TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE PISKANJA BORATE PROJECT, SERBIA

REPORT PREPARED IN ACCORDANCE WITH THE GUIDELINES OF NATIONAL INSTRUMENT 43-101 AND ACCOMPANYING DOCUMENTS 43-101.F1 AND 43-101.CP.

Prepared For

Erin Ventures Inc.

Report Prepared by

srk consulting

SRK Consulting (UK) Limited

UK5932

Effective Date: 1st September 2014

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Table of Contents

List of Tables

List of Figures

List of Technical Appendices

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1 SUMMARY

1.1 Introduction

SRK Consulting (UK) Ltd (SRK) has been commissioned by Erin Ventures Inc. (Erin, hereinafter also referred to as the Company or the Client) to prepare a Technical Report and Preliminary Economic Assessment (PEA) for the Piskanja Borate Project (Piskanja or the Project) located in Serbia. Balkan Gold doo is the current licence holder and is a wholly owned subsidiary of Erin. Erin is currently listed on the Toronto Stock Exchange Venture Exchange (TSX-V) using the code EV.

This technical report is an update to the report titled “Technical Report and Mineral Resource Estimate for the Piskanja Borate Project, Serbia” dated 29 November 2013 (the 2013 Technical Report) , which was prepared by SRK’s sister company SRK Exploration Services Ltd (SRK ES). The geological descriptions and Mineral Resource Estimate (MRE) in this updated report are essentially unchanged from that presented in the 2013 Technical Report but this update also includes a PEA based on technical work undertaken since this report was issued.

This technical report and MRE have been prepared under the guidelines of National Instrument 43-101 and accompanying documents 43-101F1 and 43-101.CP (NI43-101). The Mineral Resource statement reported herein was prepared in accordance with the Canadian Institute of Mining (CIM) “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” (CIM Guidelines).

The Piskanja Project is considered to be an “Advanced Property” as defined by NI 43-101. The Project is located in southern Serbia, 10 km north of the town of Raška and 160 km south of the capital, Belgrade.

1.2 Licence Status

The Ministry of Mining and Environmental and Spatial Planning of the Republic of Serbia (the Ministry) granted Balkan Gold doo, Exploration Licence #1934, which is the licence covering the Project, on 23 August 2010 under the 1995 Law on Mining published in Official Gazette of RS, no. 44/95. Balkan Gold doo is a wholly owned subsidiary of Erin. The Exploration Licence was initially granted for a period of one year but an extension was subsequently granted in August 2011 for an additional year until 23 August 2012.

In July 2012, Erin applied for and was granted a new three year Exploration Licence under the new Law on Mining and Geological Researches, published in Official Gazette no. 88/2011 which came into force in January 2012. The renewed Exploration Licence #1934, was granted on 05 November 2012 and is valid until 05 November 2015 and a further two extensions are potentially available, albeit with 25% size reduction of the previously granted licence area per extension. The licensed area is the same as under the previous title (3.075 km^2) and covers the Piskanja mineral deposit in its entirety. There are no other known mineral deposits within the licence area.

1.3 Geology

Piskanja is situated within the Jarandol Basin, which forms the eastern part of the larger Gradac-Baljevac Graben. The Gradac-Baljevac Graben is located in the Varder Zone, a geotectonic unit that lies east of the Dinaric Alps and continues into central Macedonia. The basins contained within the Gradac-Baljevac Graben are associated with rifting in the Miocene (23 Ma to 5 Ma) which affected the ophiolite basement of Upper Jurassic age. Prior to the formation of the Jarandol Basin (and G-B Basin) intense magmatism in the Oligocene (between 34 Ma and 23 Ma) introduced andesitic, dacitic volcanic and pyroclastic flows which extended over the Ibar and Raška River valleys for some 40 km. A number of granite stocks were also intruded some 5-10 km west of the licence area during this period.

During the Miocene, the basins in the Vardar Zone were filled by sediments associated with various facies typical of continental basins including alluvial, lacustrine and swamp settings and the transitional environments associated with them. Fluctuations in water level and sediment input gave rise to alternating units of mudstone, shale, sandstone and lignite seams, tuffaceous material was also deposited related to on-going volcanism related to extension. This rift-related volcanism was accompanied by hydrothermal activity beneath the basins, circulating fluids through the basement and sediments. It was this high thermal and tectonic activity that is thought to have led to borate mobilisation and deposition within the Jarandol Basin sediments. The age of sediments within the Jarandol Basin has been determined based on sparse fossils as Lower and Middle Miocene, however, Upper Miocene sedimentation is also possible.

