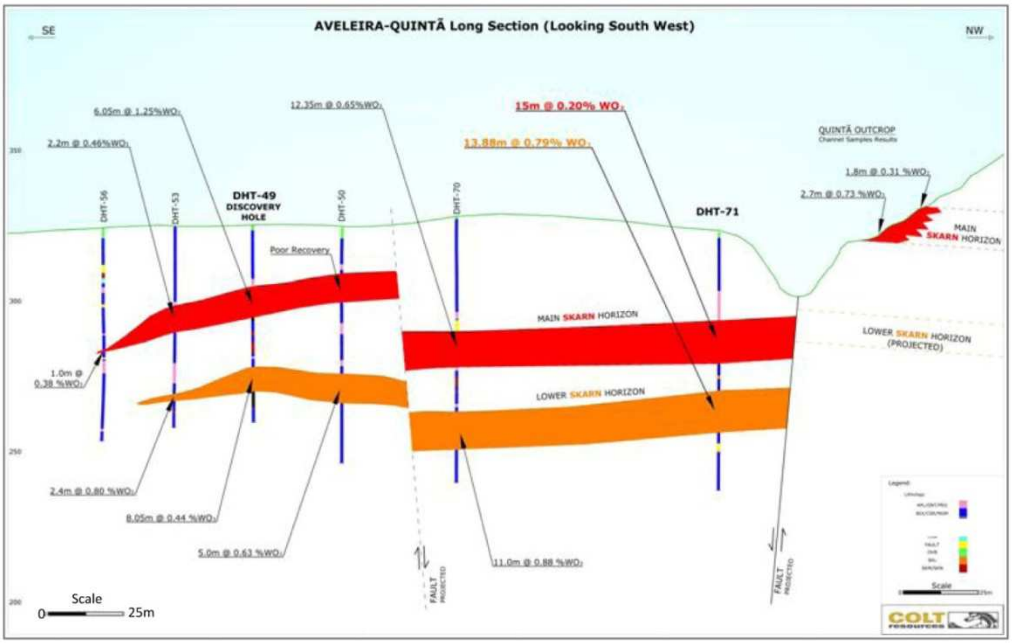


Figure 9-1: South–West Facing Long Section showing Aveleira Mineralised Zones

9.1 Surveys and Investigations

All drillholes were surveyed at the collar surface by the responsible geologist, using a handheld GPS. Data for Eastings, Northings and RL were initially recorded in UTM 29T, Datum ED50. All collar coordinated have been converted subsequently and are currently surveyed in UTM29N, Datum WGS84.

A high resolution topographic survey was conducted in 2010 by survey contractors Superfície Topographia Lda ("Superfície") of Porto, leading to a 2 m resolution digital terrain model. Holes DHT01–DHT09 were also accurately surveyed during this work. The topographic survey has enabled the continued use of handheld GPS units for collar locations, as data can be draped onto the topographic surface and corrected.

The topographic survey was updated in August 2012 by survey contractors Superfície. The DTM was updated using photogrammetric methods and remote sensing imagery. Drillhole collars for the 2012 campaign were captured using a portable DGPS using a central radio beacon for reference. Data was presented by Superfície in both ED50 and WGS84 format. The updated coordinates in ED50 datum incorporating this survey were used in the Mineral Resource Estimation.

Exploration work done beyond the Tabuaço (SPA) deposit aimed at locating additional tungsten-bearing skarn deposits and included: geologic prospecting and rock sampling; night-time Mineralight (UV-lamp) prospecting; a soil geochemical survey; and finally scout diamond drilling.

Geologic prospecting led to the discovery of skarn float at Aveleira as well as skarn outcrops at Quintã. Follow-up rock grab and chip sampling confirmed the presence of tungsten (scheelite) mineralisation in most of these rock units.

Mineralight UV light prospecting was conducted particularly along the West bank of the Távora to the Northwest of the SPA deposit. This lead to the discovery of several areas containing (fluorescing) scheelite mineral debris in the soil or in mineralised float spread from the São Pedro das Águias farm through the Aveleira farm.

In order to help in the location of likely blind tungsten deposits, a multi-element soil geochemical survey was undertaken by Colt along the West bank of the Távora. Firstly, two pilot soil traverses comprising a total of 54 soil samples were sampled at 10 m intervals across the São Pedro das Águias deposit outcrop, in order to provide indicative evidence with regards to the geochemical signature of the tungsten skarns in this particular area. Following this a soil geochemical survey was carried out to the NW of the deposit throughout the São Pedro das Águias and Aveleira farms. This comprised a total of 293 samples collected from 10 m intervals along a total of seven 100 m spaced traverses. This survey lead to the identification of a number of geochemical anomalies containing elements indicative of tungsten skarns, namely W, Ca, Mg, Mn, Ti, etc.

9.2 Sampling Methods and Quality

The following field methods have been applied to the multiple sampling programs undertaken in Colt's fieldwork, in every case executed by COLT's own field staff and under supervision of experienced senior personnel:

Soil sampling sites along planned traverses were defined in advance by a senior geologist prior to the fieldwork based on both topographic and geologic mapping. The location of the sampling sites in the field was done with the aid of a hand-held GPS with some minor site adjustments being occasionally necessary due to various obstacles (e.g. vineyards, steep terracing, stream crossings etc.).

The samples collected from each grid location were sieved on-site through a 2 mm sieve prior to bagging in order to avoid inclusion of gravel the sieve was cleaned with a thick brush between each pair of samples. Samples weighing between 500 g and 700 g were taken from each site for shipment to the laboratory; these samples were bagged in plastic bags which included a tag with the sample number inside the bag as well as the same number written in water-proof ink on the outside of the bag. The collection and sieving of the samples in the field as well as their packing and shipment to the laboratory was done by company local helpers under the supervision of a senior field assistant.

The selection of the areas for hammer prospecting and rock sampling work has been planned by Colt's senior geological personnel based on geological, geochemical and mineral occurrence data. Rock samples taken during hammer prospecting are meant only to determine whether a prospective tungsten-mineralised lithology (e.g. scheelite or cassiterite bearing skarn) yields any anomalous tin/tungsten values and not to try and determine average grades for a specific rock unit over a specific width. Where a number of float blocks are found spread over a restricted area, a composite float sample is taken from several blocks for the sake of statistics but this must also be considered as a grab sample. The rock samples are bagged for shipment to the laboratory inside plastic bags which include a tag with the sample number inside the bag as well as the same number written on the outside of the bag in both cases in water-proof ink.

Drill core has been sampled on site by Colt staff, the procedure for which is described in Section 10 of this report.

Contamination checks during sample preparation were carried out through blanks being prepared on site by the Colt technicians from known barren material (white marble) and inserted at a rate of 1 per every 30 samples into the drill core sample stream.

Certified Standards used by Colt for the 2012 season are W104 (0.202% tungsten), W105 (1.40% tungsten), W106 (2.16% tungsten), and W108 (0.72% tungsten) and were inserted at a rate of 1 per 30 samples into the sample stream. All Certified Standards have been prepared and supplied by WCM Minerals, and Certificates of Analysis are shown in the Appendix of the November 2012 NI 43-101 Report.

Colt has compiled and analysed Quality Assurance and Quality Control ("QAQC") on the submitted samples, blanks and standards which have been subject to audit by SRK for the November 2012 NI 43-101. Any samples from the returned assays that varied from the expected values by more than 1 standard deviation were investigated further before acceptance into, or rejection from, the final dataset. It was SRK's opinion, based on QAQC program results that the obtained assays are representative of the deposit. The graphs reported support the conclusion of no bias in the laboratory results, and consequently, SRK had full confidence that the samples in the assays database are representative of the deposit.

Since starting the exploration drilling programme at SPA in November 2009, Colt has drilled 101 drillholes as of December 31, 2012, completing a total of 7,829 m of diamond drilling across the Tabuaço project area.

Four diamond drilling campaigns have been conducted on the Tabuaço Tungsten project area. The first was conducted by SPE SEREM in the early 1980's and comprised six diamond drill holes labelled S1-S4, S6 and S9. This campaign was sufficient to prove the existence of scheelite bearing skarn horizons over the area but recoveries were poor thus none of the data has been utilised in a quantitative manner by Colt.

The second campaign was conducted by Colt in 2009 to 2010, primarily to verify the extent and levels of tungsten mineralisation quoted in the historical literature. A total of 9 diamond drillholes were drilled totalling some 815 m, and labelled DHT01B-DHT09.

The third campaign, also conducted by Colt between November 2010 and 11th October 2011, was conducted to establish a Mineral Resource Estimate ("MRE") for Tabuaço and comprised 23 diamond drillholes for 2,661.54 m and labelled DHT10-DHT33 (but not including DHT-31 and DHT-32). The resultant MRE can be seen in the December 2011 NI 43-101 Report.

The fourth diamond drilling campaign was again conducted by Colt between November 2011 and 4th December 2012. The reasons for this campaign were to:

- Close off and delimit the extent of mineralisation at the QSPA resource area;
- Upgrade the Inferred classified resource area estimated in the previous MRE of December 2011. To this effect no drillholes were planned within the Indicated classified resource area outlined by SRK for the November 2012 MRE between sections 1075N and 1200N;
- Produce drill core sample for bench-level metallurgical testwork, and;
- Verify and outline the extent of mineralisation in Quintã de Aveleira and the gap area between Aveleira and QSPA.

Details of the last campaign are shown in Table 9-1. The drillholes used in the November 2012 MRE are listed and illustrated in Table 9-2, Table 9-3 and Figure 9-2. Drilling has generally been planned and executed using a grid with spacings ranging from 50 m x 25 m to 25 m x 25 m.

Table 9-1: Colt Drilling Campaign 2011 to 2012

Area	Diameter	# of Holes	Metres	Drillholes
QSPA Evaluation	HQ/NQ	23	2,506.9	DHT43, 44, 45, 46,47,48, 51,52,54,55,57,58,59, 60, 61, 62, 63, 64, 65, 66, 67, 68,69
QSPA Metallurgical	PQ/HQ	8	736.0	DHMT01-05 DHMT 31,41,51
Aveleira Exploration	HQ	15	1,815.4	DHT49,50,53,56,70,71, 74, 76, 79, 80, 81, 84, 85, 87 and 86.
Total		**46**	**5058.3**	

Table 9-2: Drillholes utilised in the November 2012 MRE in ED50 format

Sondagem Drillhole	ED50/ UTM Easting	Northing	Elevation	Assay results returned and used in evaluation
DHT-01B	624,678.50	4,549,477.87	378.84	Yes
DHT-02	624,728.02	4,549,393.04	378.53	Yes
DHT-03	624,784.15	4,549,323.21	379.66	Yes
DHT-04	625,032.70	4,549,408.18	261.64	Yes
DHT-05	624,838.32	4,549,384.16	331.48	Yes
DHT-06	624,874.43	4,549,349.95	330.88	Yes
DHT-07	624,977.59	4,549,273.63	325.94	Yes
DHT-08	624,704.51	4,549,440.45	378.79	Yes
DHT-09	624,703.33	4,549,438.02	379.20	Yes
DHT-10	624,533.08	4,549,503.32	400.53	Yes
DHT-10A	624,533.48	4,549,502.76	400.67	Yes
DHT-11A	624,572.71	4,549,468.67	408.59	Yes
DHT-12	624,572.20	4,549,468.09	408.56	Yes
DHT-13	624,585.24	4,549,415.74	415.76	Yes
DHT-14	624,584.73	4,549,414.68	415.76	Yes
DHT-16	624,620.97	4,549,380.82	417.84	Yes
DHT-17A	624,741.06	4,549,364.84	380.50	Yes
DHT-18	624,665.88	4,549,325.23	418.84	Yes
DHT-19	624,748.09	4,549,354.40	380.54	Yes
DHT-20	624,748.09	4,549,354.40	380.54	Yes
DHT-21	624,761.31	4,549,350.46	379.66	Yes
DHT-22	624,912.65	4,549,251.87	356.46	Yes
DHT-23	624,535.59	4,549,504.22	400.59	Yes
DHT-24	624,824.28	4,549,299.50	374.12	Yes
DHT-25	624,701.86	4,549,437.59	379.21	Yes
DHT-26	624,629.34	4,549,500.56	393.60	Yes
DHT-27	624,906.82	4,549,328.38	326.99	Yes
DHT-28	624,511.70	4,549,466.25	419.48	Yes
DHT-29	624,741.22	4,549,420.44	363.06	Yes
DHT-30	624,569.15	4,549,472.90	407.74	Yes
DHT-31	624,512.89	4,549,465.30	419.38	Yes
DHT-33	624,652.42	4,549,503.75	384.55	Yes
DHT-34	624,742.23	4,549,420.76	363.08	Yes
DHT-36	624,626.36	4,549,345.82	427.19	Yes
DHT-37	624,766.09	4,549,259.60	403.18	Yes
DHT-38	624,707.77	4,549,471.49	370.77	Yes
DHT-39A	624,623.04	4,549,346.60	427.20	Yes
DHT-40	624,774.37	4,549,382.03	364.23	Yes
DHT-41	624,538.88	4,549,359.18	440.07	Yes
DHT-42	625,030.28	4,549,260.05	307.35	Yes
DHT-43	624,567.69	4,549,318.96	447.48	Yes
DHT-44	624,983.05	4,549,220.05	328.73	Yes

Table 9-3: Drillholes utilised in the November 2012 MRE in ED50 format (con't)

Sondagem Drillhole	ED50/ UTM Easting	Northing	Elevation	Assay results returned and used in evaluation
DHT-45	624,746.76	4,549,957.97	272.15	Yes
DHT-46	624,873.37	4,549,380.73	317.89	Yes
DHT-47	624,535.57	4,549,497.76	402.14	Yes
DHT-48	624,689.55	4,549,280.98	420.55	Yes
DHT-49	624,301.13	4,550,196.10	326.20	Yes
DHT-50	624,281.59	4,550,218.36	326.21	Yes
DHT-51	624,550.87	4,549,481.77	406.83	Yes
DHT-52	624,721.85	4,549,410.24	377.49	Yes
DHT-53	624,310.80	4,550,167.77	325.71	Yes
DHT-54	624,816.75	4,549,414.10	332.27	Yes
DHT-55	624,520.27	4,549,373.38	443.75	Yes
DHT-56	624,314.80	4,550,138.15	325.10	Yes
DHT-57	624,744.00	4,549,420.05	363.09	No
DHT-58	624,635.99	4,549,363.99	417.84	Yes
DHT-59	624,755.00	4,549,328.01	385.52	Yes
DHT-60	624,757.67	4,549,331.73	384.99	Yes
DHT-62	624,790.68	4,549,243.66	402.59	Yes
DHT-63	624,867.73	4,549,310.70	354.27	Yes
DHT-64	624,963.73	4,549,338.99	304.57	Yes
DHT-65	624,933.59	4,549,292.00	332.43	Yes
DHT-66	624,734.13	4,549,446.51	361.81	Yes
DHT-67	624,703.20	4,549,439.02	379.21	Yes
DHT-68	624,592.80	4,549,290.17	455.07	Yes
DHT-69	624,683.99	4,549,377.57	398.96	Yes
DHT-70	624,233.12	4,550,223.43	331.67	Yes
DHT-71	624,165.10	4,550,281.79	322.01	Yes
DHT-72	624,527.94	4,549,690.92	357.14	Yes
DHT-73	624,450.25	4,549,801.87	358.96	Yes
DHT-74	624,309.07	4,550,251.28	304.90	Yes
DHT-75	624,462.68	4,549,648.80	375.84	Yes
DHT-76	624,333.75	4,550,104.75	328.53	Yes
DHT-77	624,456.75	4,549,706.61	370.74	Yes
DHT-78	624,531.84	4,549,609.87	370.15	Yes
DHT-79	624,222.92	4,550,307.89	291.52	Yes
DHMT-01	624,703.23	4,549,435.78	379.20	Yes
DHMT-02	624,728.79	4,549,394.67	378.59	No
DHMT-03	624,679.35	4,549,477.01	378.83	No
DHMT-31	624,677.97	4,549,478.73	378.72	No
DHMT-04	624,572.01	4,549,465.25	408.72	No
DHMT-41	624,573.89	4,549,466.10	408.68	No
DHMT-05	624,585.12	4,549,413.66	415.84	No
DHMT-51	624,584.39	4,549,415.86	415.75	No

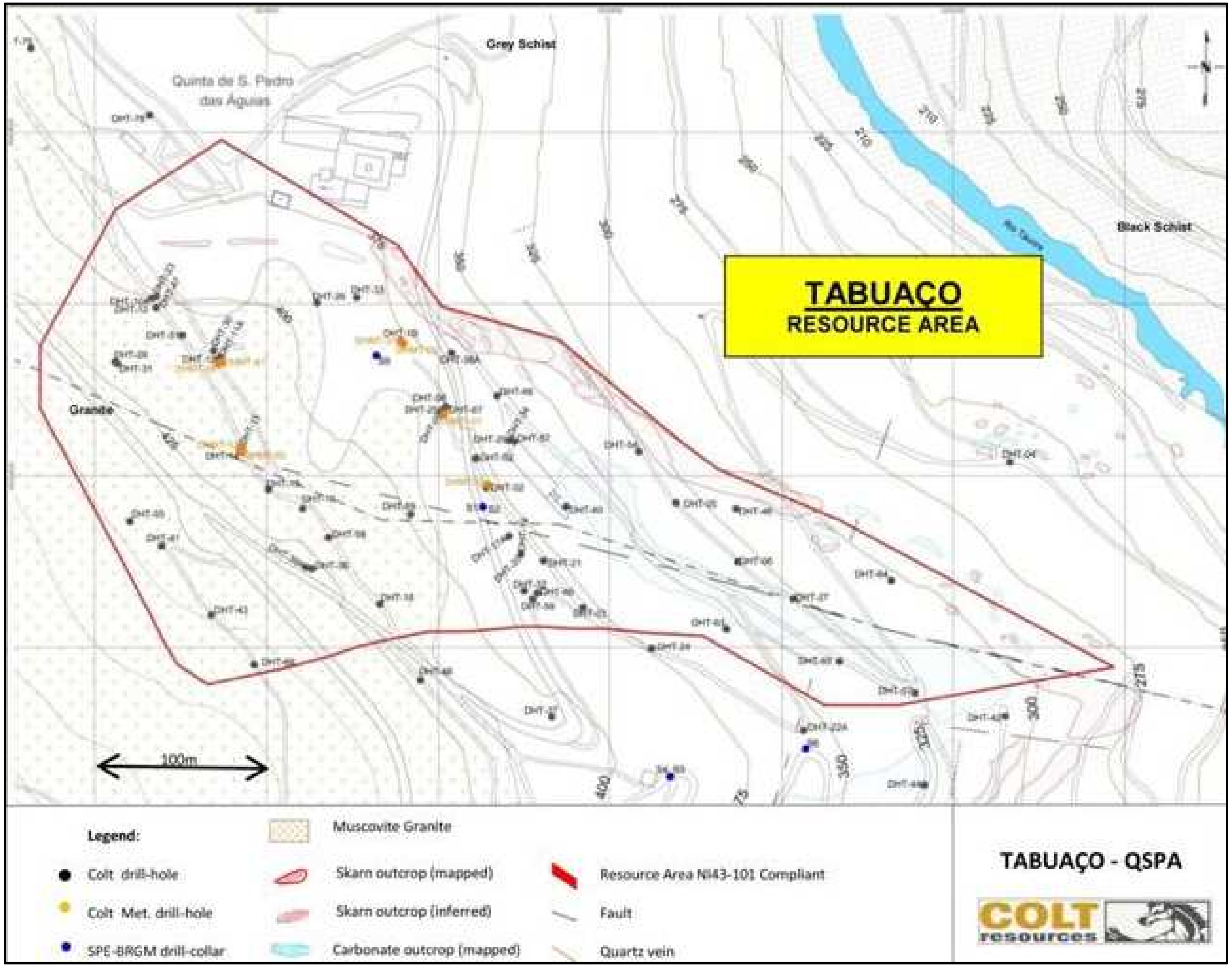


Figure 9-2: Location Map of Diamond Drilling at the Tabuaço Project (SPA)

9.3 Type and Extents

Due to the terrain and the extent of vineyard terracing, usually low impact access was required along with small, manoeuvrable rigs. Thus diamond drilling services have been provided by a number of drilling contractors but there has been a trade-off using the smaller rigs with lower power and slower penetration rates, particularly with inclined holes below 100 m depth.

Companies contracted for the previous round of drilling at Tabuaço include Laboratório Nacional de Energia e Geologia, I.P. ("LNEG"), Geoplano, and CGS Ingenierá, Betoteste and Geocontrole. Rigs were utilised during the project as follows:

- Betoteste rig#1 (Oxidrill 300): on site from 17 January to October 2012;
- Betoteste #2 (Schramm): Percussion rig on site from March 2012 to April 2012 to precollar metallurgical holes and one vertical evaluation hole;
- Betoteste rig #3 (Oxidrill 150): on site from 3rd of March to 17th of December 2012.
- LNEG rig #1 (Rieska): on site from 20th of June to 3rd of November 2011.
- LNEG rig #2 (Bonne Espérance): on site from 21st of November to 4th of May 2012.
- CGS drill rig (Acker): on site from 17 March 2012 to 20 April 2012 and again from 19 July to October 2012;
- Geoplano rig #1 (Mustang): on site from 17 January 2012 to 16 June 2012;
- Geoplano rig#2 (Apafor): on site from 17 January 2012 to 7 July 2012;
- Geoplano rig#3 (Apafor): on site from 7 March 2012 to 25 June 2012.
- Geocontrole rig #1 (Atlas Copco Mustang A32): from 28th of August 2012 to 31st of 24th of January 2013.
- Geocontrole rig# #2 (Atlas Copco Christensen CS10): from 12th of September to 23rd of November 2012.

The APAFOR™ rigs drill conventional HQ (63.5 mm diameter) vertical holes only, the remainder have wireline capability and are capable of drilling holes inclined to 45°. Evaluation holes were nominally drilled HQ reducing to NQ (47.6 mm diameter) as a last resort to advance the hole in difficult ground. Pre-collars for the metallurgical holes were drilled open hole percussion at 150 mm diameter. Metallurgical hole DHMT-01 was drilled PQ (76 mm) diameter, the remaining metallurgical holes were drilled at HQ diameter.

Geocontrole was contracted specifically to drill on the Aveleira for exploration and two geotechnical holes in QSPA in January 2013.

9.4 Procedures

Guidelines set out in Colt's standard operational procedures revised March 2012 were used by the field geologists. This set of guidelines governs all aspects of its field programs from sampling methodology through to how it processes its data. This is managed on 3 levels:

- Database Management;
- Organizational Management; and
- Operational Management.

Drill pads were levelled and prepared using a 1 t mechanical excavator with a front end blade to dig sump pits for circulation of water, level pads and cut slots into terraces where necessary. Collars were marked with a stake and flagging tape with 2 to 3 more pickets all aligned along the azimuth of the proposed hole for inclined holes. The azimuth was sighted using a Suunto TM Tandem compass or Silva TM mod 15 and fined-tuned by the geologist during set up of the rig. Twin holes for the metallurgical programme were collared within 3 m of the conjugate hole. Once the rig was set up a 20 m^2 zone was flagged off using danger tape and the site designated an industrial zone according to CE standards.

Drill core was delivered from the core barrel to a 3 m metal v-tray at the end of each run and the core transferred to a wooden core box by the geologist. A daily check on the condition of the rig and support vehicles was made and recorded according to the H&S policy. Driller sheets in duplicate were collated from the previous day's shift and the drillers' records (stick up books) checked so as to correlate with the plod. Regular checks in the field made by geologists to ensure core boxes were correctly labelled and v-trays washed to prevent cross-contamination. On completion of the hole and downhole survey the collar was capped with a ½ m^2 cement plug with the borehole stamped with its ID number.

Processing of drill core was carried out at Colt's core storage facility in Távora according to the flowsheet shown in Figure 9-3 where applicable.

Sample intervals were nominally 1 m each with sample lengths adjusted to break at lithologic boundaries from a minimum of 0.5 m to a maximum of 2.0 m in visually barren intervals in intermediate schist. Visually mineralized skarn units of the MZ and LZ were sampled including intermediate schist between mineralised skarn and 10 m into the FW and HW. For the twin holes samples were taken as close to the original sample interval from the historic hole as possible adjusted to variations in lithological boundaries. Semi-quantitative determination of scheelite content was carried out by examination under short-wave UV-light. Geotechnical logging was completed by the geologist at the time of completing the geological logging. The core was photographed under artificial lighting, using a camera cradle ensuring each photo was taken under identical settings.

The core was marked for sampling down the centre line of the core intersecting the axial trough of mineralisation. The core was then cut by diamond saw. One half of the core was sent for analysis, while the other half was retained in the core boxes for future reference. Where additional sample was required for reference assays for metallurgical testwork the core was quartered ensuring minimal bias of mineralisation in the sampled quarter. Core was stacked in Távora, later racked at a specially prepared storage facility at the site office at QSPA.

The core samples for analysis were packed into sealed cardboard boxes at the storage facility, from where they were collected and transported by a courier operator (TNT) to the ALS Laboratory in Seville, Spain. Samples were analysed for W, Sn and major oxides using a metaborate fusion followed by XRF.

A set of standards, duplicates and blanks was inserted by Colt into the sample stream on a regular basis in addition to the laboratory's own internal QAQC standards and duplicates. The standards inserted into the sampling stream were certified standards, produced by WCM Minerals of Canada. The Sample Certificates for standards W104, W105, W106, W107 and W108 are shown in the Appendix of the November 2012 NI 43-101 Report.

All drillholes were surveyed at the collar surface by the responsible geologist, using a handheld GPS. Data for Eastings, Northings and RL was initially recorded in UTM 29T, Datum ED50 and later in UTM29N, Datum WGS84.

A high resolution topographic survey was conducted in 2010 by survey contractors, Superfície, leading to a 2 m resolution digital terrain model. Holes DHT01–DHT09 were also accurately surveyed during this work.

The topographic survey was updated in August 2012 by survey contractors Superfície. The DTM was updated using photogrammetric methods and remote sensing imagery. Drillhole collars for the 2012 campaign were captured using a portable DGPS using a central radio beacon for reference. Data was presented by Superfície in both ED50 and WGS84 format which is currently being processed and verified by Colt before converting the database from ED50 to WGS84 datum. The updated coordinates in ED50 datum incorporating this survey were used in the November 2012 MRE.

The majority of drillholes have been subject to downhole surveying, to record variations from the original inclination. Surveys have been recorded at varying intervals, depending on the operator and original inclination of the drillhole and the depth drilled.

Data entry and capture was made on Excel spreadsheets with information summarised at the end of the week. Data was transferred to Colt's main office at Beloura from site for further processing.

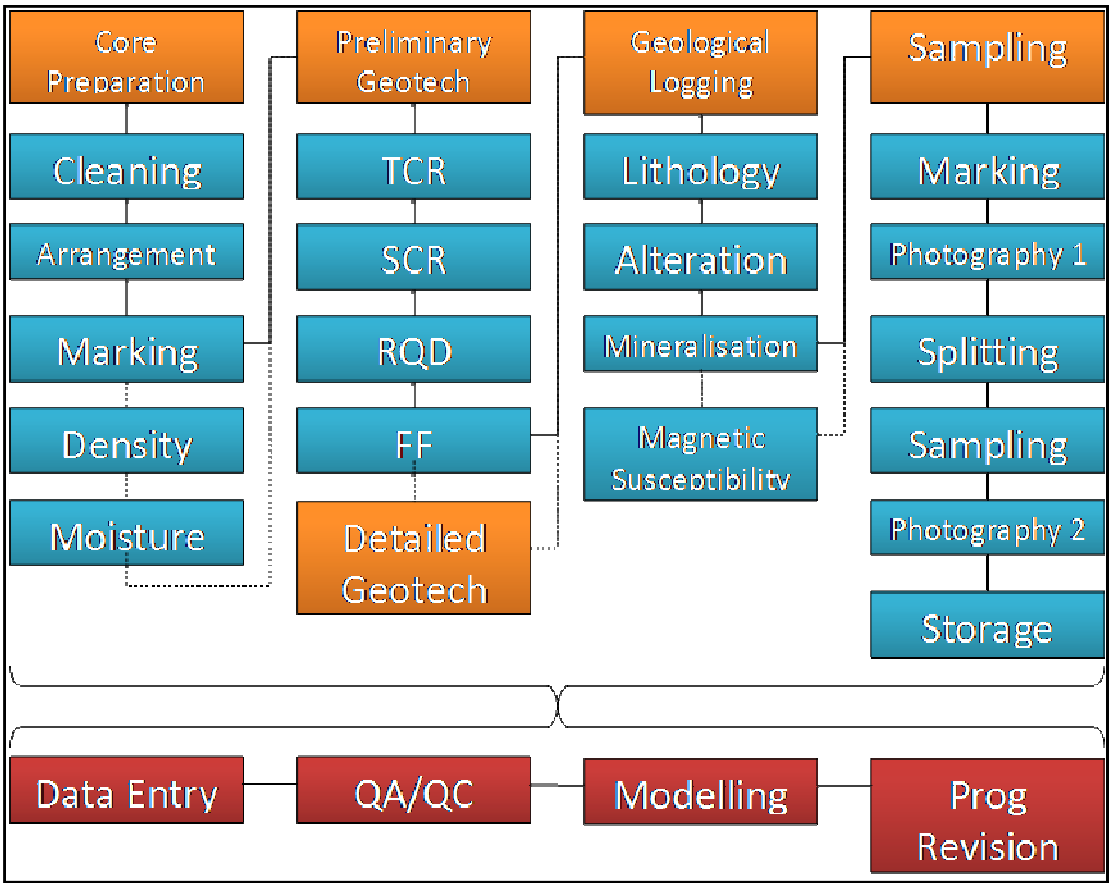


Figure 9-3: Conceptual Flowsheet for Logging of Diamond Core

9.5 Interpretation and Relevant Results

For the work conducted by COLT during the period 2009 to 2012, reputable contractors, using industry standard techniques and procedures have conducted the drilling. Results are interpreted to follow three principle tabular zones of skarn, namely the Carbonate, Main and Lower Horizons dipping gently to the south-west. The drill intercepts have been adjusted to true widths assuming an average strike of 315° and average dip -20° based on the average dip and strike of the mineralised zones at Tabuaço.

Table 9-4 and Table 9-5 display significant drillhole intercepts from the period 2009 to 2012.

Due to locally steep topography and occasional access restrictions, the diamond drillholes are arranged in a variety of orientations with the intention of intersecting the mineralised zones at approximate 50 m centres. A total of 62 assayed drillholes were used for the November 2012 MRE, with an average recovery of 86 %. 74% of the total samples used have recovery equal or greater than 80 %.

Table 9-4: Table of Significant Drill Intersections at Tabuaço

Hole	Inclination	From (m)	To (m)	% WO_3	Interval (m)	True Width (m)*
DHT-01B	Vertical	7.10	11.85	0.52	4.75	4.46
		19.15	37.95	0.73	18.8	17.66
		Including				
		24.18	**27.18**	**1.14**	**3.00**	**2.82**
		And Including				
		29.18	**35.50**	**0.99**	**6.32**	**5.94**
DHT-02	Vertical	52.60	66.20	0.93	13.60	12.78
		Including				
		57.95	**62.95**	**1.44**	**5.00**	**4.70**
DHT -04	Vertical	12.68	13.25	0.93	0.57	0.54
DHT- 05	Vertical	16.95	26.65	0.38	9.70	9.12
		Including				
		16.95	**19.45**	**1.25**	**2.50**	**2.35**
DHT-06	Vertical	12.64	17.82	0.34	5.18	4.87
DHT -08	Vertical	42.40	54.40	0.60	12.00	11.28
		Including				
		42.40	**47.75**	**1.09**	**5.35**	**5.03**
DHT- 09	-45°to 210°	93.60	115.20	0.54	21.60	20.30
		Including				
		93.60	**96.00**	**1.11**	**2.40**	**2.26**
		And Including				
		99.00	**104.00**	**0.88**	**5.00**	**4.70**
DHT-11A	-55°to 031°	46.55	52.55	0.77	6.00	5.64
		70.90	72.25	0.62	1.35	1.27
DHT-12	Vertical	52.20	67.20	0.89	15.00	14.09
		Including				
		59.20	**66.20**	**1.64**	**7.00**	**6.58**
DHT-13	-50°to 030°	69.35	72.00	0.49	2.65	
		92.80	100.45	1.08	7.65	
DHT-14	Vertical	77.30	85.65	1.29	8.35	7.84
		Including				
		79.80	**83.8**	**1.90**	**4.00**	**3.76**
		116.50	125.00	0.43	8.50	7.99
DHT-15	-60°to 055°	108.35	122.55	0.89	14.2	13.95
		Including				
		109.35	**116.50**	**1.42**	**7.15**	**7.02**
DHT-16	Vertical	116.80	122.40	0.57	5.60	5.26
DHT-19	-45 to 030°	78.10	85.10	0.95	7.00	6.29
		90.80	98.80	0.38	8.00	7.18
DHT-20	-45°to 042°	96.50	103.40	0.78	6.90	6.25
DHT-23	-45°to 030°	24.80	28.10	1.33	3.30	2.96
		33.70	44.10	0.51	10.40	9.34
DHT-25	-65°to 210°	53.78	64.62	0.95	10.84	10.75
		Including				
		57.70	**61.78**	**1.38**	**4.08**	**4.04**

Table 9-5: Table of Significant Drill Intersections at Tabuaço (con't)

Hole	Inclination	From (m)	To (m)	% WO_3	Interval (m)	True Width (m)*
DHT-26	Vertical	14.10	27.50	0.76	13.40	12.59
		Including				
		14.10	**17.65**	**1.37**	**3.55**	**3.34**
		33.10	40.70	0.47	7.60	7.14
		44.45	54.45	0.41	10.00	9.40
DHT-29	Vertical	46.05	56.25	0.53	10.20	9.58
DHT-30	-55° to 030°	49.80	52.05	0.78	2.25	2.16
DHT-31	-60° to 126°	121.60	134.60	0.29	13.00	10.93
DHT-33	-50° to 330°	6.50	10.25	0.64	3.75	2.49
		18.20	24.40	0.84	6.20	4.11
		Including				
		19.20	**22.20**	**1.16**	**3.00**	**1.99**
DHT-34	-55° to 030°	30.70	35.10	0.89	4.40	4.22
		42.80	47.80	0.55	5.00	4.80
DHT-36	Vertical	65.25	71.25	0.31	6.00	5.64
		124.33	131.10	0.52	6.77	6.36
DHT-37	Vertical	79.10	80.10	0.54	1.00	0.94
DHT-38A	Vertical	19.95	25.40	0.46	5.45	5.12
DHT-39A	-73° to 300°	125.45	132.90	0.38	7.45	6.88
		137.50	140.80	0.42	3.30	3.05
DHT-40	Vertical	84.70	88.05	0.22	3.35	3.15
DHT-41	Vertical	155.00	156.00	0.58	1.00	0.94
DHT-42	Vertical	10.45	11.50	0.11	1.05	0.99
DHT-44	Vertical	23.60	25.20	0.24	1.60	1.51
DHT-46	Vertical	13.25	16.40	0.73	3.15	2.96
		21.40	24.40	0.45	3.00	2.82
DHT-51	Vertical	60.35	66.35	0.99	6.00	5.64
		73.70	79.70	0.23	6.00	5.64
DHT-52	Vertical	54.20	61.20	1.31	7.00	6.58
DHT-54	Vertical	17.12	19.15	0.69	2.03	1.91
DHT-58	-70° to 030°	137.35	139.65	0.45	2.30	2.16
DHT-63	Vertical	3.00	9.00	0.42	6.00	5.64
		18.60	21.20	0.49	2.60	2.44
		30.90	32.90	0.47	2.00	1.88
DHT-64	Vertical	11.88	13.85	0.33	1.97	1.85

10 SAMPLE PREPARATION, ANALYSIS AND SECURITY

Samples from the 2012 programme were shipped to the principal preparation facility of ALS in Seville, Spain and then forwarded for analysis in Kamloops, Canada. Samples used in previous Colt campaigns were also prepared and analysed at Stewart Group's OMAC laboratory in Loughrea, Ireland. A third external laboratory, SGS Lakefield in Canada was commissioned in September 2012 to check pulp rejects from both laboratories.

Processing of drill core was carried out at Colt's core storage facility in Távora according to the flowsheet shown in Figure 9-3 where applicable.

Sample intervals were nominally 1 m each with sample lengths adjusted to break at lithologic boundaries from a minimum of 0.5 m to a maximum of 2.0 m in visually barren intervals in intermediate schist. Visually mineralised skarn units of the MZ and LZ were sampled including intermediate schist between mineralised skarn and 10 m into the footwall and hangingwall. For the twin holes samples were taken as close to the original sample interval from the historic hole as possible adjusted to variations in lithological boundaries.

Semi-quantitative determination of scheelite content was carried out by examination under short-wave UV-light. Geotechnical logging is completed by the geologist at the time of completing the geological logging. The core is photographed under artificial lighting using a camera cradle ensuring each photo is taken under identical settings.

A line was marked down the centre of the core intersecting the axial trough of mineralisation. The core is then cut by diamond saw along the line, splitting the core into two equal halves. One half of the core is sent for analysis, while the other half is retained in the core boxes for future reference. Where additional sample was required for reference assays for metallurgical testwork the core was quartered ensuring minimal bias of mineralisation in the sampled quarter. Core was stacked in Távora, later racked at a specially prepared storage facility at the site office at QSPA.

The core samples for analysis are packed into sealed cardboard boxes at the storage facility, from where they are collected and transported by a courier operator (TNT) to the ALS Laboratory in Seville, Spain. Samples are analysed for W,Sn and major oxides using a meta-borate fusion followed by XRF.

A set of standards, duplicates and blanks is inserted by Colt into the sample stream on a regular basis in addition to the laboratory's own internal QAQC standards and duplicates. The standards inserted into the sampling stream are certified standards, produced by WCM Minerals of Canada. The Sample Certificates for standards W104, W105, W106, W107 and W108 are shown in the Appendix of the November 2012 NI 43-101 Report.

10.1 Sample Preparation

At the ALS Laboratory in Spain, samples were prepared using the following process:

- WEI-21 – Received sample weight;
- LOG-22 – Sample login, record w/o barcode;
- CRU-31 – Fine crushing to better than 70% <2mm. Includes CRU-QC crushing efficiency test;
- PUL-32 Up to 1 kg sample split is pulverised to better than 85% 75 µm. Includes PUL_QC pulverising test; and
- LOG-24 – Pulp Login, record w/o barcode.

10.2 Analyses

Three laboratory methods were used to analyse the Tabuaço project samples:

- ICP-AES - Aqua Regia (AR/ES) and Multi Acid Digestion (MA/ES) by OMAC;
- ICP-MS (ME-MS61) – 48 element, four acid digestion of sample followed by Inductively Coupled Plasma Mass Spectrometry by ALS; and
- Fusion XRF using lithium borate disc for ore grade samples >0.2% W (ME-XRF 10 by ALS – BF ES/MS Major and Trace elements by OMAC).

Only Fusion XRF for major oxide content and minor elements was requested of SGS, method XRF78S.

10.3 Security Measures

Colt was responsible for managing the security of the entire sampling process, with the Project Geologists supervising all selection of sampling intervals, the cutting process and packing ready for shipment to ALS laboratories in sealed packages. The office and storage facility at QSPA is fully alarmed with proximity, movement, magnetic and photographic sensors. Samples bags were sealed with single use plastic cable ties to prevent tampering.

Colt used TNT couriers to provide a secure logistical train, with sample batches tracked from dispatch from the Colt core sampling facility at Távora to arrival and signed for receipt at ALS.

Samples were recorded using Colt's unique ID numbers, and a separate ID number used by the supplied laboratory submission sheet.

Assay certificates were directly issued by the laboratories to Colt Management, and to SRK. Sample tracking and downloadable certificates were also available online using ALS Webtrieve facility accessed through secure login.

10.4 Laboratories

The original laboratory used for the 2012 programme was the Stewart Group's OMAC laboratory in Loughrea, Ireland. ALS in Seville was originally utilised as a second laboratory to assist with QAQC and to accommodate the large number of samples when drilling proceeded more quickly but was then used as the main laboratory instead of OMAC. SGS has been commissioned to perform external checks on 20% of ALS and OMAC samples combined along with the requisite insertion of standards and blanks at 1 per 50 sample intervals.

All three laboratories are certified and considered to operate to internationally acceptable standards.

10.5 Results and QC Procedures

ALS Laboratory internal quality controls are based on internationally recognized (EN, ISO, US, EPA, ASTM, CEM, NIOSH, AOAC) standards and all ALS analytical procedures are fully validated and accredited to EN ISO/IEC 17025:2005 and ISO 9001:2008 registration. Under this accreditation, a systematic process of quality control is established including: regular calibration of instruments and independent verification of calibration; regular measurements of blank, laboratory control and laboratory duplicate samples. Details on ALS internal QAQC procedures, proficiency testing, independent QA monitoring and internal auditing is presented on its website[1].

[1] ALS Global. *Inspection Downloads*. [online] Available at: <http://www.alsglobal.com/en/Our-Services/Minerals/Inspection/Downloads>

Laboratory results were issued by both OMAC and ALS laboratories to the client in form of pdf certificates, with accompanying excel or .csv files for insertion into the database. Copies of these datasets were forwarded directly onto SRK for QAQC analysis and independent verification of the excel files against the laboratory certificates.

Results for tungsten were delivered by OMAC and ALS laboratories from the XRF analysis and provided to the Colt as W% assays.

The W results have been converted to WO_3% using the conversion factor of 1.2611.

10.5.1 QAQC

Colt have inserted Certified Reference Materials ("SRM"). Prior to 2012, Field Standards W104, W105, W106 and W108 were used, as well as duplicates and blanks into the sampling stream on the basis of 1 per 15 samples at random. For the 2012 drill programme Field Standards W104, W105, W106 and W107 and blanks were used with the same frequency. Standards were prepared by Colt's geologists by measuring 50 g from a central container containing homogenized standard.

Copies of the SRM Certificates are presented in the Appendix of the November 2012 NI 43-101 Report.

SRK have conducted QAQC reviews on the returned laboratory assay data. This involves checking the standards, blanks and duplicates inserted by COLT into the sampling stream, and checking the reported standards, blanks and duplicates inserted into the assaying programme by the laboratory.

The XRF data supplied by OMAC and ALS to SRK has passed the QAQC checking process and was therefore used for the MRE.

10.6 Certified Reference Materials - Field Standards – XRF

10.6.1 W104

Field standard W104 (0.202% WO_3) was used 13 times by ALS during the 2012 sampling campaign and 22 times overall. Results from the 2012 campaign were all within the 2 standard deviations (SD) warning limit and showed greater precision than the previous sampling campaign (Figure 10-1, Table 10-1 and Table 10-2).

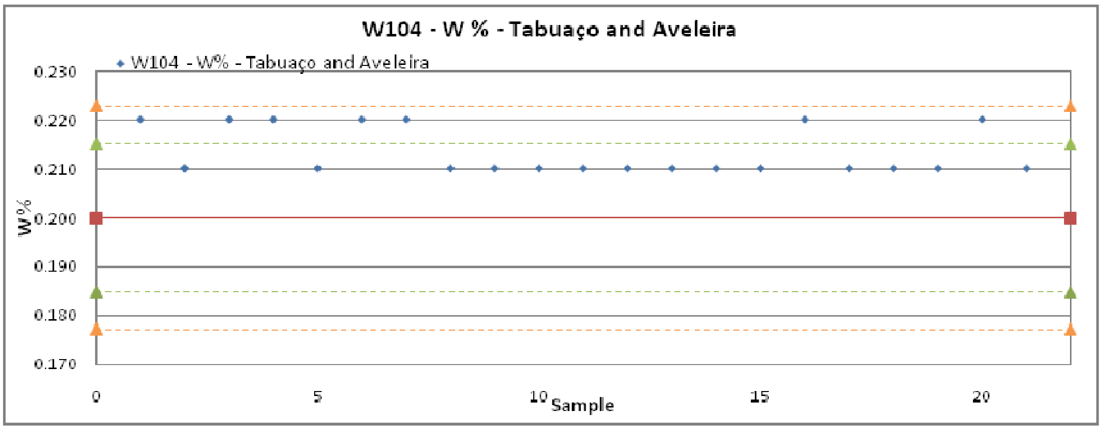


Figure 10-1: Results from Field Standard W104 for W%

Table 10-1: Data for SRM W104

Hole ID	Standard sample			Colt Sample	Laboratory		Lab W %	Relative Difference %	Lab WO_3 %	Relative Difference %
		W	WO3		Certificate	Date				
DHT-24	W104	0.2	0.26	CR-102832	SV11171973	26-09-2011	0.220	10.0	0.277	6.62
DHT-26	W104	0.2	0.26	CR-102865	SV11171973	26-09-2011	0.210	5.0	0.265	1.77
DHT-25	W104	0.2	0.26	CR-102952	SV11179423	01-10-2011	0.220	10.0	0.277	6.62
DHT-33	W104	0.2	0.26	CR-102711	SV11179679	06-10-2011	0.220	10.0	0.277	6.62
DHT-39A	W104	0.2	0.26	CR-104162	SV11222099	26-11-2011	0.210	5.0	0.277	6.62
DHT-37	W104	0.2	0.26	CR-104250	SV11224424	05-12-2011	0.220	10.0	0.265	1.77
DHT-41	W104	0.2	0.26	CR-104334	SV11243934	19-12-2011	0.220	10.0	0.277	6.62
DHT-42	W104	0.2	0.26	CR-104396	SV11250340	27-12-2011	0.210	5.0	0.265	1.77
DHT-55	W104	0.2	0.26	CR-103949	SV12088554	16-05-2012	0.210	0.005	0.265	1.769
DHT-59	W104	0.2	0.26	CR-104609	SV12109936	30-05-2012	0.210	5.00	0.26	1.769
DHT-59	W104	0.2	0.26	CR-104577	SV12109936	30-05-2012	0.210	5.00	0.26	1.769
DHT-63	W104	0.2	0.26	CR-104657	SV12119588	26-06-2012	0.210	5.00	0.26	1.769
DHT-65	W104	0.2	0.26	CR-104731	SV12141368	06-07-2012	0.21	5.00	0.26	1.769
DHT-69	W104	0.2	0.26	CR-104776	SV12154265	20-07-2012	0.2100	5.00	0.26	1.858
DHT-67	W104	0.2	0.260	CR-105001	SV12167074	07-08-2012	0.21	5.00	0.265	1.858
DHT-71	W104	0.2	0.260	CR-105080	SV12182796	29-08-2012	0.22	10.00	0.277	6.708
DHT-75	W104	0.2	0.260	CR-105243	SV12197178	03-09-2012	0.21	5.00	0.265	1.858
DHT-74	W104	0.2	0.260	CR-105259	SV12197177	04-09-2012	0.21	5.00	0.265	1.858
DHT-58	W104	0.2	0.260	CR-105145	SV12195070	04-09-2012	0.21	5.00	0.265	1.858
DHT-58	W104	0.2	0.260	CR-105209	SV12195070	04-09-2012	0.22	10.00	0.277	6.708
DHT-57	W104	0.2	0.260	CR-105358	SV12204295	14-09-2012	0.21	5.00	0.265	1.858

Table 10-2: Specifications for SRM W104

SRM	Mean	Standard Deviation	Lines		
				Warning (2σ)	Action (3σ)
W104 - W%	0.200	0.0076	Max	0.2152	0.2228
			Min	0.1848	0.1772

10.6.2 W105

Field standard W105 (1.40% WO_3) was used 19 times during Colt's 2012 drilling and sampling programme. All the results demonstrated an acceptable level of accuracy and precision (Figure 10-2, Table 10-3 and Table 10-4).