The Piskanja borate deposit is of continental lacustrine type, typical of many global boron deposits, and is considered to have formed within a closed basin with abnormally high salinity. The boron mineralisation is most likely to have been sourced from local volcanic rocks, from which it has been leached by hydrothermal fluids. Boron minerals were deposited in sedimentary successions in lacustrine conditions through the processes of evaporation and chemical precipitation. The presence of laminated dolomitic rocks and claystone in association with borate mineralisation indicates sedimentation in the deeper parts of a lake.

Ten continuous borate bearing horizons have been recognised to date and the deposit as a whole comprises a series of continuous stratiform and sub-parallel tabular layers of irregular shape which occur between 200 and 500 m below surface which are slightly folded and which dip at approximately 18° to the southwest. The lateral extent of the mineralised bodies varies up to some 950 m with longest dimensions orientated approximately north-south.

The boron bearing minerals found at Piskanja include major colemanite and ulexite with minor hydroboracite, howlite, probertite, pandermite, nobleite, meyerhofferite, inyoite, studenicite, rashite, jarandolite and tincalconite. Most minerals are considered as syn-sedimentary primary minerals. It is thought that howlite, a boron-silicate may occur as primary and diagenetic mineralisation and hydroboracite is considered to have formed during diagenesis.

1.4 Exploration Data

The exploration database provided to SRK by Erin contained information on collar coordinates, downhole deviation data and assay data for 79 drillholes totalling 27,628 meters. Not all of the drillholes in the database contained assay information; four drillholes drilled in the 1980's and 1990's by previous operators, Ibar Mines and Ras Borati, were undertaken for sterilisation purposes at the flanks of the deposit and do not therefore have assay data. As SRK decided not to use the drilling data from the Ibar Mines and Ras Borati campaigns this is not considered material. Based upon a detailed review of the core available from Rio Tinto, who also held the licence for a period, and existing analytical results SRK found that the data from the holes drilled by Rio Tinto to be reliable and appropriate for the inclusion in the geological modelling and resource estimation process. In total, therefore, the database used by SRK to produce the Mineral Resource estimate given here contained data from 53 drillholes, with a total length of 19,554 m,

1.5 Mineral Resource Estimate

The above data has been used to develop a 3D geological model for the deposit which forms the basis of the Mineral Resource Estimate presented here. The declared Mineral Resource has been restricted to material above a marginal cut-off grade of 12% B_2O_3 and a minimum mining height of 1 m so as to constrain the estimate to material which SRK considers has reasonable prospect for eventual economic extraction. This assumes that the mineralisation will be mined by underground methods.

In summary, SRK has estimated the deposit to comprise an Indicated Mineral Resource of 5.6 Mt with a mean grade of 30.8% B_2O_3 and an Inferred Mineral Resource of 6.2 Mt with a mean grade of 28.8% B_2O_3.

Mineral Resources are not Mineral Reserves as they have no demonstrated economic viability. SRK and Erin are not aware of any factors (environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors) that have materially affected the Mineral Resource Estimate.

The quantity and grade of reported Indicated and Inferred Mineral Resources in this estimation are uncertain in nature. There has been insufficient exploration to report these Mineral Resources in the Measured category and it is uncertain if further exploration will result in upgrading a part of these to this category in due course or if further technical work will enable them to reported as Mineral Reserves.

1.6 Preliminary Economic Assessment

The principal aim of this report is to present the results of a PEA completed by SRK to determine the justification for further exploration and technical work at the Project. This has included preliminary mine design work, the development of a mining schedule and conceptual processing flow sheet, initial investigations into infrastructure requirements inclusive of a tailings storage facility and preliminary assessments of the likely environmental and social impacts of the project and the measures needed to be put in place to mitigate these. In summary, this PEA envisages an underground mine feeding a processing circuit producing a colemanite concentrate and boric acid for subsequent export.

Specifically, SRK has developed a discounted cash flow model which has been used to derive a post-tax NPV for the Project at a 10% discount rate of USD428 Million (M) and an IRR of 64% which reduces to USD284M if based solely on Indicated Mineral Resources but increases to USD510M if the NPV is calculated using an 8% discount rate.

It should be noted that this PEA is preliminary in nature, that the NPV of USD428M includes Inferred Mineral Resources that are currently considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves and there is no certainty that the PEA will be realised.

1.7 Interpretation and conclusions

The exploration work undertaken by Erin to date in combination with work undertaken on the Project by previous workers has delineated a significant borate deposit which in SRK's opinion now justifies further exploration and assessment to pre-feasibility study (PFS) level.

The Project is still at an early stage of assessment and much technical work remains to be completed and many risks removed before a decision could be made on putting a mine into production. This report highlights the work the authors consider needed to address these gaps and risks which notably include, further drilling to confirm the continuity and structure of the deposit (which is still uncertain), geotechnical testwork to help develop appropriate mine design parameters, further metallurgical testwork to confirm the potential to produce a saleable colemanite concentrate (which has not yet been demonstrated) as well as additional analysis in most areas to determine the infrastructure and service requirements of the Project, the potential impacts of the Project on the environment and the measures needed to be put in place to mitigate these and also the likely construction costs for the Project (which remains preliminary at this stage).