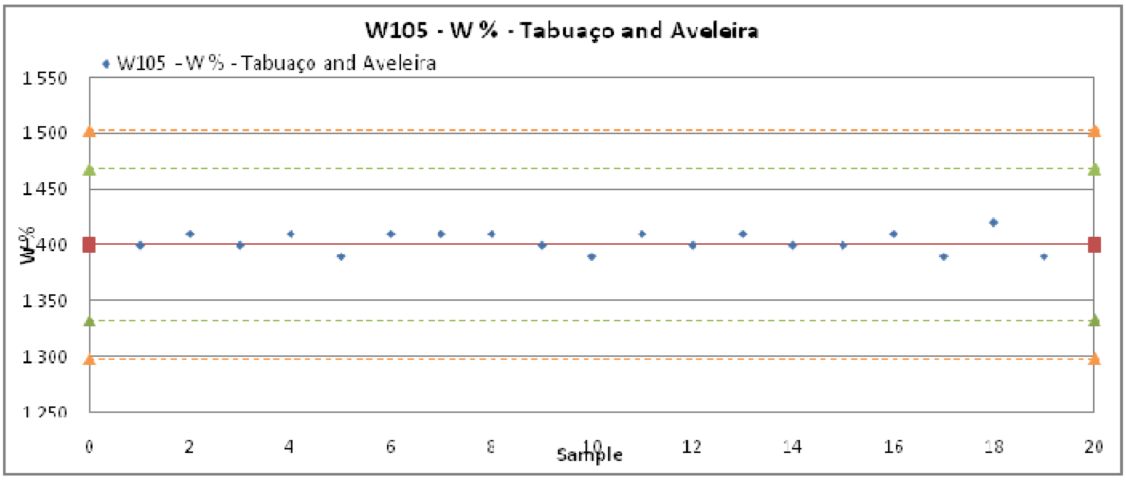


Figure 10-2: Results from field Standard W105 for W%

Table 10-3: Data for SRM W105

Hole ID	Standard sample			Colt Sample	Laboratory		Lab W %	Relative Difference %	Lab WO_3 %	Relative Difference %
		W	WO3		Certificate	Date				
DHT-46	W105	1.4	1.76	CR-103516	SV12019012	15-02-2012	1.400	0.00	1.764	0.227
DHT-43	W105	1.4	1.76	CR-104437	SV12022776	26-02-2012	1.410	0.71	1.777	0.943
DHT-48	W105	1.4	1.76	CR-103611	SV12041814	16-03-2012	1.400	0.00	1.751	-0.489
DHT-49	W105	1.4	1.76	CR-103596	SV12035481	19-03-2012	1.410	0.71	1.777	0.943
DHT-50	W105	1.4	1.76	CR-103664	SV12055358	05-04-2012	1.390	-0.71	1.764	0.227
DHT-52	W105	1.4	1.76	CR-103708	SV12060740	10-04-2012	1.410	0.71	1.777	0.943
DHT-51	W105	1.4	1.76	CR-103858	SV12081330	09-05-2012	1.41	0.714	1.78	0.943
DHT-55	W105	1.4	1.76	CR-103917	SV12088554	16-05-2012	1.410	0.714	1.78	0.943
DHT-56	W105	1.4	1.76	CR-103974	SV12092424	21-05-2012	1.400	0.000	1.76	0.227
DHT-64	W105	1.4	1.76	CR-104534	SV12106021	28-05-2013	1.390	-0.714	1.75	-0.489
DHT-63	W105	1.4	1.76	CR-104625	SV12119588	26-06-2012	1.410	0.71	1.78	0.943
DHT-65	W105	1.4	1.76	CR-104760	SV12141368	06-07-2012	1.4	0.00	1.76	0.227
DHT-69	W105	1.4	1.76	CR-104808	SV12154265	20-07-2012	1.4000	0.00	1.77	0.315
DHT-68	W105	1.4	1.77	CR-104841	SV12160884	23-07-2012	1.40	0.00	1.77	0.00
DHT-72	W105	1.4	1.766	CR-105064	SV12169162	06-08-2012	1.41	0.71	1.778	0.714
DHT-66	W105	1.4	1.766	CR-104930	SV12164801	06-08-2012	1.39	-0.71	1.753	-0.714
DHT-67	W105	1.4	1.766	CR-105033	SV12167074	07-08-2012	1.42	1.43	1.791	1.429
DHT-71	W105	1.4	1.766	CR-105112	SV12182796	29-08-2012	1.39	-0.71	1.753	-0.714

Table 10-4: Specifications for SRM W105

SRM	Mean	Standard Deviation	Lines		
				Warning (2σ)	Action (3σ)
W105 - W%	1.400	0.034	Max	1.4682	1.5023
			Min	1.3318	1.2977

10.6.3 W106

Standard W106 was used once in 2012 and nine times during the 2011 drilling campaign. Results present an acceptable precision and accuracy (Figure 10-3, Table 10-5 and Table 10-6).

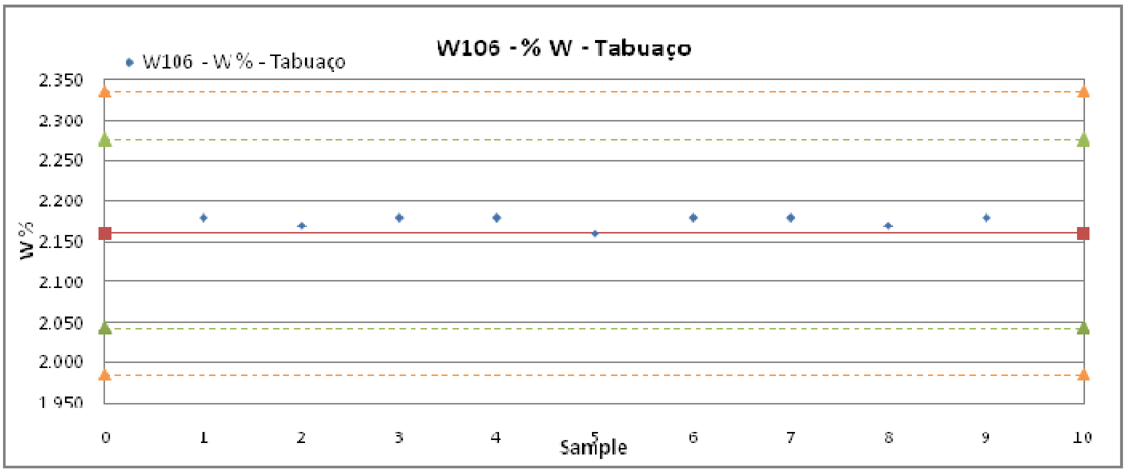


Figure 10-3: Results from field Standard W106 for W%

Table 10-5: Data for SRM W106

Hole_ID	Standard sample			Colt Sample	Laboratory		Lab W %	Relative Difference %	Lab WO_3 %	Relative Difference %
		W	WO3		Certificate	Date				
DHT-28	W106	2.16	2.73	CR-102936	SV11179423	01-10-2011	2.180	0.93	2.747	0.615
DHT-29	W106	2.16	2.73	CR-102990	SV11179423	01-10-2011	2.170	0.46	2.734	0.154
DHT-31	W106	2.16	2.73	CR-102722	SV11189354	18-10-2011	2.180	0.93	2.747	0.615
DHT-32	W106	2.16	2.73	CR-104030	SV11199029	27-10-2011	2.180	0.93	2.747	0.615
DHT-36	W106	2.16	2.73	CR-104139	SV11213274	14-11-2011	2.160	0.00	2.747	0.615
DHT-40	W106	2.16	2.73	CR-104213	SV11224424	05-12-2011	2.180	0.93	2.747	0.615
DHT-38A	W106	2.16	2.73	CR-104324	SV11237133	09-12-2011	2.180	0.93	2.747	0.615
DHT-41	W106	2.16	2.73	CR-104367	SV11243934	19-12-2011	2.170	0.46	2.734	0.154
DHT-44	W106	2.16	2.73	CR-104426	SV11261237	11-01-2012	2.180	0.93	2.722	-0.308

Table 10-6: Specifications for SRM W106

SRM	Mean	Standard Deviation	Lines	Warning (2σ)	Action (3σ)
W106 - W%	2.160	0.058	Max	2.2766	2.3349
			Min	2.0434	1.9851

10.6.4 W107

Certified reference material W107 was only used 8 times during 2012, between January and May. Results present an acceptable level of confidence (Figure 10-4, Table 10-7 and Table 10-8). Figure 10-9 shows the certified values and the average value for each SRM from the laboratory results. The unified straight line shows that there was no bias in the analysis.

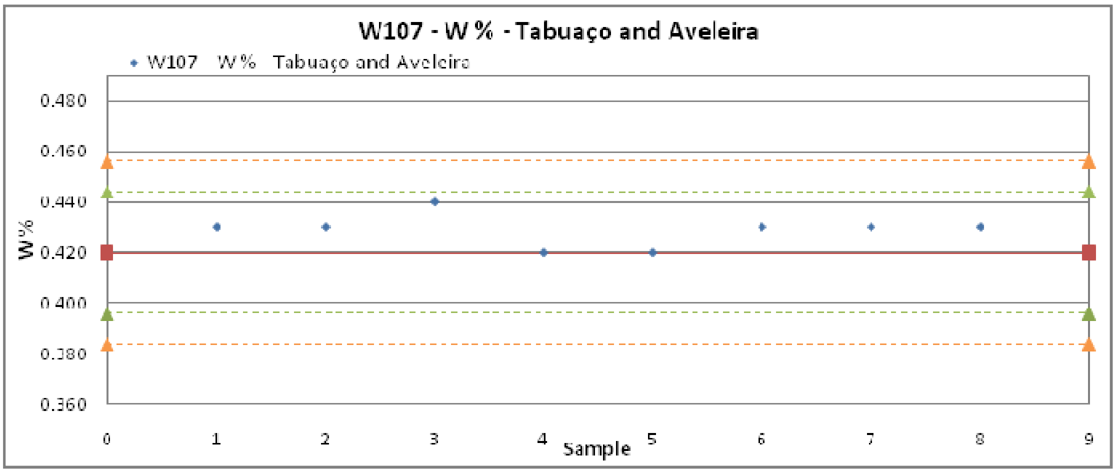


Figure 10-4: Results from field Standard W107 for W%

Table 10-7: Data for SRM W107

Hole_ID	Standard sample			Colt Sample	Laboratory		Lab W %	Relative Difference %	Lab WO_3 %	Relative Difference %
		W	WO3		Certificate	Date				
DHT-46	W107	0.420	0.530	CR-103530	SV12019012	15-02-2012	0.430	2.38	0.542	2.226
DHT-47	W107	0.420	0.530	CR-103563	SV12025435	28-02-2012	0.430	2.38	0.542	2.226
DHT-49	W107	0.420	0.530	CR-103580	SV12035481	19-03-2012	0.440	4.76	0.554	4.604
DHT-53	W107	0.420	0.530	CR-103767	SV12066521	19-04-2012	0.420	0.00	0.529	-0.151
DHMT-01	W107	0.420	0.530	CR-103733	SV12066521	19-04-2012	0.420	0.00	0.529	-0.151
DHT-54	W107	0.420	0.530	CR-103786	SV12073371	26-04-2012	0.430	2.38	0.542	2.226
DHT-51	W107	0.420	0.530	SV12081330	CR-103826	09-05-2012	0.430	2.38	0.542	2.226
DHT-51	W107	0.420	0.530	SV12081330	CR-103890	09-05-2012	0.430	2.38	0.542	2.226

Table 10-8: Specifications for SRM W107

SRM	Mean	Standard Deviation	Lines		
				Warning (2σ)	Action (3σ)
W107 - W%	0.420	0.012	Max	0.444	0.456
			Min	0.396	0.384

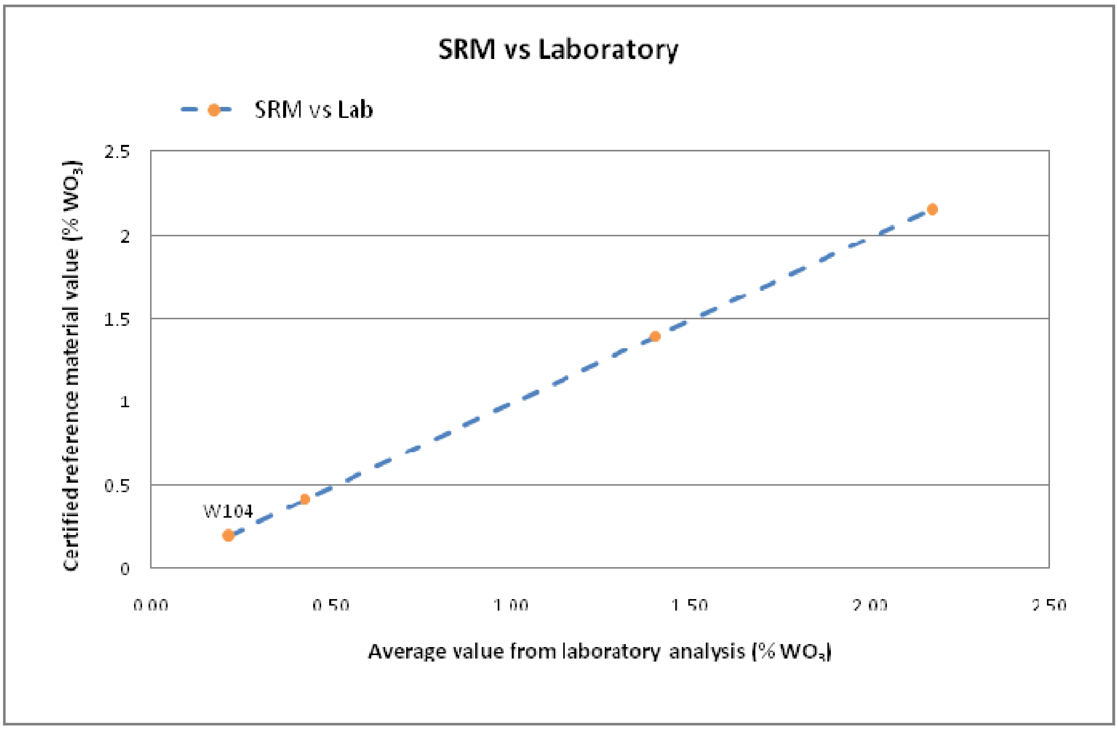


Figure 10-9: Certified Values vs the Average Value from the Several Laboratory Results for each SRM

10.7 Certified Reference Materials - Field Standards - ICP

The SRM samples reported above for XRF analysis were also tested by ICP-AES by ALS. However, given that the QAQC results for ICP-AES reported in December 2011 by SRK indicated that the ICP method is not suitable for high grade tungsten assays >0.2% WO_3 at Tabuaço (due to incomplete acid digestion), the ICP-AES results have not been reported here and have not been used in preparing the November 2012 MRE. Colt continues to compare these results with the XRF results on a regular basis to ensure integrity of sample preparation or calibration of the XRF equipment.

10.8 Laboratory QA/QC Review

Colt monitors and analyses the outcomes of internal QAQC of each laboratory that Colt employs. Assay results, from the laboratories, are treated and analysed with the same techniques, used in Colt's QAQC program. The procedure and results are shown in Figure 10-5 to Figure 10-8.

Both OMAC and ALS laboratories inserted their own quality control standards into the assay stream, in the form of repetition, or repeat, analysis and the use of blanks and standards:

- OMAC used a one in ten repeat frequency for repeats, with two in-house standards used in each batch and two blanks inserted into each batch; and
- ALS used a more random insertion of duplicates, roughly every 20 samples, with blanks and in-house standards also part of the testing.

SRK ran QAQC analysis on the provided laboratory standard assay results. These refer to supplied laboratory inserts provided by OMAC and ALS laboratories for the 2011 series samples.

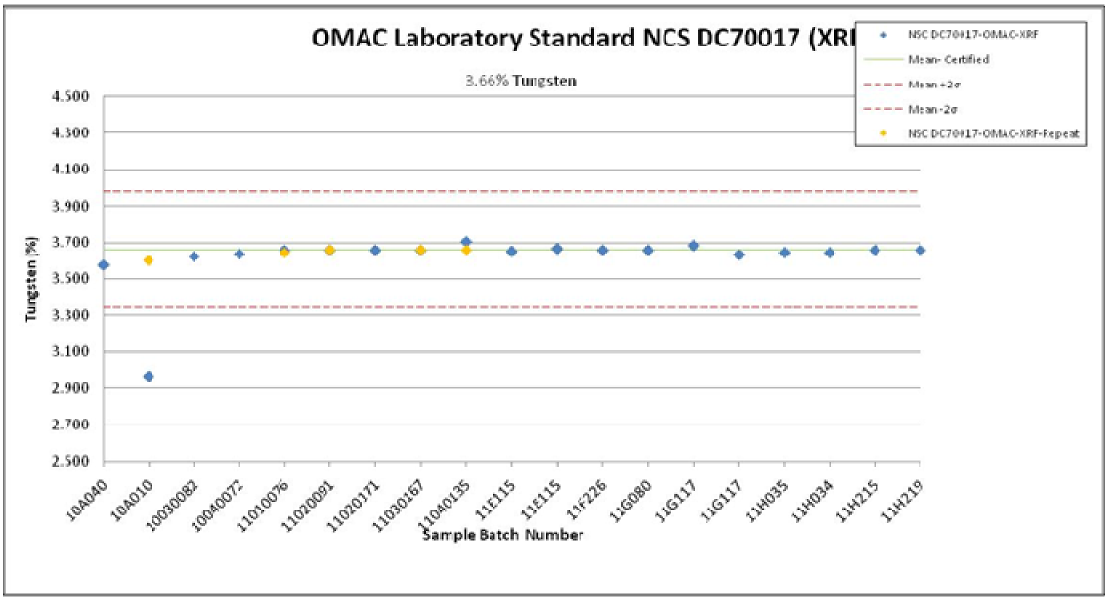


Figure 10-5: Laboratory Standards QA/QC - OMAC BF/EL

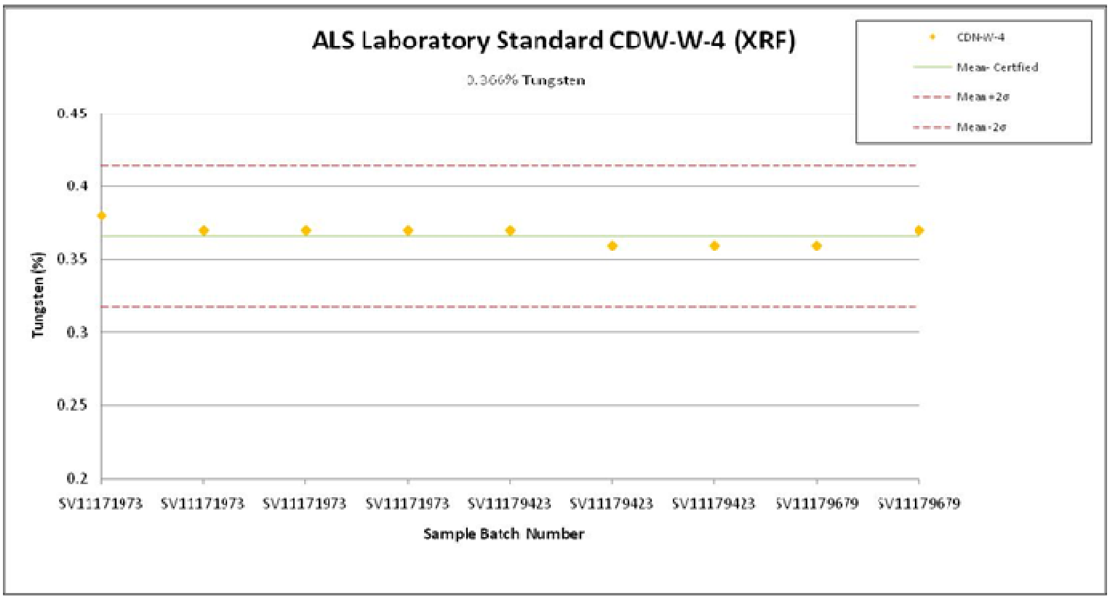


Figure 10-6: Laboratory Standards QA/QC - ALS XRF

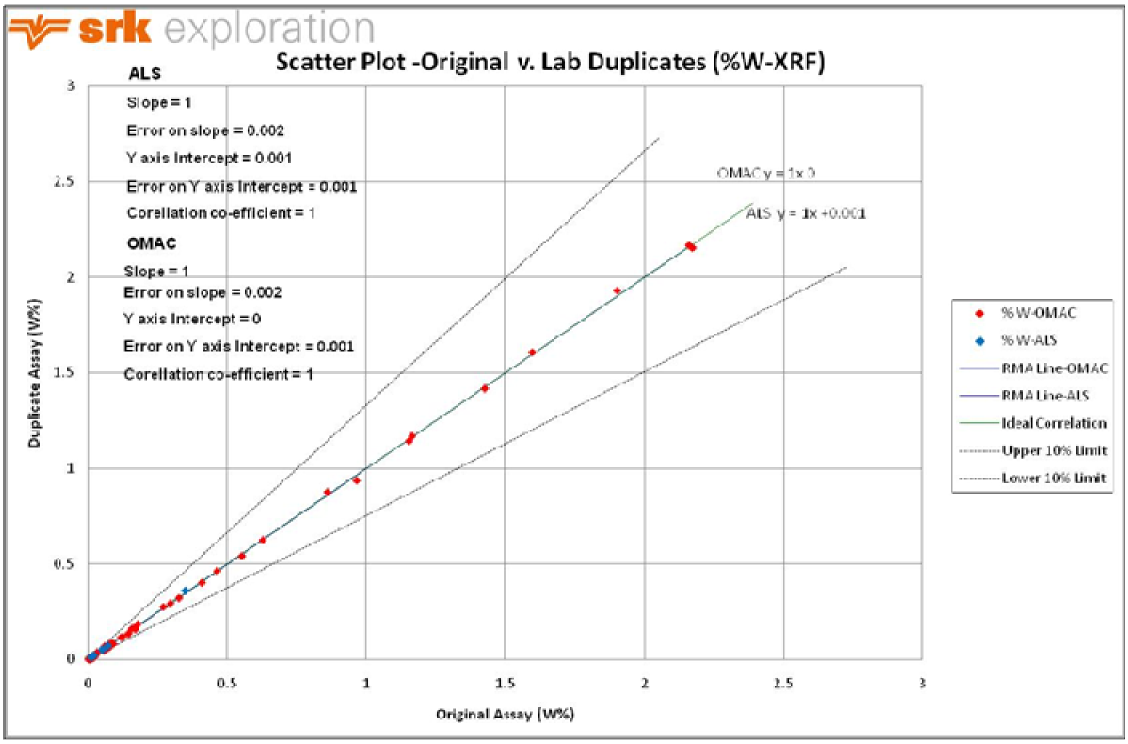


Figure 10-7: OMAC and ALS Laboratory Duplicates QAQC – XRF

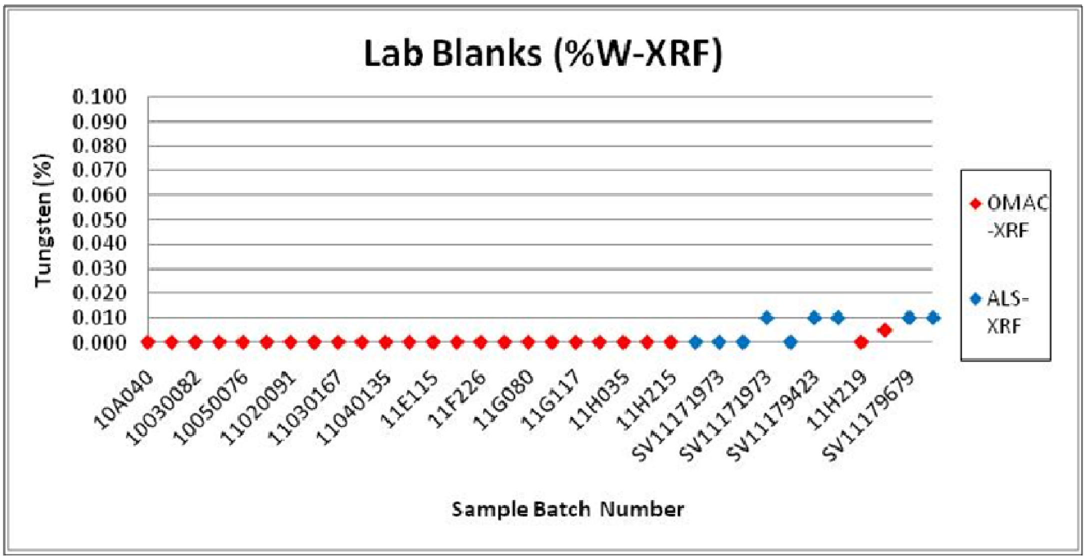


Figure 10-8: OMAC and ALS Laboratory Blanks QAQC – XRF

The ALS facility in Seville was audited by Colt on July 21 2012,The full preparation line was examined including:

- Sample reception and printing of barcodes for separate crushing and grinding schemes,
- Jaw crushers to -2mm,
- Tema mills,
- Storage and sample dispatch,
- Core cutting.

Colt's opinion was that the facility was generally clean with adequate staff operating the crushing lines with appropriate PPE equipment. Extraction and air conditioning was effective. An industrial vacuum cleaner was used to prevent accumulation of dust. The facility was operated on a 3 x 8 hr shift basis, 7 days per week.

Colt discussed their concern of the pulp storage and the condition of the grinding equipment with ALS management during the visit. The pulp is stored in paper kraft packets which risked contamination and deterioration over the long term. ALS said that kraft packets were only used for a split of 120 g of pulp for the primary analysis. Up to 900 g of the master pulp is retained in a polythene bag which is suitable for storage and retained for 3 months.

The routine use of steel tema mills for grinding appeared worn with convexing of the base of the pot compared with mills made of tunsten carbide. ALS explained that the reasons were based on cost, the alternative to use silica pots for Colt's samples made the cost uncompetitive and that wear on the steel mills was closely monitored by regular weighing. Once the pots lost 2 kg in weight they were replaced. Colt also noted that the pots were cleaned by using washed and screened inert river gravel from the local river bed.

10.9 Opinion on Adequacy

Colt's use of Stewart Group's OMAC laboratory and ALS Seville laboratory for their ICP-MS and XRF assaying meets with accepted industry standards for laboratory certification.

The XRF analysis methods utilized for detecting tungsten (WO_3%) at Tabuaço is deemed to be appropriate for the grade present. The ICP-AES method is appropriate for detecting smaller grades and should be used if a cut off grade of 0.2% or below is used. ICP-AES is not suitable or reliable for tungsten values above 0.2% due to issues with acid digestion of the sample during preparation.

The frequency and use of the laboratory repeats, blanks and standards can be used in their own right, or together with the addition of the Colt introduced repeats, blanks and standards to give an increased population for QAQC review on the laboratory instrumentation, although differing sample preparation techniques and sample states need to be taken into account.

The insertions made by Colt staff of repeats, blanks and standards give an additional check on the laboratory preparation and instrumentation quality. These insertions remain outside the laboratory control, although certain items will remain obvious in the sample stream as QAQC insertions (eg. SRM's) this is acceptable practice, as the laboratory are not made aware of the actual values of the standard being inserted into the sample stream.

SRK found no reason to reject any of the XRF assay results from the returned dataset, based on statistical analysis and QAQC. SRK was comfortable that the frequency of repeats, blanks and standards used is sufficient to give an adequate laboratory quality control check.

11 DATA VERIFICATION

SRK was appointed independent QP for the Tabuaço Project in February of 2011 and since then has been involved with reviewing 'best practice' procedures and monitoring QAQC protocols. In addition numerous visits by several personnel covering a wide range of disciplines have been made to the project to supervise the ongoing procedures and to verify the adherence to protocols. During a number of these visits SRK personnel witnessed the sampling procedures employed as well as the sealing and despatch of these samples.

SRK assisted in establishing guidelines for the standard operational procedures as documented by Colt in March 2012.

All assay certificates were issued directly by the laboratories to SRK in Cardiff for verification and assimilation into an independent database.

Thus SRK was confident that the values returned from the laboratories are an accurate reflection of those in the deposit and that there has been no opportunity for these results to be tampered with. As such SRK did not consider it worthwhile to collect further verification samples from the Tabuaço Project.

11.1 Procedures

A procedure for data logging and management was put in place by SRK for this project, which was implemented and refined by Colt.

Tabuaço drilling data is contained in excel spreadsheet format. The raw data on all activities from historical drilling, metallurgical test holes and exploration drilling is stored on a master workbook which contains all Colt geological, geotechnical, specific gravity, sampling intervals, and assay return data. This is a user access controlled database that becomes the only source of input data used in Gemcom for geological three dimensional modeling and block modeling.

User access to all or parts of the master is strictly controlled so as to prevent old slave data overwriting the new master. Two copies of the database are presen;, the master is housed on the Colt server for updating and a secondary public copy is stored on the corporate public domain for use by the exploration team and management. This maintains the integrity of the dataset and allows widespread use without compromising the integrity of the master database.

This master database has gone through multiple phases of data validation and verification checks using a number of mining software packages, including Gemcon, Micromine and LeapFrog, as well as manual checks following SRK QAQC. It is routinely validated by Colt in Gemcom.

All drilling and sampling data independent of the master database was digitally uploaded onto the SRK Sharepoint FTP site by Colt project geologists upon completion of each drillhole as individual excel files. The files were split into categories for drillhole surveying; core boxes; core photography; daily drill monitoring; drillers' logs; geology data; geotechnical data; sampling data and specific gravity data. In addition, copies of the sampling certificate PDFs held by SRK were placed on the FTP site as a record. These data files were then collated and combined by SRK into a single, independently verified database. This database was held in a read-only format, with write access limited to SRK. The finalized database was uploaded into mining software and validation checks were run against the data to check accuracy of the data ranges, correct height and depths.

11.2 Limitations

SRK was not aware of any limitations to the data collected and was comfortable for this data to be used for the November 2012 MRE.

11.3 Data Adequacy

SRK was confident that the data utilised in the November 2012 NI 43-101 Report, specifically in the MRE, was of a professionally acceptable standard and quality and that it is adequate for the purposes used herein.

Geological Data was collected from a series of 75 vertical and inclined drillholes across the mapped and predicted skarn horizon in the Tabuaço project exploration licence area. Drilling was completed on a hole spacing ranging from 50 m x 25 m to 25 m x 25 m.

Collars were surveyed using the Project Geologist's handheld differential GPS devices however the availability of detailed topography adds confidence to the collar elevation data. All collar positions were surveyed in UTM 29T, Datum ED50. Core logging was completed in sufficient detail to be able to separate and determine significant changes in alteration, lithology, structure, mineralization, alteration and geotechnical features when entered into a modelling package.

Total core recovery within the skarn zones at the Tabuaço resource area averaged nearly 87% for the MZ and 89% for the LZ.

SRK was comfortable that the methods and processes used to sample the core and the processes used to record the geological logs met the required quality and quantity parameters to complete a geological model of Tabuaço.

The assay data supplied was from reputable, certified laboratories and was subject to QAQC

12 MINERAL PROCESSING AND METALLURGICAL TESTING

Three metallurgical programs have been conducted on samples from the Tabuaço deposit. The first study was conducted by Inspectorate in 2011, under the direction of Bolu Consulting Engineering on an outcrop sample from Tabuaço. The second study was also conducted by Inspectorate in 2011/12 under the direction of SRK ES on drill core reject material from the Tabuaço property. The third study was undertaken by GRINFM in 2012 under the auspices of Colt, using samples of diamond drill core.

12.1 Metallurgical Programme on Tabuaço Outcrop (Inspectorate, 2011)

Head analyses on the test composite averaged 0.64% WO_3 with total sulphur less that 0.1%. Main elements of interest are presented in Table 12-1.

Table 12-1: Head Analyses for the Tabuaço Outcrop Sample

Comp. Sample id	WO_3 %	Sn %	F^- %	S_T %	Al_2O_3 %	CaO %	Fe_2O_3 %	P_2O_5 %	SiO_2 %
Outcrop SPA01	0.646	0.043	1.67	0.05	16.96	27.79	2.20	1.41	44.38

Mineralogical investigations indicated a predominant presence of silicate gangue minerals with fluorite present as a subordinate constituent. This observation was confirmed by the fluorine assay presented in Table 12-1, which corresponds to 3.4% fluorite (CaF_2). The microanalytical scan showed no fluorine minerals other than fluorite. Accessory minerals included apatite, scheelite and calcite, with traces of sulphides and cassiterite.

The scope of metallurgical testing conducted on the outcrop composited included:

- Heavy liquid separation studies on individual size fractions;
- Gravity release study with a laboratory Wilfley shaking table on separate size fractions;
- Gravity concentration tests; and
- Scheelite flotation tests.

12.1.1 Heavy Liquid Separation

The Heavy Liquid Separation ("HLS") studies were conducted on size fractions over the range from -12.7 mm to +0.84 mm and subjected to separation with heavy liquid specific gravities ("SG") of 2.96, 2.8 and 2.6 g/cm^3. This work was conducted to gain a preliminary understanding of the gravity concentration potential of the material, and demonstrated that at the size range tested, 92.3% of the scheelite is present in 65% of the HLS feed mass in the +2.96 SG sink product.

12.1.2 Gravity Release Study

In order to assess liberation and potential upgrading characteristics of scheelite in various size ranges, a gravity release analysis was conducted on 23 kg of Outcrop composite sample using shaking tables. The test sample was stage-crushed to -595 µm and screened at 420, 297, 210, 150, 105 and 75 µm to produce seven fractions for laboratory table tests. Tabling was conducted initially using a Wilfley Table, with the concentrate from this stage upgraded using a Mozley Mineral Separator.

The results of these tests are presented in Table 12-2, and demonstrate that very high grade concentrates were achieved in the -105 µm size ranges, while the -120 to +105 µm size range should be upgradable to marketable concentrate grades with further stages of upgrading. The coarser size ranges would most likely require regrinding to improve concentrate grade and recovery. This work indicated that future gravity testing should target 20 to 25% rougher mass pulls followed by multiple upgrading stages. Low grade intermediate products should be subjected to mineralogical investigations to determine liberation and the necessary regrind requirements for optimum gravity recoveries.

Table 12-2: Gravity Study on Individual Size Fractions from the Tabuaço Outcrop Sample

Size Fraction (µm)	Weight (%)		Mozley Concentrate Grade (%WO_3)	WO_3 Distribution (%)	
	In Size Fraction	Overall		In Size Fraction	Overall
-595+420	7.90	0.83	3.82	56.2	4.08
-420+297	11.0	1.49	3.38	66.6	6.53
-297+210	5.08	0.58	8.45	73.2	6.36
-210+150	1.53	0.32	28.9	60.4	12.0
-150+105	1.76	0.26	30.9	62.0	10.6
-105+75	0.50	0.08	72.6	36.3	7.29
-75	0.57	0.07	76.0	42.6	7.35

12.1.3 Gravity Concentration Tests

Gravity table tests were performed using two size ranges of the Outcrop composite sample: -500 to +150 µm and -150 µm. Each fraction was separately dry fed to the laboratory Wilfley shaking table, and metallurgical results are provided in Table 12-3. These results indicate that approximately 75% of the tungsten can be recovered at approximately 25% mass pull into a gravity concentrate assaying approximately 2.5% WO_3. Tungsten recoveries and grades into the table concentrate and middling products were approximately 87% and 2.2% WO_3, respectively, for the -500 to +150 µm size range, and approximately 67% and 2.9% WO_3, respectively, for the -150 µm size range.

Table 12-3: Gravity Concentration Test on -500 to +150 µm and -150 µm Size Fractions from the Tabuaço Outcrop Sample

	Weight			Assay, %	Distribution, %	% Overall Distribution,
PRODUCT	g	%	Overall %	WO_3	WO_3	WO_3
+150 Mozley conc	435.26	10.31	4.72	3.380	59.72	22.89
+150 Mozley tail	426.98	10.11	4.63	1.412	24.48	9.39
+150 Wilfley middlings 1	99.02	2.34	1.07	0.605	2.43	0.93
+150 Wilfley middlings 2	847.13	20.06	9.18	0.126	4.34	1.66
+150 Wilfley tail 1	724.00	17.14	7.85	0.101	2.97	1.14
+150 Wilfley tail 2	1690.48	40.03	18.32	0.088	6.06	2.32
+150 Total	4222.87	100.00	45.76	0.583	100.00	38.33

	Weight			Assay, %	Distribution, %	% Overall Distribution,
PRODUCT	g	%	Overall %	WO_3	WO_3	WO_3
-150 Mozley conc	162.44	3.25	1.76	14.629	59.96	36.98
-150 Mozley tail	335.96	6.71	3.64	0.404	3.42	2.11
-150 Wilfley middlings	412.08	8.23	4.47	0.328	3.41	2.10
-150 Wilfley tail 1	1746.50	34.90	18.93	0.126	5.56	3.43
-150 Wilfley tail 2	2347.90	46.91	25.44	0.467	27.65	17.05
-150 Total	5004.88	100.00	54.24	0.792	100.00	61.67

12.1.4 Scheelite Flotation Tests

Flotation tests were conducted on the Outcrop sample ground to a target grind P_{80} of 150 µm. A total of three tests was conducted under a variety of flotation conditions using soda ash, sodium hydroxide and sodium silicate for slurry conditioning, fatty acids as the scheelite collector and methyl isobutyl carbinol (“MIBC”) as the frother. The results of these tests are summarized in Table 12-4. The best test (F2) resulted in 65.8% tungsten recovery into a rougher flotation concentrate containing 7.17 % WO_3.

Table 12-4: Summary of Scheelite Rougher Flotation Tests on Tabuaço Outcrop Composite

Test	Wt%	WO_3 %	WO_3 Dist. %
F1	4.7	6.47	43.7
F2	6.6	7.17	65.8
F3	17.5	2.47	58.9

Source: Inspectorate Report - February, 2011

12.2 Metallurgical Programme on Tabuaço Drill Core Reject (Inspectorate, 2011/12)

A subsequent metallurgical program was conducted at Inspectorate on a test composite formulated from drill core reject material from selected intervals from fourteen drill holes. Head analyses on the test composite averaged 0.71% WO_3, 2.5% F and 1.84% P_2O_5. Total sulphur was 0.1%. Main elements of interest are presented in Table 12-5.

Table 12-5: Head Analyses for Tabuaço Drill Core Reject Test Composite

Elements	Units	Head 2011 Drill Core Composite
WO_3	%	0.71
Sn	ppm	590
F^-	µg/g	25700
S	%	0.1
Al_2O_3	%	15.2
BaO	%	0.03
CaO	%	25.6
Cr_2O_3	%	0.04
LOI	%	3.66
Fe_2O_3	%	3.98
K_2O	%	1.14
MgO	%	2.25
MnO	%	0.1
Na_2O	%	1.48
P_2O_5	%	1.84
SiO_2	%	43.8
TiO_2	%	0.41
Total	**%**	**99.4**

The test program was designed to more extensively evaluate scheelite flotation and the parameters required to achieve acceptable flotation recoveries and concentrate grades. This was followed by a bulk gravity/flotation test in which three separate sized fractions over the range from 300 µm to 74 µm were subjected to gravity concentration and the minus 74 µm fraction was subjected to scheelite flotation.

12.2.1 Scheelite Flotation Studies

Scheelite flotation is typically conducted at an alkaline pH, using soda ash, sodium hydroxide and sodium silicate as flotation modifiers and fatty acids (such as oleic acid and linoleic acid) as collectors. The extent to which scheelite can be concentrated by flotation is highly dependent on the mineralogy of the ore, and more specifically the extent to which fluorite, calcite and apatite occur in the ore, as these minerals have a strong tendency to float with scheelite.

A series of flotation tests were conducted by Inspectorate, under SRK ES's supervision, to define the flotation process parameters required to effectively recover scheelite into a flotation concentrate. This work included an initial evaluation of process parameters previously used by Inspectorate during the earlier studies on the Outcrop composite, followed by a systematic evaluation of key flotation parameters, such as:

- Collector dosage;
- Sodium silicate dosage; and
- Grind size.

These tests determined the optimum, conditions to be:

- A fatty acid collector dosage of 100 g/t;
- A sodium silicate dosage of 2500 g/t; and
- A grind size P_{80} of 75 µm.

12.2.2 Bulk Gravity / Flotation Study

A bulk gravity / flotation test was conducted on a 20 kg ore sample, stage-crushed to -297 µm. The test composite was then screened into four size fractions: -297 to +210 µm, -210 to +105 µm, -105 to +75 µm and -75 µm. Table 12-6 provides a summary of tungsten grade and distribution in each of the size fractions, and shows that the tungsten values are highly concentrated in the -75 µm fraction, which represents 54% of the mass and accounts for 69% of the tungsten in the test composite. However, it should be noted that the tungsten distribution into the fine fractions in this drill core reject test composite may be significantly higher than may be experienced in an actual milling operation, as the samples had already been pulverised.

Table 12-6: Bulk Composite Tungsten Grades and Distributions by Size

Size Fraction		Weight		Assay, %	Distribution
Mesh	Microns	(g)	(%)	WO_3	WO_3
-48+65m	-297+210	3474.9	17.4	0.42	10.4
-65+150m	-210+105	4188.8	20.9	0.49	14.6
-150+200m	-105+75	1508.4	7.5	0.56	6.0
-200m	-75	10827.9	54.1	0.90	69.0
Total		**20000.0**	**100.0**	**0.71**	**100.0**
Measured				0.71	

The first three size fractions were subjected to gravity concentration on a laboratory Wilfley Table followed by upgrading of the table concentrate on a Mozley Mineral Separator.

The results of the gravity concentration tests are summarized in Table 12-7 to Table 12-9.

Table 12-7: Gravity Concentration Results on the -297 to +210 μm Size Fraction

Product	Weight (g)	Weight (%)	Assay,% WO_3	Distribution,% WO_3
Mozley Conc	40.5	10.6	5.23	37.7
Mozley Middlings	184.6	48.5	1.43	47.0
Mozley Tailings	155.8	40.9	0.55	15.3
Wilfley Conc	**380.91**	**100.0**	**1.47**	**100.0**
Wilfley Table Performance				
Conc	380.9	11.0	1.47	38.2
Middling 1	84.6	2.4	2.22	12.8
Middling 2	818.3	23.6	0.25	13.9
Middling 3	1426.4	41.0	0.26	25.2
Tailings	764.6	22.0	0.19	9.8
Total -48+65mesh fraction	**3474.89**	**100.0**	**0.42**	**100.0**

Table 12-8: Gravity Concentration Results on the -210 to +105 μm Size Fraction

Product	Weight (g)	Weight (%)	Assay,% WO_3	Distribution,% WO_3
Mozley Conc	20.2	14.2	17.40	73.6
Mozley Middlings	80.9	56.7	1.34	22.7
Mozley Tailings	41.6	29.1	0.43	3.7
Table Conc	**142.70**	**100.0**	**3.35**	**100.0**
Wilfley Table Performance				
Conc	142.7	3.4	3.35	23.2
Middling 1	341.5	8.2	2.98	49.5
Middling 2	1128.1	26.9	0.19	10.4
Middling 3	358.6	8.6	0.09	1.6
Tailings	2217.9	52.9	0.14	15.3
Total-65+150mesh fraction	**4188.76**	**100.0**	**0.49**	**100.0**

Table 12-9: Gravity Concentration Results on the -105 to +75 μm Size Fraction

Product Id	Weight (g)	Weight (%)	Assay,% WO_3	Distribution,% WO_3
Mozley Cl Conc	8.1	3.6	48.95	83.7
Mozley Cl Middlings 1	6.5	2.9	6.86	9.4
Mozley Cl Middlings 2	27.4	12.1	0.33	1.9
Mozley Cl Tailings	34.0	15.0	0.22	1.6
Mozley Tailings	150.9	66.5	0.11	3.5
Wilfley Conc	**226.87**	**100.0**	**2.09**	**100.0**
Wilfley Table Performance				
Conc	226.9	15.0	2.09	55.7
Middling 1	115.4	7.6	1.87	25.4
Middling 2	376.6	25.0	0.20	8.9
Middling 3	38.2	2.5	0.13	0.6
Tailings	751.4	49.8	0.11	9.5
Total -150+200mesh fraction	**1508.42**	**100.0**	**0.56**	**100.0**

The -75 µm fraction was subjected to scheelite flotation using the optimized flotation conditions developed earlier in the study. The results of this test are summarized in Table 12-10, which shows that 78% of the contained tungsten was recovered into a first cleaner flotation concentrate containing 13.5% WO_3. At 19.9% F, the major contaminant in the flotation concentrate is fluorite. Even though 67% of the fluorite was rejected into the combined flotation tailings, due to the high concentration of fluorine in the ore (approximately 3%), in order to produce higher grade scheelite flotation concentrates, further work will need to be done to develop a flotation protocol that can be more selective against fluorite.

Table 12-10: Scheelite Flotation Test Results on the -75 µm Size Fraction

Product	Weight		Assay			Distribution		
			WO_3	F	P_2O_5	WO_3	F	P_2O_5
	(g)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
2nd Cleaner Concentrate	41.9	4.1	15.14	21.01	0.73	72.76	29.49	1.26
2nd Cleaner Tailings	8.3	0.8	5.52	14.24	3.86	5.27	3.97	1.32
1st Cleaner Tailings	42.2	4.2	1.04	6.04	3.66	5.04	8.54	6.36
Flotation Tailings	921.5	90.9	0.16	1.88	2.40	16.93	58.00	91.06
	1013.8	100.0						
Combined Product	**Weight**		**Assay**			**Distribution**		
			WO_3	F	P_2O_5	WO_3	F	P_2O_5
	(g)	(%)	(%)	(%)	(%)	(%)	(%)	(%)
2nd Cleaner Concentrate	41.9	4.1	15.14	21.01	0.73	72.76	29.49	1.26
1st Cleaner concentrate	**50.2**	**4.9**	**13.54**	**19.89**	**1.25**	**78.03**	**33.46**	**2.58**
Total Ro Concentrate	92.3	9.1	7.83	13.56	2.35	83.07	42.00	8.94
Flotation Tailings	921.5	90.9	0.16	1.88	2.40	16.93	58.00	91.06
Calculated Feed	1013.8	100.0	0.86	2.94	2.40	100.00	100.00	100.00
Measured Feed			0.90	3.24	2.67			

12.3 Metallurgical Programme on Tabuaço Drill Core Samples (GRINFM, 2012)

12.3.1 Introduction

The previous metallurgical studies had demonstrated that production of high grade scheelite flotation concentrates was difficult, using commonly accepted procedures, due to the high fluorite content of the ore. Consequently, in early 2012 Colt retained GRINFM, which specializes in tungsten metallurgy, to undertake a metallurgical investigation of the Tabuaço ore.

In May 2012, Colt supplied GRINFM with 522 kg of recent drill core material to conduct their metallurgical studies. The drill core holes and intervals used are shown in Table 12-11.

Table 12-11: Samples sent to GRINFM

Met ID	Twin ID	Starting Coordinates x	y	z	Orientation Azm (°)	Dip (°)	Length (m)	From (m)	To (m)	Interval (m)	Zone	Expected Head Grade (% WO_3)	Actual Weight (kg)
DHMT-01	DHT-08	624,697	4,549,440	379	0	-90	78.35	43.00	55.40	12.40	MZ	0.59	72.055
DHMT-02	DHT-02	624,717	4,549,394	379	0	-90	90.30	52.45	61.30	8.85	MZ	0.93	31.595
DHMT-03	DHT-01B	624,658	4,549,476	TBC	0	-90	59.85	18.60	39.58	20.98	MZ	0.73	47.435
								47.88	49.70	1.82	LZ	0.12	8.695
DHMT-41	DHT-12	624,571	4,549,467	407	0	-90	118.65	59.36	68.65	9.29	MZ	1.53	61.735
								78.66	93.73	15.07	LZ	0.45	47.350
DHMT-51	DHT-14	624,586	4,549,413	415	0	-90	152.30	75.88	85.77	9.89	MZ	1.20	55.050
								104.20	129.50	25.30	LZ	0.29	130.785
DHT-33	-	624,655	4,549,505	386	330	-50	75.50	6.00	10.25	4.25	MZ	0.55	3.480
								18.20	49.75	31.55	MZ	0.44	46.225
DHT-34	-	624,742	4,549,423	363	30	-50	79.85	42.80	49.00	6.20	MZ	0.46	11.450
								66.90	70.00	3.10	LZ	0.08	6.895
Average												**0.67**	**522.750**

12.3.2 Head Assay and Mineralogy

Multi-element analyses and tungsten mineral phase analysis are shown in Table 12-12 and Table 12-13 respectively.

Table 12-12: Multi-Element Analyses of the Raw Ore

	Units	Value
WO3	%	0.64
Pb	%	0.0067
Zn	%	0.0084
Cu	%	0.002
CaF2	%	10.20
CaCO3	%	2.97
Na2O	%	1.32
Mn	%	0.037
S	%	<0.02
Bi	%	0.0041
Sn	%	0.075
K2O	%	0.98
Au	g/t	0.12
Ag	g/t	5.5
Pb	%	0.15
TiO2	%	<0.05
U	%	0.00003
Th	%	0.0014

Table 12-13: Analytical Results of the Tungsten Phase

Phase	Units	Scheelite	Wolframite	Tungstite	Total
WO3 Grade	%	0.61	0.013	0.016	0.639
WO3 Distribution	%	95.46	2.03	2.51	100.00

Mineralogical analyses indicated that the most abundant mineral was grossular (a garnet mineral) followed by feldspar, fluorite, quartz and apatite. Sulphide minerals made up only approximately 0.2% of the sample. Scheelite grain sizes ranged from 0.02 to 1 mm, with 69% of the scheelite grains being coarser than 0.08 mm. The fluorite was slightly coarser, with a range of grain sizes between 0.02 and 1.2 mm, and with 74% of the scheelite grains being coarser than 0.08 mm.

Scheelite liberation ranged from 84% at a grind size of 60 % -74 µm to 99% at a grind size of 90% -74 µm. Intergrowths were mainly with garnet, apatite and feldspar.

12.3.3 Gravity Separation

Single stage and two stage gravity separation tests were conducted using a shaking table. The sample was ground to different percentages -74 µm. The single stage tests were passed across the table in their entirety. The two stage tests involved screening the ground sample at 100 µm, with the +100 µm and -100 µm fractions tabled separately.

The results of the tabling testwork are shown in Table 12-14.

Table 12-14: GRINFM Gravity Separation Test Results

Grind Size (% -74 µm)	Number of Stages	Mass Recovery (%)	Concentrate Grade (% WO_3)	WO_3 Recovery (%)
52.1	1	0.7	43.3	46.3
	2	0.8	40.5	52.4
60.9	1	0.7	48.5	52.9
	2	0.7	45.9	51.7
69.3	1	0.6	51.2	49.5
	2	0.7	48.3	51.0
77.2	1	0.6	56.5	48.5
	2	0.8	43.8	51.6

Neither scenario produced concentrate grades significantly greater than 50% WO_3. WO_3 recoveries were also of the order of 50%.

12.3.4 Flotation

Flotation testwork progressed through batch rougher tests to batch cleaning tests to locked cycle tests.

The rougher tests sought to optimise the type, addition level and conditioning time for the reagents – pH modifier, dispersant and collectors – and grind size. The optimum conditions resulted in the production of a rougher concentrate assaying 4.56% WO_3 at a WO_3 recovery of 90.6%. Closed circuit rougher tests, i.e. with some cleaning and recycle of scavenger concentrates, improved the rougher concentrate grade to 8.35% WO_3 at a WO_3 recovery of 91.9%.

The cleaning tests followed a similar vein, with temperature added as an operating variable. The optimum conditions in an open circuit configuration resulted in a concentrate assaying 67.2% WO_3 at a stage WO_3 recovery (i.e. with respect to the rougher concentrate) of 97.1%. Using closed circuit conditions resulted in a concentrate assaying 55.9% WO_3 at a stage WO_3 recovery of 95.4%. As this concentrate also assayed 3.4% P, acid leaching was used to reduce the P content of the concentrate. The resulting concentrate assayed 70.6% WO_3, with a WO_3 loss in the acid leaching stage of 0.3%.

These results were used to generate a mass balance for the integrated flowsheet, resulting in the estimation of a concentrate assaying 70.6% WO_3 at an overall WO_3 recovery of 87.2%.

Colt anticipates contracting with the GRINFM at a future date for pilot plant testing of a 20 t ore lot when such material is available. It is recognized that some of the reagents used during these metallurgical studies may be proprietary and Colt therefore may be dependent on Chinese sources for reagents such as the modified sodium silicate and modified fatty acid collector reagents.

12.4 Further Testwork

12.4.1 Ore Sorting

Colt has initiated ore sorting testwork with Steinert in Germany. Preliminary testwork using an X-Ray sorter on diamond drill core reported encouraging results on a qualitative basis – no quantitative results have yet been reported.

12.4.2 Downstream Hydrometallurgical Processing

Colt has commissioned an American hydrometallurgical technology provider to investigate a hydrometallurgical process route using acid leaching to produce tungsten oxide (WO_3) from the Tabuaço RoM material or concentrates produced from it. SRK understands that this testwork is at a preliminary stage.

12.5 Sample Representativeness

The test composites used for the studies conducted to date are considered characteristic of the respective deposits, and suitable for this level of study. As the project advances to the next level of study, test composites should be formulated to more closely represent each deposit area with respect to mineralisation type and mineralisation grade.

12.6 Conclusions

The results of the metallurgical testwork have demonstrated the following:

- Gravity separation only produced high concentrate grades at particle sizes of the order of 105 µm or less, however recoveries were low, with the best results being concentrates of the order of 50% WO_3 at WO_3 recoveries of the order of 50%;
- Flotation testwork conducted by Inspectorate resulted in relatively high recoveries, however final concentrate grades were quite low; and
- Flotation testwork conducted by GRINFM, who have particular expertise in tungsten processing, produced a relatively high grade final flotation concentrate (55.9% WO_3) at a relatively high WO_3 recovery. As this concentrate was high in P (3.4%), a final acid leach stage was employed, which resulted in a final concentrate grade of 70.6% WO_3, with a negligible loss of W in the acid leach stage.

On the basis of the GRINFM results, an overall flotation with concentrate leaching circuit was estimated to be capable of producing a concentrate assaying 70.6% WO_3 at an overall WO_3 recovery of 87.2%.

Ore sorting testwork has recently been commenced, with the aim of reducing the mass of material that needs to be ground ahead of flotation at a minimal W loss. Colt has also commissioned an investigation into a hydrometallurgical process route to produce tungsten oxide (WO_3) from the Tabuaço RoM material, or concentrates produced from it, using acid.

13 MINERAL RESOURCE ESTIMATE

13.1 Introduction

The Mineral Resource model presented here represents the November 2012 resource estimate for the QSPA resource area (“the Resource”). The MRE was completed by Colt and reviewed by Martin Pittuck, Director and Corporate Consultant for Mining Geology with SRK, with input provided by Filipa Matias, Colt’s Geological Engineer. Mr Pittuck is considered a Qualified Person as defined in National Instrument 43-101. The effective date of this resource estimate is 3rd October, 2012.

This section describes the work undertaken by Colt in conjunction with SRK and summarizes the key assumptions and parameters used to prepare the revised Mineral Resource models.

The Mineral Resources presented here are reported in accordance with Canadian Securities Administrators’ National Instrument 43-101 and have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserves. Mineral Reserves can only be estimated as a result of a technical-economic evaluation as part of a preliminary feasibility study or a feasibility study of a mineral project. Accordingly, at the present level of development there are no Mineral Reserves at the Tabuaço project.

13.2 Topography and Coordinate System

Geographic information used in the development of this study is projected in the Universal Transverse Mercator European Datum 1950 Zone 29 North. Pre-Colt era data has been converted from Hayford-Gauss (Datum 73).

The topographic surface used in the Gemcom model was produced by surveyor contractor Superficie. The surface was created from 2 m topographic contours from a high resolution orthorectified aerial photo mosaic encompassing Sao Pedro das Águias, Quintã and the Aveleira areas. The study was completed December 2011, with periodic updates to the drillhole database for spot locations of drilling activities.

13.3 Database Construction and Validation

Raw data was compiled by Colt field exploration team, collated into excel format spreadsheets for continual archiving. Separate tables were compiled for collar, downhole survey, assay, lithological logging, structural geology, and alteration and mineralisation occurrence. In addition, density data was also compiled in an ASCI table. Colt compiled data collected until September 2012 and imported it into the Gemcom, GEMS mining software package. Validation of data was performed during collation of field data (an ongoing process) and again inside the relational database of GEMS mining software. The data was verified as correct at the time of submission.

13.4 Geological Interpretation and Domaining

As referred previously in this report, prior to geological modelling, a series of cross sections were defined perpendicular to the strike direction of the mineralized zones within the Project. Also, a series of longitudinal sections were defined along strike. After interpreting in both directions, a set of 3D wireframes and strings were created to materialize the interpretation and domaining of the deposit.

Two major zones of tungsten bearing skarn mineralization were defined, using geological aspects like rock type, texture, structure and position criteria. Grades of WO_3 were also used to help correlating the different horizons across the section. Each major zone was then divided in two sub-zones.

The two major units defined are the Main Skarn (red horizons) and Lower Skarn (orange horizons). This separation was based mainly in the rock type defined by the core logging, although in some sections and zones it was also based on position relationship along the cross section. The Main Skarn (SKM in geological log) includes mainly a coarse grain, massive in texture and well developed skarn, with high grade mineralisation. Nevertheless, in some zones some carbonaceous lithologies ("CRB") were also included in the Main Skarn domain as they might represent a less developed skarn. In fact, it is accepted that some lateral variation may occur in this type of geological setting. The Lower Skarn was modelled based on the layered and green to pink skarn (SKL rock code in the geological log). This lithology is fine grained, rich in garnet and epidote, and corresponds most of the times to medium to low grade mineralisation.

Each unit is consistently separated by a few meters of barren biotitic schist although within the same rock type and positional trend of the main zone where it is included. Therefore, M1 and M2 were defined as the two sub-zones of the Main Skarn, while L and LL are two sub-zones of the Lower Skarn.

In Aveleira a two-layer model was also defined with a Main Skarn and Lower Skarn unit based mostly in rock type and in position relationship along section.

An approximate grade cut off of 0.1 % WO_3 was used to define the mineralisation wireframes, in conjunction with lithological logging of lower grade skarn. After completion of the cross sectional modelling, the interpretations were tied together to create 3D wireframes to be used for constraining the MRE. A fault which cuts across the modelled units has also been modelled as well as two additional but less significant faults in the south-east part of the deposit. A typical cross section through the modelled units is given in Figure 13-1. From top to bottom, wireframes are designated as M1, M2, L and LL. This denomination it is not applied to Aveleira, being the wireframes named as Main Skarn horizon (red) and Lower Skarn horizon (orange) for that project (Figure 13-2).

The wireframes generally look continuous in cross section and long section which allow a reasonable confidence to be placed on their continuity. As the wireframes are often flat dipping, their outlines in horizontal plan view are sometimes uneven but this is considered by SRK to be an artefact of the wireframing which can be improved upon before mine planning commences.

A total of 66 assayed drillholes were used for the estimation, with an average core recovery of 86%, 74% of the total samples used have a core recovery equal or greater than 80%.

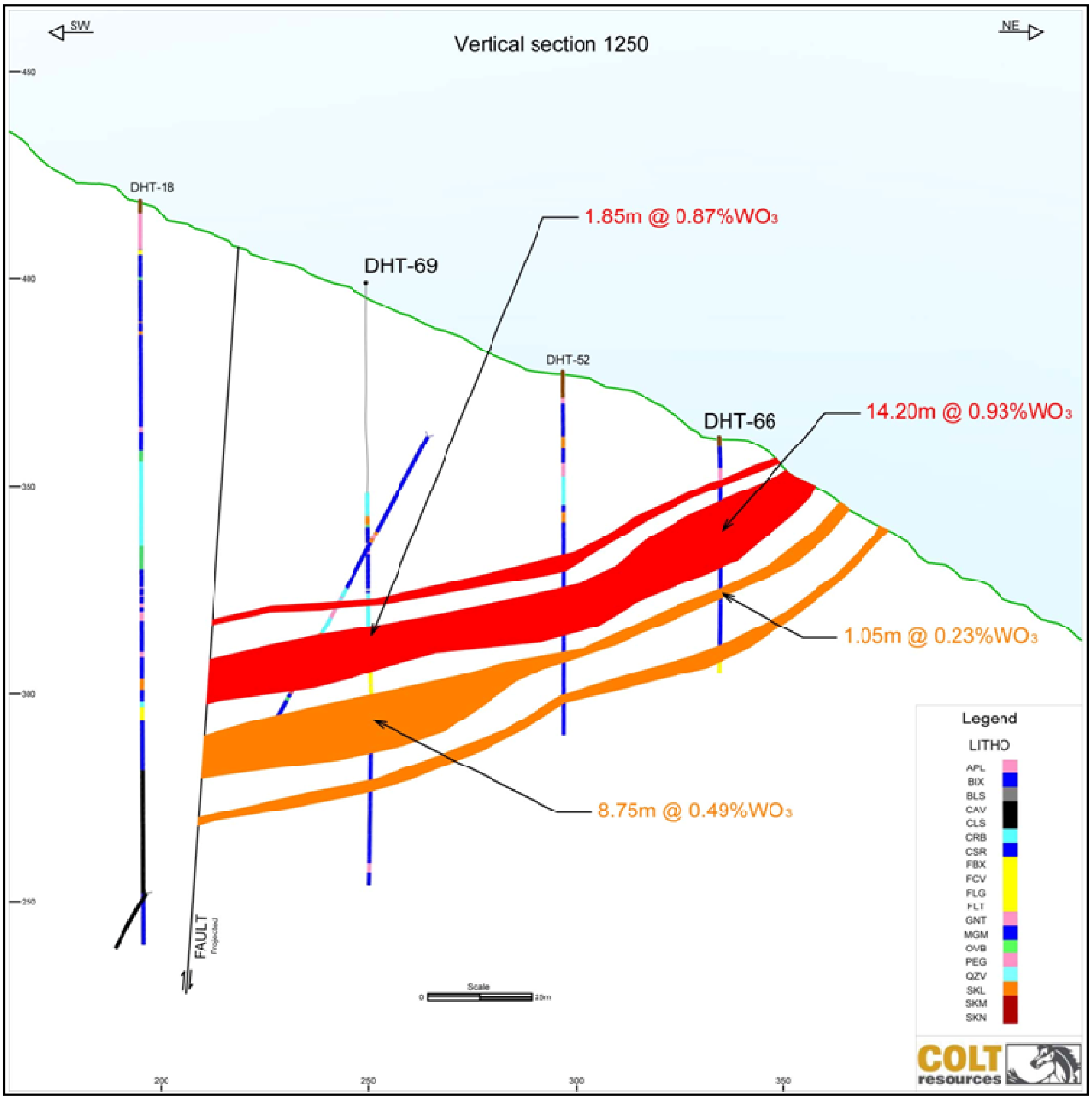


Figure 13-1: Typical Cross Section through the Modelled Units, South West - North East Facing

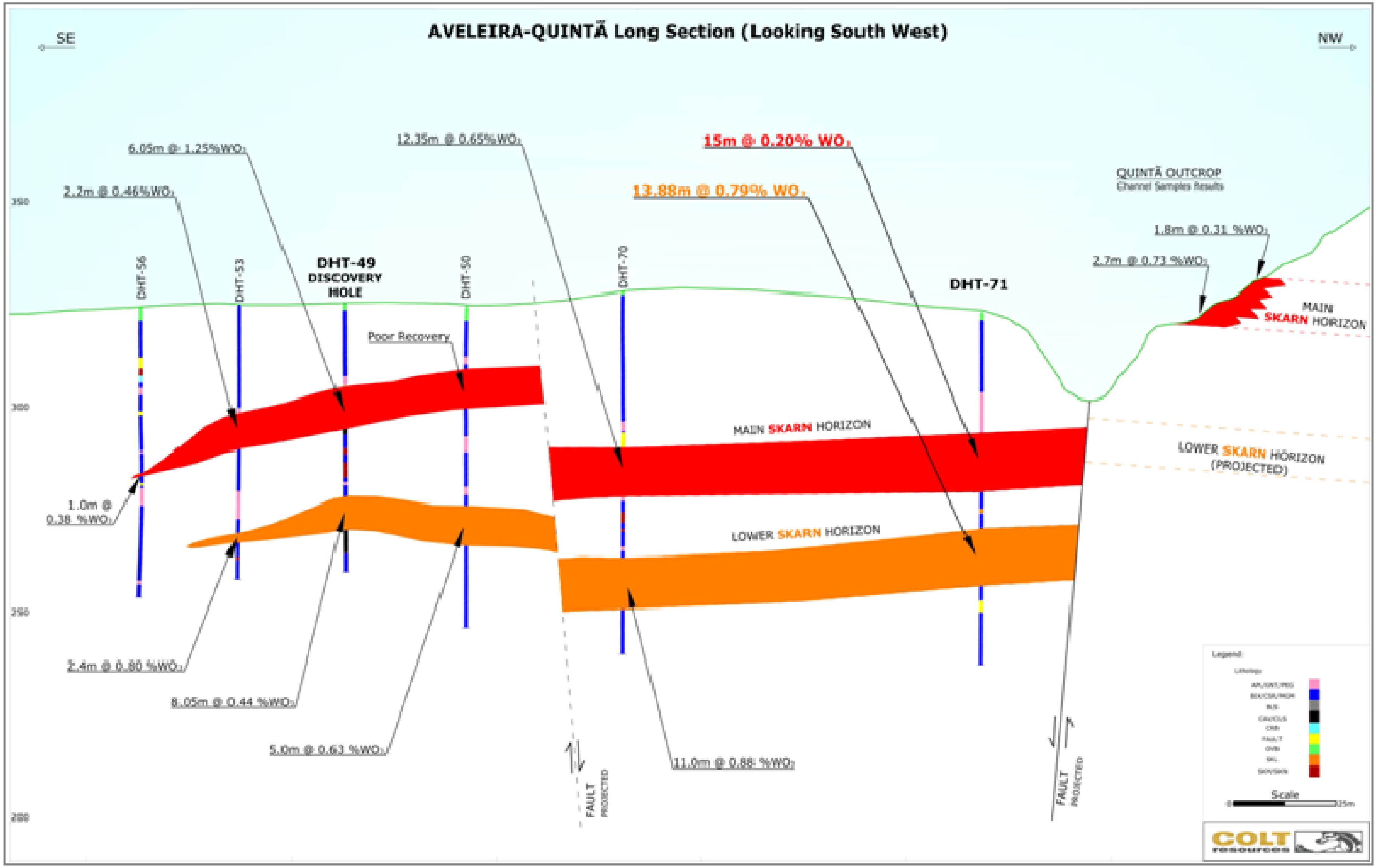


Figure 13-2: Long Section showing Intersections of the Two Mineralised Zones at Aveleira, South–West Facing

13.5 Assay Capping

A total of 284 SG, or density, measurements were taken during the course of the Colt's 2011 drilling campaign and 80 samples during the 2012 drilling campaign. Measurements were taken by the Archimedes method by displacement of water from solid core immersed in a bath. Two weights are recorded with a 1g precision scale: Weight of core ("Wc") and Weight immersed ("Wi"). By assuming SG of water as 1:

SG of core is SG = Wc/(Wc-Wi)

Table 13-1 illustrates the relationship between lithology and SG averaged over the two drilling campaigns. 3.1 is the average SG value for the skarn units samples, with a grade higher than 0.3%; this value has been used as the average density of the ore bearing blocks in the November 2012 MRE.

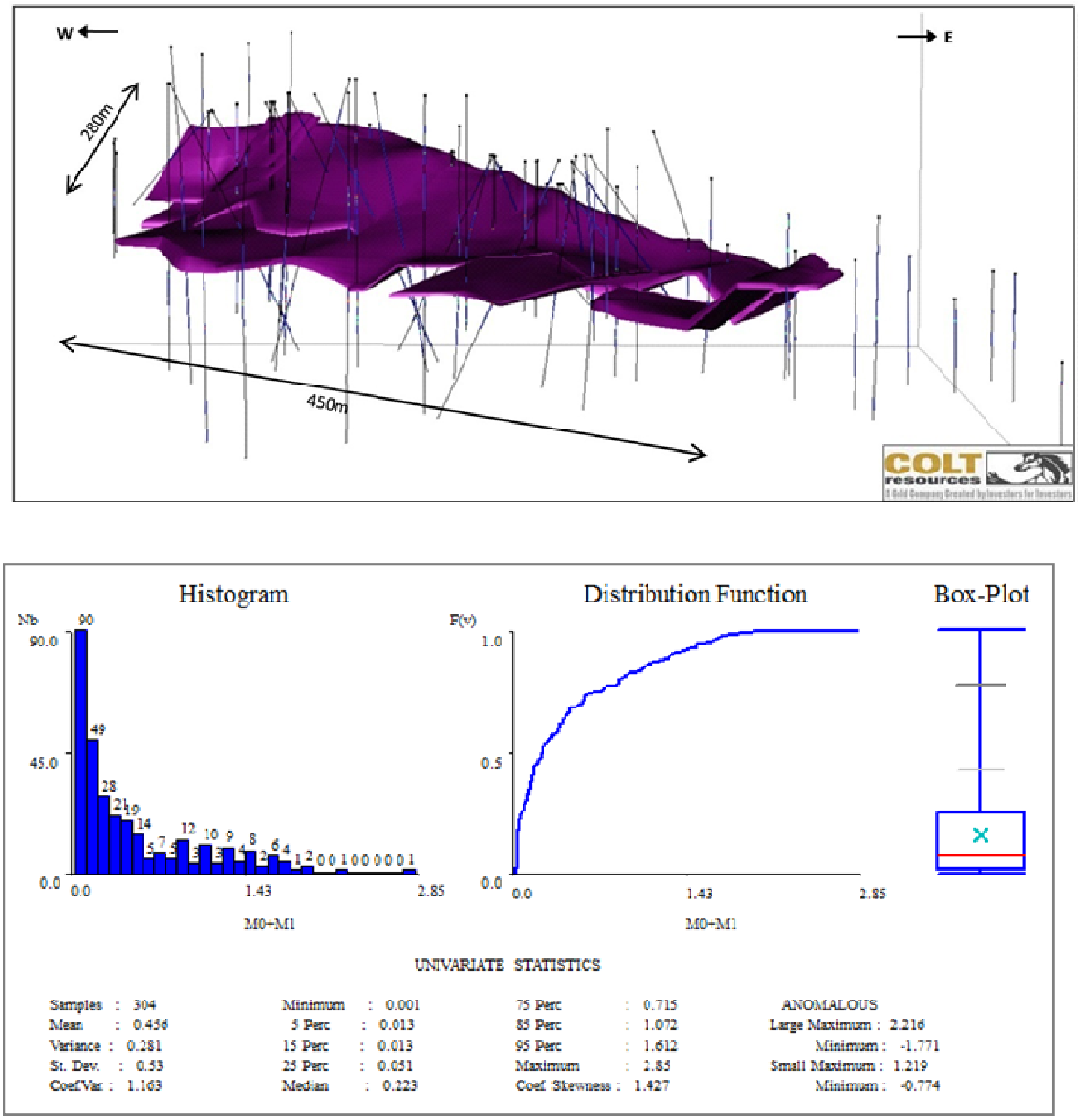
Table 13-1: Average SG by Lithology

Lithology	Average SG
APL- Aplite	2.65
BIX-Biotite Schist	2.80
BLS-Black/Dark grey schists, fine grained	2.80
CRB- Carbonates, calcschists	2.76
CSR- Calcsilicate, poorly developed skarn	2.87
GRN- Granitic	2.60
SKL- Skarn L-type	3.05
SKM- Skarn M-type	3.12
SKM (no sch)	3.07
SKN- Skarn undifferentiated	3.00

13.6 Statistical Analyses

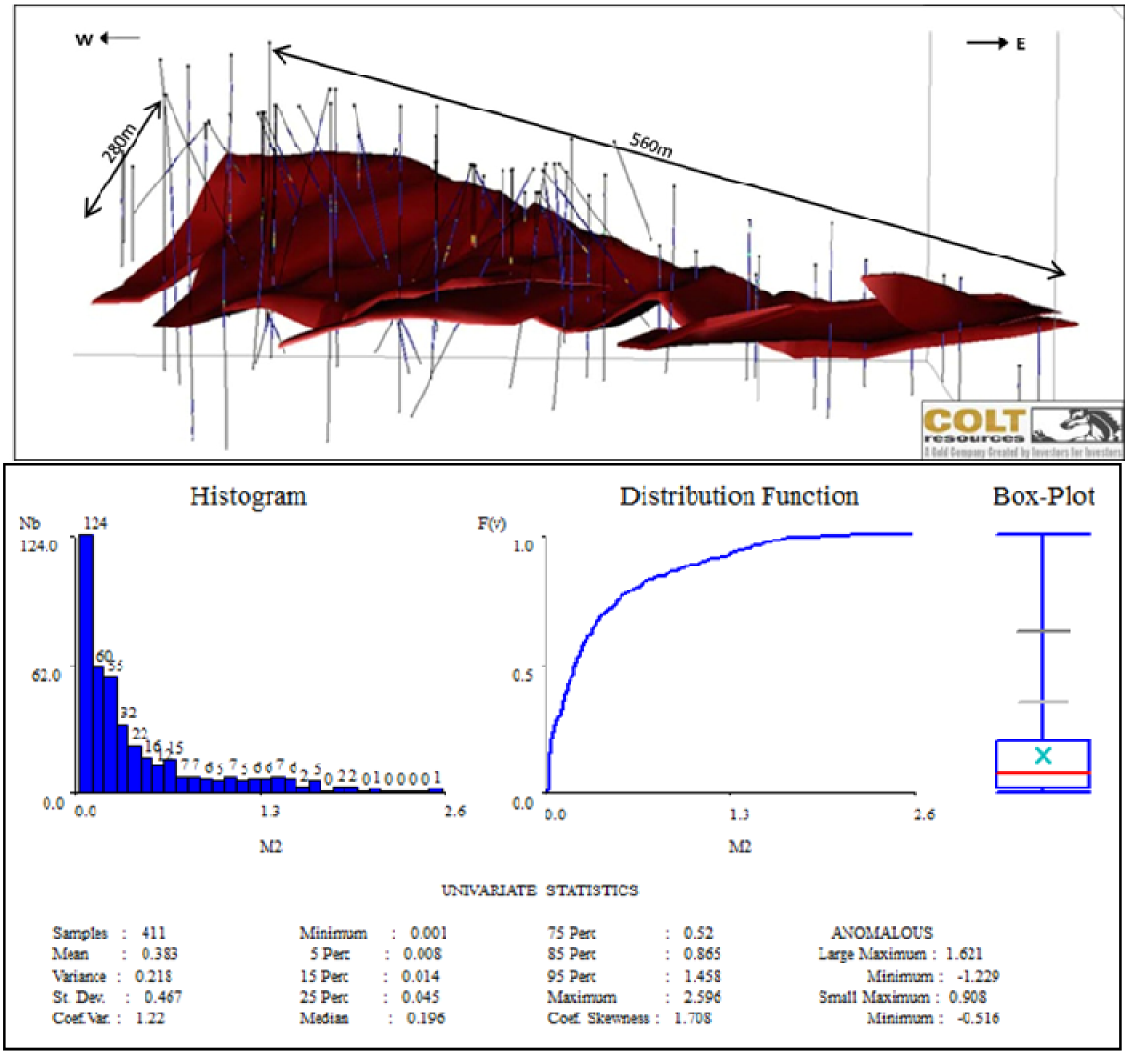
Exploratory data analysis was performed using the 1 m composite samples within the wireframes described previously. The composite process was conducted in the Gemcom Software, as well as samples extraction, with subsequent visual confirmation of interval coding by the wireframes. The short composites were checked to ensure no bias was introduced and to determine if could be included. As no bias was present, smaller samples were retained for the estimation.

Basic statistics were analysed for each layer individually, and are presented in Figure 13-3 to Figure 13-6.



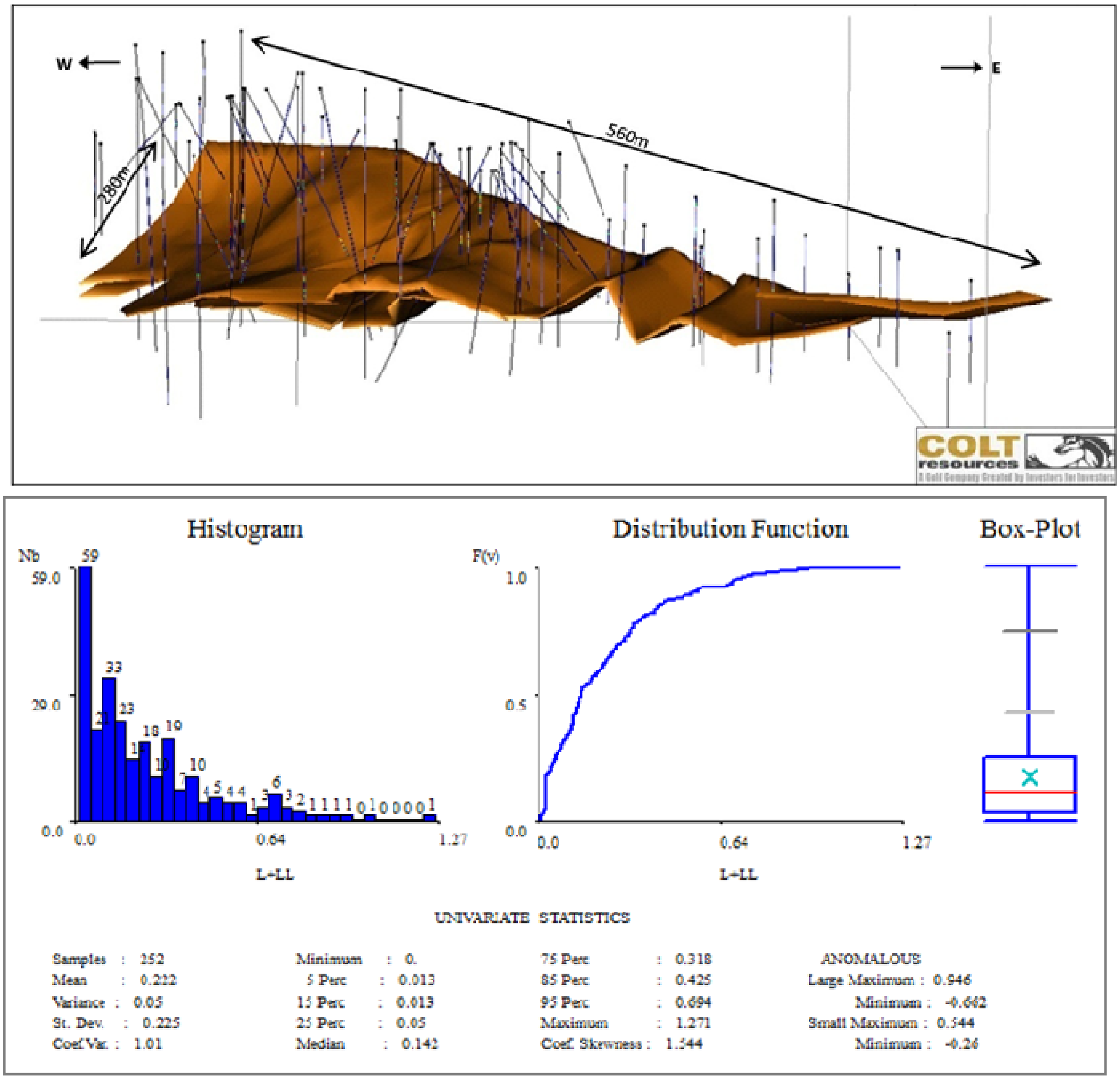
Wireframe strike distance is 450 m and dip direction is 280 m

Figure 13-3: 3D Visualization of M0 and M1 Wireframe, Looking West-East, with Histogram and Basic Statistics



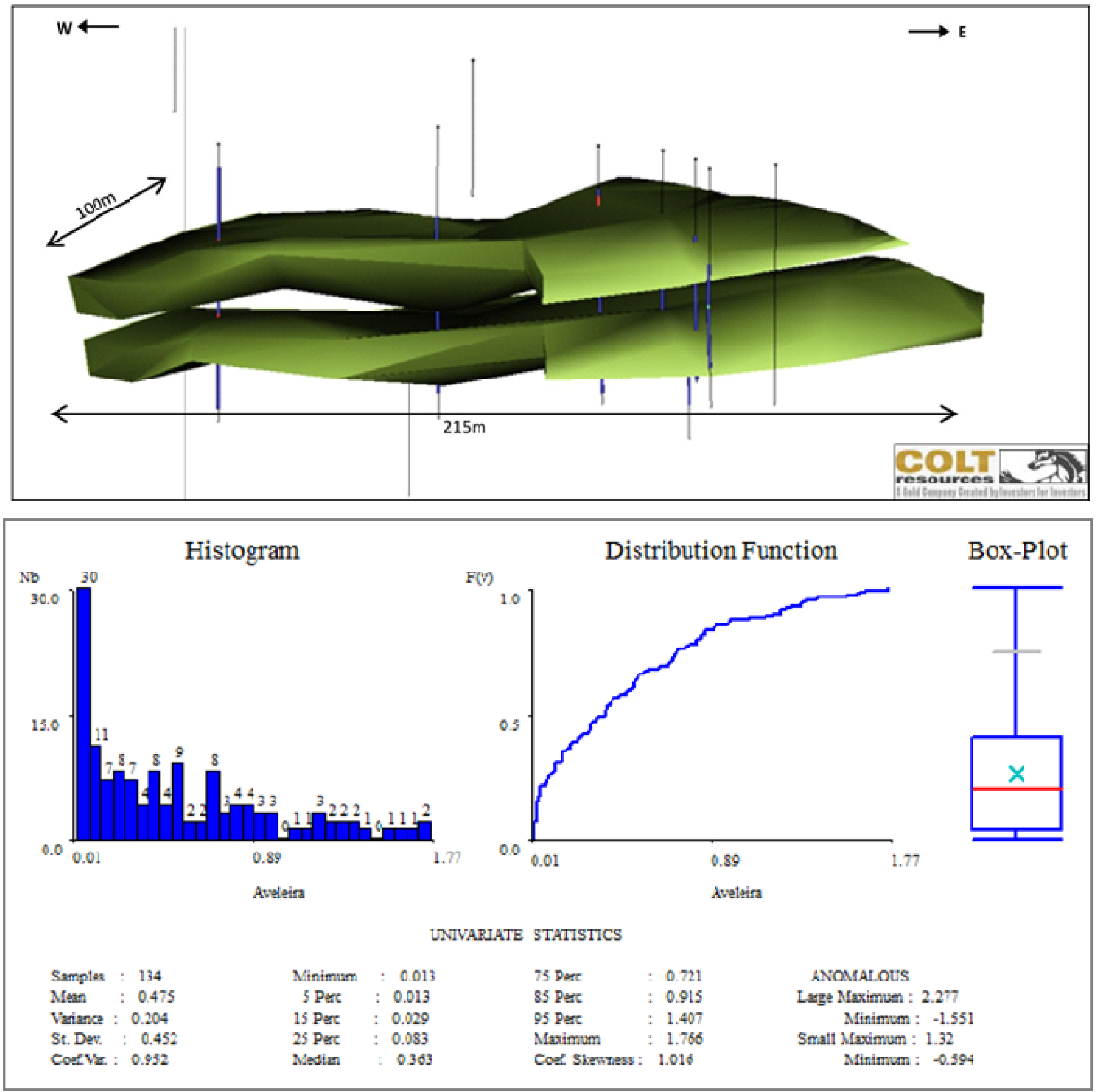
Wireframe strike distance is 560 m and dip direction is 280 m

Figure 13-4: 3D Visualization of M2 Wireframe, Looking West-East, with Histogram and Basic Statistics



Wireframe strike distance is 560 m and dip direction is 280 m

Figure 13-5: 3D Visualization of L+LL Wireframe, Looking West-East, with Histogram and Basic Statistics



Wireframe strike distance is 215 m and dip direction is 100 m

Figure 13-6: 3D Visualization of Aveleira Wireframe, Looking West-East, with Histogram and Basic Statistics

All populations show a positive skew and do not display a normal histogram. When transformed to natural logs, the populations move towards log-normality, but they are not strictly log-normal, being negatively skewed. This happens with all the wireframe populations.

The geological interpretation and knowledge of the deposits lead resulted in 3 estimation domains being used for SPA only one for Aveleira. Summary statistics for these are provided in Table 13-2

Table 13-2: Summary of Descriptive Statistics of Each Modelled Wireframe

	Mean grade (%WO_3)	Median grade (%WO_3)	Min grade (%WO_3)	Max grade (%WO_3)	Variance (%)	Std. Dev. (%)	Variance coefficient	# of samples	Volume (k m3)
São Pedro das Águias Deposit									
M0+M1	0.456	0.223	0.001	2.850	0.281	0.53	1.163	304	410.60
M2	0.383	0.196	0.001	2.596	0.218	0.467	1.22	411	698.29
L+LL	0.222	0.142	0.001	1.271	0.05	0.225	1.01	252	478.98
Aveleira Deposit									
Aveleira	0.475	0.363	0.013	1.766	0.204	0.452	0.952	134	545.98

13.7 Grade Capping

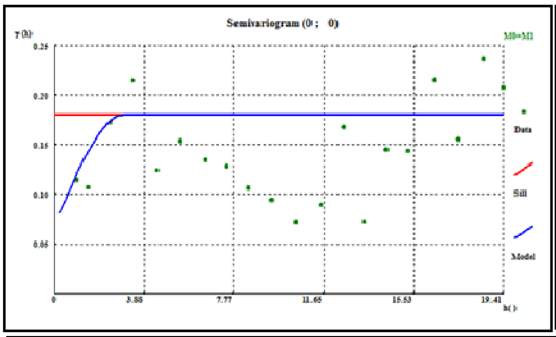
Statistical analysis of the 1 m composite data indicated that grade capping was not required since there were no extreme outliers encountered, and in most instances high grades are supported by neighbouring high grades in the same drillhole, often with additional support from neighbouring drillholes. Any isolated higher grade samples would also be accounted for in the kriging process, which requires a minimum number of composites to estimate a block value. This dependence reduces the impact of individual high grades on the block values.

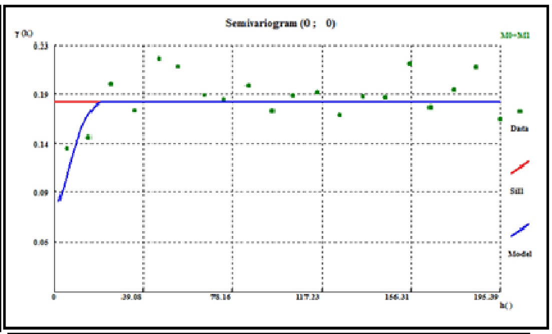
13.8 Variographic Analyses

Spatial analysis was undertaken only at the SPA deposit. Experimental directional variograms were unsuccessful in the M0+M1 and M2 domains, whereas in the L+LL domain directional variograms were obtained.

1 m lag omnidirectional variograms were used to define the nugget effect in each domain. For solids M0+M1 and M2 omnidirectional variograms, with a longer lag were used to model an appropriate range in the plane of the deposit.

Each experimental variogram was fitted with a spherical model, which despite good geological continuity showed grade continuity to have a low range on average. Experimental variograms and fitted models are presented in Figure 13-7 to Figure 13-9.

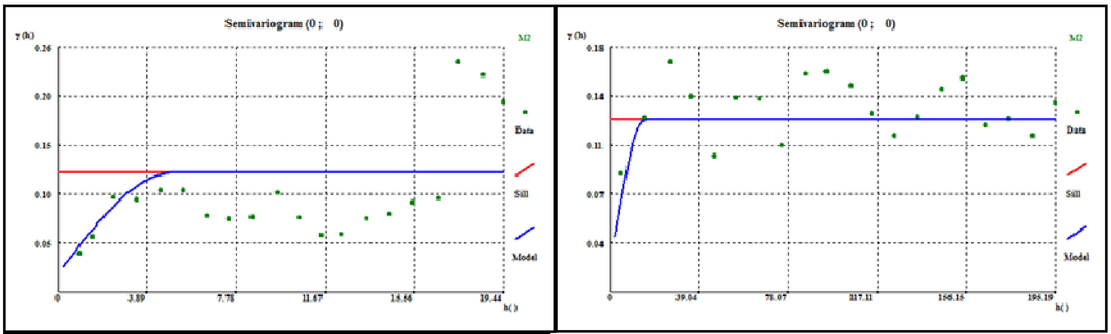




Orientation	Down-hole variogram
Lag (m)	1
Tolerance (^{0})	180
C0	0.07
C1	0.109
A1(m)	**3**
Sill (m)	0.179

Orientation	Omnidirectional variogram
Lag (m)	10
Tolerance (^{0})	180
C0	0.07
C1	0.109
A1(m)	**20**
Sill (m)	0.179

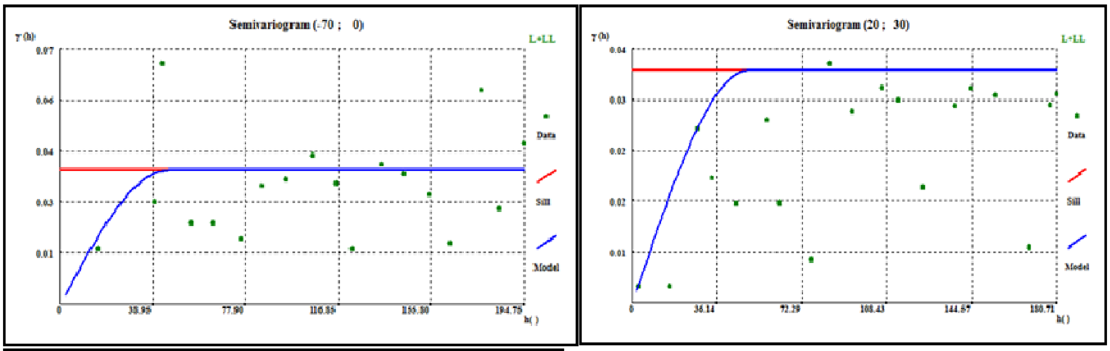
Figure 13-7: Down-Hole Variogram (left) and Omnidirectional Variogram (right) for Solids M0+M1



Orientation	Down-hole variogram
Lag (m)	1
Tolerance (^{0})	180
C0	0.02
C1	0.105
A1(m)	**5**
Sill (m)	0.125

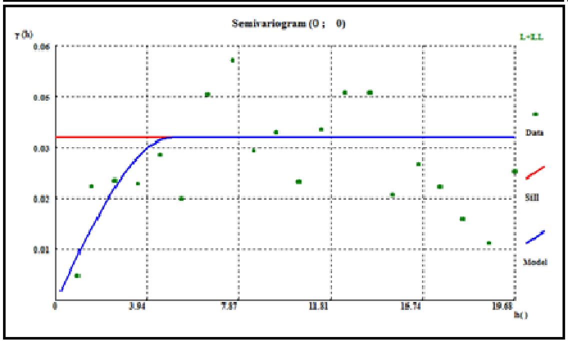
Orientation	Omnidirectional variogram
Lag (m)	10
Tolerance (^{0})	180
C0	0.02
C1	0.105
A1(m)	**15**
Sill (m)	0.125

Figure 13-8: Down-Hole Variogram (left) and Omnidirectional Variogram (right) for Solids M2



Orientation	Strike variogram
Lag (m)	10
Tolerance (^{0})	30
C0	0
C1	0.037
A1(m)	**45**
Sill (m)	0.037

Orientation	Dip variogram
Lag (m)	10
Tolerance (^{0})	30
C0	0
C1	0.037
A1(m)	**50**
Sill (m)	0.037



Orientation	Down-hole variogram
Lag (m)	1
Tolerance (^{0})	180
C0	0
C1	0.037
A1(m)	**5**
Sill (m)	0.037

Figure 13-9: Strike (top left) and Dip Variogram (top right) and Down-Hole Variogram (bottom) for Solids L+LL

13.9 Block Model Construction

A block size of 12.5 x 12.5 x 2.5 m^3 was chosen and is considered reasonable for the average drillhole spacing in the better drilled areas. Each block model is rotated 10° clockwise, to be aligned with the model edges with the strike of the mineralisation. Block model parameters are presented in Table 13-3.

Table 13-3: Block Models Parameters from Tabuaço (SPA) and Aveleira Deposits

Solid/Wireframes	Dimension	Origin	Block size (m)	Number of blocks
M0+M1	X	62,4440	12.5	37
	Y	4,549,255	12.5	28
	Z	412.5	2.5	55
M2	X	624,398	12.5	53
	Y	4,549,237	12.5	31
	Z	400	2.5	50
L+LL	X	624,429	12.5	52
	Y	4,549,270	12.5	30
	Z	387	2.5	60
Aveleira	X	624,082	12.5	26
	Y	4,550,147	12.5	22
	Z	350	2.5	45

13.10 Grade Interpolation

Grade data for each of the domains were interpolated into the respective wireframe domain using three search pass Ordinary Kriging ("OK") in SPA and two pass inverse distance squared weighting at Aveleira. The search ellipsoid was defined based on each deposit's average dip and strike orientations and drilling spacing. A minimum number of 5 samples and a maximum of 75 samples were used in each block estimation. Because Aveleira is similar to SPA in terms of geology and mineralisation, the same search ellipsoid parameters were applied, represented in Table 13-4 using Pass 1 and Pass 2 parameters.

Table 13-4: Search Ellipsoid Parameters

Search ellipsoid pass	Ellipsoid definition (rotation ZYZ) (^{0})			Ellipsoid ranges (m)		
	Z	Y'	Z	X'	Y'	Z
Pass 1	-90	30	0	40	40	5
Pass 2	-90	30	0	80	80	10
Pass 3	-90	30	0	200	200	10

At SPA the block estimation strategy ensured that all blocks in the parts of the domains which contribute to the resource were assigned a grade. At Aveleira, because a 2 pass search was employed, some blocks did not have grade assigned to them. The transformation to tonnage was done based on a density value of 3.1, which represents the average value of samples with a grade higher than 0.3%.

13.11 Block Model Validation

Block models were validated by several methods:

- Statistical comparison between block and composite data;
- Slope of regression as an indication of estimation quality;
- Visual comparison of drill-hole composites with the resource block grade; and
- Swath plots.

13.11.1 Mean Block Grade Versus Composite Mean Grade

Statistical comparison between block and composite grades was done individually, per domain (Table 13-5). To each wireframe, a cumulative function comparison was undertaken as well as means comparison, in order to determine if there is bias in the estimation process. Means are higher in the raw data, likely due to the relative clustering of drilling in higher grade areas which has been declustered by the ordinary kriging.

Table 13-5: Comparison between Block and Composite Average Grade

Solid	Raw data average grade (% WO_3)	Estimated data average grade (% WO_3)	Relative difference (%)
M0+M1	0.46	0.39	-17.37
M2	0.38	0.32	-21.71
L+LL	0.23	0.18	-30.30
Aveleira	0.47	0.45	-5.50

13.11.2 Visual Comparison

Visual comparison between the block grades and the underlying composite grades in perpendicular and longitudinal sections show acceptable agreement. An examples cross section is presented in Figure 13-10, showing block models and composite grades (% WO_3).

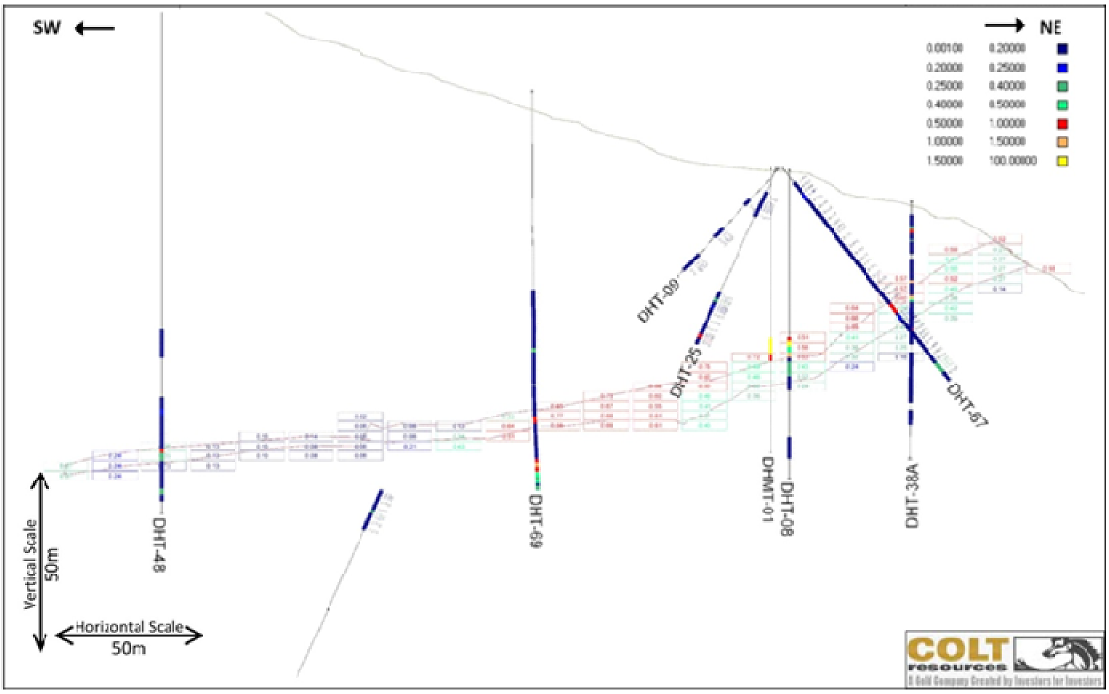


Figure 13-10: Section South West – North East of M2 Solid and Block Model

13.11.3 Swath Plot Analysis

As part of the validation process, the block model grades were compared with the composite grades within slices through the deposit. The results were displayed on graphs to check for visual agreement between grades averaged for sections cut in the X, Y and Z directions. An example is given in Figure 13-11.