1.8 Recommendations

SRK has discussed with Erin the work required to be done to advance the project towards the development stage, much of which is highlighted in this report, and based upon this Erin has developed the budget given in Table 1-1 below which culminates in the completion of a PFS by end-2015. The aim of the PFS will be to enable the various options for the development of the Project as outlined in this report to be assessed so that a feasibility study can be focussed on a single mining, processing and infrastructure option and the justification for such a study determined. Further funds though would then need to be raised to complete the feasibility study.

The exploration drilling includes both infill and extension drilling plus specific drilling to assess the presence of faulting; the bulk sampling will be done via wide diameter drilling while the environmental and hydrological work will be commenced in parallel with this. SRK is confident that the work proposed is justified by the potential of the Project and that the budget allowed is reasonable given the work planned and recommends that this work is carried out as planned.

Table 1-1: Planned Expenditure

Item	USD '000
Exploration/Resource Drilling	2,100
Bulk Sampling/Metallurgical Testwork	800
Decline Drilling	450
Environmental Studies	150
Geotechnical Testwork	60
Hydrological and Hydrogeological Analysis	150
PFS Study	550
Office Costs	540
Contingency	300
Total	5,100

2 INTRODUCTION

2.1 Background

SRK Consulting (UK) Ltd (SRK) has been commissioned by Erin Ventures Inc. (Erin, hereinafter also referred to as the Company or the Client) to prepare a Technical Report and Preliminary Economic Assessment (PEA) for the Piskanja Borate Project (Piskanja or the Project) located in Serbia. Balkan Gold doo, which holds the Exploration Licence, is a wholly owned subsidiary of Erin. Erin is currently listed on the Toronto Stock Exchange Venture Exchange (TSX-V) using the code EV.

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The Piskanja Project is considered to be an "Advanced Property" as defined by NI 43-101. The Project is located in southern Serbia, 10 km north of the town of Raška and 160 km south of the capital, Belgrade.

The information reviewed in preparing this report has largely been provided directly by Erin. SRK's opinions and recommendations expressed in this report are effective as of 1st September 2014. Where SRK has drawn upon information from public domain sources, the source of this information is given where relevant. A full reference list can be found at the end of this report.

2.2 Qualifications of Consultants

SRK is part of the larger SRK Group, which includes some 1,500 professional staff providing expertise in a wide range of exploration, mining and engineering disciplines. The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. The SRK Group has a demonstrated track record in undertaking independent assessments of mineral resources and ore reserves, project evaluations and audits, competent person's reports and independent feasibility evaluations on behalf of exploration and mining companies and financial institutions world-wide.

The SRK Group consultants include specialists in the fields of exploration, geology, Mineral Resource/Ore Reserve estimation and classification, open-pit and underground mining, geotechnical engineering, metallurgical processing, hydrogeology and hydrology, tailings management, infrastructure, environmental management and mining economics.

SRK has extensive experience in reviewing, auditing and evaluating exploration programmes at all stages of development. Much of this work is conducted as a part of stock exchange documentation, due diligence studies and project audits, as well as for internal review purposes. SRK has extensive experience in the mining and exploration industry and employs experienced geologists, engineers and scientists who are members in good standing of appropriate professional institutions.

Neither SRK nor any of its employees employed in the preparation of this report has any beneficial interest in the assets of Erin. SRK will be paid a fee for this work in accordance with normal professional consulting practice.

While this report is the result or work undertaken by a number of experts, the Qualified Person (as such term is defined in National Instrument 43-101) and principal author of this technical report is Dr Mike Armitage who is a Member of the Institute of Materials, Minerals and Mining and by virtue of his education, membership of a recognised professional association and relevant work experience a Qualified Person as defined by National Instrument 43-101. Dr Armitage is a full time employee of SRK, with over 30 years' experience in the mining industry.

The MRE work for this technical report was completed by Dr Mikhail Tsypukov and Mr Mark Campodonic. Both Dr Mikhail Tsypukov (who is a Fellow of the Institute of Materials, Minerals and Mining) and Mr Mark Campodonic (MSc. FAusIMM,) are also Qualified Persons as this term is defined by National Instrument 43-101 and take responsibility for the geological interpretation presented here and the resource modelling respectively. Dr. Tsypukov is a full time employee of SRK ES and has over 27 years' experience in the mining industry while Mark Campodonic has been employed full time by SRK since 2003 and has over 14 years of experience in the mining industry. Dr Armitage takes responsibility for all other aspects for the purpose of this report.