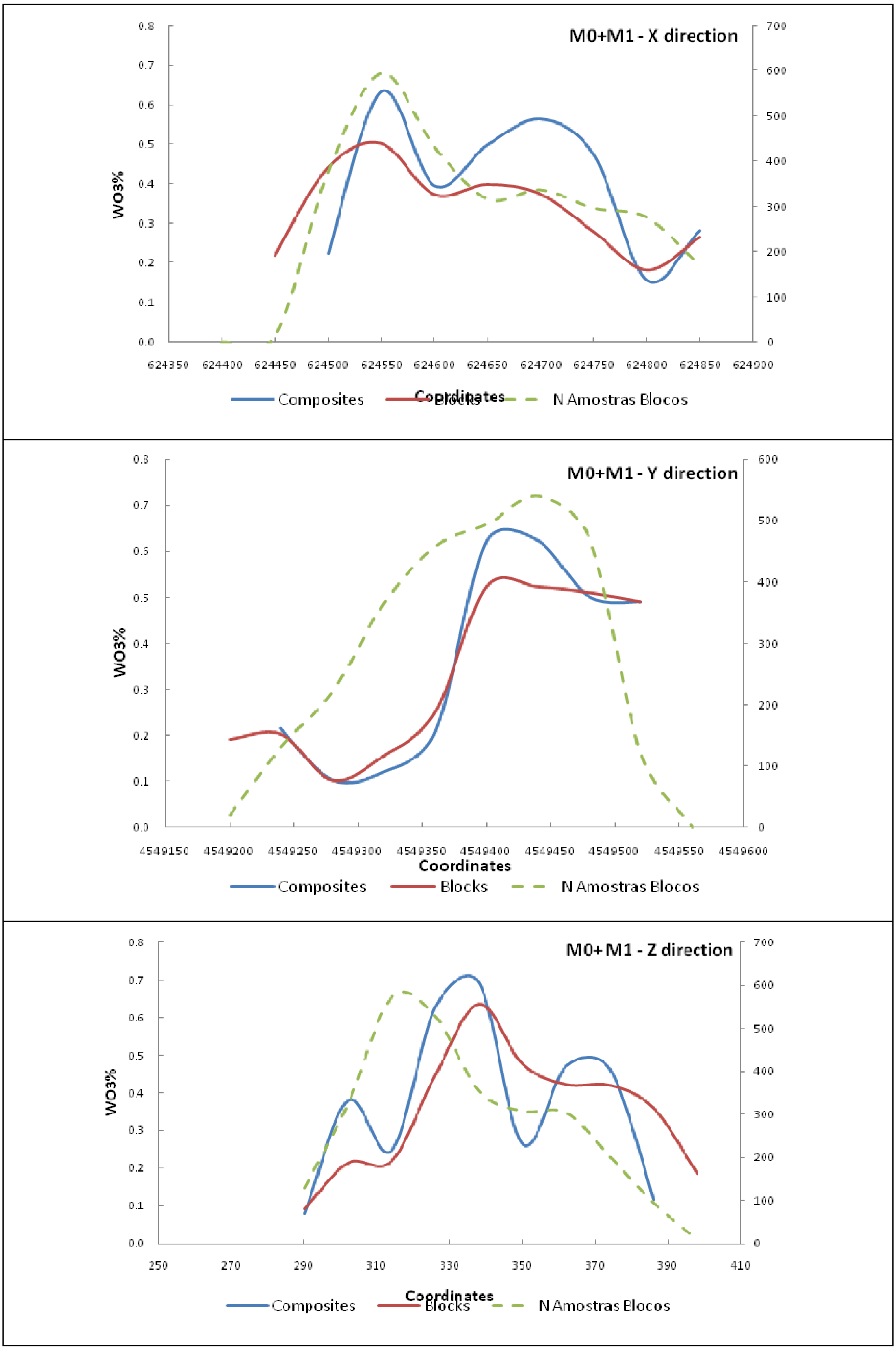


Figure 13-11: Validation Plots – M0+M1 Solid

13.12 Mineral Resource Classification

The drilling and sampling protocols and quality control results provide a high confidence in the database which has been used for the November 2012 MRE. The geological continuity of the mineralised skarns is reasonable to good in the main drilled area, some faulting may affect continuity at a small scale and some faults affecting the dip extensions in the Inferred Mineral Resource have been modelled although with low confidence; more drilling would be required to improve confidence of down dip faulting and continuity in the lower skarn units.

In the well drilled areas, whilst geological continuity is good, the grade continuity is variable; in some places there are consistent and consecutive high grade drilling intersections whilst in some other areas the grade continuity is interrupted by low grade intersections. This suggests that mineralisation is bound within the continuous skarn horizons but occasionally it can be very weak.

Colt and SRK have relied on visual inspection of the grade block models and slopes of regression values which indicate the quality of block estimates to determine substantial areas of the deposit where block grade estimates are of reasonable quality and demonstrate large continuous mineralised volumes for which the average grade of the blocks is known with a reasonable degree of confidence: these areas have been classified as Indicated Mineral Resources and other areas have been classified as Inferred Mineral Resources

13.13 Mineral Resource Statement

A Mineral Resource estimate for the Tabuaço deposit was completed by Colt and reviewed by Martin Pittuck, Director and Corporate Consultant for Mining Geology with SRK, with input provided by Filipa Matias, Colt's Geological Engineer. Mr Pittuck is considered a Qualified Person as defined in National Instrument 43-101. The effective date of this resource estimate was 3rd October, 2012.

Colt prepared a 3D geological model based on diamond drilling and mapping which resulted in a robust demonstration of the geological continuity of mineralised skarns at the Tabuaço deposit. The drilling, sampling and assaying was carried out to a high standard. The continuity of grade within the main mineralised zones is variable, but sufficient in many places to allow estimation of block grades with a reasonable level of confidence. The resultant model demonstrates that there are substantial parts of the deposit which present a continuous body of mineralisation above a cut-off grade of 0.3% WO_3 which have sufficient dimensions and grade to have reasonable prospects for eventual economic extraction according to the mining and processing studies carried out to date.

Mineral Resources have been classified and reported in accordance with standards as defined by the CIM "CIM Definition Standards - For Mineral Resources and Mineral Reserves", prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council on December 11, 2005 and as amended on November 27, 2010.

SRK has reviewed the process and outcomes in sufficient detail to adopt and report the resultant estimate given in Table 13-6.

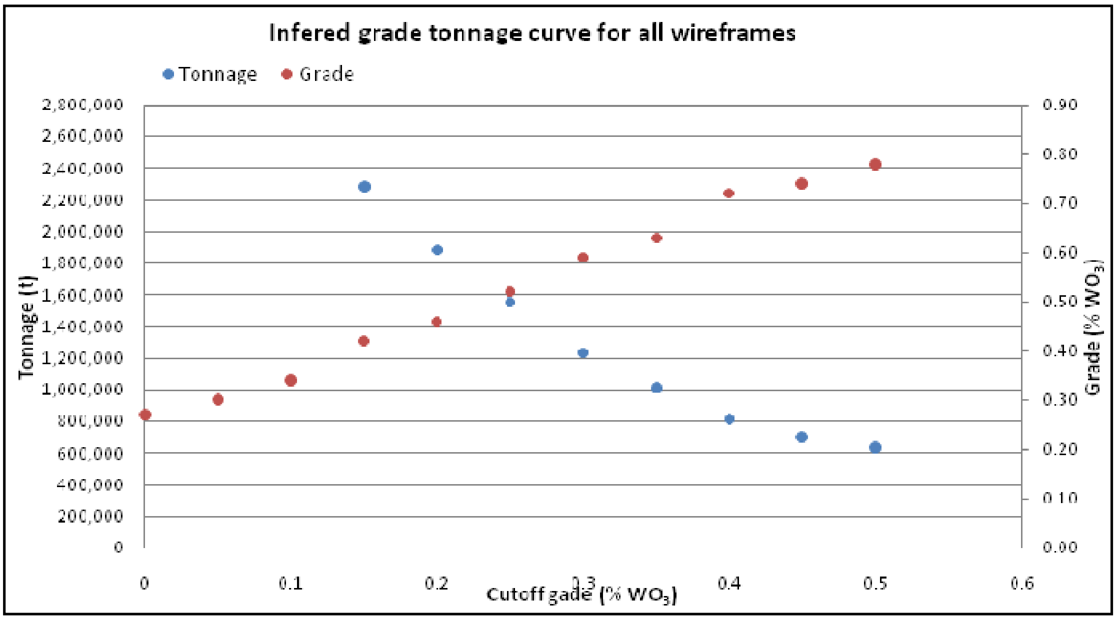
Table 13-6: Mineral Resource Estimation for the Tabuaço Project effective as the 3 October 2012 using a Cut Off of 0.3%WO_3.

Classification	Tonnage	Grade	Contained Metal	Contained Metal	Contained Metal
	kt	% WO_3	t WO_3	M lb WO_3	k MTU WO_3
Indicated Mineral Resources	1,495	0.55	8,150	18	815
Inferred Mineral Resources	1,230	0.59	7,200	16	720

1. Mineral Resources are not Mineral Reserves and there is no assurance that any, or all of the Mineral Resources will be converted to Mineral Reserves.
2. The tonnage, grade and contained metal values have been rounded to reflect the accuracy of the Mineral Resource Estimate. Numbers may not add due to rounding.
3. The Mineral Resources are stated above a cut-off grade of 0.3%WO_3, based on realistic mining and processing cost and recovery assumptions.

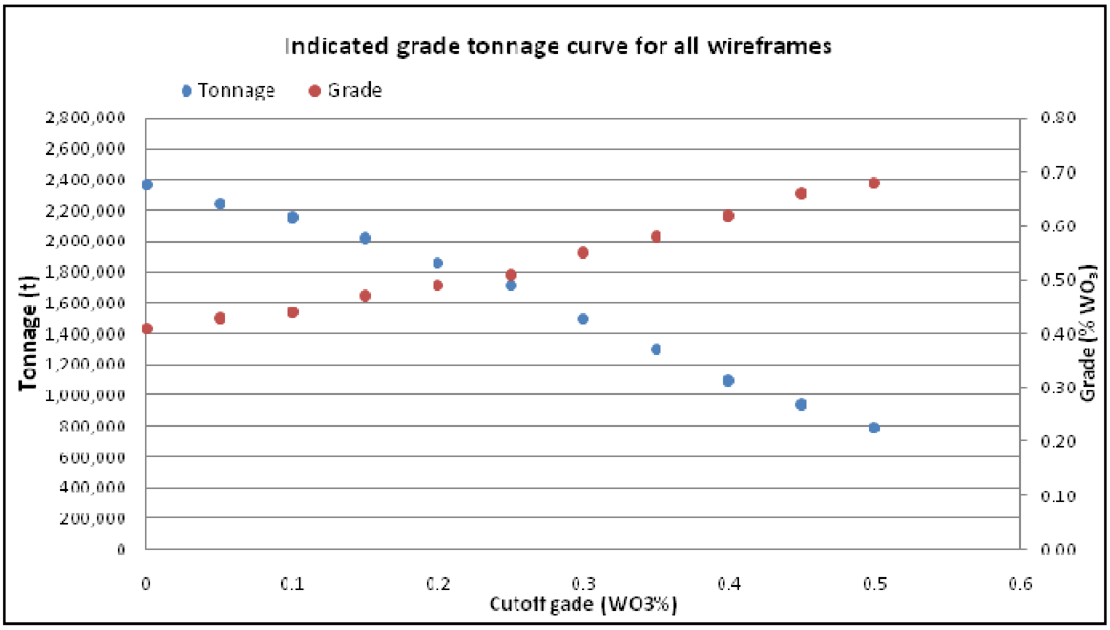
13.14 Grade Tonnage Curves

The MRE is sensitive to cut-off grades. A breakdown of the tonnage and grade of the defined domains is presented below in form of table and grade-tonnage curves presented separately for the Indicated and Inferred classifications (Figure 13-12 and Figure 13-13).



Inferred Resources in all wireframes						
Cutoff	Tonnage (t)	Grade (% WO_3)	WO_3 (mtu)	WO_3 (t)	WO_3 (lb)	Volume (m^3)
0	3,950,000	0.27	1,080,000	10,800	23,760,000	1270000
0.05	3,550,000	0.30	1,080,000	10,800	23,760,000	1150000
0.1	2,990,000	0.34	1,030,000	10,300	22,660,000	960000
0.15	2,280,000	0.42	950,000	9,500	20,900,000	740000
0.2	1,880,000	0.46	870,000	8,700	19,140,000	610000
0.25	1,550,000	0.52	800,000	8,000	17,600,000	500000
0.3	1,230,000	0.59	720,000	7,200	15,840,000	400000
0.35	1,010,000	0.63	640,000	6,400	14,080,000	330000
0.4	810,000	0.72	580,000	5,800	12,760,000	260000
0.45	700,000	0.74	520,000	5,200	11,440,000	230000
0.5	630,000	0.78	490,000	4,900	10,780,000	200000

Figure 13-12: Grade Tonnage Curve for Inferred Resources



Indicated Resources in all wireframes						
Cutoff	Tonnage (t)	Grade (% WO_3)	WO_3 (mtu)	WO_3 (t)	WO_3 (lb)	Volume (m^3)
0	2,365,000	0.41	965,000	9,650	21,230,000	765000
0.05	2,240,000	0.43	965,000	9,650	21,230,000	725000
0.1	2,150,000	0.44	950,000	9,500	20,900,000	695000
0.15	2,020,000	0.47	940,000	9,400	20,680,000	650000
0.2	1,855,000	0.49	910,000	9,100	20,020,000	600000
0.25	1,710,000	0.51	880,000	8,800	19,360,000	550000
0.3	1,495,000	0.55	815,000	8,150	17,930,000	480000
0.35	1,300,000	0.58	760,000	7,600	16,720,000	420000
0.4	1,095,000	0.62	675,000	6,750	14,850,000	355000
0.45	945,000	0.66	620,000	6,200	13,640,000	305000
0.5	790,000	0.68	540,000	5,400	11,880,000	255000

Figure 13-13: Grade Tonnage Curve for Indicated Resources

14 MINERAL RESERVE ESTIMATE

There are currently no Mineral Reserve estimates at the Tabuaço Project.

15 MINING METHODS

15.1 Mining

15.1.1 Introduction

An underground mining method has been selected for the Tabuaço project due to significant surface constraints formed by environmental and cultural heritage restrictions. Given the depth, poor ground conditions and geometry of the deposits, a drift and fill mining method has been selected for the purposes of this PEA. Drift and fill has the added advantage of providing a high degree of ore selectivity, maximising ore recovery and grade compared to bulk mining methods. This method is also compatible with the variable ground conditions expected during production.

The following sources were reviewed and provided the basis of the mining inputs for the PEA:

- *Tabuaço Mining Project – Adaptation of Mining Underground Infrastructure to the Conditions of NI 43-101, issued by SRK Exploration on the 15 November 2012* (“Updated Mining Underground Infrastructure Report”) completed by Fernando Real and Diogo Caupers in 2013;
- *2013_05_24_Tabuaço Plan-Annex1.xlxs* (“Preliminary Mining Plan”) completed by Fernando Real in 2013;
- *Figuras.pptx* completed by Fernando Real in 2013;
- Various AutoCAD (dwg) design files including polylines and triangulations for the mine design completed by Fernando Real in 2013;
- *Analise Comparativa de Alternativas de Operaçao e Transporte de Minério e Estéril, para os Niveis de Extracçao de 1000 e 1500 Toneladas por Dia* (“Mine Production Trade-off Study”) completed by Fernando Real and Diogo Caupers in 2012; and
- *São Pedro das Águias Mining Project: Experimental Mine Plan – Descriptive Document* (“EMP”) completed by Fernando Real, A. Martins and Diogo Caupers in 2012.

15.1.2 Historical Trade-off Studies

The 2012 Mine Production Trade-off Study compared the capital and operating costs of two production rates: 1,000 tpd and 1,500 tpd. This study was based on a preliminary mining plan and not a detailed production schedule. The results showed that a production rate of 1,500 tpd would cost approximately EUR 34/mtu less than a 1,000 tpd operation.

Various ore and waste transport methods along with the location of the processing facilities were also evaluated to determine the most cost effective solution. The following options were evaluated:

- Loading ore and waste directly with trucks and direct transportation to Passa Frio;
- Loading ore and waste for primary crushing underground and transportation to Passa Frio by underground conveyor;
- Loading ore and waste for primary crushing underground and transportation to Passa Frio by trucks;
- Loading ore and waste for primary and secondary crushing underground and transportation to Passa Frio by trucks; and

- Loading ore and waste for primary and secondary crushing followed by grinding underground and transportation to Passa Frio by pipeline.

The options were evaluated using the following criteria:

- Mine development capital expenditure;
- Mine equipment capital expenditure;
- Transport logistics and simplicity of the operation;
- Personnel requirements;
- Impact on production start date;
- Potential safety risks;
- Environmental impacts; and
- Transport operating costs.

According to the 2012 Mine Production Trade-off Study, the best option was determined to be loading ore and waste with trucks and direct transportation to Passa Frio with the crushing, grinding and concentration facilities located at Passa Frio. SRK has based the PEA on this option.

15.1.3 Mining Method

Access

The main SPA deposit will be accessed via a decline from surface. Previous studies (such as the *Trade-off Study*) established the best locations for surface mine infrastructure including portal access, taking into account the surface disturbance restrictions. The initial underground infrastructure was designed by F. Real, a mining engineering consultant engaged directly by the Client, and provided to SRK for mine planning purposes. Further details on this access and infrastructure design can be found in the 2012 Updated Mining Underground Infrastructure Report. The main decline and accesses to the SPA deposits were redesigned by SRK in order to reduce development requirements.

In addition to the main haulage way, underground support infrastructure such as ventilation raises, workshops and sumps have been designed. An adit has been appended to the decline below the mineralisation to serve the multiple purposes of mine drainage, ventilation exhaust and secondary egress.

A decline access to the Aveleira deposit has been designed, tied into the proposed SPA mine infrastructure.

The conceptual mine infrastructure is shown in Figure 15-1 along with the mineralised areas above the 0.22% WO_3 CoG.

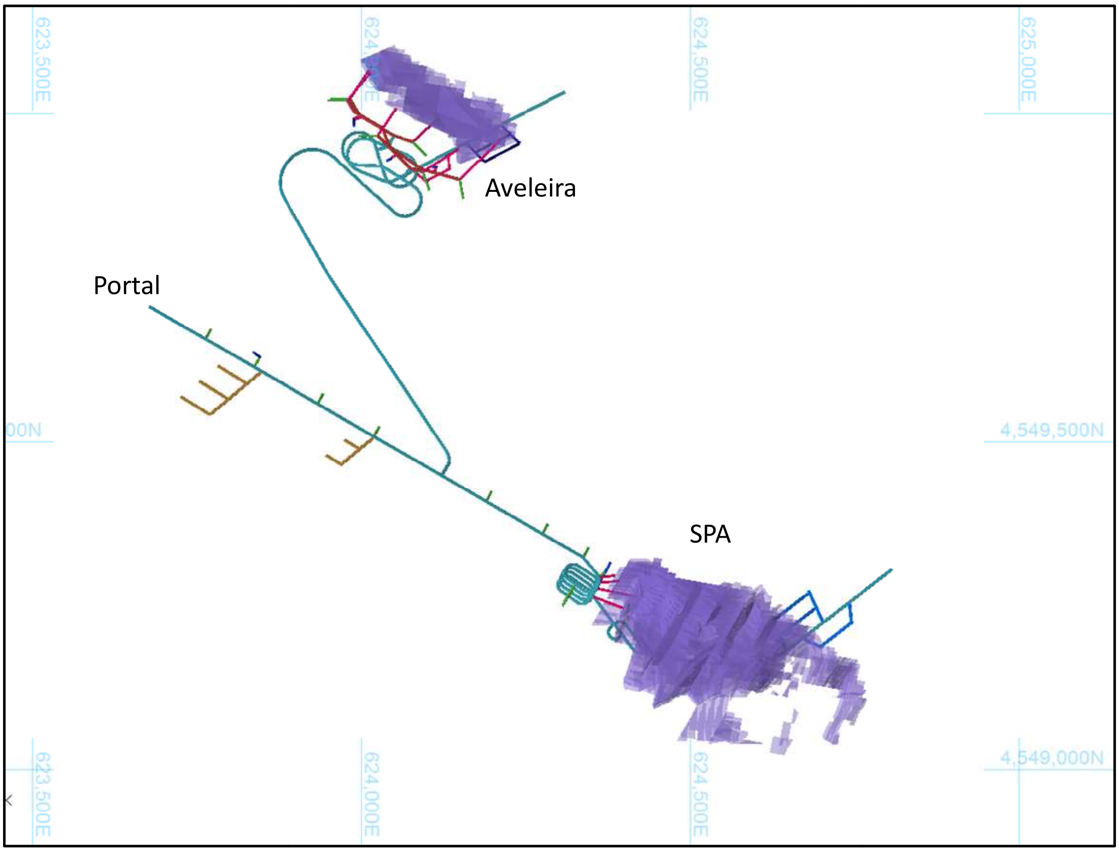


Figure 15-1: Conceptual Mine Infrastructure

Production

Mining of the SPA and Aveleira deposits will be performed using an overhand cut and fill (drift and fill) mining method as presented in Figure 15-2. In order to maximise production, mining will take place simultaneously at four horizons. Mining will advance in a bottom-up direction, with backfill forming the working platform for each cut The first cut from each of the mining horizons will require an engineered solution, such as a sill mat, to create more stable backs for the final production cut from the horizon below. Each level will have secondary development that will enable it to produce from multiple headings simultaneously. Given the shape of the deposit lenses and the variation of the grades within a level, it is recommended that the operation establishes a rigorous grade control program in order to minimise dilution.

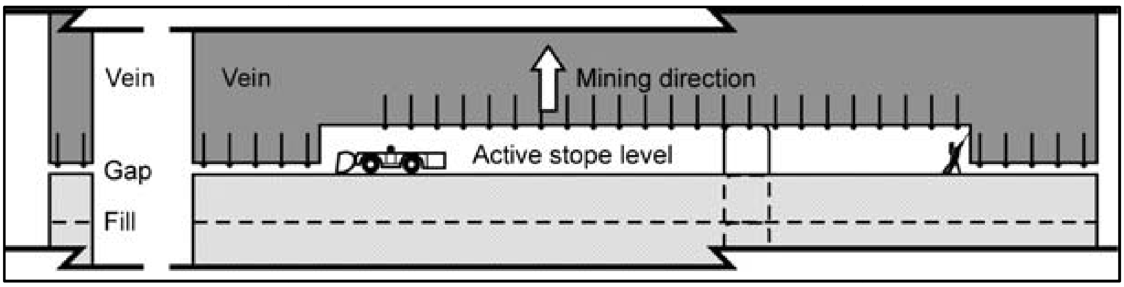


Figure 15-2: Overhand Drift and Fill Mining Method

An underhand cut and fill mining method (Figure 15-3) will be used as an alternative, should the geotechnical conditions decrease the production rate. This will allow most of the production activities to take place under backfill with a higher cement content. The necessity of using this method is to be determined during the proposed trial mining.

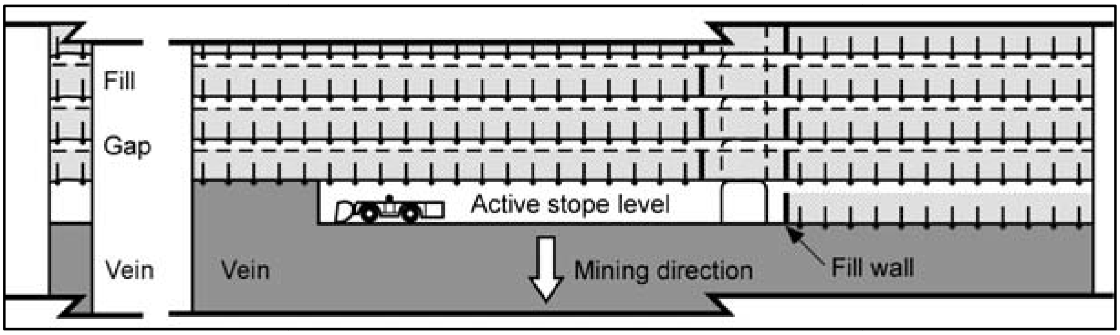


Figure 15-3: Underhand Drift and Fill Mining Method

Material will be excavated using mechanised trackless mining equipment such as jumbos, bolters, load-haul-dump units and mining haul trucks. Material will be loaded into haul trucks to be transported to surface via the decline (Figure 15-4), where it will either be routed to a waste dump or to the mineral processing facilities.

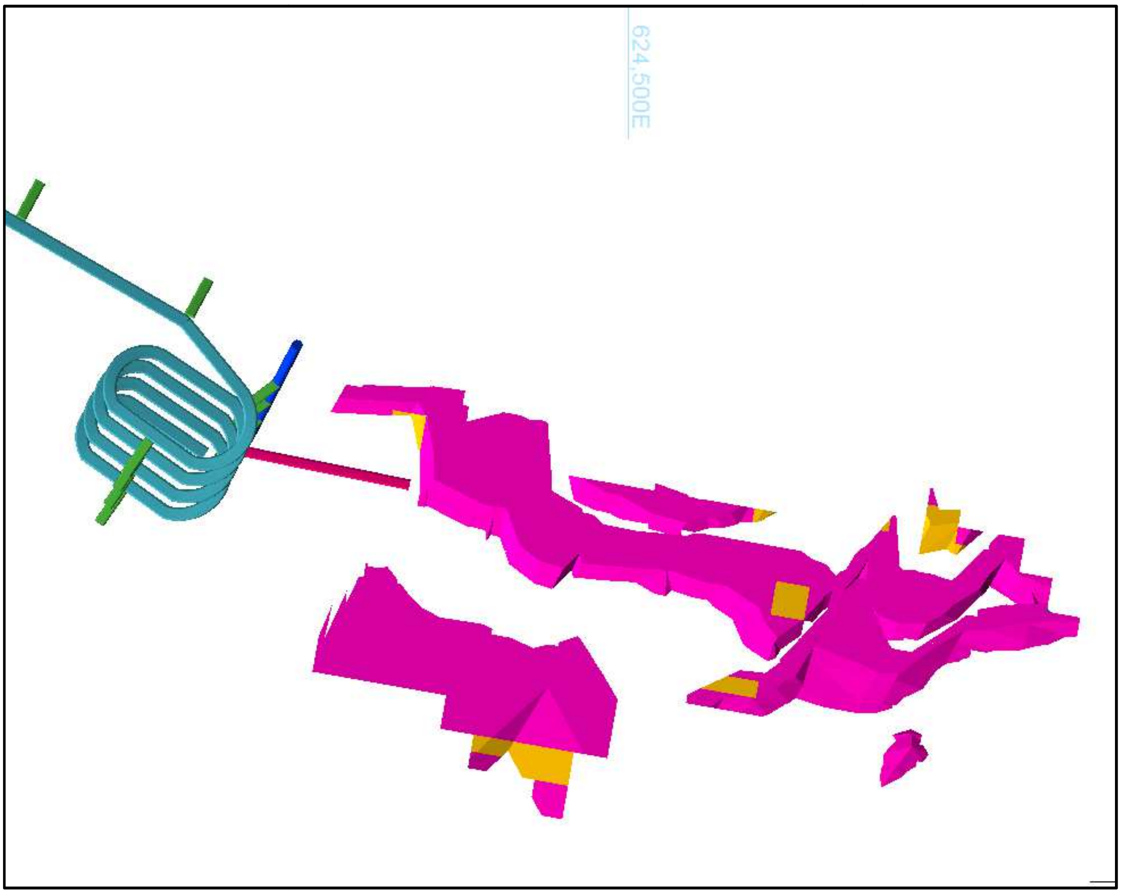


Figure 15-4: Plan View of a Conceptual Mining Horizon at SPA showing decline access

Backfill

Based on information provided by the Client, the mining method will incorporate the use of paste backfill which will make use of tailings generated at the processing facilities. A backfill density of 1.8 t/m^3 has been assumed in order to calculate backfill quantity requirements.

Equipment

Based on discussions with the Client, it has been assumed that the operation will use contract mining and therefore the mobile mining equipment will be provided by a mining contractor. Ventilation fan and dewatering pumping equipment will need to be purchased, as well as light vehicles for mine management and technical services.

Labour

A labour estimate was performed as part of the conceptual mining trade off study. A five day week, three shifts per day work roster is proposed. The mine will produce material during 265 days per year and the plant will process 360 days per year. SRK has examined the labour estimates and finds that operator requirements proposed in the conceptual mining study are sufficient to meet the needs of the proposed mining schedule. Mine labour costs have been included in the mining unit costs, as quoted by a local contractor.

Dewatering

Mine water will be pumped to intermediate sumps throughout the mine, from where it is to be pumped to the processing facilities.

Ventilation

Ventilation intake shafts have been included in the capital development design. The 4 m diameter shafts can be developed with a raisebore machine once the connecting horizontal development is in place. Mine air will be exhausted to the drainage adit. The capital equipment budget includes main and auxiliary fan costs, while the operating cost estimate has made an allowance for on-going ventilation related costs.

SRK recommends that in future studies the proposed ventilation circuit be revisited to allow provisions for ventilating the main decline from the portal to the workshop and to reduce the number of ventilation raises.

15.1.4 Life of Mine Plan

Modifying Factors

The following modifying factors were used to generate the mine plan: a mining recovery factor of 95% and a dilution factor of 17% based on a cross-sectional analysis. SRK believes that the 95% mining recovery factor is acceptable due to the high selectivity of the mining method. SRK believes this dilution factor is appropriate due to the geometry of the resource and the relatively poor quality of the ground conditions. Furthermore, this dilution factor takes into account the lack of a detailed design of the deposits' ore drives, as SRK expects that waste will be mined to access the high grade zones of the deposit at each production level. This material is expected to be routed to surface for processing (if above the marginal cut-off grade) or for disposal at the waste dump.

The following Figure 15-5 and Figure 15-6 present the graphical results of the cross-sectional analysis undertaken by SRK to determine the dilution and loss factors. Blocks above the CoG are being mined using 4 m by 4 m drives (shown in green), positioned horizontally to reduce dilution.

Where possible, the backs of the drives are following the hanging wall to reduce dilution up to a maximum of 6 m (shown in orange). If the drive height would be higher than 6 m, another drive above it is required which leads to more dilution (shown in purple) or if the additional drive is not economic, more losses (shown in red). SRK notes that this is a preliminary estimation of dilution and that future studies require a three dimensional design to determine the dilution and loss factors, taking into consideration the morphology of the mineralisation along strike.

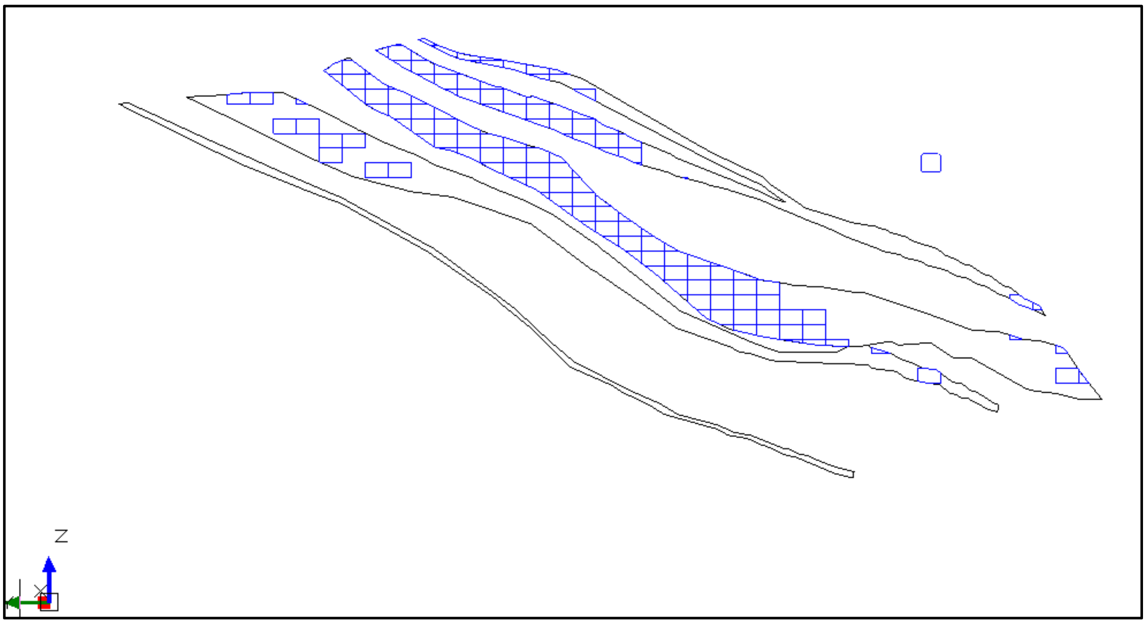


Figure 15-5: Cross-sectional view of blocks with WO_3>COG within the mineralisation wireframes

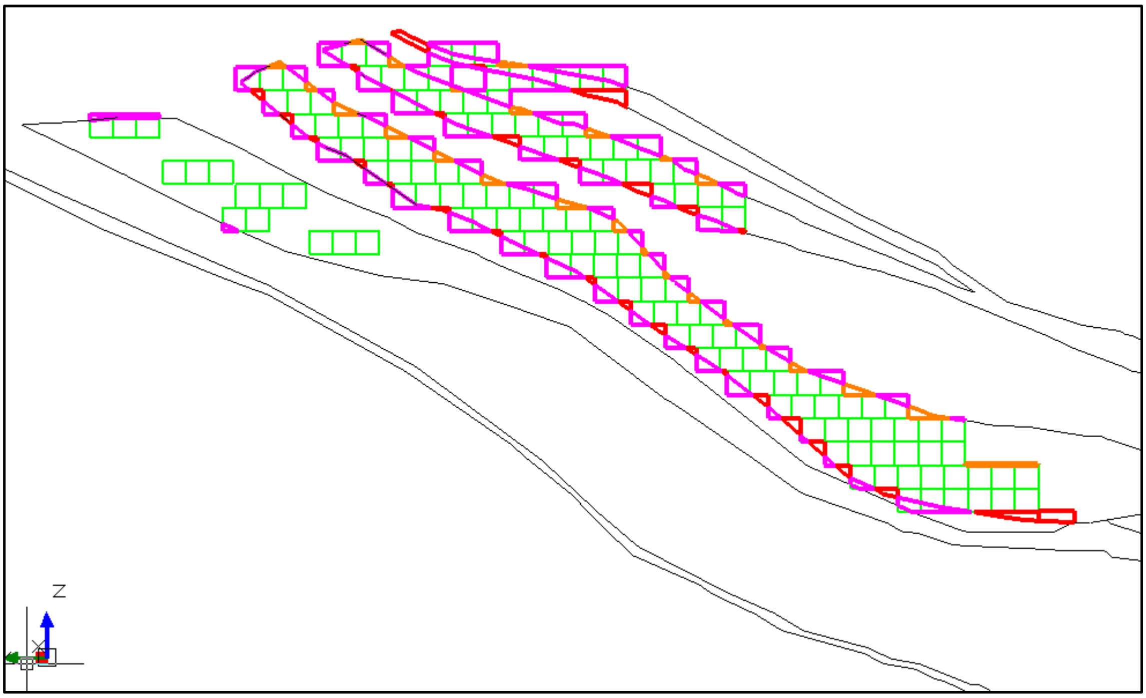


Figure 15-6: Cross-sectional loss and dilution analysis

Given that the deposit intersects with surface topography, a crown pillar of 15 m was removed from the resources in order to maintain geotechnical stability. A further exclusion zone was applied to the SPA deposit surrounding the convent to prevent blast vibration damage. The exclusion zone comprises a 60 m stand-off pillar from the convent as shown in Figure 15-7.

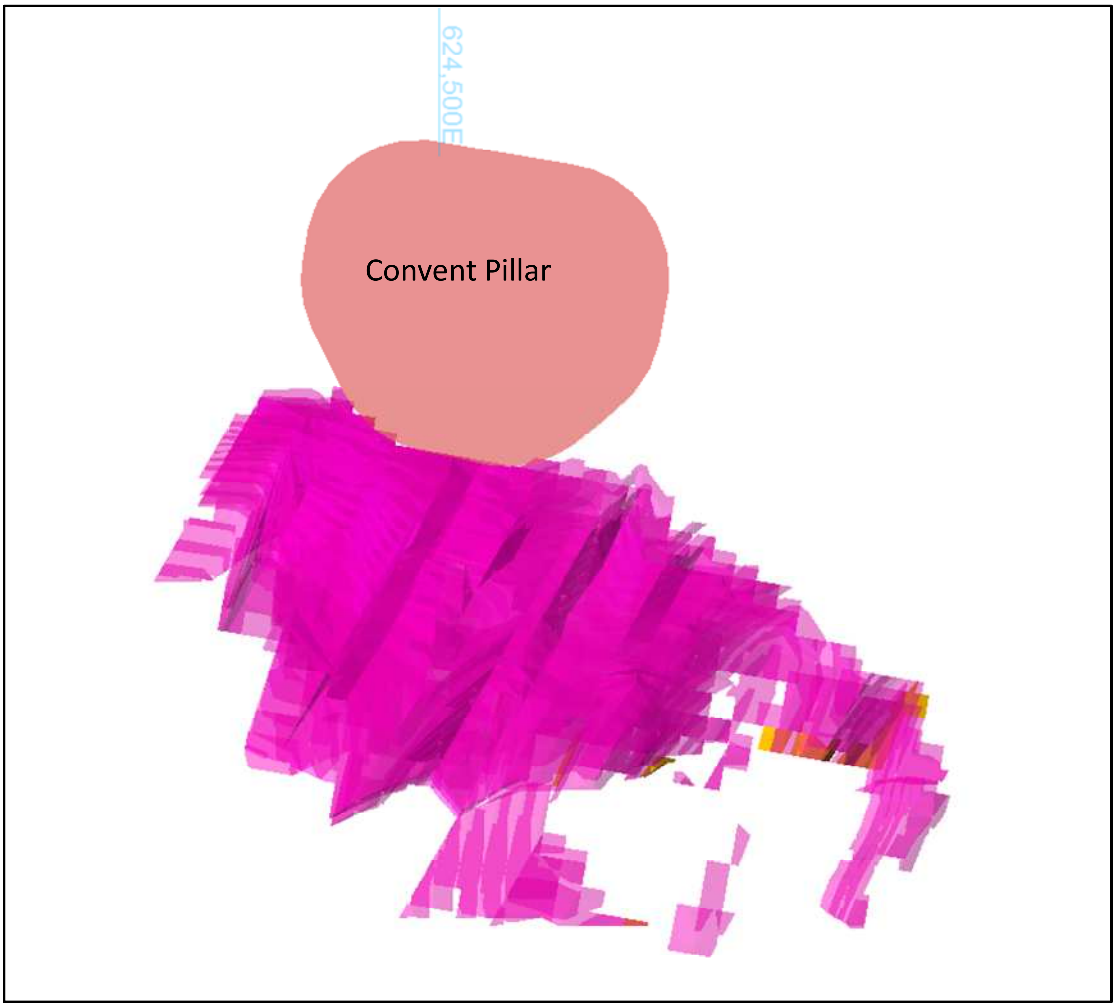


Figure 15-7: Mining Exclusion Zone for Convent Preservation

Cut-Off Grade

Based on the mining and processing costs provided by the Client, the break-even CoG for mining the Tabuaço mineral deposits is 0.22% WO_3. The marginal cut-off grade for processing mined material is 0.10% WO_3.

Both CoGs were considered in delineating material for the mining schedule. Mining solids were first generated by applying a 12.5 m x 6.25 m grid along the strike of the deposits at each mining level, that is, in 4 m intervals. These mining solids were evaluated against the block models and classified as ore for scheduling if they met the following CoG selection criteria:

- Above break-even cut-off grade; or
- Above marginal cut-off grade if the material must be mined to access the higher grade material previously selected.

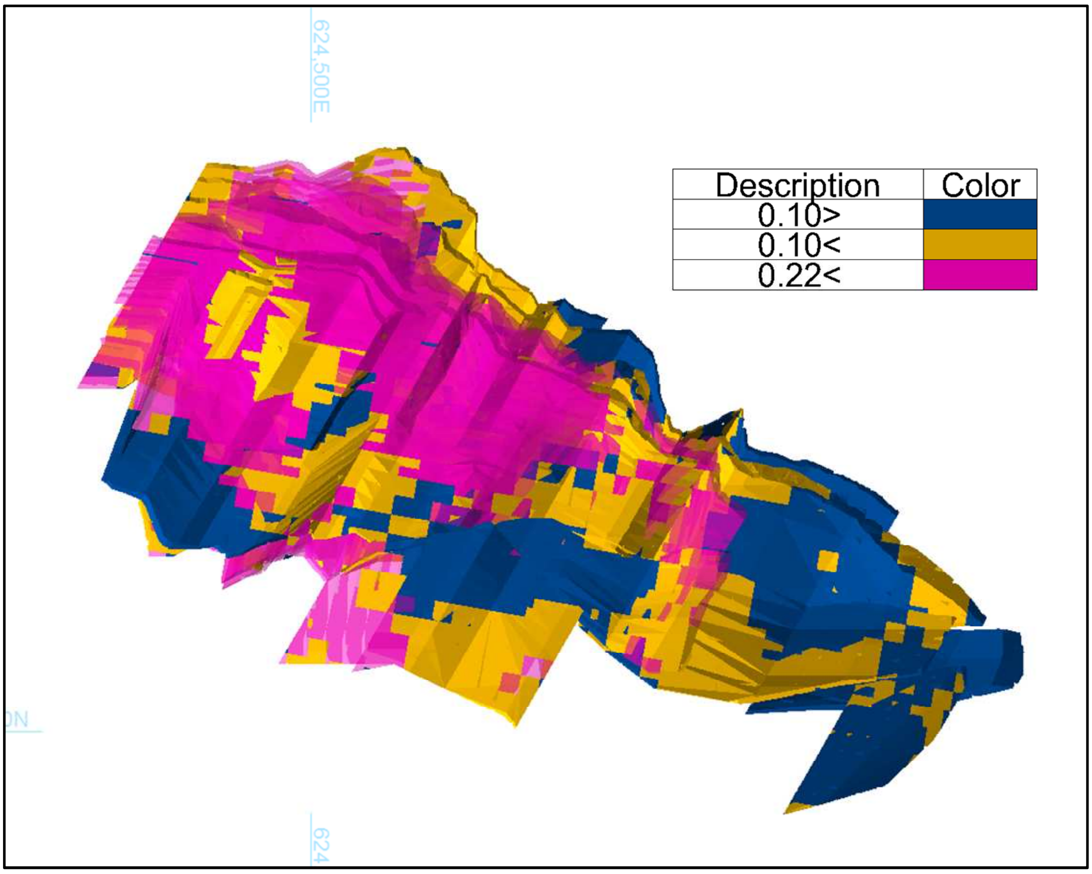


Figure 15-8: Unit mining blocks prior to selection

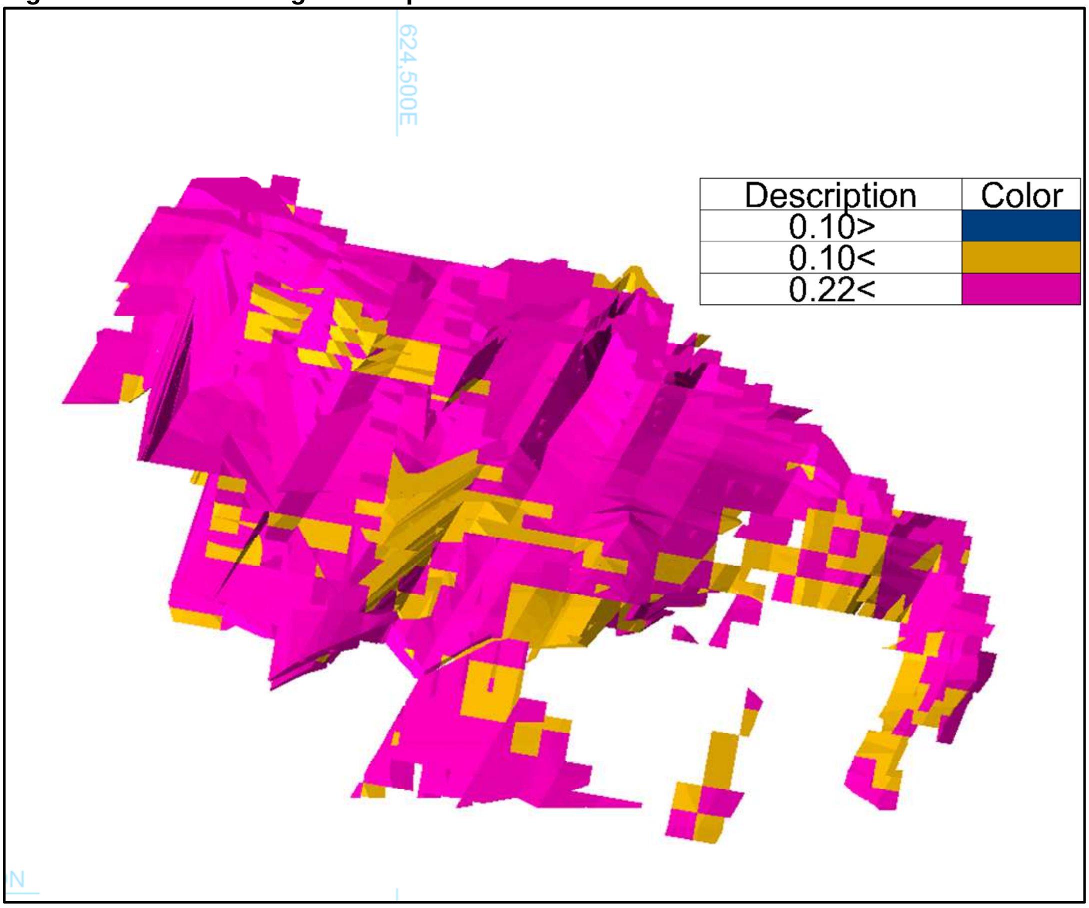


Figure 15-9: Unit mining blocks after selection

Mine Schedule

Capital development has been sequenced and scheduled with a nominal rate of 100 m per month per active face, with an overall limit of 32 m per day advance. These rates translate to a cut per shift, assuming a 3.5 m advance length. Given these assumptions, SRK estimates that mine development will take approximately two years. Production can commence once the capital development is completed. SRK assumes that mine development will be performed by a mining contractor in possession of its own mining equipment.

A 1,500 tpd ore production was used as a target for production scheduling purposes. This rate was identified as an option in the conceptual mining trade-off study and recommended for further study in the PEA. The average mining rate for the first five years resulting from the scheduling exercise is 1,406 tpd, which SRK feels is the maximum achievable given the mining method and spatial constraints.

The main focus of the schedule is on the SPA deposit. Development and mining of the Aveleira deposit was sequenced such that it complements the production profile once mining at full production rate in the SPA deposits becomes unsustainable.

For mine scheduling purposes, it was assumed that production will not be constrained by backfill placement.

Life of Mine Plan

The Life of Mine plan was generated using Deswik's mine planning software. The results of the mine scheduling exercise are presented in Table 15-1.

Table 15-1: Tabuaço Life of Mine Plan

	Units	Total	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Total Production													
Ore	t	**3,556,718**	-	328,253	364,285	393,863	392,254	384,631	385,963	383,612	395,394	347,453	181,010
Diluted WO_3 Grade	%	**0.386**	-	0.44	0.41	0.44	0.36	0.40	0.34	0.38	0.41	0.34	0.33
Horizontal Development	m	**5,949**	1,037	1,402	357	357	396	615	1,377	409	-	-	-
Vertical Development	m	**435**	63	107	-	-	-	-	266	-	-	-	-
SPA													
Ore	t	**2,582,073**	-	328,253	364,285	393,863	392,254	384,631	360,909	237,821	120,058	-	-
Diluted WO_3 Grade	%	**0.375**	-	0.44	0.41	0.44	0.36	0.40	0.28	0.29	0.33	-	-
Horizontal Development	m	**2,865**	1,037	1,402	357	69	-	-	-	-	-	-	-
Vertical Development	m	**169**	-	208	551	76	-	-	-	-	-	-	-
Aveleira													
Ore	t	**974,645**	-	-	-	-	-	-	25,054	145,791	275,336	347,453	181,010
Diluted WO_3 Grade	%	**0.418**	-	-	-	-	-	-	0.49	0.50	0.53	0.34	0.33
Horizontal Development	m	**3,084**	-	-	-	288	396	615	1,377	409	-	-	-
Vertical Development	m	**266**	-	-	-	-	-		266	-			
Backfill Placement	t	**1,919,966**	-	157,476	216,364	125,609	243,906	252,218	192,812	215,377	204,780	165,868	145,556

15.1.5 Conclusion

Based on a scoping level of study the following conclusions can be made:

- With the current resource, the mine life at a 0.22% WO_3 CoG is one year of pre-production followed by ten years of production;
- The average RoM grade over the mine life is 0.39% WO_3;
- The average daily ore production rate for the life of mine is 1,415 tpd; and
- The peak ore production rate is in year 9, at 1,582 tpd.

The following recommendations should be undertaken in the next phase of the project:

- Review mining operating costs, especially the development costs;
- Review mine infrastructure designs in order to optimise access to material above the CoG;
- Review equipment productivities to ensure that the mining rate is achievable given the access and deposit geometry;
- Review the ventilation concept in order to ensure adequate air flow throughout the mine; and
- Evaluate various backfill systems to optimise the proposed mining method.

15.2 Geotechnical Assessment

15.2.1 Geological and Geotechnical Conditions

The engineering geological assessment is based on a review of core photographs and basic geotechnical logs provided by the Client, which contains Rock Quality Designation (“RQD”) and core recovery information.

The rock mass is dominated by four main lithologies: carbonates, skarn, schists and granites. Each lithology has very different geotechnical properties and must be considered separately when assigning geotechnical parameters. Table 15-2 lists the individual lithological units recorded in the geological logs together with their proposed geotechnical domain allocation (Column GT).

Table 15-2: Lithological Descriptions to Geotechnical Domains

Lithology	LITH	GT	HW	ORE	FW
Overburden	**OVB**	OVB	HW		
Solution cavity	**CAV**	CAV	HW	ORE	FW
Fault, obvious	**FLT**	FLP	HW	ORE	FW
Fault, probable	**FLP**	FLP		ORE	FW
Fault, Filled with gauge	**FLG**	FLP	HW	ORE	FW
Fault Breccia	**FBX**	FLP	HW	ORE	FW
Quartz vein	**QZV**	QTZ	HW	ORE	FW
Quartz-chlorite vein	**QCV**	QTZ			
Calcite vein	**CRV**	CAB			
Feldspar-chlorite vein	**FCV**	GRN			
Aplite	**APL**	GRN	HW	ORE	FW
Pegmatite	**PEG**	GRN	HW	ORE	FW
Granite	**GNT**	GRN	HW	ORE	FW
Carbonates, calcschists	**CRB**	CAB	HW	ORE	FW
Skarn, L-type	**SKL**	SKN	HW	ORE	FW
Skarn, M-type	**SKM**	SKN	HW	ORE	FW
Skarn, indifferentiated	**SKN**	SKN	HW	ORE	FW
Calcsilicate lithology, poorly developed skarn	**CSR**	SKN	HW	ORE	FW
Biotite schists	**BIX**	SCH	HW	ORE	FW
Biotite schists with psamitic laminations	**BLX**	SCH			
Sericitic schists	**SRX**	SCH			
Black/dark-grey schists, fine grained	**BLS**	SCH	HW	ORE	FW
Migmatites	**MGM**	GRN	HW	ORE	
Core Loss	**CLS**	NA	HW	ORE	FW
Re-collar RC	**PRC**	NA			

Engineering geological descriptions and core photograph examples of each of the four major lithologies and geotechnical domains are presented in the following sections.

Carbonates

The carbonates geotechnical domain includes the carbonates and calcschists (“CRB”) and the calcite veining (“CRV”). The materials can generally be described as very strong to moderately strong (R4/R5), bedded, light cream to grey (light and dark) fine to moderate grained, exhibiting a degree on folding (folds axis can be seen in core).

Discontinuities are generally undulating smooth to planar smooth (as estimated from core photographs), staining is present with calcite veining/infilling. Infill could in part be clay, but this is considered minimal in the overall deposit from core photographs. Joint number ranges from 2 to 3, 3R in parts.

An example of the carbonate unit intersected in Borehole DHT-16 is shown in Figure 15-10.

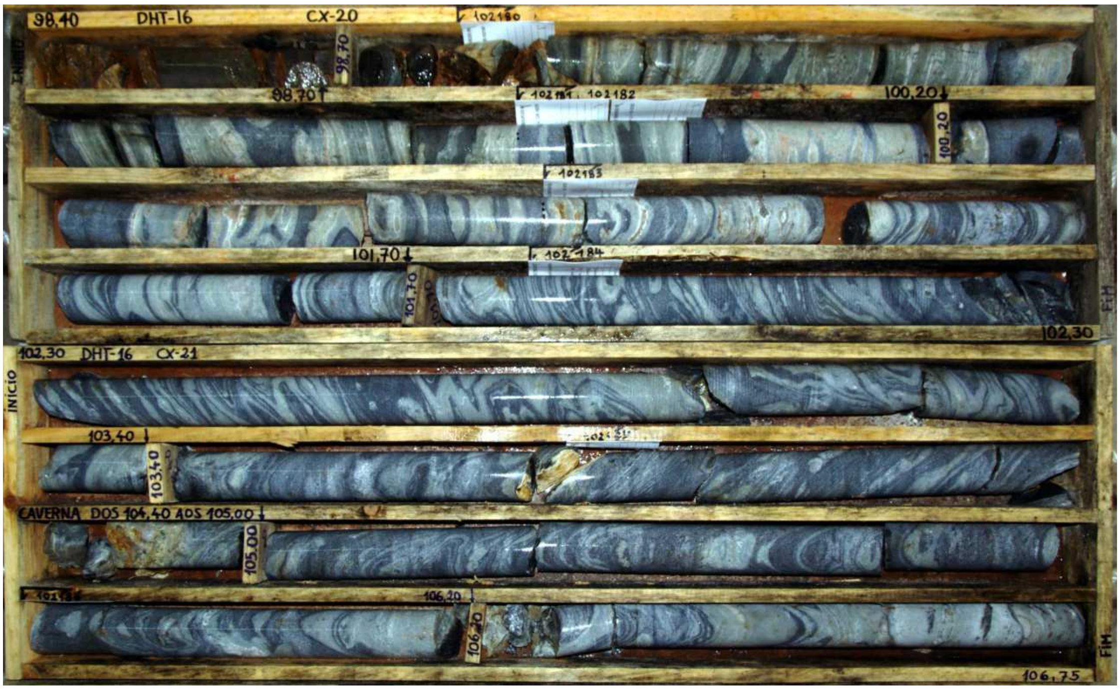


Figure 15-10: Carbonate DHT-16 98.40 to 106.75 m

Skarn

The skarn geotechnical domain (Figure 15-11) includes the L-type and M-type skarns with undifferentiated and calc-silicate material that exhibit poorly developed Skarnation. These materials can be described as strong to weak (R4 to R1, generally R2 and R3), greenish cream to light grey moderately to fine grained, structureless in parts with minor folding.

Discontinuities are generally undulating rough to planar smooth, stained (Fe) to calcite infill (visual identification), discontinuity wall alteration is present on occasion, with a low incidence of clay (core photographs). RQD percentage is low (generally compared with other geotechnical domains). The structures within the skarn range from two visible joint sets plus random joints to completely crushed. However on average there are generally three joint sets with a series of high and low angle cross cutting discontinuities.

The skarn geotechnical domain in some instances appears highly broken as shown in Figure 15-12, but it is unclear whether this is a product of poor drilling or a function of the rock mass. There is significant core loss (logging database and core photographs) which can be attributed to very poor ground conditions. This material is within the Middle mineralised zone.

Figure 15-11: Skarns DHT-12 85.10 to 92.55 m

Figure 15-12: Skarns DHT-09 98.00 to 111.30 m

Schist

The schist geotechnical domain includes biotite schist with and without psamitic laminations and sericitic materials. The schist (Figure 15-13) can be described as very strong to moderately strong (R3 to R5), occasionally weak to very weak (R0 to R2) depending on degree of weathering, dark grey with occasional lighter layers/bands, fine grained, occasionally veined with quartz (visual observation).

Discontinuities are undulating smooth to planar smooth (visual), Fe straining dominates infill (generally) with a low clay probability (core photographs), joint number is 2 to 3 plus random. Fabric/schistosity is the dominant discontinuity with a high angle set mixed with shattered zones.

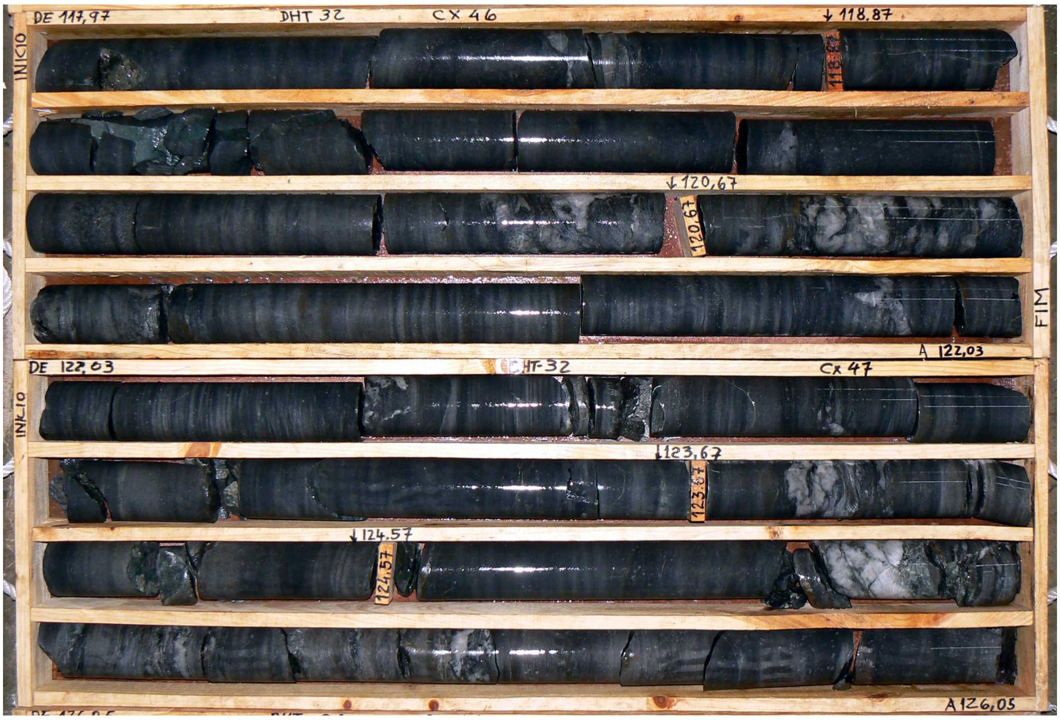


Figure 15-13: Schists DHT-32 117.97 to 126.05 m

Granite

The granite geotechnical domain includes the feldspar-chlorite veins, apilite, pegmatite (course grained granite) and granites. The rock mass (Figure 15-14) is generally characterised as very strong to strong (R4 to R5, depending on alteration and weathering) spotted light cream, pinky, black coarse to moderate grained, occasionally containing porphyritic feldspar.

Discontinuities are undulating, rough to stepped, rough with granular (silty sand) infill to staining (Fe predominates), rarely associated with clay, fracture frequency is low compared to other geotechnical domains with a higher RQD. Joint number ranges from 2 plus random to 3 plus random in parts.

Figure 15-14: Granite from DHT-28: 82.45 to 89.60 m

15.2.2 Mining Domains

The rock mass has been divided into three mining zones. These are Hangingwall ("HW"), Footwall ("FW") and mineralised zone ("ORE"). The HW is defined as being 15 m vertically above the top of the mineralised zone with the FW being 10 m below the base of the mineralised zone. Note that the orebody comprises three mineralised zones: Upper, Middle and Lower. The upper and lower zones are further sub-divided into two zones, Upper 1 and Upper 2, Lower 1 and Lower 2, producing 5 mineralised zones in total. The In order to develop geotechnical parameters for each mining zone, Datamine was used to determine the intersection of the orebody wireframes with the borehole traces. A database of RQD values for each of the main geotechnical domains within each mining zone was extracted from the Datamine model.

15.2.3 Estimation of Q parameters

The resource logging programme collected RQD data which could be utilised in the Q classification system (Barton 2002[2]). The equation used to calculate Q is presented below:

$$Q = \frac{RQD}{Jn} x \frac{Jr}{Ja} x \frac{Jw}{SRF}$$

Where:

- RQD is the Rock Quality Designation (where RQD <10%, 10% was utilised)
- J_n is the Joint set number
- J_r is the joint roughness number
- J_a is the joint alteration number
- J_w is the joint water reduction factor
- SRF is the stress reduction factor.

[2] Barton, N. 2002. Some new Q-value corrections to assist in site characterisation and tunnel design. International Journal of Rock Mechanics & Mining Sciences. 39 (2002) 185-216.

RQD was validated against the logging database and core photographs. The Q classification system can only be used on rock masses which have an RQD of 10% or greater. Where the RQD of any logged core run was less than 10% a default value of 10% was used in the calculation of Q for that run.

As no specific Q geotechnical parameters were logged, SRK has estimated these parameter values from inspection of the core photographs. The parameters Jn, Jr and Ja were estimated from core photographs and are presented in Table 15-3 for each of the main geotechnical domains.

A constant J_w factor of 0.66 was applied to all geotechnical domains. This value describes a rock mass with medium joint water inflow inflow or pressure, occasional out washing of joint infillings.

A constant SRF of 4 has been applied to all geotechnical domains. This value describes a near surfaces rock mass containing single weak zones with clay or chemically disintegrated rock.

Table 15-3: Q Input Parameters for Geotechnical Domains

GT Domain	GT	Joint Number	Jn	Joint Roughness	Jr	Joint Alteration Description	Ja	Joint Water (Jw)	Stress Reduction Factor (SRF)
Fault	FLP	CR	20	PS	1	clay, rock fragments	4	0.66	4
Quartz	QTZ	3R	12	UR	3	clean, little to no fill, clay free, sandy particles	2	0.66	4
Carbonate	CAB	3	9	US	2	slight at depth, no infill, Fe staining, potential for clay due to minerology perhaps	2	0.66	4
Granite	GRN	3	9	UR	3	slight at depth, no infill, staining, sand perhaps	2	0.66	4
Skarn	SKN	3	9	US	2	slight at depth, Fe staining, predominately no clay, not really a geotechnical Schist - more mineral fabric.	2	0.66	4
Schist	SCH	3	9	US	2	slight at depth, Fe staining, predominately no clay,	2	0.66	4

The dominant controlling factor on the Q value is RQD, so although the values of J_n, J_r and J_a have been estimated, and are not definitive or absolute values for the geotechnical domain, their contribution to the final Q value is small compared to RQD. When averaged in the Q formulation J_r/J_a, the change is very small overall. The main contributing factor is the RQD (which appear to be low as a result of poor geotechnical drilling) and the impact it has on the final outcome of the Q value (Table 15-4). There is a very slight improvement from the HW to ORE, but due HW being a mix of HW and Ore the overall difference is small. Table 15-4 shows that the rock mass in each domain and mining zone can be classified as very poor to poor.

Table 15-4: Q Values for Geotechnical Domains and Mining Zones

Lithology	GT	Hangingwall	Q Description	Upper/Middle Ore Interburden	Q Description	Middle/Lower Ore Interburden	Q Description
Carbonate, Carbonate silicate,	CAB	0.814	Very Poor	0.740	Very Poor	0.839	Very Poor
Granite, Pegmatite, Aplite, Feldspar-chlorite vein, migmatites	GRN	0.917	Very Poor	1.197	Poor	0.993	Very Poor
Skarn - All	SKN	0.600	Very Poor	0.766	Very Poor	0.702	Very Poor
Schist - Biotite with psamitic laminations, Sericitic, black/dark	SCH	0.478	Very Poor	0.658	Very Poor	0.604	Very Poor

Lithology	GT	Upper ORE	Q	Middle ORE	Q	Lower ORE	Q
Carbonate, Carbonate silicate,	CAB	0.870	Very Poor	1.157	Poor	0.707	Very Poor
Granite, Pegmatite, Aplite, Feldspar-chlorite vein, migmatites	GRN	0.986	Very Poor	1.064	Poor	1.032	Poor
Skarn - All	SKN	0.788	Very Poor	0.819	Very Poor	1.062	Poor
Schist - Biotite with psamitic laminations, Sericitic, black/dark	SCH	0.796	Very Poor	0.591	Very Poor	0.822	Very Poor

Lithology	GT					Footwall	Q
Carbonate, Carbonate silicate, Calcite vein	CAB						
Granite, Pegmatite, Aplite, Feldspar-	GRN					0.888	Very Poor
Skarn - All	SKN					0.766	Very Poor
Schist - Biotite 9with psamitic laminations, Sericitic, black/dark	SCH					0.715	Very Poor

Lithology	GT	Total	Q Description	Fresh	Q Description	Slightly Weathered	Q Description
Granite, Pegmatite, Aplite, Feldspar-chlorite vein, migmatites	GRN	1.327	Poor	1.574	Poor	1.235	Poor

15.2.4 Stope and Development Stability Condition

The standard underground drift and fill stope and development dimension being used for preliminary design is 4 m by 4 m. The stability condition for an excavation is related to the Q value and a value known as the equivalent excavation dimension (De). The parameter De is calculated by dividing the excavation span or height by the Excavation Support Ratio ("ESR"). The ESR is a function of the duty requirement of an excavation as indicated in Table 15-5. The more permanent or critical an excavation is the lower the ESR. ESR is effectively an inverse of the factor of safety.

Table 15-5: ESR for Different Excavation Categories

Excavation Category		Suggested ESR Value
A	Temporary mine openings	3 – 5
B	Permanent mine openings, water tunnels for hydro power (excluding high pressure penstocks), pilot tunnels, drifts and headings for large excavations.	1.6
C	Storage rooms, water treatment plants, minor road and railway tunnels, surge chambers, access tunnels	1.3
D	Power stations, major road and railway tunnels, civil defense chambers, portal intersections	1
E	Underground nuclear power stations, railway stations, sports and public facilities, factories	0.8

An estimate of unsupported span can be made using the following equation:

$$\textit{Unsupported span (m)} = 2 * ESR * Q\text{^}^{0.14}$$

An ESR of 3 is considered appropriate for stope excavations, whilst an ESR of 1.6 is considered appropriate for permanent excavations and mine access development. Stope excavation boundaries will be formed either in the orebody or immediate hangingwall. Permanent and access development will be formed either in the footwall, hangingwall or granite. Within the orebody, hangingwall and footwall the dominant rock type is the schist. Schist comprises over 50% of borehole intersections in this unit. It can be seen from Table 15-4 that the Q value of the schist is generally lower than the Q value of the other lithologies. It is therefore the rock mass quality of this unit that will control the stability of the stopes and development. Unsupported spans for development in each mining zone are presented in Table 15-6.

Table 15-6: Maximum Unsupported Spans

	Granite	Hanging wall access	Hanging wall	Upper ORE	Upper/Middle Ore Interburden	Middle ORE	Middle/Lower Ore Interburden	Lower ORE	Footwall
Q	1.57	0.48	0.48	0.80	0.66	0.59	0.60	0.82	0.71
ESR	1.6	1.6	3	3	3	3	3	3	1.6
Unsupported Span (m)	3.4	2.9	5.4	5.8	5.7	5.6	5.6	5.8	3.1

It can be seen from the data in Table 15-6 the maximum unsupported span for the stope development exceeds the design stope span whilst the maximum unsupported span for the permanent and access development is smaller than the stope span.

Figure 15-15 shows the Q support design chart for stope excavations. Table 15-7 shows that the Q values for the bulk of the rock mass lies in Category 1 "Unsupported", however with a small change in either Span or ESR the support requirements fall into Category 4 to 5. From the core photographs, additional support is likely to be required due to the RQD of the material and the proximity to the surface. Based on this assessment for 4 m span excavations, the minimum support required will be 50 mm thick shotcrete with 1.8 m long rock bolts installed at between 1.3 to 1.8 m centres. Bolt length increases to 2.4 m with a decrease in ESR from 3 to 1.6 m, the remaining support requirements are constant. The ore drives are to be supported with 5 cm of shotcrete only, with spot bolting where required.

*Bolt length (m) = 2 + (0.15 * Span / ESR)*

Table 15-7: Q Support Requirements – Temporary development (ESR 3.0) and Permanent Development (ESR 1.6)

Decline Q Parameters			Support requirements		ESR	3.00	Mining Domain		
Range		Q Description	Reinforcement categories	Bolt Length (m)	Shotcrete thickness	Bolt spacing	HW	ORE	FW
0.001	0.01	Exceptionally Poor	6/7 - Sfr +B	2.20	105-155mm	<1m	0%	0%	0%
0.01	0.1	Extremely Poor	5/6 - Sfr +B	2.20	50-105mm	1-1.3m	0%	0%	0%
0.1	1	Very Poor	1 - Unsupported			1.3-1.7m	84%	61%	67%
1	4	Poor	1 - Unsupported			1.7-2.1m	16%	39%	33%
4	10	Fair	1 - Unsupported				0%	0%	0%
10	40	Good	1 - Unsupported				0%	0%	0%
40	100	Very Good	1 - Unsupported				0%	0%	0%
100	400	Extremely Good	1 - Unsupported				0%	0%	0%
400	1000	Exceptionally Good	1 - Unsupported				0%	0%	0%

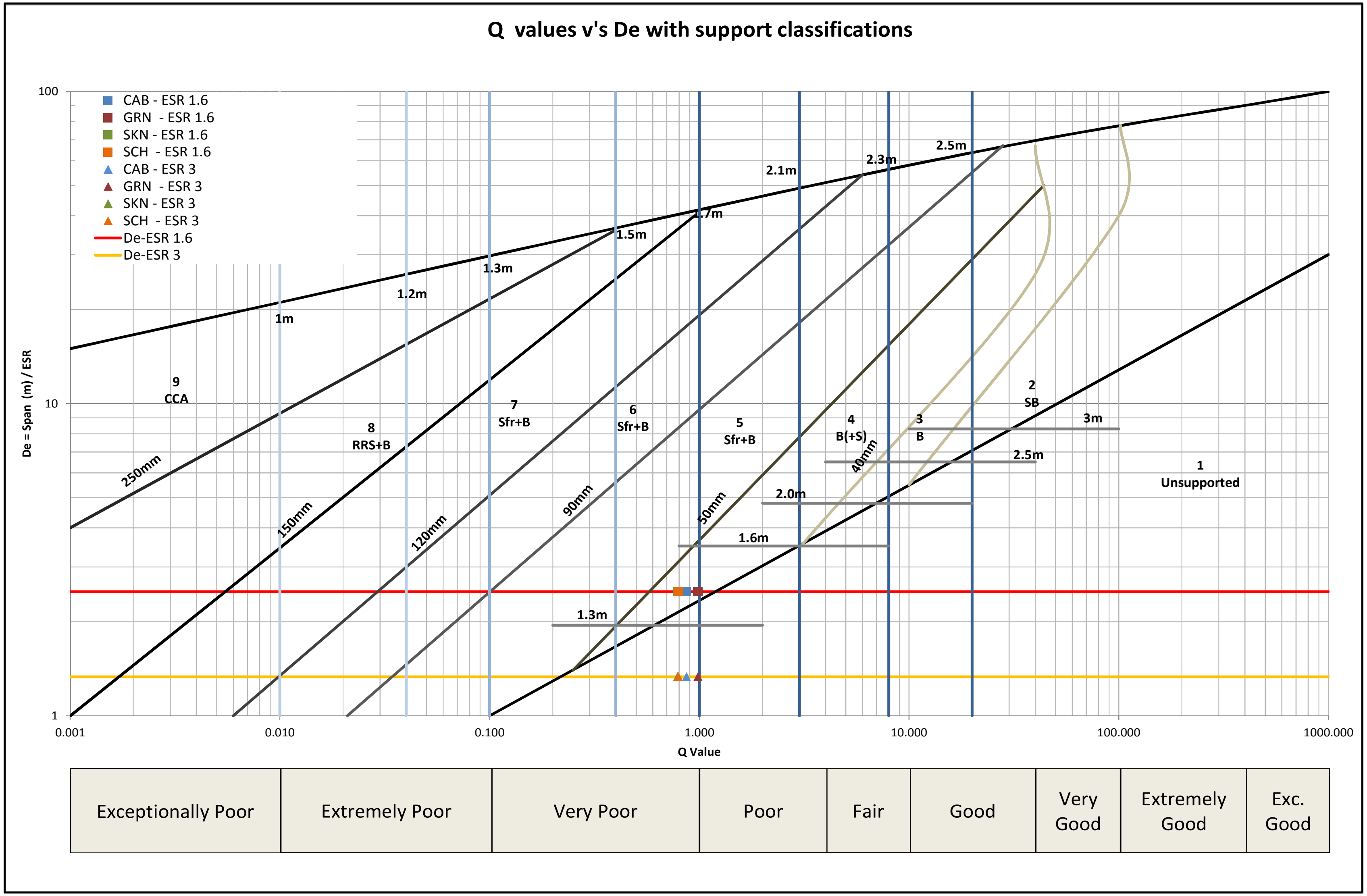


Figure 15-15: Q Values and the Required Support Classification

15.2.5 Conclusions

The following conclusions can be made from the work carried out for the PEA:

- The rock is within the very poor to poor range of the Q graph. These represent the average value and some areas will be significantly lower;
- Changing the ESR from temporary (3) to permanent (1.6) environments increases the support requirements and bolt length increases from 2.20 m to 2.40 m;
- Shotcrete and bolting should be in accordance with Barton's Category 4 - "Bolting + shotcrete", and should be considered in all capital development areas as a minimum, until greater confidence in the material can be determined; and
- The oredrives should be shotcreted with spot bolting where required.

The following recommendations should be undertaken in the next phase of the project:

- It is recommended that specific triple tube geotechnical boreholes are drilled and logged to ascertain the true rock mass properties opposed to doubling resource drilling where core management was secondary to meterage. Logging Q parameters would enhance the confidence in the database leading to a greater understand of the rock mass stability. This will provide greater confidence in the material properties and has the potential to improve the Q support requirements;
- Relocating the underground mine infrastructure in the granite (HW) could enhance the excavations stability through improved rock mass properties and increased RQD. This has not been proven in this study as there are insufficient boreholes that intercept the granite at depth in the HW; and
- Increased analysis needs to be undertaken into the material properties to zone the underground workings spatially.

15.3 Hydrological and Hydrogeological Related Issues

15.3.1 Introduction

A hydrogeological study has been initiated to characterize the area of interest. Monitoring of surface and groundwater has commenced and preliminary conceptual hydrogeological models have been developed to further assess permanent monitoring points with respect to final layout of the mine site. SRK consider the hydrogeological studies to be well advanced for this stage of the project and the approach adopted for the hydrogeological study is fully supported by SRK. Figure 15-16 shows a schematic of the Tabuaço deposit area with the resource area in red, the piping in dark blue, surface water runoff in light blue, dams in grey, waste dump in green and tailings in brown.

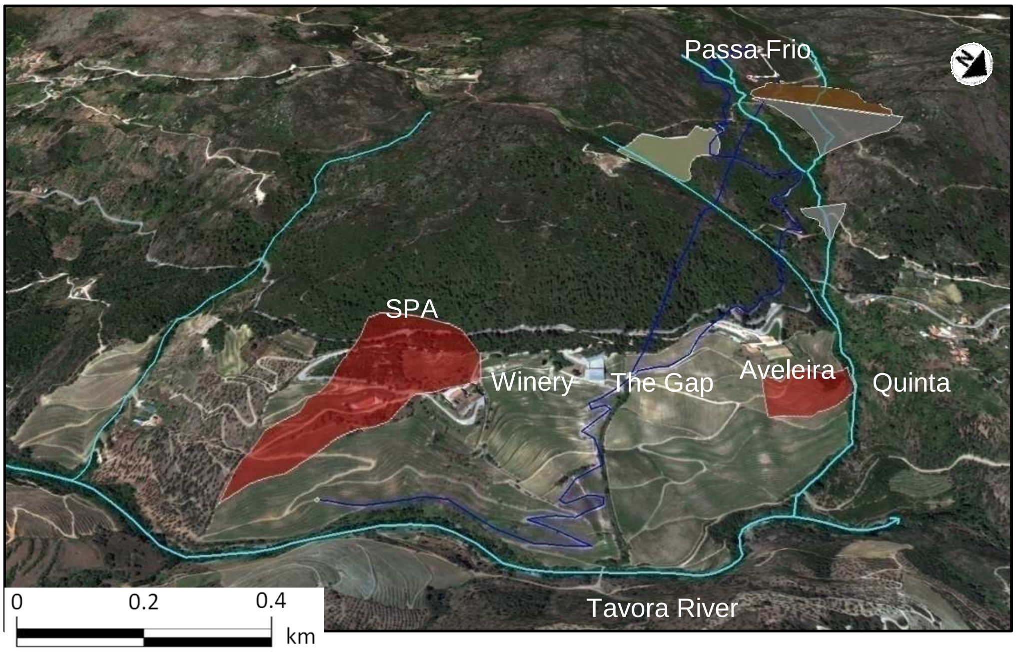


Figure 15-16: Schematic overview of conceptual mine site and block classification

The mine water management plan is at an early in its development however the principal water related issues currently facing the project are described below:

- There is evidence of conductive and confined structures that could cause sudden inrushes of groundwater during mine development if appropriate cover drilling, dewatering and monitoring is not carried out beforehand;
- High levels of fluoride may lead to poor discharge water quality from the workings, and groundwater impacts beneath the waste rock dumps and TSF;
- Seasonal climate variations may temporarily limit water supplies during the summer months and result in the need for mine water discharge during the wetter winter months;
- Sources of makeup water have not yet been identified, although water demand is likely to be heavily dependent on the selected tailings option; and
- River abstraction may be limited due to imposed limits through permits and regulated river stages from the upstream Vilar dam.

Some surface water diversions will be required, although the catchment areas involved are relatively small and all project infrastructure is located above the flood plain of the Tavora river.

15.3.2 Dewatering

Conceptual Groundwater Model

The Aveleira Block is crossed by a small river that originates from Passa Frio with the Quintã Fault extending below. Exploration well DHT-85 located approximately 50 m above the Aveleira block is artesian which signifies penetration of a confined aquifer, infiltrated at a higher elevation. The aquifer is believed to be formed through fractured rocks and demonstrates the potential for sudden inrushes of groundwater during mine development.

The study area consists of the granite massif extending above the tree line with a conceptualised 45° dipping contact zone to the meta sedimentary sequences containing the SPA deposit. The granite massif appears to be of low permeability as it supports elevated groundwater levels and springs are also present on the higher ground around Passa Frio.

Assessments of the water levels together with drill core data from the exploration boreholes indicate deeper water levels and higher permeability in the metasedimentary sequence. Groundwater flow does appear to be controlled by faults and fractured zones. Results from the provisional water level monitoring of exploration wells indicate large water level variations over relatively short distances which is indicative of a heavily compartmentalised groundwater system.

Four major faults has been identified; Quintã Fault, Aveleira Fault, Winery Fault and the Covent Fault. All but the Winery Fault are located within or next to the resource areas and extends to higher elevations in to the granite massif where groundwater levels are shallow.

Shallow groundwater flowing from the higher elevated granite rock towards the Tavora river line is likely to drain into the underground mine area through hydraulically conductive faults and fracture zones in the metasedimentary sequences and thus dewatering flow rates will therefore most likely be strongly correlated to seasonal variations in climate.

Flow Rates

Table 15-8 contains indicative estimates of available surface water resources and possible dewatering flow rates based on approximate monthly precipitation as well as general recharge, evapotranspiration and runoff coefficients. The SPA resource catchment area represents the area directly above the underground mine, the immediate surroundings, and the higher grounds where groundwater is likely to drain into underground facilities. The purpose of these estimates is to broadly estimate water resources available as natural inflows to the underground mine as well as surface water resources.

Catchment to the TSF has been approximated to give an estimated extent of river diversion. Runoff and recharge to the TSF area are generally expected to be low but still significant as water resource or potential source of infiltration through the Quintã Fault to the Aveleira Block. The Aveleira block is adjacent to the Quintã fault (East) and the Aveleria Fault (West), with both extending from the granite to the Tavora River. The Quintã fault is located directly underneath the river that runs from Passa Frio to the Tavora River. Runoff and recharge to the Aveleira and Quintã blocks will be influenced by water management in Passa Frio regarding river diversion and reservoir development

Table 15-8: Estimated Flow Rates Under Steady State Conditions (after complete drainage)

	Catchment to SPA, The Gap, Winery Block			**Catchment to TMF**			**Catchment Area: Passa Frio, SPA & Aveleira Block incl. Quinta zone**		
	Ave.	**S**	**W**	**Ave.**	**S**	**W**	**Ave.**	**S**	**W**
Precipitation [mm/month]	50	15	100	50	15	100	50	15	100
Evapotranspiration coeff.	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8	0.8
Recharge coeff.	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Recharge [m^3/hr]*	6.1	1.8	12.3	3.3	1.0	6.5	20.3	6.1	40.6
Runoff coeff.	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
Runoff [m^3/hr]*	6.1	1.8	12.3	3.3	1.0	6.5	20.3	6.1	40.6
Dewatering Capacity**	**16.0**	**4.8**	**32.0**	**8.5**	**2.5**	**17.0**	**52.7**	**15.8**	**105.5**

*total for catchment area
**incl. contingency capacity (30%) and full runoff infiltration to the mine after mine development
Ave - Average, S - Summer, W - Winter

The next stage will be to estimate peak flow rates to determine the seasonality of required dewatering. Return flows from service and backfill water will also need to be quantified and it is sensible to assume that the installed dewatering capacity will need to be greater than the rates indicated in Table 15-8.

Rock formation volume has been estimated to provide specific yield volumes that will drain into the mine before constant dewatering rates are reached (Table 15-9). The upper estimates suggest that volumes are significant, and further evaluation will be required to refine these estimates. The results will be used to determine the practicalities of using dewatering flows for a water supply, and also to evaluate whether advance dewatering measures will be required prior to the initiation of experimental or full scale mining.

Table 15-9: Estimated Storage Volumes for Dewatering

	SPA, Winery & The Gap	Aveleira & Quinta
Total rock volume [m3]*	51,000,000	4,500,000
Specific Yield (1%)	510,000	45,000
Specific Yield (2.5%)	1,275,000	112,500

*Estimated rock volumes with respect to conceptual mine designs and deposit areas. Excl. catchment above granite rock surface boundary.

For the purposes of PEA costing and pump sizing SRK considers an installed dewatering capacity of $180m^3$/hr to be reasonable.

Considerations for dewatering system design

It is anticipated that the compartmentalised groundwater system will tend to reduce the overall dewatering rates and restrict the propagation of drawdown and associated impacts. The risk of groundwater inrushes will need to be mitigated through continuous review of water level monitoring data during mining, and a programme cover hole drilling can be implemented during mine development where water level monitoring data indicates it to be necessary.

The viability of dewatering boreholes should also be evaluated during the feasibility programme. A dewatering borehole test programme will also demonstrate the viability of groundwater wells as a source of make-up water supply.

It is considered likely that relatively high groundwater inflows will be encountered during mine development, and this may also be an issue during the experimental mining phase of the project. This would potentially result in excess water and a requirement for discharge. Mine development will almost certainly occur prior to commissioning of the process plant and the project water demand will therefore be very low. Water quality of discharge from dewatering boreholes is likely to be better than contact water from within the workings. The dewatering strategy and water treatment requirements therefore require evaluation early in the project. These requirements should be evaluated as part of the ongoing hydrogeological programme.

Water Quality Considerations

The deposit is rich on carbonates and the risk of acid rock drainage is considered to be low. However, the drill assay results indicate high levels of arsenic and fluoride. Water quality from mine discharge will require further evaluation during the feasibility study to determine the extent of water treatment required, if discharge to the environment is to occur. This will need to consider both the operational and post-closure water treatment requirements. Some residual drainage from the mine to the Tavora River post-closure is likely given the high topographic relief. Passive treatment systems may be required order to manage this long-term discharge.

A review of regulatory requirements for mine water discharge to the Tavora River will also be required.

15.3.3 Water Supply and Site Water balance

The TSF design option will have a major impact on the final water demand. It is currently anticipated that the process water demand will be between 50 and 100 m^3/hr. The option of dry tailings will reduce demand but will also limit the availability of water reservoirs on site thus making operations more reliant on other sources. It is assumed at this point that both treatment for discharge and make up water for process are required due to seasonal variability. It will therefore be necessary to further investigate the need for water reservoirs to reduce the sensitivity to seasonal changes in water supply and treatment.

A likely source of makeup water is the Tavora River. The river is part of the Douro basin flowing in to the Douro River. It is partly regulated by the upstream Vilar dam and reported having elevated nutrients as a result of agricultural activities. Flood occurrences with return periods have been addressed in the environmental scoping report and will require additional hydrological studies for seasonal variations with respect to abstraction.

15.3.4 Conclusions

Based on a scoping level of study the following recommendations should be undertaken. A hydrogeological field program is required to:

- Investigate surface water diversions depending on mine water balance and choice of TMF;
- Set up a permanent baseline monitoring program;
- Provide data to support the dewatering system design;
- Characterise groundwater conditions at the tailings management facility and waste rock dump locations; and
- Provide data to support rock mechanical assessment.

It is understood that this work is part of the existing hydrogeological study but this should be confirmed, and the existing study extended as necessary.

Following completion of the field campaigns it will be necessary to:

- Design closure plans for the EIA;
- Review the conceptual design of the TMF and WRD taking account of seepage and runoff management requirements;
- Construct a Site Wide Water Balance Model to evaluate the need for makeup water and mine water discharges;
- Develop a site wide water management plan;
- Construct numerical groundwater and geochemical models to assess inflow rates to the underground mine and mine site seepage post closure, if geochemical testwork indicates the risk to be elevated; and
- Confirm environmental impacts and permitting limits for river abstraction and mine water discharge.

16 RECOVERY METHODS

Colt has presented four processing options for consideration for the PEA:

- Option A1/A2: Production of a scheelite concentrate by flotation;
- Option A3: Use of ore sorting to reduce the plant feed tonnage, followed by flotation to produce a scheelite concentrate;
- Option B1/B2: Production of a scheelite concentrate by flotation and leaching of the concentrate using acid to produce WO_3; and
- Option B3: Use of ore sorting to reduce the plant feed tonnage, followed by production of a scheelite concentrate by flotation and leaching of the concentrate using acid to produce WO_3.

The following effective recovery estimates have been used in the economic analysis for the process options (which include 95% ore metal recovery for the ore sorting options):

- Option A1/A2: 87%;
- Option A3: 83%;
- Option B1/B2: 95%; and
- Option B3: 90%.

For Options B1, B2 and B3 Colt proposes to use an acid based processing path to produce high purity tungsten oxide end product. The process path is estimated to be less capital and operating cost intensive than the traditional ammonium paratungstate ("APT") processing path.

SRK believes that, with the exception of Option A1/A2, the flowsheets as proposed have not been tested at either laboratory or any larger scale, and that therefore the proposed recovery figures are not supported by the metallurgical testwork reported in Section 12. Specifically:

- Options A3 and B3 are based on the use of ore sorting with, SRK understands, the assumption of a mass rejection of 55.8% with an accompanying loss of WO_3 of 5% based on a external dilution of 17%. Given that no quantitative testwork has been conducted on ore sorting, this assumption is unverifiable. While the difference between the Option B1/B2 and B3 recoveries is 5%, the difference in recovery between Options A1/A2 and A3 implies an ore sorting rejection of only 2%. Given that the 87% recovery for Option A1/A2 is supported by the testwork, SRK believes that the recovery for Option A3 should be 83%.
- As no acid leaching testwork results have been reported on the Tabuaço ore or concentrates, the assumed recovery of 95% for Option B1/B2 (and implied for Option B3) is also unverifiable.

SRK therefore believes that a significant program of further testwork is required in order to verify the technical feasibility of the proposed process options, and then to confirm the recovery figures assumed for this PEA.

17 PROJECT INFRASTRUCTURE

17.1 Introduction

Figure 17-1 shows a map of the suggested in the preliminary underground infrastructure study by Real and Caupers (2012). The bypass is designed to avoid haul trucks crossing the main N323 road.

The main underground infrastructure of the mine, such as main ramps, underground workshops and hanging wall galleries will be located whenever possible within the granite to improve the stability of the excavation and reduce support costs.

Real and Caupers (2012) propose a phase of trial mining to include:

- Gallery access at 332.5 m level, located at the footwall of the deposit with a length of approximately 60 m; and
- Sinking an exploratory shaft, 4 m in diameter and 60 m deep.
- At present the proposed gallery and shaft are not connected. SRK proposes to expand the design to connect the two elements and trial raise boring to determine the applicability for future ventilation development. Sinking a shaft is not recommended in the poor rock conditions.

Figure 17-2 shows the underground mining infrastructure development on an annual basis for both SPA and Aveleira deposits.

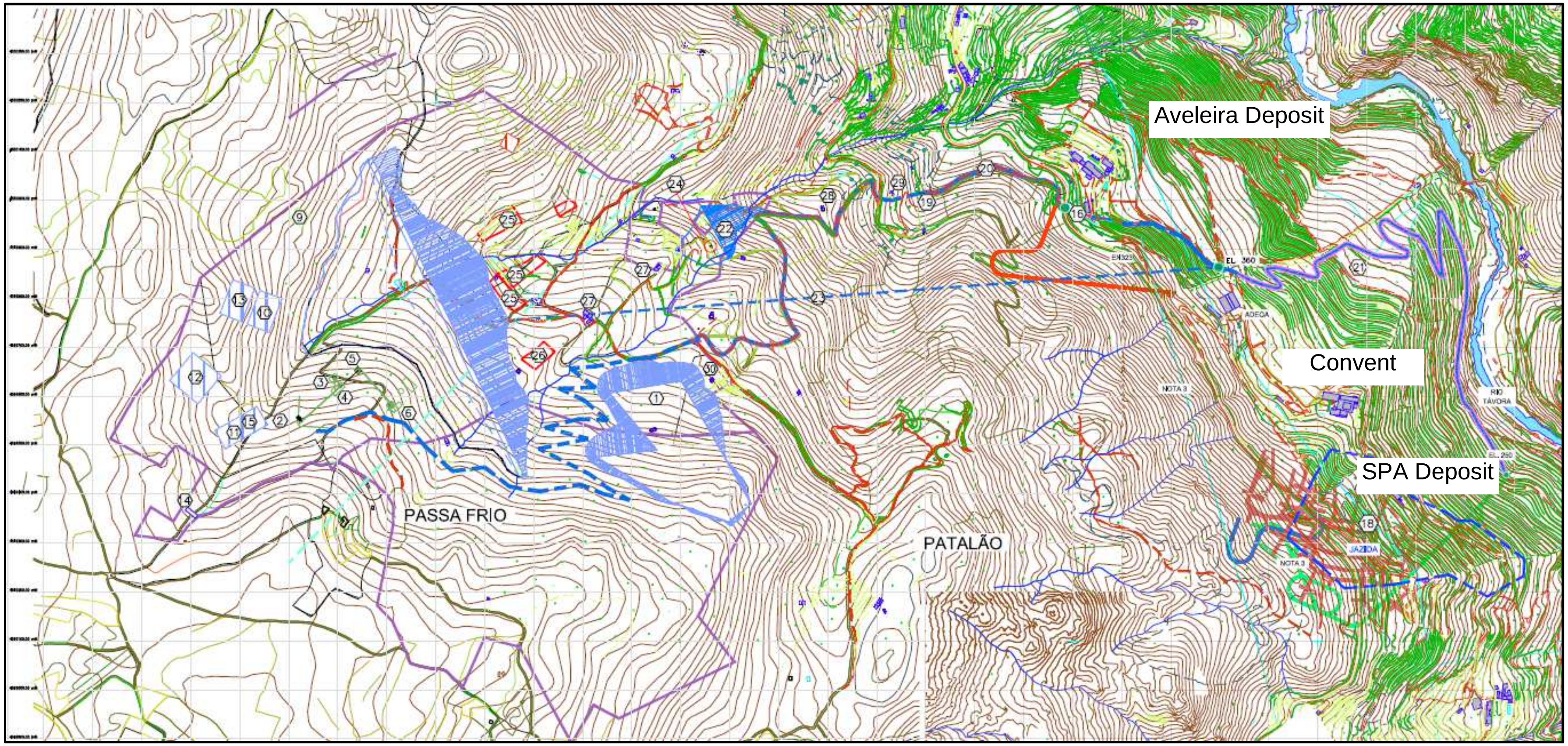


Figure 17-1: Map of Surface Infrastructure (Colt)

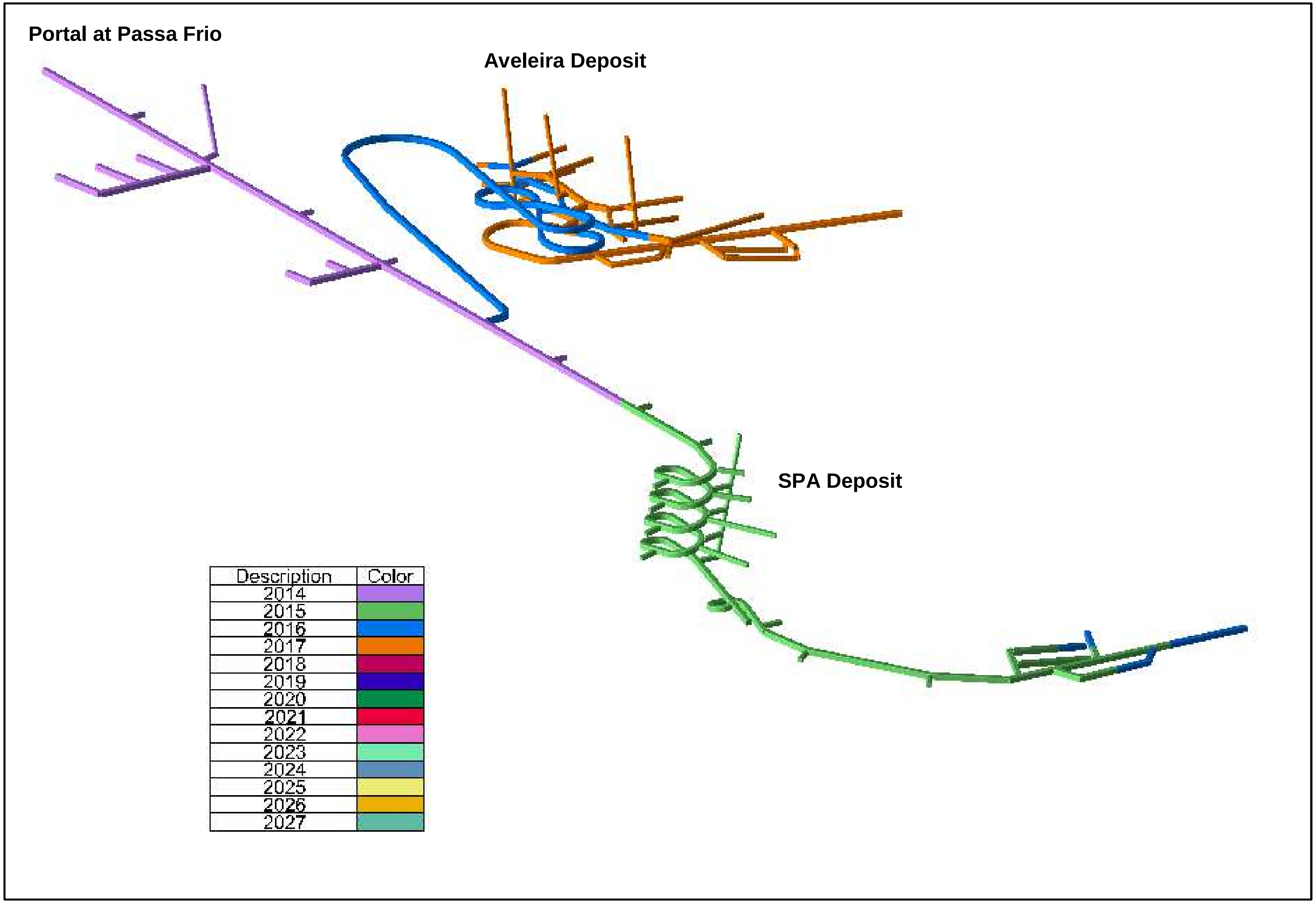


Figure 17-2: Isometric view of Underground Mining Infrastructure (SRK)

17.2 Power

The project area is well serviced by the national grid, with a high-voltage 60 kVa power line passing immediately adjacent to the project along the N323 highway between Granjinha and Quintã. The entity responsible for the power infrastructure and main supplier of both base load and renewable power in Portugal is, respectively, REN and EDP, formally known as Electricidade de Portugal.