2.3 Site Visits

Dr Mikhail Tsypukov, an exploration geologist employed by SRK ES, and Ms Liubov Egorova visited the Project between 12 and 14 June 2013. Previously, Dr. Mikhail Tsypukov had visited the Project from 10 June 2012 to 15 June 2012. The site visit included inspection of drill core, discussion with Erin personnel and assessment of Erin's technical protocols and methodologies. During the site visit discussions were held with the project personnel and the relevant information was collected for the preparation of this technical report and the MRE.

SRK was given full access to relevant data and discussed with Erin personnel any changes in the geological and structural understanding of the deposit, the drill core logging and core sampling procedures and submission of these samples for geochemical assay, and the management and interpretation of assay results returned from laboratories.

More recently, three more members of the SRK team, Max Brown, Louise Bland and Colin Chapman all visited the Project between 22 and 25 April 2014. This visit focussed more on mining, infrastructure and environmental aspects of the Project and involved meetings with Erin personnel and consultants at the project site. Notably the existing infrastructure and utilities supply in the area were observed and discussions were held regarding potential product off-take agreements and corresponding production rates and export scenarios.

2.4 Declaration

SRK's opinion contained herein, and effective 1st September 2014, is based on information collected by SRK throughout the course of its investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

SRK has confirmed that the MRE reported herein is within the licence boundaries given below. SRK has not, however, conducted any legal due diligence on the ownership of the licences themselves.

SRK has not undertaken any detailed investigations into the legal status of the Project nor any potential environmental issues and liabilities that the Project may have at this stage.

SRK is not aware of any other information that would materially impact on the findings and conclusions of the report. SRK was informed by Erin that there are no known litigations potentially affecting the Piskanja Project.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of Erin, and neither SRK nor any affiliate has acted as advisor to Erin, its subsidiaries or its affiliates in connection with this Project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

SRK cannot accept any liability, either direct or consequential for the validity of information that has been accepted in good faith.

3 RELIANCE ON OTHER EXPERTS

The information reviewed in preparing this report has been provided directly by Erin and a compilation of proprietary and publicly available information. SRK has referenced information and data sourced from reports and documents where applicable.

Some of the sources of information and reports used by SRK in the creation of this technical report are authored by persons who are not recognised as independent Qualified Persons as this term is defined by National Instrument 43-101. In this case, SRK has relied upon the professional measures used by the companies who completed the work. The information in those reports is assumed to be accurate based on the data review conducted by the author, but is not NI43-101 compliant. Notwithstanding the above, SRK has reviewed this information, and has included only the information considered appropriate and of suitable quality for inclusion. These reports are as follows:

- 2006, Geosystem srl, Magnetotelluric Survey, Jarandol Basin, Serbia;
- 2012, University of Belgrade, Faculty of Mining and Geology, Testing of samples from the Piskanja borate deposit (translation from Serbian);
- 2012, University of Belgrade, Faculty of Mining and Geology, Petrological characteristics of holes 104, 105, 106, 107, IBM-4 and IBM-6 – Piskanja (in Serbian);
- 2012, SGS Minerals Services), Report on magnetic and HTE testing of borate samples from Serbia;
- 2013, University of Belgrade, Faculty of Mining and Geology, Study of engineering properties rock masses and terrains of the Piskanja borate deposit (translation of concluding remarks from Serbian), and;
- 2013, MWH UK Ltd, Interim Hydrogeological Report (Phase II), Piskanja boron, near Baljevac, Raška, Serbia

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The Project covers an area of 305.7 hectares. The approximate centre of the project area is 43º 22’ 43”N and 20º 38’ 50”E in standard degrees, minutes, seconds format. The Project is located in southern Serbia, some 160 km south of the Serbian capital Belgrade. Nearby towns include: Kraljevo, 40 km to the north; Novi Pazar, 28 km to the south, and; Raška, 11 km to the south, (Figure 4-1).

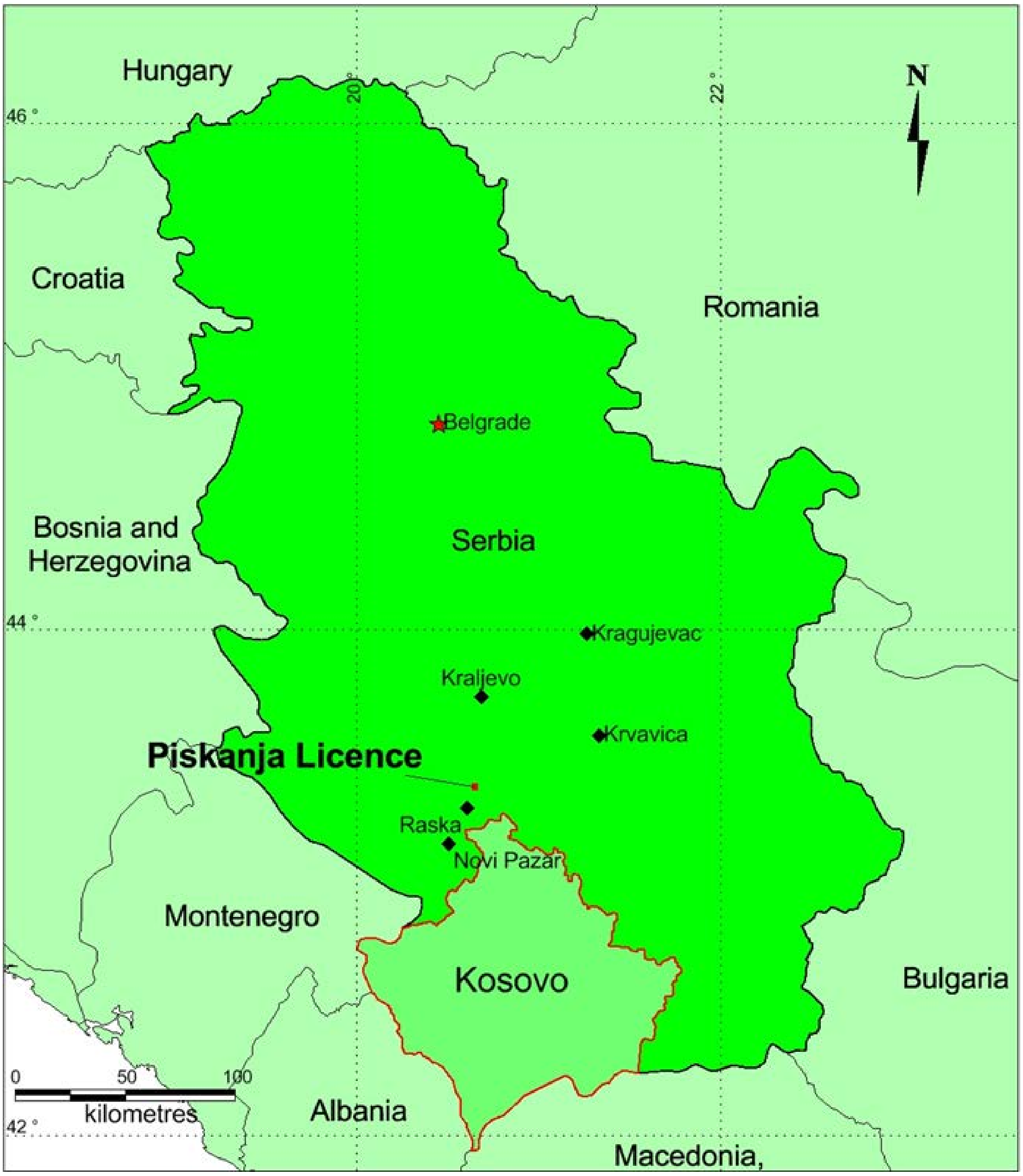


Figure 4-1: Location of the Piskanja licence area

4.2 Mineral Licence Tenure

The Ministry of Mining and Environmental and Spatial Planning of the Republic of Serbia (the Ministry) granted Balkan Gold doo, Exploration Licence #1934 on 23 August 2010 under the 1995 Law on Mining published in Official Gazette of RS, no. 44/95. Balkan Gold doo is a wholly owned subsidiary of Erin. The licence was initially granted for a period of one year but an extension was subsequently granted in August 2011 for an additional year until 23 August 2012.

In July 2012, Erin applied for and was granted a new three year Exploration Licence under the new Law on Mining and Geological Researches, published in Official Gazette no. 88/2011 which came into force in January 2012. The renewed Exploration Licence #1934, was granted on 05 November 2012 and is valid until 05 November 2015. The licensed area is the same as under the previous title (3.075 km^2) and is defined by the coordinates in Table 4-1 and Figure 4-2. The licence covers the Piskanja mineral deposit in its entirety and there are no other known mineral deposits within the licence area.

On 10 December 2012, through Erin's 100% owned subsidiary Balkan Gold doo, Erin was granted Exploration Licence #2065 which covers an area of 35.22 km^2. This Exploration Licence (#2065) is adjacent to the existing Piskanja #1934 Exploration Licence. Exploration Licence #2065 is valid until 10 December 2015 and allows Erin to continue exploration in the Jarandol Basin for boron mineralisation and associated elements (Li, Na, Sr and K), with a view to expand the Piskanja Project to the west. This Technical Report and MRE is only concerned with Exploration Licence #1934 as SRK understands that no exploration work has been undertaken by the Company on Exploration Licence (#2065) since it was granted.

Table 4-1: Licence boundary coordinates for the Piskanja Project, licence #1934, given in UTM WGS84 zone 34N datum and projection.