17.3 Water

Supply and regulation of the drinking water and sewerage is controlled by the Water and Sanitation Regulation Agency Entidade Reguladora de Águas e Resíduos (“ERSAR”). Service provision is shared between the local municipalities and the national water company, Águas de Portugal (“AdP”).

Potable water is available from the local municipal system and the Távora River passes through the project licence area, making it an option for sourcing larger volumes of drilling water subject to permitting. A further option is pumping water from a number of existing reservoirs which are located close to the project area.

17.4 Mining Personnel

The area of northern Portugal has a long and varied mining past, with a significant number of stone quarries used for building or road building still evident in operation in the area. The metalliferous mining industry in northern Portugal is relatively small scale compared to global standards, however the area hosts the Panasqueria Tin-Tungsten mine, located in Covilha, Castelo Branco, which is one of the world’s main producers of Tungsten. The nearest concentration of modern metalliferous mines in Portugal is found largely concentrated in the Iberian Pyrite Belt to the south. Additional skilled workers are potentially available from Spain and other EU countries with areas of existing or historic Tungsten or other metalliferous underground mines.

17.5 Port

The closest ports to the project site are The Port of Leixões and The Port of Lisbon. The Port of Leixões is an artificial harbour comprising two breakwaters, 1.6 km and 1.1 km long, located 9 km north of Porto at Matosinhos and has container-park gantries and heavy cranes capable of handling Panamax-type container ships. The Port of Lisbon handles over 12 Mt of containerized cargo and solid bulk materials (2010). Container facilities are developed on both banks of the River Tejo equipped to deal with over 3,000 vessels annually.

A third port with deep water facilities is The Port of Setúbal which formerly had a concession to Almina/Somincor to export ore concentrates from Aljustrel and Neves-Corvo mines. It has a 126 m long pier near a 19,000 m^2 park with dedicated railroad and service facilities.

17.6 Rail

If the transport of the scheelite concentrate or WO_3 product is to be made by train, the nearest railway station from Passa Frio, with direct connection to Spain, is “Vila Franca das Naves” about 70 km (by road) SSE from Passa Frio.

This route only allows a low trucking speed of around 50 km/h, and the route is crossing several small cities (Moimenta, Trancoso, Vila Franca das Naves).

17.7 Buildings and Ancillary Facilities

The preliminary infrastructure study reports that there are currently no buildings located on the property adaptable for mining and campsite purposes. Lodging for the operations personnel required during any trial mining phase will need to be supported in the nearby town of Tabuaço.

17.8 Potential Waste Disposal Areas

Real and Caupers (2012) estimated that 2,300 t of waste rock will be extracted from the trial mine phase. The waste rock disposal site must have the least possible impact on the vineyard, using to the uncultivated land or land currently planted with olive trees.

17.9 Potential Processing Plant Sites and Tailings Management Areas

A multi-criteria analysis is currently ongoing to establish the optimal location for processing facilities and tailings management area. Four areas are being considered using criteria including, but not limited to, the following:

- Location at an economically cost-effective distance from the mine site;
- Relief not too uneven to accommodate tailings and minimize entrainment of solids during heavy rainfall;
- Sufficient capacity to accommodate waste which is not suitable to be used as fill material underground;
- Location of the infrastructure within the same catchment area with minimal impact on watercourses and runoff;
- Shortest possible distances and differences in elevation so as to minimise cost on roads, pipes, pumping etc.;
- Economically cost effective distances to source of fill material for construction of dykes and platforms;
- Areas not affected by the existence of habitats of endangered species;
- Location that minimises impacts on flora, fauna and aquifers, and containable in case of accidents and has minimal risk from geological or seismic hazards;
- Land use of low economic value or potential and within the same administrative area;
- Minimal negative visual impacts to the community and from noise, dust generation and effluent discharge;
- Constraints related to power transmission lines, water pipes, roads, National Ecological Reserve (“REN”) and National Agricultural Reserve (“RAN”), PDM and other instruments of planning and land management; and
- Ability to negotiate and deal sensibly with affected landowners.

The four areas currently under consideration are Azenha Velha, Lavandeiras, Passa Frio and Patalão. The sites that currently best meet the criteria are Passa Frio and Patalão, which are both located on the edge of the granite plateau between 600 m and 800 m elevation on the western side of the Távora Valley, which is the same side of the valley as the deposits and located within the same administrative municipality of Tabuaço.

17.10 Tailings Storage

17.10.1 Introduction

Previous studies commissioned by the Client include the NI 43-101 Technical Report completed by SRK ES November 2012. The results of the previous work with approval of the Client inferred that the best area for the TSF would be in the valley of Passa Frio (river) east of the proposed plant location. SRK followed this selection as most appropriate at this stage, however because of difficult terrain and high volumes of dam construction SRK also evaluated a dry stacking concept as viable alternative. The TSF will be a valley type of facility enclosed by the main embankment.

17.10.2 Previous Studies

The Passa Frio site has been chosen based on many criteria presented in the previous studies:

- Location at an economically cost-effective distance from the mine site;
- Relief not too uneven to accommodation tailings and minimize entrainment of solids during heavy rainfall;
- Sufficient capacity to accommodate waste which is not suitable to be used as fill material underground;
- Location of the infrastructure within the same catchment area with minimal impact on watercourses and runoff;
- Shortest possible distances and elevation changes so as to minimize cost on roads, pipes, pumping, etc;
- Economically cost effective distance to fill material source for construction of dykes and platforms;
- Areas not affected by the existence of habitats of endangered species;
- Location that minimises impacts on flora, fauna and aquifers, and containable in case of accidents;
- Minimal risk from geological or seismic hazards;
- Land use of low economic value or potential and within the same administrative area;
- Minimal negative impacts to the community in terms of visual, noise, dust generation and effluent discharge;
- Constraints related to power transmission lines, water pipes, roads, National Ecological Reserve ("NER") and National Agricultural Reserve ("RAN"), PDM and other instruments of planning and land management;
- Ability to negotiate and deal sensibly with affected landowners; and
- Ease of implementing solutions for mine closure and reclamation.

Four alternative sites were considered with the Passa Frio site deemed the most suitable. The site is located on the edge of the granite plateau between 600 and 800 m elevation on the western side of the Tavora Valley, which is the same side of the valley as the deposits.

The selected site option allows the expansion of the tailings storage beyond the quantity indicated in this study.

Issues regarding land ownership on the area designated for the process plant and tailings management facility at Passa Frio are being addressed by the Client and there is a risk that consensus will not be reached with several landowners.

17.10.3 Tailing Characteristics and Volumes

The unit mass of the ore rock (SG) is expected to be about 3,100 kg/m^3. The porosity of the tailings will likely vary between 0.4 to 0.5, depending on the deposition method and the consolidation conditions. A bulk dry density of 1,490 kg/m^3 was selected for the tailings material, which corresponds to a porosity value of 0.52. The total volume of tailings which needs to be stored externally would therefore be about 1.07 Mm^3 based on 1.6 Mt.

The yearly production of tailings destine for external storage will be 375 ktpa or 250 km^3 per annum. As such it has been assumed that the starter dam will have to hold 250 km^3 of tailings (one year of storage) and further raises will be possible in year round construction.

The quantities of tailings for the external storage may change as the final plan for the mine backfilling with tailings has not been finalised.

Table 17-1 (from Research on Samples from Tabuaco Scheelite Project of Cold Resources Inc by GRINFM, November 2012), shows that the fines contents in tailings vary from 52 to 85%.

Table 17-1: Grinding Fineness Test Results

Grinding fineness -0.074mm	Products	Weight / g	Yield	WO_3 Grade	Recovery
No.1: 52.07	Scheelite rough conc	18.95	7.70	6.18	74.12
	Tailings	227.13	92.30	0.18	25.88
	Feeding	246.08	100.00	0.64	100.00
No.2: 60.90	Scheelite rough conc	24.39	9.71	5.80	87.51
	Tailings	226.89	90.29	0.089	12.49
	Feeding	251.28	100.00	0.64	100.00
No.3: 69.26	Scheelite rough conc	29.48	11.79	4.81	88.32
	Tailings	220.65	88.21	0.085	11.68
	Feeding	250.13	100.00	0.64	100.00
No.4: 77.18	Scheelite rough conc	31.97	12.77	4.56	90.63
	Tailings	218.47	87.23	0.069	9.37
	Feeding	250.44	100.00	0.64	100.00
No.5: 85.29	Scheelite rough conc	36.59	14.57	3.98	90.78
	Tailings	214.48	85.43	0.069	9.22
	Feeding	251.07	100.00	0.64	100.00

If this gradation is to be representative of the final tailings then the tailings scenario would most likely involve the downstream construction method for the containment embankment.

17.10.4 Geotechnical Setting

Although SRK has not found any site investigation information in the subject area of Passa Frio valley it is expected that the geotechnical conditions will be good.

It is expected that below surficial soils, there will be a layer of schist/granite with varying degree of weathering.

17.10.5 Design Considerations

As mentioned earlier two external storage scenarios were considered in the design of the TSF:

1. Wet tailings storage and;
2. Dry tailings stacking.

Wet tailings

In this case it has been assumed that the tailings will be stored sub-aerially behind the rock fill dam with the upstream liner as shown in Figure 17-3.

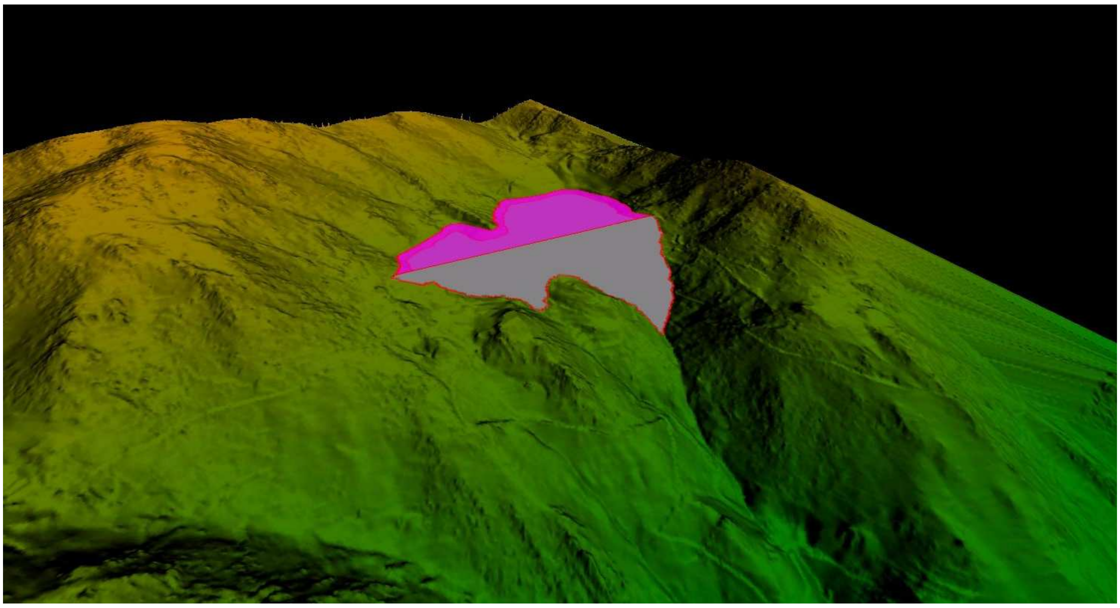


Figure 17-3: Wet Tailings Schematic

For the wet tailings option, the valley dam is assigned a slope of 2.5H:1V and 1.5H:1V for the upstream and downstream slopes, respectively.

- SRK proposes that the containment embankment be built in stages rather than in one stage to reduce up front capital costs. The containment will be predominantly constructed from the rock fill from the borrow pit. A provision should be made, however for the production period to allow for utilisation of some waste rock from the underground mine.
- 2 mm thick high-density polyethylene ("HDPE") liner will be installed on the upstream slopes of the confining embankment and at some distance from the dam. The inundation area on the bottom of the pond may also require a liner, however for the purpose of the PEA we have assumed that no liner will be required for this area (refer to Section 0). A membrane will be installed on the upstream slope of the tailings dam on the suitable subgrade with adequate filtration layer to prevent piping failure in case of the membrane rupture. The liner will be protected by the geotextile. In addition the upstream perimeter of the tailings storage will be supported with the upstream tailings beach constructed and maintained above water at all times with the minimum width of 100 m. The layout and cross-section of embankment is shown in Figure 17-4.
- The water balance has been addressed in Section 15.3; however the tailings facility will be designed as tailings storage, not water storage facility, with minimum area of effluent pond.
- It has been assumed that the clear water surface ditch will be necessary around the tailings facility to divert all water away from the storage area and discharge via spillway to the original bed of the creek. The length of this diversion ditch will be approximately 1,880 m. The TSF will be equipped with spillway capable of handing the probable maximum flood (1 in 10,000 years).

The further analysis of the dynamic loading will be required during the prefeasibility study to finalise the geometry of the TSF.

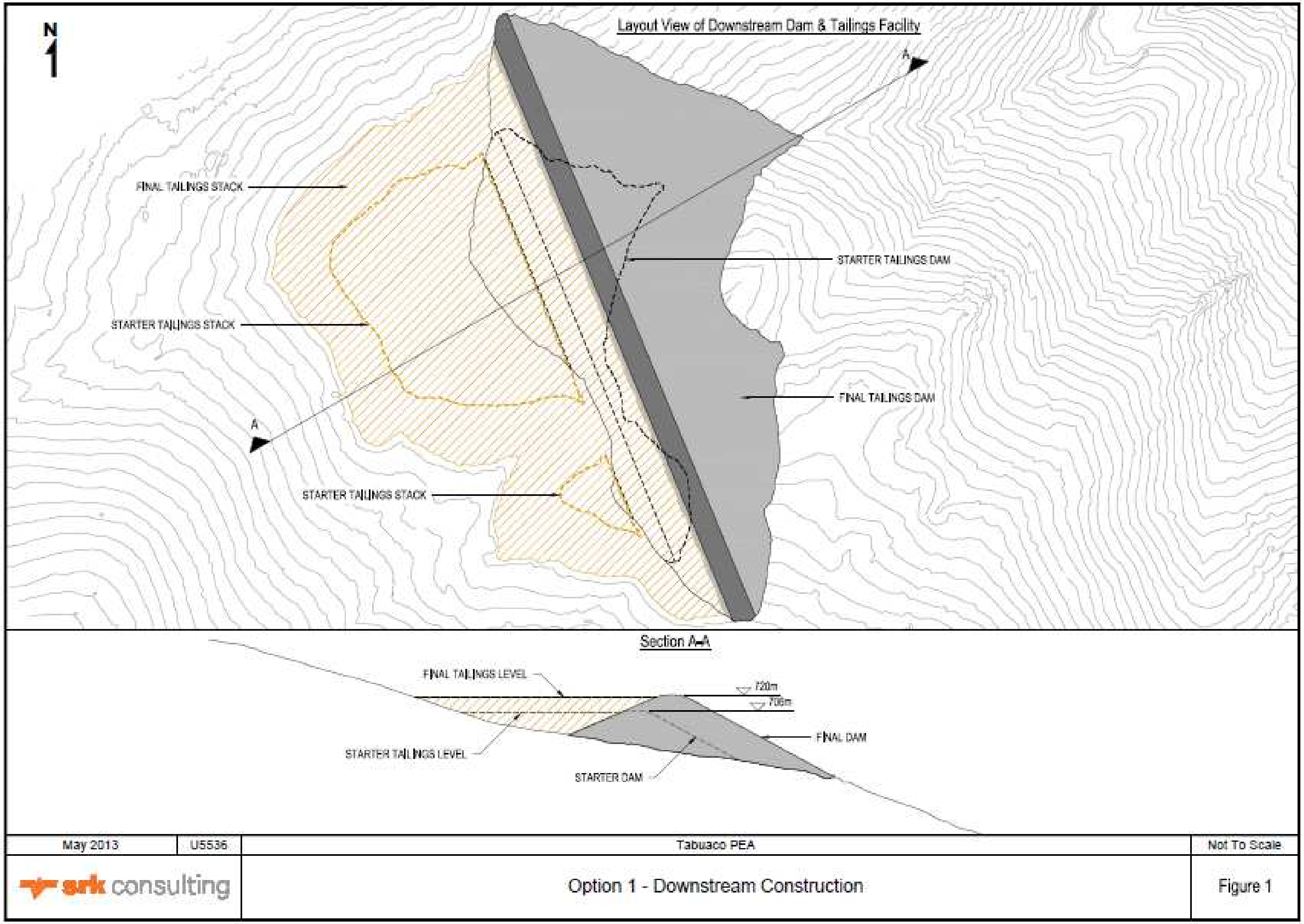


Figure 17-4: Wet Tailings Embankment Layout and Cross Section

Table 17-2 below shows the basic dimensions of the TSF. The starter and final dam embankments will be 60 m and 100 m high, respectively, with construction volumes of 0.3 and 1.6 Mm^3 for capital and sustaining capital construction costs.

Table 17-2: Basic Parameters for Wet Tailings Option

	Units	Starter Dam Embankment	Final Dam Embankment
Elevation	m	706	720
Height (toe to crest)	m	60	100
Length	m	410	590
Embankment footprint	m^2	32,414	90,732
Volume (embankment)	m^3	308,375	1,621,847
Surface area of liner	m^2	35,394	100,786

Dry stacking

SRK has carried out a high level study of dry stacking concept for the same location as the wet tailings disposal site. The concept of dry disposal has been developed with mine waste rock (approximately 300 km^3) to be co-disposed with the filter pressed tailings. The starter wall embankment is essentially reduced in size at the front of the disposal area and the mine waste rock can be sequentially dumped with the tailings. Figure 17-5 shows the location of the dry stack.

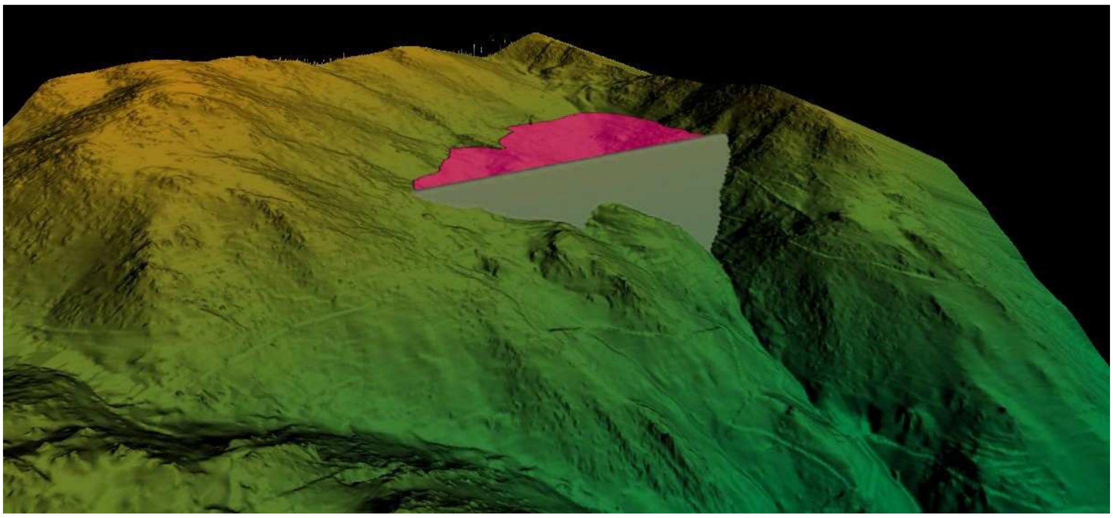


Figure 17-5: Dry Tailings Schematic

To evaluate the feasibility of co-disposal, the following physical characteristics need to be determined.

- Geotechnical classification of the two waste products including shear strength and permeability; and
- Slope stability of the combined products with the proper dimensioning of the downstream waste rock berms.

The planning for this option involves the coordination between the raise of the downstream berms and the filter cake storage schedule. It has been assumed that the downstream slope of the dry stack of 2H:1V are achievable with the rock berms. The layout and cross-section of embankment is shown in Figure 17-6.

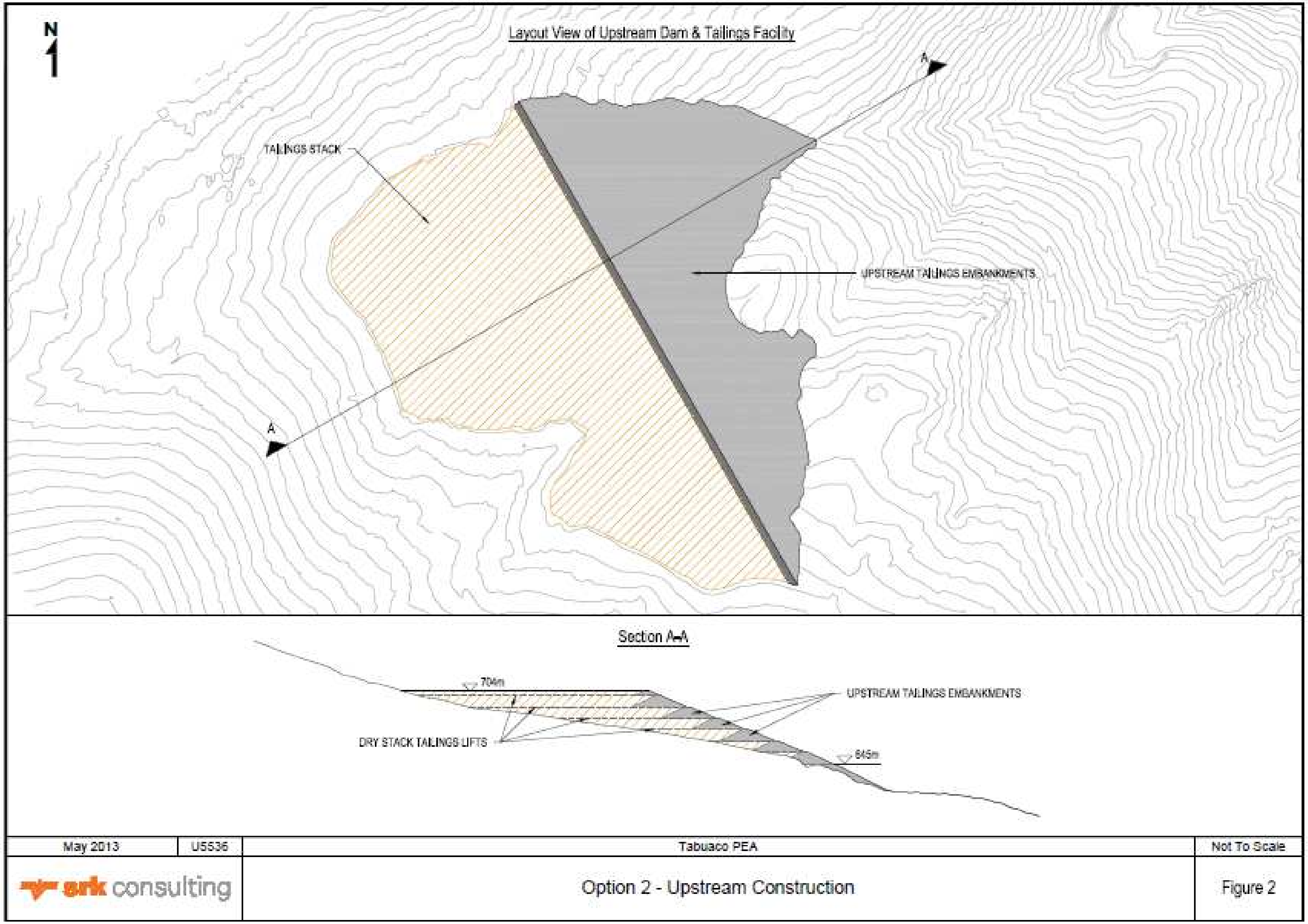


Figure 17-6: Dry Tailings Embankment Layout and Cross Section

For the PEA, SRK proposes that the containment embankment be built in stages:

- Initial three berms will be built to elevation 665 m (three 10 m high lifts each) with no tailings disposal. At this stage, the embankment/platform will be built using the borrow rock fill (68,000 m^3 of fill); and
- The rest of the rock containment will be built over life of mine in 10 m lifts utilising waste rock from the underground mine (over 300 km^3).
- There has been no provision for any geo-membrane at this stage as both the waste rock and tailings waste are assumed to be benign for the economic analysis. However, following further geochemical assessment this requirement may change. Table 17-3 shows the basic geometric characteristics of dry-stacking. The total volume of fill for the rock berms is estimated at 392.5k m^3 to store behind 1.1 Mm^3 of filter cake tailings waste.

Table 17-3: Basic Parameters of Dry Stacking

	Units	Lift 1	lift 2	Lift 3	Lift 4	Lift 5	Lift 6	Lift 7	Lift 8 (Final)
Elevation	m	645	655	665	675	685	695	705	709
Height (toe to crest from previous lift)	m	10	10	10	10	10	10	10	4
Length	m	111	72	145	295	375	420	478	490
Embankment footprint	m^2	2,941	3,679	5,300	6,830	13,320	18,280	21,165	11,770
Volume (embankment)	m^3	22,419	20,220	27,325	36,155	60,468	93,063	113,031	29,599
Storage volume	m^2	6,044	13,735	29,405	45,521	99,500	237,297	425,260	253,731

Similar to the wet tailings option, it has been assumed that the clear water surface ditch will be necessary around the tailings facility to divert all water away from the storage area and discharge via spillway to the original bed of the creek. The length of this diversion ditch will be approximately 1,379 m.

17.10.6 Further Work

Additional design and investigative work will be required for the next stage of the project. The design of the TSF or the co-storage impoundment will require various studies for the following fields:

- Tailings alternative assessment to include the dry tailings disposal option;
- Groundwater condition at and near the tailings impoundment and dam;
- Geochemical properties of the tailings and the leachate originating from the tailings;
- Geotechnical conditions of the dam foundations and properties of the tailings;
- Borrow source investigation; and
- Engineering.

The hydrology aspects are linked to the site water management. It will require precipitation data that would be representative of the site. The analysis of the site precipitation patterns and the development of the site storm scenarios will require a dataset over 50 years.

The hydrogeological investigation will consist of characterising the hydraulic properties of the overburden and bedrock primarily at the dam and along the perimeter on the tailings impoundment. This will include the drilling several boreholes that will be used to perform in situ hydraulic conductivity testing and to install monitoring wells/piezometers for monitoring groundwater quality and levels. Groundwater samples would be recovered as part of the characterisation work. Six to ten boreholes would typically be part of the hydrogeological investigation.

Geochemical investigations would need to consist of performing laboratory tests on samples that would be representative of the tailings and waste rock that would be produced once the mine is in operation. The laboratory program would consist of static tests, kinetic tests, mineralogy and the related chemical analyses. This information is essential for assessing the potential impacts on the receiving environment and to determine the necessity for lining of the facility.

The geotechnical investigation will involve reviewing the structural geology in the vicinity of the dam, performing geophysical testing, and developing trial pit and drilling programs.

The trial pit program will be used to characterise the shallow overburden and recover soil bulk samples. The trial pits would also be used for the borrow source investigation. About 10 trial pits would be required for the dam and probably a similar quantity would be required for the borrow source investigation. The drilling program would consist of drilling about 5 boreholes along the alignment of the proposed dam. Representative soil and rock samples would be recovered and selected samples would be tested in the laboratory. Piezometers would be installed in those boreholes for monitoring piezometric levels and water quality.

The design of the tailing facilities will rely on the data provided by the hydrologists, the hydrogeologist, the geochemist and the geotechnical engineer. It will require the analysis of groundwater movement through the structure(s), the foundations. It will also require detailed analysis of slope stability and deformation of the dam under static and dynamic loads. The seismic conditions of the region combined with the proximity of the local people would justify a detailed stability analysis that would then feed into the project's risk assessment.

18 MARKET STUDIES AND CONTRACTS

There are currently no market studies or contracts for the Tabuaço Project.

19 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

19.1 Introduction

The environmental and social input to this PEA is based on a desktop review of the following documents:

- Tabuaço Tungsten Project, Portugal. NI 43-101 Technical Report (SRK, November 2012);
- Environmental and Social Scoping Study of the SPA Mining Project: Executive Summary (DHV, 2012);
- São Pedro das Águias Mining Project Experimental Mining Plan Descriptive Document (Colt Resources, 2012); and
- Information from Colt Resources website (http://www.coltresources.com/).

19.2 Environmental and Social Setting

The Tabuaço project is located approximately 3 km southeast of the village of Távora and within the municipality of Tabuaço in the Viseu district. The project is located along the western flank of the Távora River, a tributary of the Douro River, the main river in the north of Portugal. The topography is represented by steep peaks and deep v-shaped valleys with elevations ranging from 225 m to 500 m above mean sea level.

The climate of Portugal is described as typically Mediterranean, consisting of dry hot summers and temperate wet winters. However, the climate of the Tabuaço project area is distinct from regional averages due to the steep valley setting producing a microclimate effect. The average annual mean temperature is 12.6℃ and the average annual rainfall is approximately 500 to 700 mm, the majority of which usually falls during November to March. Maximum monthly temperatures are usually above 30℃ between June and September and winters are relatively mild, with infrequent snowfall on the higher peaks. The prevailing wind blows from the west.

Soils of the project area are skeletal. The vegetation is a mix of terraced grape vines and orchards, olive and almond trees in the area of proposed mine access, and shrubs and pine forest in the area of the proposed TSF and waste rock dump (Passa Frio). Local agricultural activities affect the quality and quantity of water in the project area, by discharging nitrogen and phosphorous into the water courses and representing 70% of the water usage of the area. The project is not located within a protected area but is within the buffer zone of the Alto Douro Vinhateiro UNESCO World Heritage Site. This wine region is listed by UNESCO for a long tradition of viticulture that is reflected in a cultural landscape of outstanding beauty.

The 12th century convent of San Pedro das Águias has been identified as a site of municipal interest within the licence area. A vineyard is also located within the proposed mine area. Colt has purchased the vineyard and the convent. Buildings are located to the north of the mine licence, in the same catchment as the potential waste storage location in the Passa Frio valley.

Colt owns 100% of the surface rights for the proposed mining area that is sufficient for current requirements. SRK understands Colt has announced the execution of a binding letter of intent to purchase the land rights for the Passa Frio area, which has been identified as the preferred location for the process plant, TSF and waste rock dump.

19.3 Environmental and Social Approvals

Colt has an experimental mining licence for the Tabuaço project, which covers operations up to five hectares in size and/or less than 150,000 tons.

In accordance Mining Law 88/90 of 16 March 1990, an application for a full mining licence requires the submission of a Mine Pre-Feasibility Study that includes an EIA. The EIA part of the application requires approval from the APA. Colt intends to complete the Feasibility Study in Q4 2014 and obtain the required approvals in 2015.

The EIA process is administered by the APA through Decree Law no. 69/2000, of 3 May, amended by Decree-Law no. 197/2005, of 8 November. The EIA process includes the following procedural steps:

- Screening;
- Scope definition (scoping report);
- EIA report;
- Evaluation of the EIA by an evaluation committee and public consultation; and
- Decision.

An environmental scoping report was issued to APA in August 2012 together with the application for the experimental mining licence and a response to the report was received in October 2012. The response indicated that the EIA should carefully evaluate the stability of the waste rock dump and TSF and include appropriate characterisation of waste material ensuring compliance of the project with Decree Law 10/2010 of 4 February on Regulating the Management of Wastes from Extractive Industries. The response also commented on the terms of reference for the baseline studies and stated that cultural heritage impacts should be evaluated following the methodology provided in the Guidelines of the International Council on Monuments and Sites (“ICOMOS”).

Colt intends to initiate the baseline phase of the EIA in January 2014, although a hydrogeological baseline study has already commenced.

19.4 Approach to Management

The Experimental Mining Plan Descriptive Document, submitted as part of the application for the experimental mining licence, states that an Environmental Management Plan will be prepared prior to the bulk sampling works commencing in Q2 2014. SRK understands that the plan has not yet been prepared but will reflect the relevant environmental guidelines and legislation.

Currently, a water treatment plant has been designed to treat the water discharging from the drainage level of the proposed mine; that only removes oils and suspended solids from the mine water drainage.

At the current stage of mine development, geochemical evaluation of acid rock drainage and metal leaching (ARDML) and its potential impact on water quality has not been undertaken for the Tabuaco deposit. Therefore the requirement for additional water treatment / mitigation controls other than those include in the PEA cannot be addressed. It is expected that a detailed assessment of the ARDML potential of this deposit will be assessed during upcoming feasibility studies.

19.5 Stakeholder Engagement

The November 2012 NI 43-101 states that Colt holds regular meetings with the local community, however, no information was available from Colt on stakeholder engagement activities associated with the Tabuaço project.

19.6 Key Issues

Potential key environmental and social issues for the Tabuaço project, which should be considered during subsequent phases of project development, are as follows:

- *Potential risks from waste storage facilities*: The location of the TSF and waste rock dump presents significant risks to down gradient buildings and ecological systems if the facilities were to fail. These risks should be thoroughly evaluated in the EIA and by project engineers and managed through the project design. As requested by APA, the project should plan for appropriate characterisation of waste rock, tailings material and seepage quality in accordance with requirements of Decree Law 10/2010. The findings of these studies will need to be incorporated into the EIA to ensure that impacts have been appropriately identified and adequate management is incorporated into project design. The potential impact of seepage and runoff from the facilities should also be evaluated in conjunction with the waste characterization studies, and all necessary mitigation measures should be incorporated into the engineering design.
- *Potential impacts on cultural heritage:* The project is located within the buffer zone of Alto Douro Vinhateiro UNESCO world heritage site. Colt is not aware of any development restrictions within the buffer zone but this should be confirmed during the EIA process. Construction and operation of the project may create impacts on the visual character of this area through surface disturbance, air emission and noise.
- Vibrations from underground blasting have the potential to affect the convent and Quintã structures and therefore the impact of vibrations should be evaluated in the EIA In the Experimental Mining Plan, Colt indicated that the stability of the convent would be monitored during the exploration activities and the Company would promote repair and maintenance measures that may be necessary for the preservation of the building. As requested by APA, cultural heritage impacts should be addressed following the methodology provided in the Guidelines of the International Council on Monuments and Sites.
- *Potential impacts from mine dewatering on volumes of surface water flow and water availability for other groundwater users*: Dewatering of the underground mine is likely to have a negligible impact on the Tavora River in terms of flow depletion as the base of the workings are at a higher elevation than the river. The compartmentalised groundwater system is likely to limit the cone of depression and thus minimise the impacts on other groundwater users. A detailed groundwater study should be conducted to model the effects of dewatering and confirm whether an impact on water availability is expected. An inventory of all groundwater users in the area will also be required for EIA purposes.
- *Potential impacts from mine discharge on downstream water users*: According to the current mine design, water will be released from the discharge level of the mine during operations. As the area has a high topographic variation, the mine workings may also produce mine water discharge post closure. A water treatment plant has been designed but there is insufficient characterisation of the predicted water quality to confirm whether this treatment method is suitable and whether subsequent water users are likely to be affected by the proposed discharge. High arsenic concentrations in the core are cause for concern if present in the discharge water, especially if the water is to be used to irrigate vineyards or agricultural areas. Water quality prediction via geochemical modelling should be conducted to confirm the potential impact of mine discharge during operations and post-closure.

- *Apparent lack of stakeholder engagement:* Given the lack of information regarding stakeholder engagement, there is a risk that there may be local opposition to the project, which may delay the permitting process. The response from the APA stresses the importance of including public opinion in the EIA process. Proactive stakeholder engagement should be implemented in a structured way to ensure that key stakeholder issues are identified and are addressed by the EIA and project design.
- *Timing of the baseline phase of the EIA*: SRK understands the feasibility study is due to be completed in Q4 2014 and submitted as part of the mining licence application. An EIA must be included in this application. SRK recommends that the EIA schedule should allow for a period of baseline data to be included in the EIA report that meets the expectations of the APA. The schedule should also allow for time required to address stakeholder concerns raised during the EIA process, which could be lengthy. The scope of the baseline studies should also include the recommendations provided by the APA to avoid delays to the permitting process.

19.7 Closure Requirements

A closure plan has not yet been prepared for the project but will be included in the EIA submission. The Experimental Mining Plan provides closure actions if the experimental mining activities were to finish prematurely. These include removal of oils and toxic materials for appropriate disposal, prevention of access to the workings and rehabilitation of waste disposal areas for agricultural use. There are no costs associated with these proposed activities.

It is not possible to provide an accurate closure cost without a closure plan but on the basis of SRK's experience of closure costs for similar types of operations in similar environments a provisional ballpark estimate for the closure of this site would be in the region of USD 5m, excluding the requirement for post-closure water treatment. The requirement for potentially require long term treatment of this water should be evaluated.

20 CAPITAL AND OPERATING COSTS

20.1 Exchange Rates

The cost estimates have been completed in USD. An exchange rate of USD 1.30/EUR was used in this cost estimate.

20.2 Mining

20.2.1 Approach

The mine schedules, equipment and labour requirements were used to develop mine operating and capital costs. Previous studies completed for the Tabuaço project were used to develop the cost estimates.

A diesel price of EUR 1.50/L was provided by the Client and has been applied in the cost estimate.

20.2.2 Capital Costs

The lateral capital development costs are estimated at EUR 2,425/m derived from benchmarking comparable underground operations.

The vertical development costs are estimated at EUR 1,850/m and the waste transport costs have been estimated at EUR 3.00/t.

Based on discussions with the Client it has been assumed that a mining contractor will be used for all mining operations. The following ancillary equipment capital costs have been used in the cost estimate and have been sourced from the Trade-Off Study (Table 20-1).

Table 20-1: Equipment Capital Costs

Equipment	Units	Cost
Backfill Materials Transport	EURk	75
Ambulance	EURk	35
Light Vehicles	EURk	35
Secondary Fans	EURk	24

20.2.3 Operating Costs

The unit operating costs are shown in Table 20-2.The direct mining costs have been derived from the benchmarking comparable Tungsten operations. The indirect mining costs include ventilation and mine services and have been estimated at 2.5% of the direct mining costs. The ore transport cost has been based on the Trade-Off Study adjusted by SRK. The backfill cost has been based on similar cut and fill operations.

Table 20-2: Unit Operating Costs

	Units	Unit Cost
Direct Mining	EUR/t ore	32.53
Indirect Mining	EUR/t ore	0.81
Ore Transport	EUR/t ore	3.00
Backfill	EUR/t ore	4.23
Total	**EUR/t ore**	**40.57**

20.3 Processing

The following capital and operating cost estimates for the six options have been derived from documentation provided by the Client (Table 20-3 and Table 20-4). The processing costs for the ore sorter options have been increased by 10% based on the assumption that there will be less material to process with decreasing economies of scale.

Table 20-3: Process Plant Capital Cost Estimates

	Units	Processing Plant	Paste Backfill Plant	Ore Sorter	Acid Plant
Option A1	USDm	44.20	7.0	-	-
Option A2	USDm	44.20	7.0	-	-
Option A3	USDm	26.52	7.0	2.0	-
Option B1	USDm	44.95	7.0	-	8.0
Option B2	USDm	44.95	7.0	-	8.0
Option B3	USDm	26.97	7.0	2.0	8.0

Table 20-4: Processing Operating Cost Estimates

	Units	Operating Cost
Ore Sorting	EUR/t RoM	1.54
Reject Handling	EUR/t reject	0.86
Crushing	EUR/t crushed	1.15
Processing		
Option A1/A2	EUR/t processed	18.03
Option A3 (Ore Sorter)	EUR/t processed	19.83
Option B1/B2 (Acid leach)	EUR/t processed	14.42
Option B3 (Ore Sorter and Acid leach)	EUR/t processed	15.86

The process plant capital cost estimate for Option A1 and A2 is based on a conceptual estimate prepared by Contecmina, based on a gravity and flotation circuit from a tungsten operation in Brazil. The capital cost estimate for Option A3 is scaled by a factor of 0.6 for throughput from Option A1, with the cost for ore sorting added on.

No information has been provided as to the basis for the Option B1 and B2 process plant capital cost estimate. As with Option A3, Option B3 has been scaled by 0.6 from Option A1.

The process plant operating estimate for Option A1 is also based on the Contecmina conceptual estimate. The Option B1 process plant operating cost estimate is taken as being 80% of the Option A cost. The operating cost estimate for Options A3 and B3 have been included the ore sorting and reject handling costs.

The capital cost estimates for Options A1, A2 and A3 appear to be reasonable based on benchmark data available to SRK. Given the developmental nature of the flowsheet for Options B1, B2 and B3, and the absence of detail provided for the cost estimates, SRK is not able to independently verify these capital cost estimates, other than to say that they are probably of an appropriate order of magnitude.

It has been assumed, based on the Contecmina report, that the water drainage, water treatment plant and pilot holes for water supply, backfilling and pumping to Passa Frio have been included in the plant capital costs. It is possible that a more sophisticated water treatment plant may be required however this will only be confirmed following the completion of the geochemical characterisation studies.

As with the capital cost estimates, the operating cost estimates for Options A1, A2 and A3 appear to be reasonable based on benchmark data available to SRK. However, again given the developmental nature of the flowsheet for Options B1, B2 and B3, SRK is not able to independently verify these operating cost estimates, other than to say that they may be of an appropriate order of magnitude. SRK does note that the operating cost for the acid process proposed for Options B1, B2 and B3 is highly dependent on the efficient generation and regeneration of the acid reagent. Should the generation and regeneration of this reagent not be optimised, then the operating costs for these Options could be significantly higher than the values shown in Table 20-3.

20.4 Tailings

The TSF costs have been evaluated for the two options: wet and dry tailings.

20.4.1 Wet Tailings

The capital costs for the project are estimated at EUR 6.7m with sustaining capital costs of EUR 14.8m and shown in Table 20-5 excluding preliminaries, general items and contingencies.

Table 20-5: Wet Tailings Capital and Sustaining Capital Costs

	Units	Capital (Pre-production)	Sustaining Capital (Year 1 and 6)
Ground Clearance	EURm	0.3	0.3
Earthworks including fill placement	EURm	3.1	13.0
Liner	EURm	1.0	1.5
Surface water diversion	EURm	0.3	0
Spillway	EURm	1.5	0
Pumps and pipelines, road and powerline	EURm	0.5	0
Total	**EURm**	**6.7**	**14.8**

The capital cost to construct the TSF is estimated at about EUR 6.7m based on a unit cost of EUR 10/m^3 for the dam material when fill material is to be gained from the separate borrow source. The operating cost will be in large part controlled by the pumping and the related infrastructure used to transport the tailings. It has been assumed that the operating costs will be around EUR 0.5/t of tailings waste (EUR 0.8m in total). The closure of the tailings impoundment could be achieved by placing a soil cover over the entire footprint. A closure cost of EUR 1m has been assumed.

20.4.2 Dry Tailings

The capital costs for the dry option are estimated at EUR 7.7m with sustaining capital costs of EUR 0.8m and shown in Table 20-6.

The capital cost to construct the TSF is estimated at about EUR 7.7m based on a unit cost of EUR 10/m^3 for the dam material and EUR 5m for the filtering press. It has been assumed that the operating costs for running the dry disposal operation will be around EUR 4/t of tailings waste (total EUR 5.6m). The closure of the tailings impoundment could be achieved by placing a soil cover over the entire footprint. A tailings closure cost of EUR 0.8m has been assumed.

Table 20-6: Dry Tailings Capital and Sustaining Capital Costs

	Units	Capital (Pre-production)	Sustaining Capital (Year 1 and 6)
Ground Clearance	EURm	0.3	0.3
Earthworks including fill placement	EURm	0.68	0.5
Tailings filter press	EURm	5.0	0
Surface water diversion	EURm	0.3	0
Spillway	EURm	1.5	0
Total	**EURm**	**7.78**	**0.8**

20.5 Waste Dumps

SRK has undertaken a mass balance analysis, based on maximising tailings use for backfilling the mined-out areas. For the Options A1, A2, B1 and B2, 1.5 Mt of external tailings storage is require after backfilling and a waste dump is required for the 0.4 Mt of development waste. For the Options A3 and B3 (ore sorting options), all tailings and a portion of the waste reject from the ore sorter (0.6 Mt) can be used as backfill. A larger waste dump is therefore required for the remaining reject waste (1.4 Mt) and the development waste (0.4 Mt).

For Options A1, A2, B1 and B2 USD 500k has been estimated as the cost for constructing the waste dumps. For the ore sorting options (A3 and B3) USD 1,500k has been estimated as the waste management capital cost.

20.6 Dewatering

For the purposes of this PEA, an operational expenditure of EUR 100k per annum has been assumed for operation of the dewatering system and provision of water to the process plant at Passa Frio. This estimate will need to be revised to reflect actual dewatering rates, process water demand and water treatment costs during the Feasibility Study.

20.7 Environment

The costs for the recommended waste characterisation (including testwork costs but without other disbursements), numerical prediction of water quality and specification of treatment requirements are as follows:

- Phase 1: Sampling, static testing and initial characterisation - approximately EUR 70,000;
- Phase 2: Kinetic testing - approximately EUR 60,000; and
- Phase 3: Water quality prediction / geochemical modelling and specification of treatment requirements – approximately EUR 60,000.

20.8 Closure

On the basis of SRK's experience of closure costs for similar types of operations in similar environments a provisional ballpark estimate for the closure of this site would be in the region of USD 5m, excluding the requirement for post-closure water treatment.

20.9 General and Administrative

A general and administrative ("G&A") cost of EUR 3/t of ore has been assumed for the cost estimate.

20.10 Drilling Programmes and Engineering Studies

Table 20-7 represents a high level summary of the work program and budget for upcoming studies provided by Colt and amended based on SRK's experience with similar projects.

Table 20-7: Planned Budget for Studies

Item	# of Holes	Depth (m)	Unit cost (EUR/m)	Estimated Cost (EUR)
Exploration Drilling	10	125	250	312,500
Infill Drilling (Resource/Reserve)	24	125	250	750,000
Deposit Expansion	20	125	250	625,000
Metallurgy Sampling				20,000
Metallurgical Testwork				120,000
Geotechnical Drilling	20	125	270	675,000
Hydrology/Hydrogeology				100,000
Geochemistry				180,000
Environmental Impact Assessment				190,000
NI43-101 Resource Update				60,000
Design Feasibility/Feasibility Study				1,200,000
Market Study				25,000
Surface Rights Acquisition				500,000
Trial Mining				765,000
Total				**5,522,500**

20.11 Capital and Operating Cost Summary

A summary of the total capital and operating costs for the four options is shown in Table 20-8. The main findings from the capital and operating costs are summarised below:

- The processing costs show the most variation between the options (Options A3 and B3 have the lowest costs);
- The tailings costs show that the wet tailings options (A2 and B2) are higher than the dry tailings options (A1 and B1);
- Option B3 has the lowest operating costs, while Option A1 has the highest; and
- Option A3 has the lowest capital requirements, while Option B2 has the highest.

Table 20-8: Capital and Operating Costs Summary

	Units	A1	A2	A3	B1	B2	B3
Operating Expenditure							
Mining	USDm	150.4	150.4	150.4	150.4	150.4	150.4
Mine Services	USDm	3.8	3.8	3.8	3.8	3.8	3.8
Ore Sorting	USDm	-	-	7.1	-	-	7.1
Reject handling	USDm	-	-	4.0	-	-	4.0
Crushing	USDm	5.3	5.3	5.3	5.3	5.3	5.3
Processing	USDm	83.3	83.3	40.5	66.7	66.7	32.4
Ore Transport	USDm	13.9	13.9	13.9	13.9	13.9	13.9
Backfill	USDm	19.6	19.6	19.6	19.6	19.6	19.6
Tailings	USDm	11.1	1.4	0.3	11.1	1.4	0.3
Dewatering	USDm	1.1	1.1	1.1	1.1	1.1	1.1
G&A	USDm	13.9	13.9	13.9	13.9	13.9	13.9
Total Operating Expenditure	**USDm**	**302.3**	**292.6**	**259.8**	**285.7**	**276.0**	**251.7**
Capital Expenditure							
Mining	USDm	23.2	23.2	23.2	23.2	23.2	23.2
Plant	USDm	44.2	44.2	26.5	45.0	45.0	27.0
Paste Backfill Plant	USDm	7.0	7.0	7.0	7.0	7.0	7.0
Tailings - Construction	USDm	12.2	47.2	0.3	12.2	47.2	0.3
Tailings - Closure	USDm	1.0	1.3	-	1.0	1.3	-
Waste Management	USDm	0.5	0.5	1.5	0.5	0.5	1.5
Ore Sorter	USDm	-	-	2.0	-	-	2.0
Acid Plant	USDm	-	-	-	8.0	8.0	8.0
Mine Closure	USDm	5.0	5.0	5.0	5.0	5.0	5.0
Drilling/Eng. Studies	USDm	7.2	7.2	7.2	7.2	7.2	7.2
Total Capital Expenditure	**USDm**	**100.3**	**135.6**	**72.7**	**109.1**	**144.3**	**81.2**

21 ECONOMIC ANALYSIS

21.1 Introduction

The economic analysis for the Tabuaço project is based on the production schedule presented in Section 15.1.4 and the cost structure outlined in Section 20.11. An economic analysis has been undertaken for all six project options.