Point	Easting (X), m	Northing, (Y), m
1	470,574.94	4,803,821.38
2	472,324.40	4,803,821.38
3	472,324.40	4,802,071.91
4	470,574.94	4,802,071.91

Table 4-2: The history of the validity of the Tenement covering the Piskanja Project area.

Licence Number.	Tenement Name	Date Valid from	Date of Expiry	Licence Area
1934	Piskanja	08/12/2010	08/23/2012 (extended from 08/23/2011)	307.5 ha
1934	Piskanja	05/11/2012	05/11/2015	307.5 ha

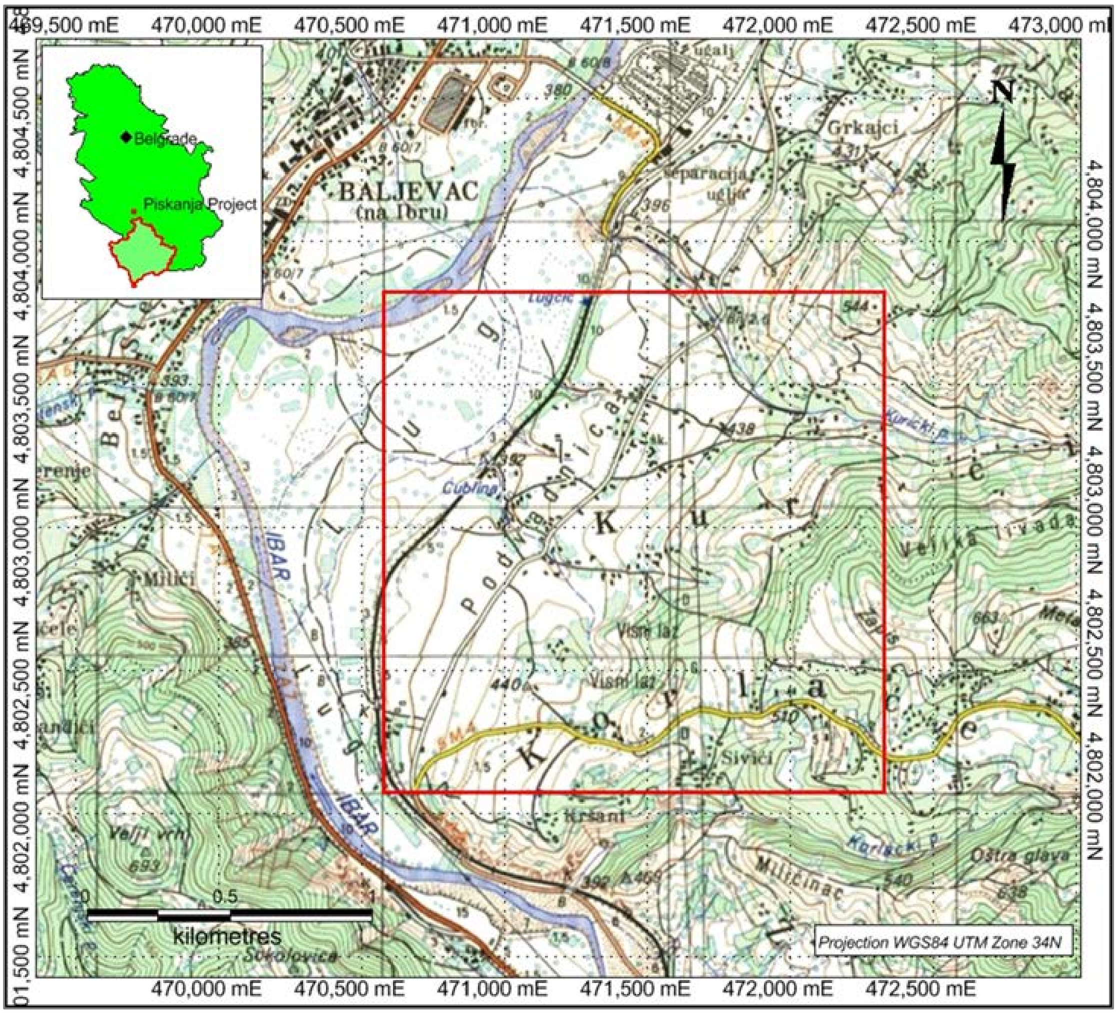


Figure 4-2: Geographical map of the Piskanja Exploration Licence #1934 (red line).

Erin's main responsibilities as licence holder are described in the "Decision of the Ministry of Natural Resources, Mining and Spatial Planning" dated 05 November 2012. It is understood by SRK that this decree states that Erin, through its 100% owned subsidiary Balkan Gold doo, is committed to performing exploration activities initially for three years in accordance with the Exploration Programme submitted by Erin to the Ministry at the time of licence application.