21.2 Metal Price

A flat tungsten price of USD 400/MTU WO_3 has been used in the PEA. This price is based on other studies on similar projects as published on SEDAR, and sits in the higher end of the range of prices used. It compares with a 3-year average APT price of USD 388/MTU WO_3 as shown in Figure 21-1. SRK notes that this price is the base price in the PEA, however a 9% price deduction has been applied to options A1, A2, and A3, as the product from these Options ($CaWO_4$) is different from the product from the Options B1, B2 and B3 (WO_3) and will hence not realise as high a price on a WO_3 basis as the product from these latter options.

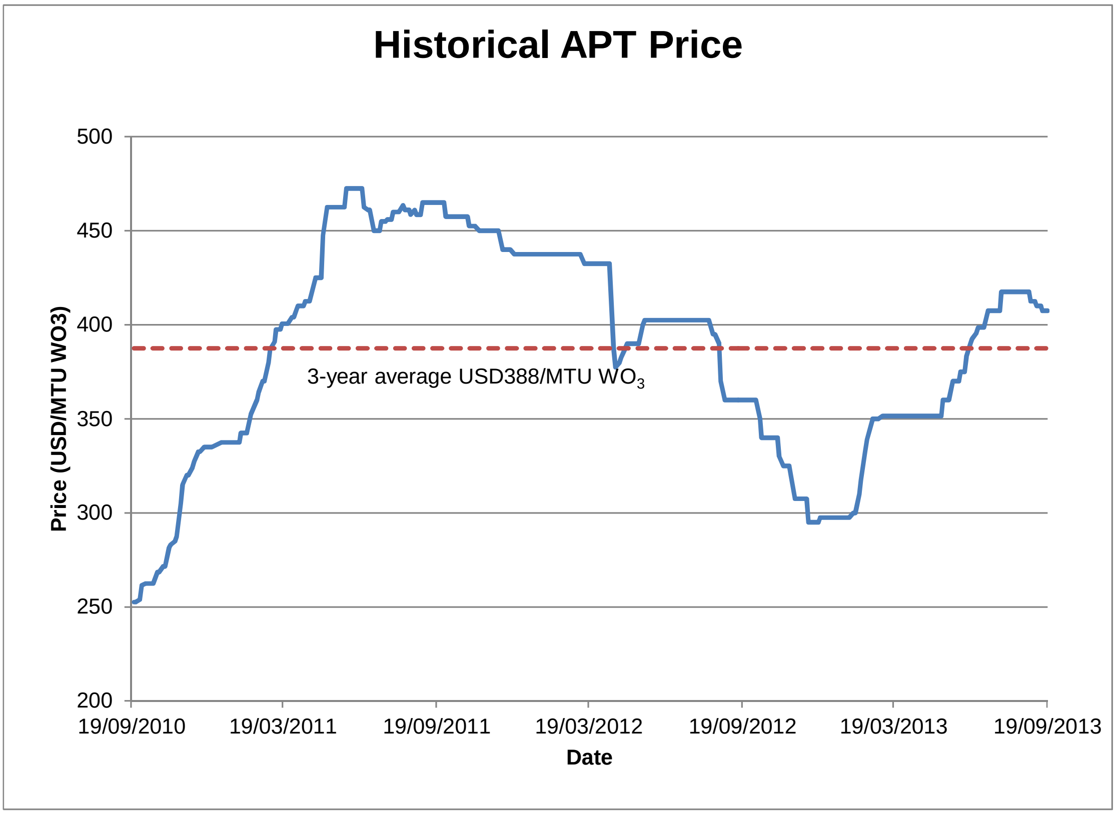


Figure 21-1: Historical Tungsten Prices

The study aims to identify the potential opportunities for the development of the Tabuaço project based on limited geological and technical information. A USD 400/MTU WO_3 price is therefore considered appropriate when considering the current stage, however sensitivities around this commodity price should be considered.

21.3 Royalties

The royalty payments applicable for the Tabuaço EML are:

- 10% of the net profit derived from its mining activities or, alternatively, pay a
- Net Smelter Return up to 4% depending on the price per mtu:

- 0% up to USD$300/mtu,
- 1% from USD$301 to 400/mtu,
- 2% from USD$ 401 to 450/mtu,
- 3% from 451 to 500/mtu,
- 4% from >USD$500/mtu.

SRK understands that the selection of the applied royalty option is at the discretion of the Portuguese government. SRK's economic model assumes a 1% royalty on the sales revenue in line with the rate for a price of USD400/mtu.

21.4 Corporate Income Tax

The corporate income tax has been assessed using a 25% state and a municipal tax rate (total 26.6%). Depreciation has been applied on a straight line basis with the designations and annual rates shown in Table 21-2.

Table 21-1: Depreciation Designations and Rates

Designation	Rate (%)
Buildings/Land	5
Plant/Machinery	15
Heavy Equipment	20
Development Projects	33

21.5 Contingencies

No contingencies have been applied in the financial model over the estimated operating and capital expenditures.

21.6 Implementation Schedule

The financial model has been developed based on the assumption that plant and infrastructure construction will be completed in 2015, with underground development commencing in the second half of 2015 and ore production beginning in 2016. SRK believes that this schedule is aggressive and will require stringent compliance to the project milestones.

21.7 Financial Model

A discounted cashflow ("DCF") model has been developed based on the production schedule and costs discussed above. The summary results from each scenario are shown in Table 21-2 while the detailed financial models can be found in Appendix B. The start date of the model is 1 September 2013, and it is in real 2013 money terms.

Table 21-2: Project Scenario Comparison

Economic Model Summary	Units	A1	A2	A3	B1	B2	B3
Production							
Ore Processed	(kt)	3,557	3,557	3,557	3,557	3,557	3,557
	(% WO_3)	0.39	0.39	0.39	0.39	0.39	0.39
	(kt WO_3)	13.7	13.7	13.7	13.7	13.7	13.7
Recovered Metal	(MTU WO_3)	1,195,807	1,195,807	1,136,016	1,305,766	1,305,766	1,240,478
Financial							
Metal Price	(USD/MTU WO_3)	364.0	364.0	364.0	400.0	400.0	400.0
Revenue	(USDm)	435.3	435.3	413.5	522.3	522.3	496.2
Operating Expenditure	(USDm)	302.3	292.6	259.8	285.7	276.0	251.7
Royalty	(USDm)	4.4	4.4	4.1	5.2	5.2	5.0
Operating Profit	(USDm)	128.6	138.3	149.6	231.4	241.1	239.6
Net Profit	(USDm)	119.4	136.1	127.2	196.4	213.1	194.7
Capital Expenditure	(USDm)	100.3	135.6	72.7	109.1	144.3	81.2
Cashflow	(USDm)	19.1	0.6	54.5	87.3	68.8	113.6
Pre-Tax Reporting							
NPV @ 5%	(USDm)	7.1	-12.8	45.3	71.1	51.1	100.6
NPV @ 10%	(USDm)	-5.5	-21.4	25.6	39.2	23.3	64.1
NPV @ 15%	(USDm)	-13.1	-25.9	13.0	19.0	6.1	40.5
IRR	(%)	7.5	0.7	24.3	23.3	17.5	37.5
Post-Tax Reporting							
NPV @ 5%	(USDm)	1.6	-14.0	30.5	48.0	33.1	70.3
NPV @ 10%	(USDm)	-8.9	-22.1	15.4	23.6	11.3	43.0
NPV @ 15%	(USDm)	-15.2	-26.4	5.8	8.1	-2.1	25.3
IRR	(%)	5.6	0.1	19.6	18.9	14.1	30.7
Cash Cost							
Cash Cost	(USD/t ore)	86.2	83.5	74.2	81.8	79.1	72.2
	(USD/MTU WO_3)	256.5	248.3	232.3	222.8	215.3	206.9

SRK notes that the economic assessment is preliminary in nature and the production schedules are inclusive of material classified as Inferred Mineral Resources. Inferred Mineral Resources are considered too geologically speculative to have economic considerations applied to them that would enable them to be classified as Mineral Reserves. There is no certainty that the preliminary economic assessment will be realised.

The sensitivities of the $NPV_{5\%}$ relative to commodity price, operating costs and capital expenditures are shown in Figure 21-2 through to Figure 21-7 inclusive for the various options. This evaluation demonstrates that the production schedules are most sensitive to commodity price followed by operating costs. SRK notes that impact of changes to the processing recovery can be assessed based on the results of the commodity price sensitivity curve.

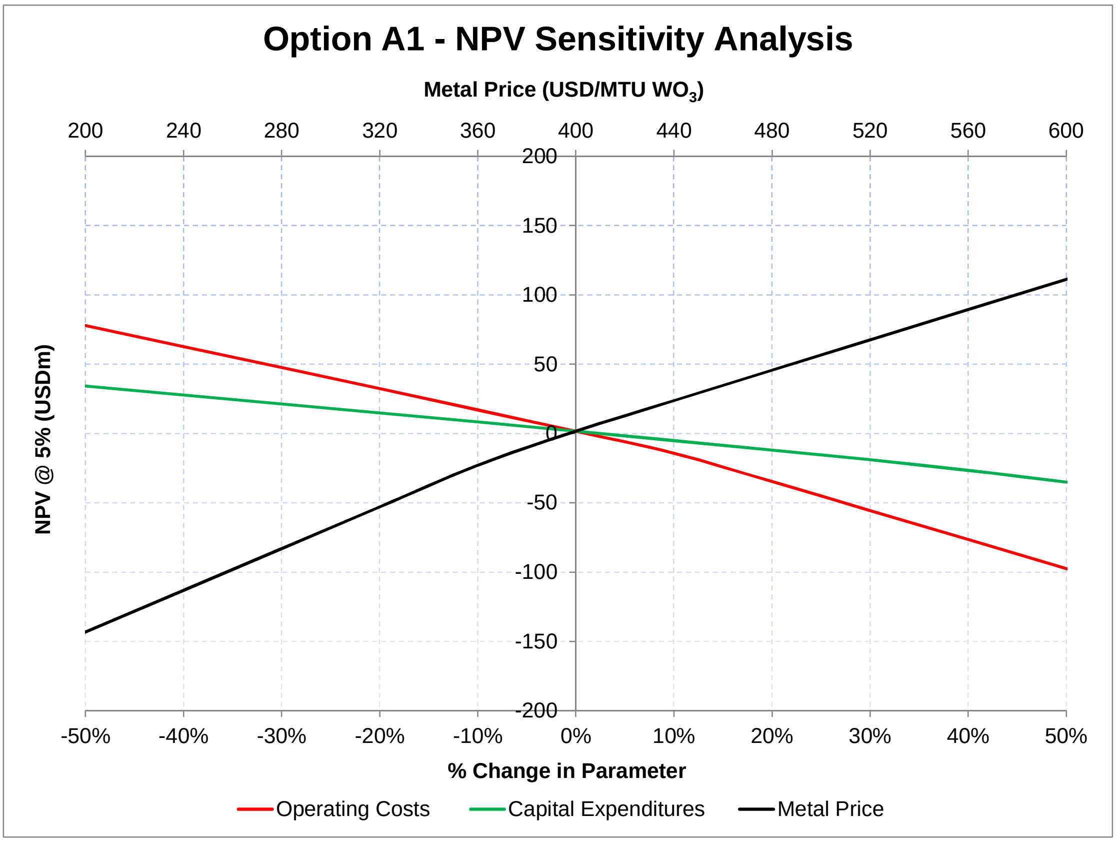


Figure 21-2: Option A1 - Sensitivity Analysis

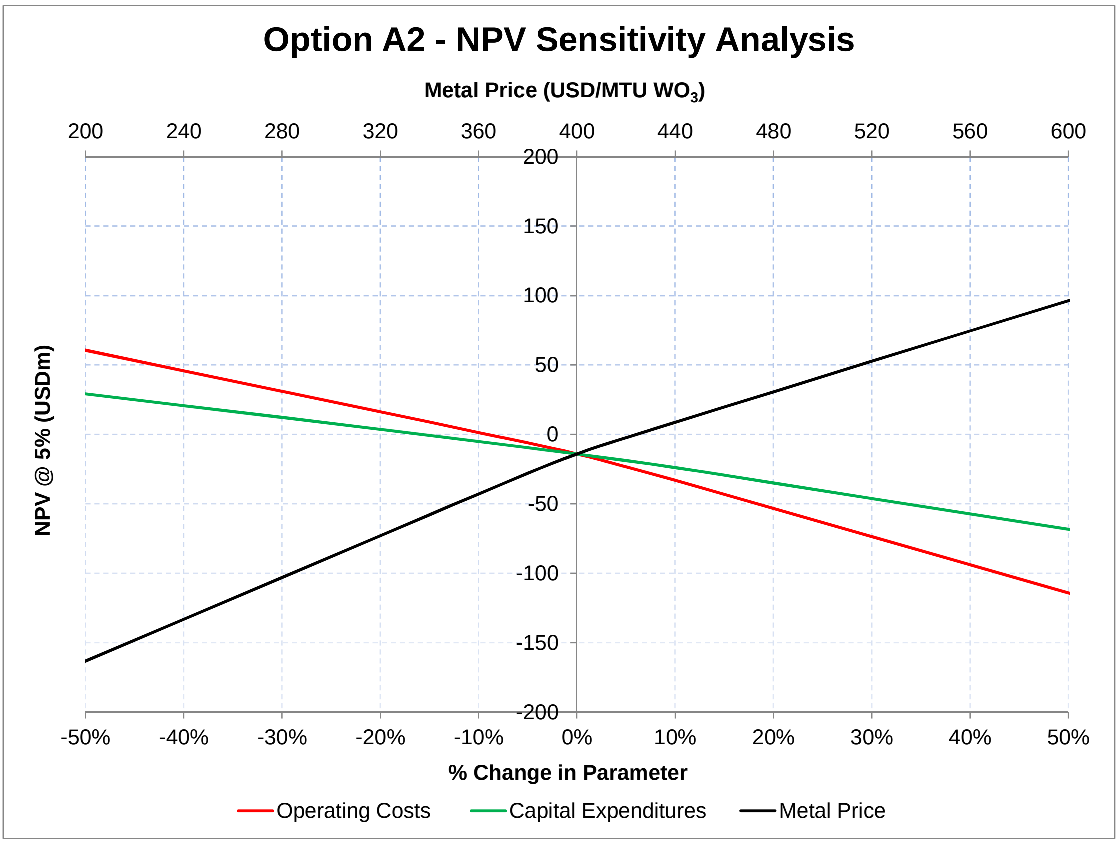


Figure 21-3: Option A2 - Sensitivity Analysis

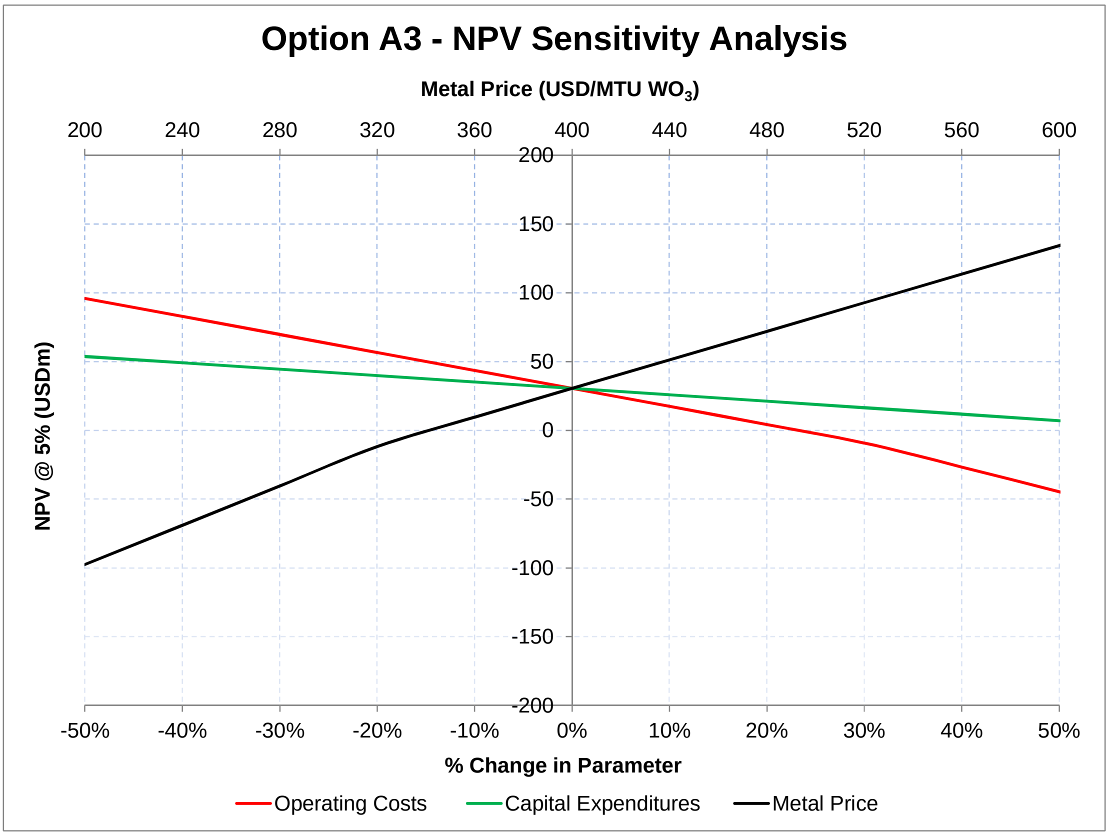


Figure 21-4: Option A3 - Sensitivity Analysis

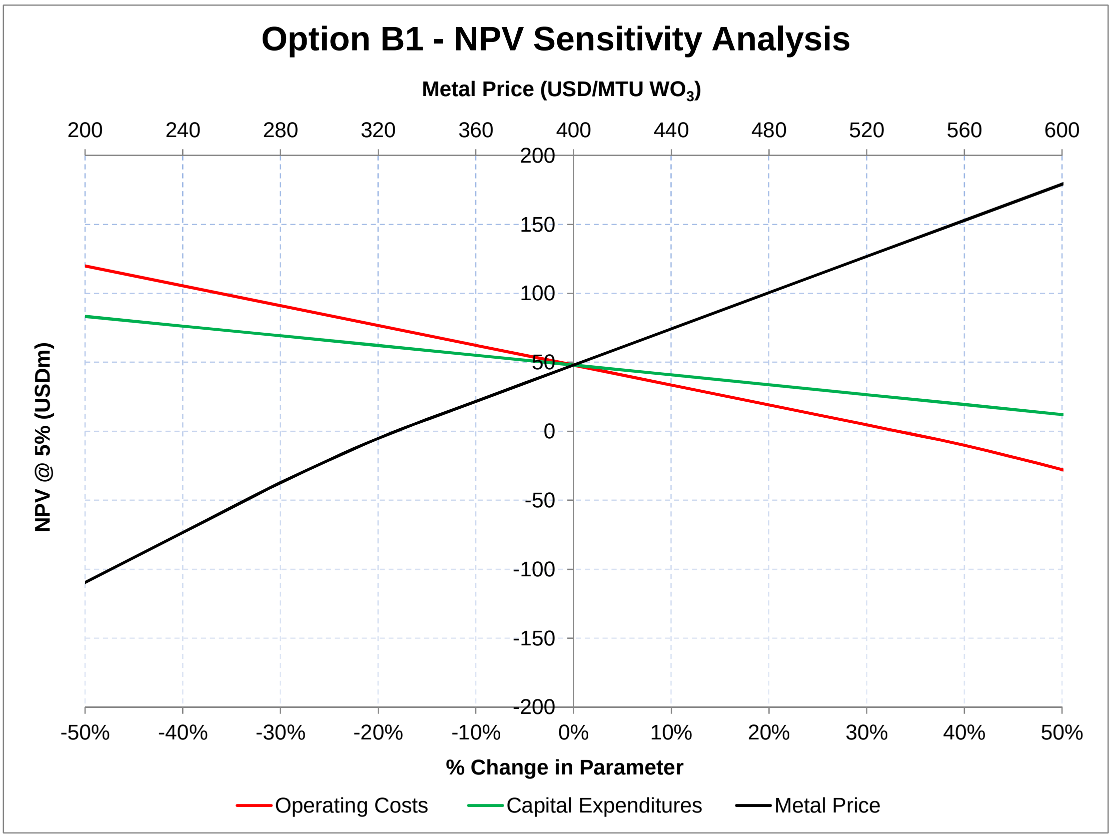


Figure 21-5: Option B1 - Sensitivity Analysis

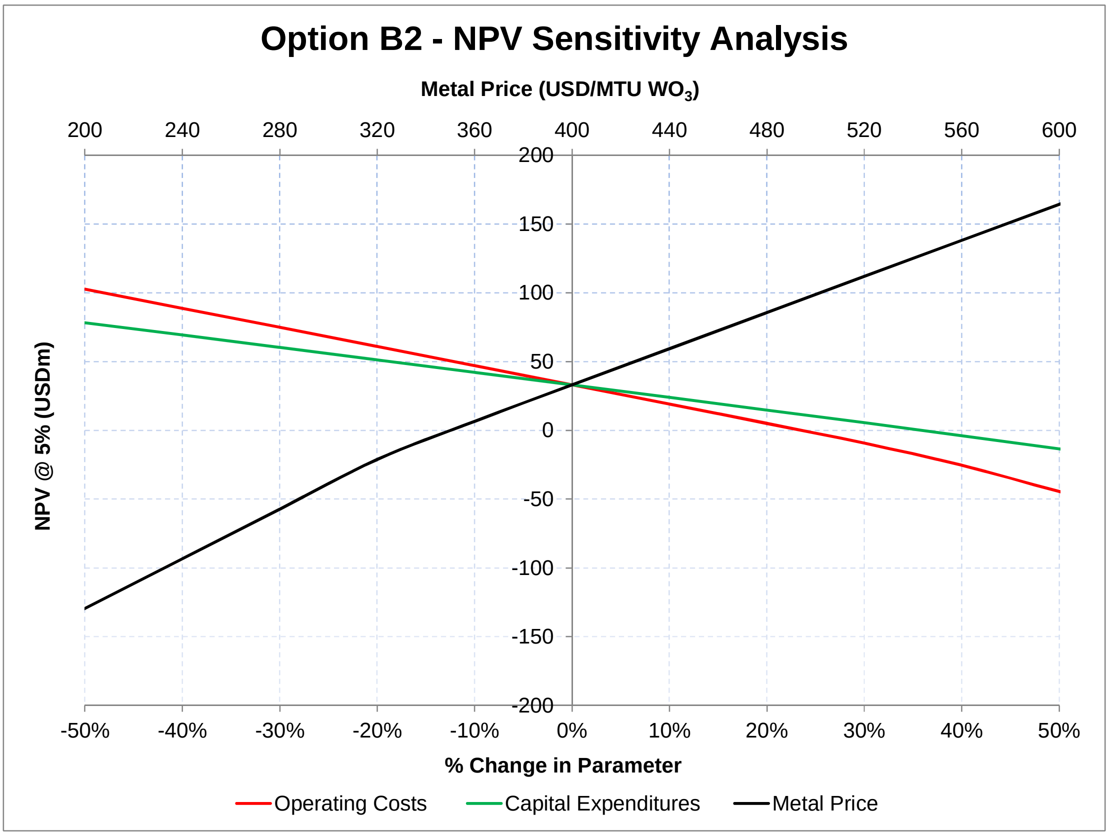


Figure 21-6: Option B2 - Sensitivity Analysis

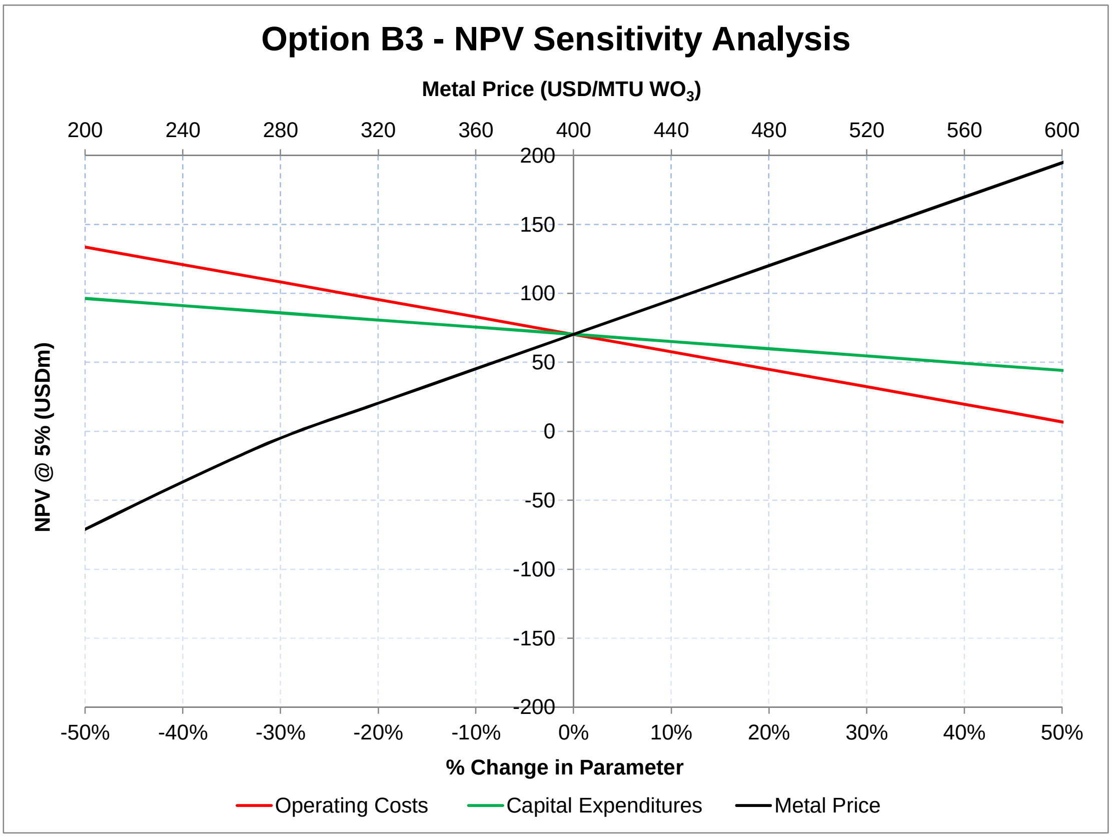


Figure 21-7: Option B3 - Sensitivity Analysis

21.8 Conclusion

The project options demonstrate $NPV_{5\%}$ ranging between negative USD 14.0m and positive USD 70.3m, with the maximum IRR achieved of 30.7%. The most positive scenario is option B3, with a total cash operating cost of USD 206.9/MTU WO_3.

The project is most sensitive to tungsten price, with an approximate break-even price of USD 290/MTU WO_3 for option B3. Operating costs also have a significant role in the project economics, and additional work should be undertaken to support the parameters for any preferred case taken forward.

Based on the limited technical work that has been undertaken and the assumptions which underlie this economic analysis, SRK concludes that there is potential for economically viable project options for the deposit. The positive financial indicators suggest that further studies and field work for this project are justified.

22 ADJACENT PROPERTIES

Whilst there are no active mining operations in the proximity of the Armamar Meda licence or the Tabuaço Project there are several other companies exploring in the region, notably:

- To the Northwest: Iberian's Régua exploration concession - also skarn tungsten/scheelite advanced project;
- To the South and East: COLT also holds Penedono (Au), Cedovim (Au, Sn-W) and Moimenta-Almendra (Au, Sn-W) exploration concessions; and
- To the Northeast: Minaport holds the Numão exploration concession (Au).

23 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information which would materially impact the conclusions of this report.

24 INTERPRETATION AND CONCLUSIONS

24.1 Geology and Exploration

SRK would anticipate that further exploration and drilling on the Quinta da Aveleira deposit and adjacent areas has high potential to outline and define additional tungsten resources.

There are numerous surface exposures of similar rocks in the region and several of these have already been sampled and shown to carry anomalous tungsten grades. The prospects of further economic mineralisation within the Tabuaço Project Area are therefore good, though in all probability a large part of these will be small in size.

24.2 Mineral Processing and Metallurgical Testwork

Three metallurgical programs have been conducted on samples from the Tabuaço deposit. The first two studies were conducted by Inspectorate Exploration and Mining Services Ltd ("Inspectorate") in 2011 and 2012, the first on an outcrop sample and the second on drill core reject material from the Tabuaço property. The third study was undertaken by Guangzhou Research Institute of Non-Ferrous Metals ("GRINFM") in 2012 under the auspices of Colt, using samples of diamond drill core.

The results of this series of tests demonstrated the following:

- Gravity separation only produced high concentrate grades at particle sizes of the order of 105 µm or less, however recoveries were low, with the best results being concentrates of the order of 50% WO_3 at WO_3 recoveries of the order of 50%;
- Flotation testwork conducted by Inspectorate resulted in relatively high recoveries, however final concentrate grades were quite low; and
- Flotation testwork conducted by GRINFM, who have particular expertise in tungsten processing, produced a relatively high grade final flotation concentrate (55.9% WO_3) at a relatively high WO_3 recovery. As this concentrate was high in P (3.4%), a final acid leach stage was employed, which resulted in a final concentrate grade of 70.6% WO_3, with a negligible loss of W in the acid leach stage.

On the basis of the GRINFM results, an overall flotation with concentrate leaching circuit was estimated to be capable of producing a concentrate assaying 70.6% WO_3 at an overall WO_3 recovery of 87.2%.

Ore sorting testwork has recently been commenced, with the aim of reducing the mass of material that needs to be ground ahead of flotation at a minimal W loss. Colt has also commissioned an investigation into a hydrometallurgical process route to produce tungsten oxide (WO_3) from the Tabuaço ore, or concentrates produced from it, using acid.

24.3 Mineral Resource Estimate

The Mineral Resources estimate for the Tabuaço deposit was completed by SRK and the Client in 2012 at 1,495 kt of Indicated classified resource at 0.55% WO_3 and 1,230 kt of Inferred classified resource at 0.59% WO_3.

24.4 Mining

The following conclusions can be made from the work carried out for the PEA:

- With the current resource, the mine life at a 0.22% WO3 CoG is one year of pre-production followed by ten years of production;
- The average RoM grade over the mine life is 0.39% WO_3;
- The average daily ore production rate for the life of mine is 1,415 tpd; and
- The peak ore production rate is in year 9, at 1,582 tpd.

24.5 Geotechnics

The following conclusions can be made from the work carried out for the PEA:

- The materials are within the very poor to poor range of the Q graph. These represent the average value and some areas will be significantly lower;
- Shotcrete and bolting should be in accordance with Barton's Category 4 - "Bolting + shotcrete", and should be considered in all areas as a minimum, until greater confidence in the material can be determined; and
- The oredrives should be shotcreted with spot bolting where required; and
- Changing ESR from temporary (3) to permanent (1.6) environments increases the support requirements.

24.6 Recovery Methods

The following effective recovery estimates for the process options have been used in the economic analysis (including 95% metal recovery for the ore sorter options):

- Option A1/A2: 87%;
- Option A3: 83%;
- Option B1/B2: 95%; and
- Option B3: 90%.

For Options B1, B2 and B3 Colt proposes to use an acid based processing path to produce high purity tungsten oxide end product. The process path is estimated to be less capital and operating cost intensive than the traditional ammonium paratungstate ("APT") processing path.

SRK believes that, with the exception of Option A1/A2, the flowsheets as proposed have not been tested at either laboratory or any larger scale, and that therefore the proposed recovery figures are not supported by the metallurgical testwork. Specifically:

- Options A3 and B3 are based on the use of ore sorting with, SRK understands, the assumption of a mass rejection of 55.8% with an accompanying loss of WO_3 of 5% based on an external dilution of 17%. Given that no quantitative testwork has been conducted on ore sorting, this assumption is unverifiable. While the difference between the Option B1/B2 and B3 recoveries is 5%, the difference in recovery between Options A1/A2 and A3 implies an ore sorting rejection of only 2%. Given that the 87% recovery for Option A1/A2 is supported by the testwork, SRK believes that the effective recovery for Option A3 should be 83%.
- As no acid leaching testwork results have been reported on the Tabuaço ore, the assumed recovery of 95% for Option B1/B2 (and implied for Option B3) is also unverifiable.

24.7 Project Infrastructure

The results of the previous work with approval of the Client inferred that the best area for the TSF would be in the valley of Passa Frio (stream) east of the proposed plant location. SRK followed this selection as most appropriate at this stage, however because of difficult terrain and high volumes of dam construction SRK also evaluated a dry stacking concept as viable alternative. The TSF will be a valley type of facility enclosed by the main embankment.

For wet tailings it has been assumed that the tailings will be stored sub-aerially behind the rock fill dam with the upstream liner

The concept of dry disposal has been developed with mine waste rock to be co-disposed with the filter pressed tailings. The starter wall embankment is essentially reduced in size at the front of the disposal area and the mine waste rock can be sequentially dumped with the tailings.

24.8 Environmental Studies, Permitting and Social or Community Impact

A review of current environmental and social documents was undertaken to determine the level of studies completed and provide recommendations for future work.

The following conclusions can be made from the review carried out for the PEA:

- Colt has an experimental mining licence for the Tabuaço project, which covers operations up to five ha in size and/or less than 150,000 ktpa of RoM;
- In accordance with Mining Law 88/90 of 16 March 1990, an application for a full mining licence requires the submission of a Mine Feasibility Study that includes an Environmental Impact Assessment ("EIA");
- An environmental scoping report was issued to Agencia Portuguesa de Ambiente (Portuguese Environmental Agency) ("APA") in August 2012. Colt intends to initiate the baseline phase of the EIA in January 2014, although a hydrogeological baseline study has already commenced; and
- Colt intends to obtain the required permits by 2015.

24.9 Capital and Operating Costs

The main findings from the capital and operating costs are summarised below:

- The processing costs show the most variation between the options (Options A3 and B3 have the lowest costs);
- The tailings costs show that the wet tailings options (A2 and B2) are higher than the dry tailings options (A1 and B1);
- Option B3 has the lowest operating costs, while Option A1 has the highest; and
- Option A3 has the lowest capital requirements, while Option B2 has the highest.

24.10 Economic Analysis

The project options demonstrate $NPV_{5\%}$ ranging between negative USD 14.0m and positive USD 70.3m, with the maximum IRR achieved of 30.7%. The most positive scenario is option B3, with a total cash operating cost of USD 206.9/MTU WO_3.

The project is most sensitive to tungsten price, with an approximate break-even price of USD 290/MTU WO_3 for option B3. Operating costs also have a significant role in the project economics, and additional work should be undertaken to support the parameters for any preferred case taken forward.

Based on the limited technical work that has been undertaken and the assumptions which underlie this economic analysis, SRK concludes that there is potential for economically viable project options for the deposit. The positive financial indicators suggest that further studies and field work for this project are justified.

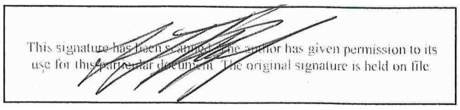
25 RECOMMENDATIONS

Based on the work carried out for this PEA, SRK recommends the following:

- Drilling
  - SRK notes that there is a lack of grade continuity in some parts of the upper Main Skarn horizon. A short programme of 4 further infill holes is proposed to resolve this between section lines 1250N and 1325N;
  - Colt has planned 10 holes to expand the inferred resource area at Aveleira. Should this programme demonstrate continuity then a further 20 holes may be warranted to infill and expand the resource into the Aveleira gap area;
  - A total of 20 holes would be warranted to explore and test extent of mineralisation at Quintã and north-westward to Távora and beyond along strike;
- Mineral Processing and Metallurgical Testing
  - SRK recognises the need to expand upon results from scoping level testwork, and that additional metallurgical studies need to be conducted, including the possibility of pilot plant testing of a 20 t sample to be taken from either the centreline of the proposed gallery or the shaft in advance of trial extraction.;
- Mineral Resource Estimate
  - SRK recommends a further update of the resource during 2013 and early 2014 to lead into a Feasibility Study;
- Mining
  - Review mining operating costs, especially the development costs;
  - Review mine infrastructure designs in order to optimise access to material above the CoG;
  - Review equipment productivities to ensure that the mining rate is achievable given the access and deposit geometry
  - Review the ventilation concept in order to ensure adequate air flow throughout the mine; and
  - Evaluate various backfill systems to optimise the proposed mining method.
- Geotechnical
  - It is recommended that specific triple tube geotechnical boreholes are drilled and logged to ascertain the true rock mass properties opposed to doubling resource drilling where core management was secondary to meterage. Logging Q parameters would enhance the confidence in the database leading to a greater understand of the rock mass stability. This will provide greater confidence in the material properties and has the potential to improve the Q support requirements;
  - Relocating the underground mine infrastructure in the granite (HW) could enhance the excavations stability through improved rock mass properties and increased RQD. This has not been proven in this study as there are insufficient boreholes that intercept the granite at depth in the HW;
  - Increased analysis needs to be undertaken into the material properties to zone the underground workings spatially;
- Hydrological and Hydrogeological Related Issues
  - A hydrogeological field program is required to investigate surface water diversions, to set up a permanent baseline monitoring program, to provide data to support the dewatering system design, to characterise the groundwater conditions for the TSF

    - and waste dump locations, and to provide data to support rock mechanical assessment;
    - Design closure plants for the EIA
    - Review the conceptual design of the TMF and WRD taking account of seepage and runoff management requirements;
    - Construct a Site Wide Water Balance Model to evaluate the need for makeup water and mine water discharges;
    - Develop a site wide water management plan;
    - Construct numerical groundwater and geochemical models to assess inflow rates to the underground mine and mine site seepage post closure, if geochemical testwork indicates the risk to be elevated; and
    - Confirm environmental impacts and permitting limits for river abstraction and mine water discharge.
- Recovery Methods
    - A significant program of further testwork is required in order to verify the technical feasibility of the proposed process options, and then to confirm the recovery figures assumed for this PEA;
- Project Infrastructure
    - Tailings alternative assessment to include the dry tailings disposal option;
    - Groundwater condition at and near the tailings impoundment and dam;
    - Geochemical properties of the tailings and the leachate originating from the tailings;
    - Geotechnical conditions of the dam foundations and properties of the tailings;
    - Borrow source investigation;
    - Engineering for the TSF;
- Environmental Studies, Permitting and Social or Community Impact
    - The project should plan for appropriate characterisation of waste rock, tailings material and seepage quality in accordance with requirements of Decree Law 10/2010;
    - As the project is located within a buffer zone of a United Nations Education, Scientific and Cultural Organisation (“UNESCO”) world heritage site the APA has requested that impacts on cultural heritage should be addressed following the methodology provided in the Guidelines of the International Council on Monuments and Sites;
    - Water quality prediction via geochemical modelling should be conducted to confirm the potential impacts of mine water discharge;
    - Proactive stakeholder engagement should be planned and implemented in a structured way to ensure that key stakeholder issues are identified and are addressed by the EIA and project design;
- Capital and Operating Costs
    - Manufacturer quotes should be obtained for major mining and processing equipment to validate the assumptions used in this cost estimate;
    - Detailed studies should be conducted to confirm the cost estimates made for dewatering, tailings, waste dump clearance and closure plans;
- Economic Analysis
    - Based on the limited technical work that has been undertaken and the assumptions which underlie this economic analysis, SRK concludes that there is potential for economically viable project options for the deposit. The positive financial indicators suggest that further studies and field work for this project are justified.

For and on behalf of SRK Consulting (UK) Limited



Jurgen Fuykschot,
Principal Consultant (Mining Engineering),
SRK Consulting (UK) Limited

Mike Beare,
Corporate Consultant (Mining Engineering),
SRK Consulting (UK) Limited

26 REFERENCES

Borrego, J.M., 2011. *São Pedro das Águias Tungsten Deposit Structural Mapping Results*. GeoLog Internal Report.

British Columbia Ministry of Energy and Mines, 2011. *Appendix VII – Conversion Factors*. [online] Available at:

<http://www.em.gov.bc.ca/Mining/Geoscience/MINFILE/ProductsDownloads/MINFILEDocumentation/CodingManual/Appendices/Pages/VII.aspx>. [Accessed 09 December 2011].

Charley, M.J., 1985. *Mineralogical and Petrological Descriptions of a Suite of Rocks from Tabuaço, Portugal*. BP Minerals Internal Report.

Colt Resources Inc., 2013. *Colt awarded two mining licences and two additional exploration concessions in Portugal*. Press Release, 20 February 2013.

Contecmina, 2012. [pdf] *SPA/Tabuaço Tungsten Project: Trade-Off Study – Executive Summary*. Translated from Portuguese.

Geoquest Consulting LTD, 2006. [pdf] *Armamar Meda NI 43-101 Report Sep 6 2010-Scanned Signature*.

Faria, A.F., 2008. [GeoLog Internal Report] *Summary Reports on the Tabuaço Tungsten Deposits – July 2008*.

Hinde, C. and Peters, T., 2008. *Tungsten Mining Journal Special Publication*. [online] London: Mining Journal. Available at: <http://www.mining-journal.com/__data/assets/supplement_file_attachment/0014/123314/Tungsten-scr.pdf>. [Accessed 09 December 2011].

Kimmel, JR, 2000. *GRG301K – Weather and climate, Köppen Climate classification flow chart*. [online] University of Texas at Austin, Department of Geography. Available at: <http://www.utexas.edu/depts/grg/kimmel/GRG301K/grg301kkoppen.html>. [Accessed 09 December 2011].

Lane, G.R., 2011. A Mineralogical Description of One Composite Head Sample from the Tabuaço Property, Portugal. *Process Mineralogical Consulting Report*, SEP2011-03, 24 October, 2011.

Pinto, A., 2010. Calcsilicate Rocks Rich in Tungsten, Tabuaço – Samples DHT01 and DHT05. Preliminary Mineralogical Characteristics (Portuguese).

Real, F. and Caupers, D., 2012. [pdf] *São Pedro das Águias Mining Project: Experimental Mine Plan – Descriptive Document*.

Real, F. and Caupers, D., 2013. [pdf] *Tabuaço Mining Project – Adaptation of Mining Underground Infrastructure to the Conditions of NI 43-101, issued by SRK Exploration on the 15 November 2012*.

Real, F., Martins A. and Caupers, D., 2012. [pdf] *Analise Comparativa de Alternativas de Operaçao e Transporte de Minério e Estéril, para os Niveis de Extracçao de 1000 e 1500 Toneladas por Dia*.

SRK Exploration, 2011. *NI 43-101 Technical Report on The Tabuaço Tungsten Project, Portugal, December 21 2011*. [online] SEDAR. Available at: <http://sedar.com/CheckCode.do;jsessionid=00001yw1yApIa_4B2tzRuW4hjti:17lkkk26t> [Accessed May 2013].

Wernerm B.T., Sinclair, D, and Amey, E.B., 1998. *International Strategic Mineral Issues Summary Report – Tungsten*. [online] US Geological Survey Circular 930-0. Available at: <http://pubs.usgs.gov/pdf/circular/c930-o.pdf>. [Accessed 09 December 2011].

27 GLOSSARY

27.1 Mineral Resources

The mineral resources and mineral reserves have been classified according to the "CIM Standards on Mineral Resources and Reserves: Definitions and Guidelines" (November 27, 2010). Accordingly, the Resources have been classified as Measured, Indicated or Inferred, the Reserves have been classified as Proven, and Probable based on the Measured and Indicated Resources as defined below.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilized organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

An 'Inferred Mineral Resource' is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

27.2 Mineral Reserves

A Mineral Reserve is the economically mineable part of a Measured or Indicated Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified. A Mineral Reserve includes diluting materials and allowances for losses that may occur when the material is mined.

A 'Probable Mineral Reserve' is the economically mineable part of an Indicated, and in some circumstances a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction can be justified.

A 'Proven Mineral Reserve' is the economically mineable part of a Measured Mineral Resource demonstrated by at least a Preliminary Feasibility Study. This Study must include adequate information on mining, processing, metallurgical, economic, and other relevant factors that demonstrate, at the time of reporting, that economic extraction is justified.

27.3 Definition of Terms

The following general mining terms may be used in this report (Table 27-1).

Table 27-1: Definition of Terms

Term	Definition
Assay	The chemical analysis of mineral samples to determine the metal content.
Capital Expenditure	All other expenditures not classified as operating costs.
Composite	Combining more than one sample result to give an average result over a larger distance.
Concentrate	A metal-rich product resulting from a mineral enrichment process such as gravity concentration or flotation, in which most of the desired mineral has been separated from the waste material in the mineralisation.
Crushing	Initial process of reducing particle size to render it more amenable for further processing.
Cut-off Grade (CoG)	The grade of mineralised rock, which determines as to whether or not it is potentially economic to recover its gold content by further concentration.
Dilution	Waste, which is unavoidably mined with mineralised material.
Dip	Angle of inclination of a geological feature/rock from the horizontal.
Fault	The surface of a fracture along which movement has occurred.
Footwall	The underlying side of a deposit or stope.
Gangue	Non-valuable components of the mineralisation.
Grade	The measure of concentration of gold within mineralised rock.
Hangingwall	The overlying side of a deposit or slope.
Haulage	A horizontal underground excavation which is used to transport mined material.
Hydrocyclone	A process whereby material is graded according to size by exploiting centrifugal forces of particulate materials.
Igneous	Primary crystalline rock formed by the solidification of magma.
Kriging	An interpolation method of assigning values from samples to blocks that minimizes the estimation error.
Level	Horizontal tunnel the primary purpose is the transportation of personnel and materials.
Lithological	Geological description pertaining to different rock types.
LoM Plans	Life-of-Mine plans.
LRP	Long Range Plan.
Material Properties	Mine properties.
Milling	A general term used to describe the process in which the extracted material is crushed and ground and subjected to physical or chemical treatment to extract the valuable metals to a concentrate or finished product.
Mineral/Mining Lease	A lease area for which mineral rights are held.
Mining Assets	The Material Properties and Significant Exploration Properties.
Ongoing Capital	Capital estimates of a routine nature, which is necessary for sustaining operations.
Ore Reserve	See Mineral Reserve.
Pillar	Rock left behind to help support the excavations in an underground mine.
RoM	Run-of-Mine.
Sedimentary	Pertaining to rocks formed by the accumulation of sediments, formed by the erosion of other rocks.
Shaft	An opening cut downwards from the surface for transporting personnel, equipment, supplies, mineralisation and waste.

Term	Definition
Sill	A thin, tabular, horizontal to sub-horizontal body of igneous rock formed by the injection of magma into planar zones of weakness.
Smelting	A high temperature pyrometallurgical operation conducted in a furnace, in which the valuable metal is collected to a molten matte or doré phase and separated from the gangue components that accumulate in a less dense molten slag phase.
Stope	Underground void created by mining.
Stratigraphy	The study of stratified rocks in terms of time and space.
Strike	Direction of line formed by the intersection of strata surfaces with the horizontal plane, always perpendicular to the dip direction.
Sulphide	A sulfur bearing mineral.
Tailings	Finely ground waste rock from which valuable minerals or metals have been extracted.
Thickening	The process of concentrating solid particles in suspension.
Total Expenditure	All expenditures including those of an operating and capital nature.
Variogram	A statistical representation of the characteristics (usually grade).

27.4 Abbreviations

The following abbreviations may be used in this report (Table 27-2).