Erin contracted Ibarski Rudnici Coal Company, which is a subsidiary of the State-owned JP PEU Resavica, to design its current (2012-2015) Exploration Programme. In accordance with the 2012 Law on Mining and Geological Researches of Serbia, this Exploration Programme was approved by the Institute for Nature Conservation of Serbia and the Institute for Cultural Heritage and Preservation, Kraljevo, prior to it being submitted to the Ministry. Previously, Erin's 2010 Exploration Programme was designed and submitted by private exploration consultancy, Jantar Group, Belgrade.

SRK understands that the obligations of the 2012-2015 Exploration Programme, which Erin must fulfil according to the Law on Mining and Geological Researches of Serbia, include completion of the following:

- The drilling of 7 holes to validate the pre-1997 exploration (completed by November 2013);
- Hydrogeological and hydrological studies (on-going);

- Mineralogical and petrological studies including SEM-EDS and XRD (completed by November 2013);
- Geotechnical studies of the core (completed by November 2013);
- Preliminary metallurgical testing using SGS and SCL laboratories (2 x25 kg) (completed by November 2013);
- Analytical tests (ICP, titration and XRF) on all new core samples (completed on all holes drilled in 2011/2012);
- Maintenance of a GIS model and data base (on-going);
- Preparation of a Mineral Resource/Ore Reserve estimate report (Serbian report completed);
- The drilling of 14 further holes using a 50m x 50m grid of drill collars (planned for 2014/2015);
- Topographic surveying at 1:1000 scale over the exploitation area (150-200 ha) (planned for 2014/2015); and
- The preparation of annual reports and a final report on geological exploration for the validity of the licence period (on-going).

Any changes in the Exploration Programme are required to be discussed with the Ministry before their fulfilment.

SRK understands that it is required by Serbian Law on Mining and Geological Researches that exploration activities and annual reports submitted to the Ministry must be monitored by a third party company. The following organisations have been responsible for such monitoring of Erin's Exploration Programme, although it should be noted that SRK has not verified the listed reports or third party companies involved:

- Monitoring of the technical programme in 2010 was conducted by Geoprofesional Ltd, based in Belgrade;
- Technical monitoring of the programme and the annual report in 2011 were completed and submitted by Silur doo, based in Kraljevo, and;
- Technical monitoring of the programmes and the annual reports in 2012 and 2013 were completed and submitted by South Danube Metals (a wholly-owned subsidiary of Euromax Resources Ltd), based in Belgrade.

4.3 Mining Rights in Serbia

The laws relating to Mining Rights in Serbia are described in the document titled "Law on Mining and Geological Researches" which is published in "Official Gazette of RS" #88/2011. This document states that an Exploration Licence may be granted for an initial period of three years, and then be extended twice more for a further two years on each renewal. Each extension should be accompanied by a 25% size reduction of the previously granted licence area. Therefore, by the end of the seventh year of ownership, the licence area would cover no more than 56.25% of the originally licenced area. Under Serbian Law, the Exploration Company must submit annual reports of the work completed which evidence that not less than 75% of the planned work has been completed.

In accordance with agreed exploration programme for the Piskanja Project, Erin completed a Mineral Resource Estimate (MRE) using the guidelines as stated by the Serbian Mining Law, in July 2013. Erin began the verification process of the MRE with the Commission for Investigation and Verification of Mineral Resources, however, in March 2014 Erin chose to delay the verification process until its current work phase is completed. This MRE has not been verified by SRK and although it is compliant with Serbian resource and reserve classifications, SRK do not consider that it is compliant with the CIM code. According to Serbian Law, it is necessary to undertake a feasibility study prior to applying for a Mining Licence. Erin plans to complete all necessary steps in order to apply for a Mining Licence as soon as possible following approval by the Ministry of the Serbian MRE.

Article 57 of the Serbian Law on Mining and Geological Researches defines the items that must be addressed and attached to a Mining Licence application as:

1. Proof of paid administrative fee;
2. Situational map in the scale of 1:25,000 or in the appropriate scale with marked borders of the exploitation field, public roads and other facilities located in that area and marked cadastral parcels in the written and/or digital form;
3. Certificate on resources and reserves of mineral raw materials or on geothermal resources, issued based on the performed researches in accordance with the existing regulations on classification of resources and reserves;
4. Feasibility study of exploitation of deposits of mineral raw materials or geothermal resources;
5. Act of the municipal authority in charge of urbanism with regard to harmonization of exploitation with appropriate spatial, i.e. urban plans;
6. Act of the Ministry in charge of environmental protection and the act of the institution in charge of cultural heritage protection;
7. Act of the Ministry in charge of water management, in the event exploitation has effects on the water regime, and;
8. Proof of the ownership or user right, i.e. easement for the terrains designated for surface exploitation of reserves of mineral raw materials. In the event of underground exploitation of reserves of mineral raw materials, when the proof of the ownership or use right, or easement shall be submitted only for the land designated for construction of mining facilities, plants and equipment, and in case of exploitation of resources of mineral raw materials and geothermal resources of importance for the Republic of Serbia, a specific Government act on the establishment of public interest for five-year exploitation period shall be submitted".