Table 27-2: Abbreviations

Abbreviation	Unit or Term
A	ampere
AA	atomic absorption
A/m^2	amperes per square meter
ANFO	ammonium nitrate fuel oil
Ag	silver
Au	gold
AuEq	gold equivalent grade
℃	degrees Centigrade
CCD	counter-current decantation
CIL	carbon-in-leach
CoG	cut-off grade
cm	centimeter
cm^2	square centimeter
cm^3	cubic centimeter
cfm	cubic feet per minute
ConfC	confidence code
CRec	core recovery
CSS	closed-side setting
CTW	calculated true width
°	degree (degrees)
dia.	diameter
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
EURm	million EUR
EURk	thousand EUR
FA	fire assay
ft	foot (feet)
ft2	square foot (feet)
ft3	cubic foot (feet)

Abbreviation	Unit or Term
g	gram
gal	gallon
g/L	gram per liter
g-mol	gram-mole
gpm	gallons per minute
g/t	grams per tonne
ha	hectares
HDPE	Height Density Polyethylene
hp	horsepower
HTW	horizontal true width
ICP	induced couple plasma
ID2	inverse-distance squared
ID3	inverse-distance cubed
IFC	International Finance Corporation
ILS	Intermediate Leach Solution
kA	kiloamperes
kg	kilograms
km	kilometer
km2	square kilometer
koz	thousand troy ounce
kt	thousand tonnes
ktpd	thousand tonnes per day
ktpa	thousand tonnes per annum
kV	kilovolt
kW	kilowatt
kWh	kilowatt-hour
kWh/t	kilowatt-hour per metric tonne
L	liter
L/sec	liters per second
L/sec/m	liters per second per meter
lb	pound
LHD	Long-Haul Dump truck
LLDDP	Linear Low Density Polyethylene Plastic
LOI	Loss On Ignition
LoM	Life-of-Mine
m	meter
m^2	square meter
m^3	cubic meter
masl	meters above sea level
MARN	Ministry of the Environment and Natural Resources
MDA	Mine Development Associates
mg/L	milligrams/liter
mm	millimeter
mm^2	square millimeter
mm^3	cubic millimeter
MME	Mine & Mill Engineering

Abbreviation	Unit or Term
Moz	million troy ounces
Mt	million tonnes
Mtpa	million tonnes per annum
MTW	measured true width
mtu	Metric tonne unit
MW	million watts
m.y.	million years
NGO	non-governmental organization
NI 43-101	Canadian National Instrument 43-101
OSC	Ontario Securities Commission
oz	troy ounce
%	Percent
PLC	Programmable Logic Controller
PLS	Pregnant Leach Solution
PMF	probable maximum flood
ppb	parts per billion
ppm	parts per million
QA/QC	Quality Assurance/Quality Control
RC	rotary circulation drilling
RoM	Run-of-Mine
RQD	Rock Quality Description
SEC	U.S. Securities & Exchange Commission
sec	second
SG	specific gravity
SPT	standard penetration testing
st	short ton (2,000 pounds)
t	tonne (metric ton) (2,204.6 pounds)
t/h	tonnes per hour
t/d	tonnes per day
t/y	tonnes per year
TSF	tailings storage facility
TSP	total suspended particulates
USDm	million USD
USDk	thousand USD
µmm	micron or microns
V	volts
VFD	variable frequency drive
W	watt
XRD	x-ray diffraction
y	year

APPENDIX

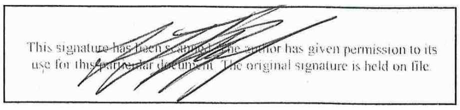
A CERTIFICATE & CONSENT OF AUTHOR

CERTIFICATE AND CONSENT

To Accompany the report entitled: "**A Preliminary Economic Assessment on the Tabuaço Tungsten Project, Portugal"**, **effective date 1st October 2013,**

I, Jurgen Fuykschot, residing at 26 Maes Yr Annedd, Canton, Cardiff, UK, CF5 1GR do hereby certify that:

1) I am a Principal Consultant (Mining Engineering) with the firm of SRK Consulting (UK) Ltd ("SRK") with an office at Churchill House, 17 Churchill Way, Cardiff, UK CF10 2HH;
2) I am a graduate of Delft University of Technology, The Netherlands in 1991 (MSc Mining Engineering), and qualified with an MBA from Deakin University, Melbourne, Australia in 2001. I have practiced my profession continuously since 1992, initially as a mining engineer based in the Netherland and Australia from 1995 until joining SRK (UK) in 2006 where I have worked extensively on technical studies on metal projects. My main areas of expertise are mine design and scheduling;
3) I am registered with the Australian Institute of Mining and Metallurgy and have achieved 'Competent Person' status. (membership number 306269);
4) I have personally inspected the subject project between the 6th and 8th of May 2013;
5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of National Instrument 43-101;
6) I, as a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
7) I have had no prior involvement with the subject property;
8) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
9) I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Tabuaço Tungsten Project or securities of Colt Resources Inc.;
10) That, as of the date of this technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading;
11) I consent to the filing of the technical report with any stock exchange and other regulatory authority and any publication for regulatory purposes, including electronic publication in the public company files on their websites accessible to the public of extracts from the technical report; and
12) I confirm that I have read the news release dated 4th September 2013 in which the findings of the technical report have been disclosed publically and have no reason to believe that there are any misrepresentations in the information derived from the report or that the press release dated 4th September 2013 contains any misrepresentations of the information contained in the report.



Jurgen Fuykschot (MAusIMM(CP) MSc, MBA)
Cardiff UK
1 October 2013

APPENDIX

B ECONOMIC ANALYSIS

Financial Model – Option A1

A1 - Base Case Dry Tailings	Units		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026
Production																
Mining																
Ore	(kt)	3,557	0	0	0	328	364	394	392	385	386	384	395	347	181	0
	(% WO_3)	0.386	0.000	0.000	0.000	0.437	0.414	0.440	0.356	0.395	0.339	0.379	0.412	0.337	0.325	0.000
	(kt WO_3)	13.7	0.0	0.0	0.0	1.4	1.5	1.7	1.4	1.5	1.3	1.5	1.6	1.2	0.6	0.0
Processing																
Ore	(kt)	3,557	0	0	0	328	364	394	392	385	386	384	395	347	181	0
	(% WO_3)	0.386	0.000	0.000	0.000	0.437	0.414	0.440	0.356	0.395	0.339	0.379	0.412	0.337	0.325	0.000
	(kt WO_3)	13.7	0.0	0.0	0.0	1.4	1.5	1.7	1.4	1.5	1.3	1.5	1.6	1.2	0.6	0.0
Sales																
Recovered Metal	(MTU WO_3)	1,195,807	0	0	0	124,858	131,094	150,905	121,462	132,303	113,945	126,535	141,703	101,787	51,213	0
Adjusted Price	(USD/MTU WO_3)	364	364	364	364	364	364	364	364	364	364	364	364	364	364	364
Revenue	(USDk)	435,274	0	0	0	45,448	47,718	54,929	44,212	48,158	41,476	46,059	51,580	37,051	18,642	0
Operating Expenditure																
Mining	(USDk)	(150,388)	-	-	-	(13,879)	(15,403)	(16,654)	(16,586)	(16,263)	(16,320)	(16,220)	(16,718)	(14,691)	(7,654)	-
Mine Services	(USDk)	(3,760)	-	-	-	(347)	(385)	(416)	(415)	(407)	(408)	(406)	(418)	(367)	(191)	-
Ore Sorting	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reject handling	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Crushing	(USDk)	(5,335)	-	-	-	(492)	(546)	(591)	(588)	(577)	(579)	(575)	(593)	(521)	(272)	-
Processing	(USDk)	(83,348)	-	-	-	(7,692)	(8,537)	(9,230)	(9,192)	(9,013)	(9,045)	(8,990)	(9,266)	(8,142)	(4,242)	-
Ore Transport	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Backfill	(USDk)	(19,562)	-	-	-	(1,805)	(2,004)	(2,166)	(2,157)	(2,115)	(2,123)	(2,110)	(2,175)	(1,911)	(996)	-
Tailings	(USDk)	(11,108)	-	-	-	(879)	(760)	(1,384)	(763)	(679)	(996)	(866)	(2,046)	(1,799)	(938)	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Dew atering	(USDk)	(1,100)	-	-	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	-
Selling Cost	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
G&A	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Operating Costs - Subtotal	**(USDk)**	**(302,343)**	**-**	**-**	**(100)**	**(27,756)**	**(30,576)**	**(33,613)**	**(32,860)**	**(32,155)**	**(32,581)**	**(32,258)**	**(34,399)**	**(30,242)**	**(15,803)**	**-**
Royalty - Mine Gate	(USDk)	(4,353)	-	-	-	(454)	(477)	(549)	(442)	(482)	(415)	(461)	(516)	(371)	(186)	-
Working Capital	(USDk)	0	-	-	7	(1,912)	(6)	(393)	831	(366)	572	(398)	(313)	924	561	493
Operating Costs - Total	**(USDk)**	**(306,696)**	**-**	**-**	**(93)**	**(30,122)**	**(31,059)**	**(34,555)**	**(32,471)**	**(33,002)**	**(32,423)**	**(33,117)**	**(35,228)**	**(29,689)**	**(15,429)**	**493**
Operating Profit	**(USDk)**	**128,578**	**-**	**-**	**(93)**	**15,326**	**16,659**	**20,374**	**11,741**	**15,157**	**9,053**	**12,942**	**16,352**	**7,361**	**3,213**	**493**
Corporate Income Tax & Profit Royalty																
Profit tax	(USDk)	(9,171)	-	-	-	-	-	-	-	(146)	(131)	(2,606)	(3,737)	(1,848)	(703)	-
Effective Tax Rate	(%)	7.1	-	-	-	-	-	-	-	1.0	1.4	20.1	22.9	25.1	21.9	-
Net Profit (Pre-Royalty)	(USDk)	119,407	-	-	(93)	15,326	16,659	20,374	11,741	15,010	8,922	10,336	12,615	5,514	2,510	493
Royalty - Profit Related	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Operating Profit	**(USDk)**	**119,407**	**-**	**-**	**(93)**	**15,326**	**16,659**	**20,374**	**11,741**	**15,010**	**8,922**	**10,336**	**12,615**	**5,514**	**2,510**	**493**
Capital Expenditure																
Site Capital Expenditures																
Mining	(USDk)	(23,191)	-	-	(3,703)	(5,964)	(1,224)	(1,223)	(1,355)	(2,105)	(6,242)	(1,375)	-	-	-	-
Plant	(USDk)	(44,200)	-	-	(44,200)	-	-	-	-	-	-	-	-	-	-	-
Paste Backfill Plant	(USDk)	(7,000)	-	-	(7,000)	-	-	-	-	-	-	-	-	-	-	-
Tailings - Construction	(USDk)	(12,194)	-	-	(10,114)	(1,040)	-	-	-	-	(1,040)	-	-	-	-	-
Tailings - Closure	(USDk)	(1,040)	-	-	-	-	-	-	-	-	-	-	-	-	(1,040)	-
Waste Management	(USDk)	(500)	-	-	(500)	-	-	-	-	-	-	-	-	-	-	-
Ore Sorter	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Nitric Acid Plant	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Closure	(USDk)	(5,000)	-	-	-	-	-	-	-	-	-	-	-	-	(2,500)	(2,500)
Drilling/Eng. Studies	(USDk)	(7,179)	(897)	(6,282)	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital Expenditure	**(USDk)**	**(100,305)**	**(897)**	**(6,282)**	**(65,517)**	**(7,004)**	**(1,224)**	**(1,223)**	**(1,355)**	**(2,105)**	**(7,282)**	**(1,375)**	**-**	**-**	**(3,540)**	**(2,500)**
Cashflow - Pre Finance																
Cashflow	**(USDk)**	**19,103**	**(897)**	**(6,282)**	**(65,610)**	**8,322**	**15,435**	**19,151**	**10,386**	**12,906**	**1,641**	**8,960**	**12,615**	**5,514**	**(1,030)**	**(2,007)**
Cumulative Cashflow	(USDk)		(897)	(7,179)	(72,789)	(64,467)	(49,032)	(29,881)	(19,495)	(6,589)	(4,949)	4,012	16,626	22,140	21,109	19,103
Discounted Cashflow (NPV) @ 5%	**(USDk)**	**1,602**														
IRR	**(%)**	**5.6%**														
Reporting Statistics																
Total Cash Costs	(USD/$t_{ore\ mined}$)	86.23	-	-	-	85.94	85.24	86.74	84.90	84.85	85.49	85.29	88.31	88.11	88.34	-
Total Cash Costs	(USD/MTU WO_3)	256.5	-	-	-	225.9	236.9	226.4	274.2	246.7	289.6	258.6	246.4	300.8	312.2	-

Financial Model – Option A2

A2 - Base Case Wet Tailings	Units		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026
Production																
Mining																
Ore	(kt)	3,557	0	0	0	328	364	394	392	385	386	384	395	347	181	0
	(% WO_3)	0.386	0.000	0.000	0.000	0.437	0.414	0.440	0.356	0.395	0.339	0.379	0.412	0.337	0.325	0.000
	(kt WO_3)	13.7	0.0	0.0	0.0	1.4	1.5	1.7	1.4	1.5	1.3	1.5	1.6	1.2	0.6	0.0
Processing																
Ore	(kt)	3,557	0	0	0	328	364	394	392	385	386	384	395	347	181	0
	(% WO_3)	0.386	0.000	0.000	0.000	0.437	0.414	0.440	0.356	0.395	0.339	0.379	0.412	0.337	0.325	0.000
	(kt WO_3)	13.7	0.0	0.0	0.0	1.4	1.5	1.7	1.4	1.5	1.3	1.5	1.6	1.2	0.6	0.0
Sales																
Recovered Metal	(MTU WO_3)	1,195,807	0	0	0	124,858	131,094	150,905	121,462	132,303	113,945	126,535	141,703	101,787	51,213	0
Adjusted Price	(USD/MTU WO_3)	364	364	364	364	364	364	364	364	364	364	364	364	364	364	364
Revenue	(USDk)	435,274	0	0	0	45,448	47,718	54,929	44,212	48,158	41,476	46,059	51,580	37,051	18,642	0
Operating Expenditure																
Mining	(USDk)	(150,388)	-	-	-	(13,879)	(15,403)	(16,654)	(16,586)	(16,263)	(16,320)	(16,220)	(16,718)	(14,691)	(7,654)	-
Mine Services	(USDk)	(3,760)	-	-	-	(347)	(385)	(416)	(415)	(407)	(408)	(406)	(418)	(367)	(191)	-
Ore Sorting	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reject handling	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Crushing	(USDk)	(5,335)	-	-	-	(492)	(546)	(591)	(588)	(577)	(579)	(575)	(593)	(521)	(272)	-
Processing	(USDk)	(83,348)	-	-	-	(7,692)	(8,537)	(9,230)	(9,192)	(9,013)	(9,045)	(8,990)	(9,266)	(8,142)	(4,242)	-
Ore Transport	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Backfill	(USDk)	(19,562)	-	-	-	(1,805)	(2,004)	(2,166)	(2,157)	(2,115)	(2,123)	(2,110)	(2,175)	(1,911)	(996)	-
Tailings	(USDk)	(1,388)	-	-	-	(110)	(95)	(173)	(95)	(85)	(125)	(108)	(256)	(225)	(117)	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Dew atering	(USDk)	(1,100)	-	-	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	-
Selling Cost	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
G&A	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Operating Costs - Subtotal	**(USDk)**	**(292,624)**	**-**	**-**	**(100)**	**(26,987)**	**(29,911)**	**(32,402)**	**(32,193)**	**(31,561)**	**(31,709)**	**(31,501)**	**(32,610)**	**(28,668)**	**(14,983)**	**-**
Royalty - Mine Gate	(USDk)	(4,353)	-	-	-	(454)	(477)	(549)	(442)	(482)	(415)	(461)	(516)	(371)	(186)	-
Working Capital	(USDk)	0	-	-	7	(1,962)	0	(429)	867	(361)	554	(390)	(381)	938	610	547
Operating Costs - Total	**(USDk)**	**(296,976)**	**-**	**-**	**(93)**	**(29,403)**	**(30,388)**	**(33,380)**	**(31,768)**	**(32,403)**	**(31,570)**	**(32,352)**	**(33,506)**	**(28,100)**	**(14,559)**	**547**
Operating Profit	**(USDk)**	**138,297**	**-**	**-**	**(93)**	**16,045**	**17,330**	**21,549**	**12,444**	**15,755**	**9,906**	**13,707**	**18,074**	**8,950**	**4,082**	**547**
Corporate Income Tax & Profit Royalty																
Profit tax	(USDk)	(2,181)	-	-	-	-	-	-	-	-	-	-	-	(1,263)	(919)	-
Effective Tax Rate	(%)	1.6	-	-	-	-	-	-	-	-	-	-	-	14.1	22.5	-
Net Profit (Pre-Royalty)	(USDk)	136,116	-	-	(93)	16,045	17,330	21,549	12,444	15,755	9,906	13,707	18,074	7,688	3,164	547
Royalty - Profit Related	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Operating Profit	**(USDk)**	**136,116**	**-**	**-**	**(93)**	**16,045**	**17,330**	**21,549**	**12,444**	**15,755**	**9,906**	**13,707**	**18,074**	**7,688**	**3,164**	**547**
Capital Expenditure																
Site Capital Expenditures																
Mining	(USDk)	(23,191)	-	-	(3,703)	(5,964)	(1,224)	(1,223)	(1,355)	(2,105)	(6,242)	(1,375)	-	-	-	-
Plant	(USDk)	(44,200)	-	-	(44,200)	-	-	-	-	-	-	-	-	-	-	-
Paste Backfill Plant	(USDk)	(7,000)	-	-	(7,000)	-	-	-	-	-	-	-	-	-	-	-
Tailings - Construction	(USDk)	(47,190)	-	-	(8,710)	(19,240)	-	-	-	-	(19,240)	-	-	-	-	-
Tailings - Closure	(USDk)	(1,300)	-	-	-	-	-	-	-	-	-	-	-	-	(1,300)	-
Waste Management	(USDk)	(500)	-	-	(500)	-	-	-	-	-	-	-	-	-	-	-
Ore Sorter	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Nitric Acid Plant	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Closure	(USDk)	(5,000)	-	-	-	-	-	-	-	-	-	-	-	-	(2,500)	(2,500)
Drilling/Eng. Studies	(USDk)	(7,179)	(897)	(6,282)	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital Expenditure	**(USDk)**	**(135,561)**	**(897)**	**(6,282)**	**(64,113)**	**(25,204)**	**(1,224)**	**(1,223)**	**(1,355)**	**(2,105)**	**(25,482)**	**(1,375)**	**-**	**-**	**(3,800)**	**(2,500)**
Cashflow - Pre Finance																
Cashflow	**(USDk)**	**556**	**(897)**	**(6,282)**	**(64,206)**	**(9,159)**	**16,106**	**20,326**	**11,089**	**13,651**	**(15,576)**	**12,332**	**18,074**	**7,688**	**(636)**	**(1,953)**
Cumulative Cashflow	(USDk)		(897)	(7,179)	(71,385)	(80,545)	(64,439)	(44,112)	(33,023)	(19,372)	(34,948)	(22,617)	(4,543)	3,145	2,509	556
Discounted Cashflow (NPV) @ 5%	**(USDk)**	**(14,050)**														
IRR	**(%)**	**0.1%**														
Reporting Statistics																
Total Cash Costs	(USD/$t_{ore\ mined}$)	83.50	-	-	-	83.60	83.42	83.66	83.20	83.31	83.23	83.32	83.78	83.58	83.80	-
Total Cash Costs	(USD/MTU WO_3)	248.3	-	-	-	219.8	231.8	218.4	268.7	242.2	281.9	252.6	233.8	285.3	296.2	-

Financial Model – Option A3

A3 - Base with Sorter	Units		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026
Production																
Mining																
Ore	(kt)	3,557	0	0	0	328	364	394	392	385	386	384	395	347	181	0
	(% WO_3)	0.386	0.000	0.000	0.000	0.437	0.414	0.440	0.356	0.395	0.339	0.379	0.412	0.337	0.325	0.000
	(kt WO_3)	13.7	0.0	0.0	0.0	1.4	1.5	1.7	1.4	1.5	1.3	1.5	1.6	1.2	0.6	0.0
Processing																
Ore	(kt)	1,572	0	0	0	145	161	174	173	170	171	170	175	154	80	0
	(% WO_3)	0.831	0.000	0.000	0.000	0.940	0.889	0.947	0.765	0.850	0.730	0.815	0.886	0.724	0.699	0.000
	(kt WO_3)	13.1	0.0	0.0	0.0	1.4	1.4	1.6	1.3	1.4	1.2	1.4	1.5	1.1	0.6	0.0
Sales																
Recovered Metal	(MTU WO_3)	1,136,016	0	0	0	118,616	124,539	143,359	115,389	125,688	108,248	120,208	134,618	96,698	48,653	0
Adjusted Price	(USD/MTU WO_3)	364	364	364	364	364	364	364	364	364	364	364	364	364	364	364
Revenue	(USDk)	413,510	0	0	0	43,176	45,332	52,183	42,002	45,751	39,402	43,756	49,001	35,198	17,710	0
Operating Expenditure																
Mining	(USDk)	(150,388)	-	-	-	(13,879)	(15,403)	(16,654)	(16,586)	(16,263)	(16,320)	(16,220)	(16,718)	(14,691)	(7,654)	-
Mine Services	(USDk)	(3,760)	-	-	-	(347)	(385)	(416)	(415)	(407)	(408)	(406)	(418)	(367)	(191)	-
Ore Sorting	(USDk)	(7,113)	-	-	-	(657)	(729)	(788)	(785)	(769)	(772)	(767)	(791)	(695)	(362)	-
Reject handling	(USDk)	(3,970)	-	-	-	(366)	(407)	(440)	(438)	(429)	(431)	(428)	(441)	(388)	(202)	-
Crushing	(USDk)	(5,335)	-	-	-	(492)	(546)	(591)	(588)	(577)	(579)	(575)	(593)	(521)	(272)	-
Processing	(USDk)	(40,513)	-	-	-	(3,739)	(4,149)	(4,486)	(4,468)	(4,381)	(4,396)	(4,370)	(4,504)	(3,958)	(2,062)	-
Ore Transport	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Backfill	(USDk)	(19,562)	-	-	-	(1,805)	(2,004)	(2,166)	(2,157)	(2,115)	(2,123)	(2,110)	(2,175)	(1,911)	(996)	-
Tailings	(USDk)	(293)	-	-	-	-	-	(30)	-	-	-	-	(112)	(99)	(52)	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Dewatering	(USDk)	(1,100)	-	-	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	-
Selling Cost	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
G&A	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Operating Costs - Subtotal	**(USDk)**	**(259,776)**	**-**	**-**	**(100)**	**(23,946)**	**(26,564)**	**(28,743)**	**(28,596)**	**(28,042)**	**(28,139)**	**(27,968)**	**(28,936)**	**(25,440)**	**(13,301)**	**-**
Royalty - Mine Gate	(USDk)	(4,135)	-	-	-	(432)	(453)	(522)	(420)	(458)	(394)	(438)	(490)	(352)	(177)	-
Working Capital	(USDk)	0	-	-	7	(1,975)	(11)	(420)	827	(339)	523	(369)	(367)	908	636	581
Operating Costs - Total	**(USDk)**	**(263,912)**	**-**	**-**	**(93)**	**(26,354)**	**(27,028)**	**(29,685)**	**(28,189)**	**(28,839)**	**(28,010)**	**(28,775)**	**(29,794)**	**(24,884)**	**(12,842)**	**581**
Operating Profit	**(USDk)**	**149,598**	**-**	**-**	**(93)**	**16,822**	**18,304**	**22,498**	**13,813**	**16,912**	**11,392**	**14,981**	**19,207**	**10,314**	**4,867**	**581**
Corporate Income Tax & Profit Royalty																
Profit tax	(USDk)	(22,415)	-	-	-	-	(606)	(3,984)	(1,934)	(2,706)	(1,300)	(3,262)	(4,617)	(2,660)	(1,254)	(91)
Effective Tax Rate	(%)	15.0	-	-	-	-	3.3	17.7	14.0	16.0	11.4	21.8	24.0	25.8	25.8	15.7
Net Profit (Pre-Royalty)	(USDk)	127,184	-	-	(93)	16,822	17,698	18,514	11,879	14,206	10,092	11,719	14,590	7,654	3,614	490
Royalty - Profit Related	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Operating Profit	**(USDk)**	**127,184**	**-**	**-**	**(93)**	**16,822**	**17,698**	**18,514**	**11,879**	**14,206**	**10,092**	**11,719**	**14,590**	**7,654**	**3,614**	**490**
Capital Expenditure																
Site Capital Expenditures																
Mining	(USDk)	(23,191)	-	-	(3,703)	(5,964)	(1,224)	(1,223)	(1,355)	(2,105)	(6,242)	(1,375)	-	-	-	-
Plant	(USDk)	(26,520)	-	-	(26,520)	-	-	-	-	-	-	-	-	-	-	-
Paste Backfill Plant	(USDk)	(7,000)	-	-	(7,000)	-	-	-	-	-	-	-	-	-	-	-
Tailings - Construction	(USDk)	(325)	-	-	(325)	-	-	-	-	-	-	-	-	-	-	-
Tailings - Closure	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Management	(USDk)	(1,500)	-	-	(1,500)	-	-	-	-	-	-	-	-	-	-	-
Ore Sorter	(USDk)	(2,000)	-	-	(2,000)	-	-	-	-	-	-	-	-	-	-	-
Nitric Acid Plant	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Closure	(USDk)	(5,000)	-	-	-	-	-	-	-	-	-	-	-	-	(2,500)	(2,500)
Drilling/Eng. Studies	(USDk)	(7,179)	(897)	(6,282)	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital Expenditure	**(USDk)**	**(72,716)**	**(897)**	**(6,282)**	**(41,048)**	**(5,964)**	**(1,224)**	**(1,223)**	**(1,355)**	**(2,105)**	**(6,242)**	**(1,375)**	**-**	**-**	**(2,500)**	**(2,500)**
Cashflow - Pre Finance																
Cashflow	**(USDk)**	**54,468**	**(897)**	**(6,282)**	**(41,141)**	**10,858**	**16,474**	**17,291**	**10,524**	**12,101**	**3,850**	**10,343**	**14,590**	**7,654**	**1,114**	**(2,010)**
Cumulative Cashflow	(USDk)		(897)	(7,179)	(48,320)	(37,462)	(20,988)	(3,697)	6,826	18,928	22,778	33,121	47,711	55,365	56,478	54,468
Discounted Cashflow (NPV) @ 5%	**(USDk)**	**30,485**														
IRR	**(%)**	**19.6%**														
Reporting Statistics																
Total Cash Costs	(USD/$t_{ore\ mined}$)	74.20	-	-	-	74.27	74.17	74.30	73.97	74.10	73.93	74.05	74.42	74.23	74.46	-
Total Cash Costs	(USD/MTU WO_3)	232.3	-	-	-	205.5	216.9	204.1	251.5	226.7	263.6	236.3	218.6	266.7	277.0	-

Financial Model – Option B1

B1 - Nitric Dry Tailings	**Units**		**2013**	**2014**	**2015**	**2016**	**2017**	**2018**	**2019**	**2020**	**2021**	**2022**	**2023**	**2024**	**2025**	**2026**
Production																
Mining																
Ore	(kt)	3,557	0	0	0	328	364	394	392	385	386	384	395	347	181	0
	(% WO_3)	0.386	0.000	0.000	0.000	0.437	0.414	0.440	0.356	0.395	0.339	0.379	0.412	0.337	0.325	0.000
	(kt WO_3)	13.7	0.0	0.0	0.0	1.4	1.5	1.7	1.4	1.5	1.3	1.5	1.6	1.2	0.6	0.0
Processing																
Ore	(kt)	3,557	0	0	0	328	364	394	392	385	386	384	395	347	181	0
	(% WO_3)	0.386	0.000	0.000	0.000	0.437	0.414	0.440	0.356	0.395	0.339	0.379	0.412	0.337	0.325	0.000
	(kt WO_3)	13.7	0.0	0.0	0.0	1.4	1.5	1.7	1.4	1.5	1.3	1.5	1.6	1.2	0.6	0.0
Sales																
Recovered Metal	(MTU WO_3)	1,305,766	0	0	0	136,340	143,149	164,781	132,631	144,469	124,423	138,171	154,733	111,147	55,922	0
Adjusted Price	(USD/MTU WO_3)	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400
Revenue	(USDk)	522,306	0	0	0	54,536	57,259	65,912	53,053	57,788	49,769	55,268	61,893	44,459	22,369	0
Operating Expenditure																
Mining	(USDk)	(150,388)	-	-	-	(13,879)	(15,403)	(16,654)	(16,586)	(16,263)	(16,320)	(16,220)	(16,718)	(14,691)	(7,654)	-
Mine Services	(USDk)	(3,760)	-	-	-	(347)	(385)	(416)	(415)	(407)	(408)	(406)	(418)	(367)	(191)	-
Ore Sorting	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reject handling	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Crushing	(USDk)	(5,335)	-	-	-	(492)	(546)	(591)	(588)	(577)	(579)	(575)	(593)	(521)	(272)	-
Processing	(USDk)	(66,679)	-	-	-	(6,154)	(6,829)	(7,384)	(7,354)	(7,211)	(7,236)	(7,192)	(7,413)	(6,514)	(3,393)	-
Ore Transport	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Backfill	(USDk)	(19,562)	-	-	-	(1,805)	(2,004)	(2,166)	(2,157)	(2,115)	(2,123)	(2,110)	(2,175)	(1,911)	(996)	-
Tailings	(USDk)	(11,127)	-	-	-	(881)	(762)	(1,386)	(765)	(681)	(998)	(868)	(2,048)	(1,801)	(938)	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Dew atering	(USDk)	(1,100)	-	-	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	-
Selling Cost	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
G&A	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Operating Costs - Subtotal	**(USDk)**	**(285,693)**	**-**	**-**	**(100)**	**(26,219)**	**(28,871)**	**(31,769)**	**(31,024)**	**(30,354)**	**(30,773)**	**(30,462)**	**(32,549)**	**(28,616)**	**(14,956)**	**-**
Royalty - Mine Gate	(USDk)	(5,223)	-	-	-	(545)	(573)	(659)	(531)	(578)	(498)	(553)	(619)	(445)	(224)	-
Working Capital	(USDk)	(0)	-	-	7	(2,757)	(57)	(521)	1,008	(426)	679	(472)	(407)	1,179	913	855
Operating Costs - Total	**(USDk)**	**(290,916)**	**-**	**-**	**(93)**	**(29,522)**	**(29,500)**	**(32,949)**	**(30,546)**	**(31,358)**	**(30,592)**	**(31,488)**	**(33,575)**	**(27,881)**	**(14,267)**	**855**
Operating Profit	**(USDk)**	**231,390**	**-**	**-**	**(93)**	**25,014**	**27,759**	**32,963**	**22,506**	**26,430**	**19,177**	**23,781**	**28,318**	**16,578**	**8,102**	**855**
Corporate Income Tax & Profit Royalty																
Profit tax	(USDk)	(35,010)	-	-	-	-	-	(5,587)	(3,317)	(4,315)	(2,674)	(5,587)	(7,028)	(4,382)	(2,048)	(71)
Effective Tax Rate	(%)	15.1	-	-	-	-	-	16.9	14.7	16.3	13.9	23.5	24.8	26.4	25.3	8.3
Net Profit (Pre-Royalty)	(USDk)	196,381	-	-	(93)	25,014	27,759	27,376	19,189	22,114	16,503	18,194	21,291	12,196	6,054	784
Royalty - Profit Related	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Operating Profit	**(USDk)**	**196,381**	**-**	**-**	**(93)**	**25,014**	**27,759**	**27,376**	**19,189**	**22,114**	**16,503**	**18,194**	**21,291**	**12,196**	**6,054**	**784**
Capital Expenditure																
Site Capital Expenditures																
Mining	(USDk)	(23,191)	-	-	(3,703)	(5,964)	(1,224)	(1,223)	(1,355)	(2,105)	(6,242)	(1,375)	-	-	-	-
Plant	(USDk)	(44,950)	-	-	(44,950)	-	-	-	-	-	-	-	-	-	-	-
Paste Backfill Plant	(USDk)	(7,000)	-	-	(7,000)	-	-	-	-	-	-	-	-	-	-	-
Tailings - Construction	(USDk)	(12,194)	-	-	(10,114)	(1,040)	-	-	-	-	(1,040)	-	-	-	-	-
Tailings - Closure	(USDk)	(1,040)	-	-	-	-	-	-	-	-	-	-	-	-	(1,040)	-
Waste Management	(USDk)	(500)	-	-	(500)	-	-	-	-	-	-	-	-	-	-	-
Ore Sorter	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Nitric Acid Plant	(USDk)	(8,000)	-	-	(8,000)	-	-	-	-	-	-	-	-	-	-	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Closure	(USDk)	(5,000)	-	-	-	-	-	-	-	-	-	-	-	-	(2,500)	(2,500)
Drilling/Eng. Studies	(USDk)	(7,179)	(897)	(6,282)	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital Expenditure	**(USDk)**	**(109,055)**	**(897)**	**(6,282)**	**(74,267)**	**(7,004)**	**(1,224)**	**(1,223)**	**(1,355)**	**(2,105)**	**(7,282)**	**(1,375)**	**-**	**-**	**(3,540)**	**(2,500)**
Cashflow - Pre Finance																
Cashflow	**(USDk)**	**87,326**	**(897)**	**(6,282)**	**(74,360)**	**18,009**	**26,535**	**26,153**	**17,834**	**20,010**	**9,221**	**16,818**	**21,291**	**12,196**	**2,514**	**(1,716)**
Cumulative Cashflow	(USDk)		(897)	(7,179)	(81,539)	(63,530)	(36,995)	(10,842)	6,992	27,002	36,223	53,042	74,332	86,528	89,042	87,326
Discounted Cashflow (NPV) @ 5%	**(USDk)**	**48,047**														
IRR	**(%)**	**18.9%**														
Reporting Statistics																
Total Cash Costs	(USD/$t_{ore\ mined}$)	81.79	-	-	-	81.54	80.82	82.33	80.44	80.42	81.02	80.85	83.88	83.64	83.86	-
Total Cash Costs	(USD/MTU WO_3)	222.8	-	-	-	196.3	205.7	196.8	237.9	214.1	251.3	224.5	214.4	261.5	271.4	-

Financial Model – Option B2

B2 - Nitric Wet Tailings	Units		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026
Production																
Mining																
Ore	(kt)	3,557	0	0	0	328	364	394	392	385	386	384	395	347	181	0
	(% WO_3)	0.386	0.000	0.000	0.000	0.437	0.414	0.440	0.356	0.395	0.339	0.379	0.412	0.337	0.325	0.000
	(kt WO_3)	13.7	0.0	0.0	0.0	1.4	1.5	1.7	1.4	1.5	1.3	1.5	1.6	1.2	0.6	0.0
Processing																
Ore	(kt)	3,557	0	0	0	328	364	394	392	385	386	384	395	347	181	0
	(% WO_3)	0.386	0.000	0.000	0.000	0.437	0.414	0.440	0.356	0.395	0.339	0.379	0.412	0.337	0.325	0.000
	(kt WO_3)	13.7	0.0	0.0	0.0	1.4	1.5	1.7	1.4	1.5	1.3	1.5	1.6	1.2	0.6	0.0
Sales																
Recovered Metal	(MTU WO_3)	1,305,766	0	0	0	136,340	143,149	164,781	132,631	144,469	124,423	138,171	154,733	111,147	55,922	0
Adjusted Price	(USD/MTU WO_3)	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400
Revenue	(USDk)	522,306	0	0	0	54,536	57,259	65,912	53,053	57,788	49,769	55,268	61,893	44,459	22,369	0
Operating Expenditure																
Mining	(USDk)	(150,388)	-	-	-	(13,879)	(15,403)	(16,654)	(16,586)	(16,263)	(16,320)	(16,220)	(16,718)	(14,691)	(7,654)	-
Mine Services	(USDk)	(3,760)	-	-	-	(347)	(385)	(416)	(415)	(407)	(408)	(406)	(418)	(367)	(191)	-
Ore Sorting	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Reject handling	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Crushing	(USDk)	(5,335)	-	-	-	(492)	(546)	(591)	(588)	(577)	(579)	(575)	(593)	(521)	(272)	-
Processing	(USDk)	(66,679)	-	-	-	(6,154)	(6,829)	(7,384)	(7,354)	(7,211)	(7,236)	(7,192)	(7,413)	(6,514)	(3,393)	-
Ore Transport	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Backfill	(USDk)	(19,562)	-	-	-	(1,805)	(2,004)	(2,166)	(2,157)	(2,115)	(2,123)	(2,110)	(2,175)	(1,911)	(996)	-
Tailings	(USDk)	(1,391)	-	-	-	(110)	(95)	(173)	(96)	(85)	(125)	(108)	(256)	(225)	(117)	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Dew atering	(USDk)	(1,100)	-	-	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	-
Selling Cost	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
G&A	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Operating Costs - Subtotal	**(USDk)**	**(275,956)**	**-**	**-**	**(100)**	**(25,449)**	**(28,204)**	**(30,556)**	**(30,355)**	**(29,758)**	**(29,900)**	**(29,703)**	**(30,757)**	**(27,040)**	**(14,135)**	**-**
Royalty - Mine Gate	(USDk)	(5,223)	-	-	-	(545)	(573)	(659)	(531)	(578)	(498)	(553)	(619)	(445)	(224)	-
Working Capital	(USDk)	0	-	-	7	(2,808)	(50)	(557)	1,044	(421)	661	(465)	(475)	1,194	962	909
Operating Costs - Total	**(USDk)**	**(281,180)**	**-**	**-**	**(93)**	**(28,802)**	**(28,827)**	**(31,772)**	**(29,842)**	**(30,757)**	**(29,737)**	**(30,721)**	**(31,851)**	**(26,291)**	**(13,396)**	**909**
Operating Profit	**(USDk)**	**241,127**	**-**	**-**	**(93)**	**25,734**	**28,433**	**34,140**	**23,211**	**27,031**	**20,032**	**24,547**	**30,042**	**18,168**	**8,973**	**909**
Corporate Income Tax & Profit Royalty																
Profit tax	(USDk)	(28,016)	-	-	-	-	-	(1,675)	(3,511)	(4,481)	(1,241)	(4,129)	(5,833)	(4,820)	(2,263)	(62)
Effective Tax Rate	(%)	11.6	-	-	-	-	-	4.9	15.1	16.6	6.2	16.8	19.4	26.5	25.2	6.8
Net Profit (Pre-Royalty)	(USDk)	213,111	-	-	(93)	25,734	28,433	32,465	19,700	22,550	18,791	20,418	24,209	13,349	6,709	847
Royalty - Profit Related	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Operating Profit	**(USDk)**	**213,111**	**-**	**-**	**(93)**	**25,734**	**28,433**	**32,465**	**19,700**	**22,550**	**18,791**	**20,418**	**24,209**	**13,349**	**6,709**	**847**
Capital Expenditure																
Site Capital Expenditures																
Mining	(USDk)	(23,191)	-	-	(3,703)	(5,964)	(1,224)	(1,223)	(1,355)	(2,105)	(6,242)	(1,375)	-	-	-	-
Plant	(USDk)	(44,950)	-	-	(44,950)	-	-	-	-	-	-	-	-	-	-	-
Paste Backfill Plant	(USDk)	(7,000)	-	-	(7,000)	-	-	-	-	-	-	-	-	-	-	-
Tailings - Construction	(USDk)	(47,190)	-	-	(8,710)	(19,240)	-	-	-	-	(19,240)	-	-	-	-	-
Tailings - Closure	(USDk)	(1,300)	-	-	-	-	-	-	-	-	-	-	-	-	(1,300)	-
Waste Management	(USDk)	(500)	-	-	(500)	-	-	-	-	-	-	-	-	-	-	-
Ore Sorter	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Nitric Acid Plant	(USDk)	(8,000)	-	-	(8,000)	-	-	-	-	-	-	-	-	-	-	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Closure	(USDk)	(5,000)	-	-	-	-	-	-	-	-	-	-	-	-	(2,500)	(2,500)
Drilling/Eng. Studies	(USDk)	(7,179)	(897)	(6,282)	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital Expenditure	**(USDk)**	**(144,311)**	**(897)**	**(6,282)**	**(72,863)**	**(25,204)**	**(1,224)**	**(1,223)**	**(1,355)**	**(2,105)**	**(25,482)**	**(1,375)**	**-**	**-**	**(3,800)**	**(2,500)**
Cashflow - Pre Finance																
Cashflow	**(USDk)**	**68,801**	**(897)**	**(6,282)**	**(72,956)**	**530**	**27,208**	**31,242**	**18,345**	**20,445**	**(6,691)**	**19,043**	**24,209**	**13,349**	**2,909**	**(1,653)**
Cumulative Cashflow	(USDk)		(897)	(7,179)	(80,135)	(79,606)	(52,397)	(21,155)	(2,810)	17,635	10,944	29,987	54,196	67,545	70,454	68,801
Discounted Cashflow (NPV) @ 5%	**(USDk)**	**33,083**														
IRR	**(%)**	**14.1%**														
Reporting Statistics																
Total Cash Costs	(USD/$t_{ore\ mined}$)	79.06	-	-	-	79.19	78.99	79.25	78.74	78.87	78.76	78.87	79.35	79.10	79.32	-
Total Cash Costs	(USD/MTU WO_3)	215.3	-	-	-	190.7	201.0	189.4	232.9	210.0	244.3	219.0	202.8	247.3	256.8	-

Financial Model – Option B3

B3 - Nitric with Sorter	Units		2013	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026
Production																
Mining																
Ore	(kt)	3,557	0	0	0	328	364	394	392	385	386	384	395	347	181	0
	(% WO_3)	0.386	0.000	0.000	0.000	0.437	0.414	0.440	0.356	0.395	0.339	0.379	0.412	0.337	0.325	0.000
	(kt WO_3)	13.7	0.0	0.0	0.0	1.4	1.5	1.7	1.4	1.5	1.3	1.5	1.6	1.2	0.6	0.0
Processing																
Ore	(kt)	1,572	0	0	0	145	161	174	173	170	171	170	175	154	80	0
	(% WO_3)	0.831	0.000	0.000	0.000	0.940	0.889	0.947	0.765	0.850	0.730	0.815	0.886	0.724	0.699	0.000
	(kt WO_3)	13.1	0.0	0.0	0.0	1.4	1.4	1.6	1.3	1.4	1.2	1.4	1.5	1.1	0.6	0.0
Sales																
Recovered Metal	(MTU WO_3)	1,240,478	0	0	0	129,523	135,991	156,542	126,000	137,246	118,202	131,262	146,996	105,590	53,126	0
Adjusted Price	(USD/MTU WO_3)	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400
Revenue	(USDk)	496,191	0	0	0	51,809	54,397	62,617	50,400	54,898	47,281	52,505	58,799	42,236	21,251	0
Operating Expenditure																
Mining	(USDk)	(150,388)	-	-	-	(13,879)	(15,403)	(16,654)	(16,586)	(16,263)	(16,320)	(16,220)	(16,718)	(14,691)	(7,654)	-
Mine Services	(USDk)	(3,760)	-	-	-	(347)	(385)	(416)	(415)	(407)	(408)	(406)	(418)	(367)	(191)	-
Ore Sorting	(USDk)	(7,113)	-	-	-	(657)	(729)	(788)	(785)	(769)	(772)	(767)	(791)	(695)	(362)	-
Reject handling	(USDk)	(3,970)	-	-	-	(366)	(407)	(440)	(438)	(429)	(431)	(428)	(441)	(388)	(202)	-
Crushing	(USDk)	(5,335)	-	-	-	(492)	(546)	(591)	(588)	(577)	(579)	(575)	(593)	(521)	(272)	-
Processing	(USDk)	(32,410)	-	-	-	(2,991)	(3,320)	(3,589)	(3,574)	(3,505)	(3,517)	(3,496)	(3,603)	(3,166)	(1,649)	-
Ore Transport	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Backfill	(USDk)	(19,562)	-	-	-	(1,805)	(2,004)	(2,166)	(2,157)	(2,115)	(2,123)	(2,110)	(2,175)	(1,911)	(996)	-
Tailings	(USDk)	(294)	-	-	-	-	-	(30)	-	-	-	-	(113)	(99)	(52)	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Dewatering	(USDk)	(1,100)	-	-	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	(100)	-
Selling Cost	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
G&A	(USDk)	(13,871)	-	-	-	(1,280)	(1,421)	(1,536)	(1,530)	(1,500)	(1,505)	(1,496)	(1,542)	(1,355)	(706)	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Operating Costs - Subtotal	**(USDk)**	**(251,675)**	**-**	**-**	**(100)**	**(23,199)**	**(25,734)**	**(27,846)**	**(27,702)**	**(27,166)**	**(27,260)**	**(27,094)**	**(28,036)**	**(24,649)**	**(12,889)**	**-**
Royalty - Mine Gate	(USDk)	(4,962)	-	-	-	(518)	(544)	(626)	(504)	(549)	(473)	(525)	(588)	(422)	(213)	-
Working Capital	(USDk)	-	-	-	7	(2,732)	(53)	(537)	995	(398)	625	(440)	(455)	1,144	947	899
Operating Costs - Total	**(USDk)**	**(256,637)**	**-**	**-**	**(93)**	**(26,449)**	**(26,332)**	**(29,009)**	**(27,212)**	**(28,112)**	**(27,108)**	**(28,060)**	**(29,079)**	**(23,928)**	**(12,155)**	**899**
Operating Profit	**(USDk)**	**239,554**	**-**	**-**	**(93)**	**25,360**	**28,065**	**33,608**	**23,188**	**26,786**	**20,173**	**24,445**	**29,719**	**18,308**	**9,095**	**899**
Corporate Income Tax & Profit Royalty																
Profit tax	(USDk)	(44,829)	-	-	-	-	(4,592)	(6,691)	(4,164)	(5,073)	(3,483)	(5,865)	(7,508)	(4,858)	(2,416)	(179)
Effective Tax Rate	(%)	18.7	-	-	-	-	16.4	19.9	18.0	18.9	17.3	24.0	25.3	26.5	26.6	19.9
Net Profit (Pre-Royalty)	(USDk)	194,726	-	-	(93)	25,360	23,473	26,917	19,024	21,713	16,690	18,580	22,211	13,450	6,679	721
Royalty - Profit Related	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Net Operating Profit	**(USDk)**	**194,726**	**-**	**-**	**(93)**	**25,360**	**23,473**	**26,917**	**19,024**	**21,713**	**16,690**	**18,580**	**22,211**	**13,450**	**6,679**	**721**
Capital Expenditure																
Site Capital Expenditures																
Mining	(USDk)	(23,191)	-	-	(3,703)	(5,964)	(1,224)	(1,223)	(1,355)	(2,105)	(6,242)	(1,375)	-	-	-	-
Plant	(USDk)	(26,970)	-	-	(26,970)	-	-	-	-	-	-	-	-	-	-	-
Paste Backfill Plant	(USDk)	(7,000)	-	-	(7,000)	-	-	-	-	-	-	-	-	-	-	-
Tailings - Construction	(USDk)	(325)	-	-	(325)	-	-	-	-	-	-	-	-	-	-	-
Tailings - Closure	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Waste Management	(USDk)	(1,500)	-	-	(1,500)	-	-	-	-	-	-	-	-	-	-	-
Ore Sorter	(USDk)	(2,000)	-	-	(2,000)	-	-	-	-	-	-	-	-	-	-	-
Nitric Acid Plant	(USDk)	(8,000)	-	-	(8,000)	-	-	-	-	-	-	-	-	-	-	-
Water Treatment	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mine Closure	(USDk)	(5,000)	-	-	-	-	-	-	-	-	-	-	-	-	(2,500)	(2,500)
Drilling/Eng. Studies	(USDk)	(7,179)	(897)	(6,282)	-	-	-	-	-	-	-	-	-	-	-	-
Contingency	(USDk)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total Capital Expenditure	**(USDk)**	**(81,166)**	**(897)**	**(6,282)**	**(49,498)**	**(5,964)**	**(1,224)**	**(1,223)**	**(1,355)**	**(2,105)**	**(6,242)**	**(1,375)**	**-**	**-**	**(2,500)**	**(2,500)**
Cashflow - Pre Finance																
Cashflow	**(USDk)**	**113,560**	**(897)**	**(6,282)**	**(49,591)**	**19,396**	**22,248**	**25,694**	**17,669**	**19,609**	**10,448**	**17,205**	**22,211**	**13,450**	**4,179**	**(1,779)**
Cumulative Cashflow	(USDk)		(897)	(7,179)	(56,770)	(37,374)	(15,126)	10,568	28,237	47,846	58,294	75,499	97,710	111,160	115,339	113,560
Discounted Cashflow (NPV) @ 5%	**(USDk)**	**70,324**														
IRR	**(%)**	**30.7%**														
Reporting Statistics																
Total Cash Costs	(USD/$t_{ore\ mined}$)	72.16	-	-	-	72.25	72.14	72.29	71.91	72.06	71.85	72.00	72.39	72.16	72.38	-
Total Cash Costs	(USD/MTU WO_3)	206.9	-	-	-	183.1	193.2	181.9	223.9	201.9	234.6	210.4	194.7	237.4	246.6	-