4.4 Surface Rights

The Surface Rights over the Piskanja mineral deposit are held by private individuals and by local/state governments. Land access therefore has to negotiated with the individual landowners, for which they are reimbursed according to a payment scheme approved by the State.

At the effective date of this report, Erin does not hold any surface rights in the Project area. Erin has informed SRK that some drill hole collar locations have had to be moved due to private land owners refusing access and while this has not affected the exploration programme to date, the progression to a tighter 50 m drill spacing will likely result in further such issues with landowners.

It is understood by SRK that many local residents are uneducated about the type of minerals being explored for and have concerns regarding the pollution of water supplies.

SRK understands that Erin intends to acquire the surface rights for a portion of municipal building land currently owned by the State-owned Ibarski Rudnici Coal Company for mining operations and construction.

4.5 Permits and Authorisation

Under the Serbian Law, a permit must be obtained from the relevant government department to ensure that known heritage sites are not impacted upon by exploration or mining activities. To satisfy this regulation an assessment was made by the Institute for Cultural Heritage Preservation, Kraljevo on 4 June 2010 and a permit subsequently granted. The permit is valid until 5 November 2015.

Further permits may be required as the project develops and prior to commencing any mining operations.

4.6 Environmental Considerations

Prior to Erin commencing exploration, site conditions were assessed by the Institute for Nature Conservation of Serbia on 08 June 2010 and an Environmental Permit granted as a result. The current Exploration Programme approved by the Ministry on 05 November 2012 was also approved by the Institute for Nature Conservation of Serbia and the Institute for Cultural Heritage Preservation, Kraljevo. No site inspection was required by either institution to approve the Exploration Programme as the licence area had not been altered. A renewal of approval is required by the Institute for Nature Conservation of Serbia if Erin continues with exploration beyond 2015. A renewal of approval is required every year by the Institute for Cultural Heritage Preservation, Kraljevo and is therefore needed in 2015.

4.7 Agreements and Royalties

Article 136 of the Law on Mining and Geological Researches states that entities undertaking mining activities shall pay a fee for the use of the mineral deposit. The Law states that “this revenue shall be the amount gained by the exploiting entity from used or natural mineral raw materials, determined on the basis of income gained from sale of non-refined mineral raw material, or income gained from the sale of technologically refined mineral raw material”. The fee will be split between the Republic of Serbia, the local government and the Ministry of Natural Resources, Mining and Spatial Planning

As the law does not stipulate the commodities classified under metallic and non-metallic minerals, it is not known whether the Ministry will impose a royalty on borates similar to that of other evaporite minerals such as gypsum (salt), or select to impose a levy specific to borate. Erin expects confirmation of the royalty status of borates towards the end of 2014. Table 4-3 shows some of the Royalties due on certain commodities under Serbian Law. For the purposes of the economic analysis presented in this report, SRK has assumed that a 5% royalty rate is applicable to the Project for the duration of the assumed life of mine.

Table 4-3: Royalties due on various extracted minerals (from Law on Mining and Geological Researches, 2012)

Commodity	Fee/Royalty
All types of coal and oil shale	3% of income
All metallic raw materials	5% of smelting plant net income
Technogenic raw materials resulting from exploitation and refining of mineral raw materials	1% of income
Non-metallic raw materials	5% of income
All types of salts and salty solutions	1% of income

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Project is located in the Jarandol Basin in the Raška region of south central Serbia approximately 160 km south of the Serbian capital Belgrade, and approximately 17 km north of the Kosovo border. The nearest settlement is the town of Baljevac na Ibru (literally "Baljevac on the Ibar River") which is some 1.7 km northwest from the centre of the Exploration Licence #1934. Baljevac na Ibru (Baljevac) has a population of 1,482 (2011 census), (refer back to Figure 4-2 for location). The regional capital, Raška, lies 10 km to the south of the Project area and has a population of 6,500 (2011 census).

Access to the Project is by paved road from Belgrade, a journey that takes approximately 4 hours and passes through the towns of Kragujevac and Kraljevo. Access around site is by vehicle/foot as the terrain is not steep and the land cleared for agriculture.

A standard gauge railway accommodating passenger and freight rolling stock passes through the western part of the licence area and runs from Belgrade, through the towns of Kraljevo and Raška to Poije in Kosovo.

5.2 Local Resources and Infrastructure

Erin has an office in the town of Baljevac, located at the Ibarski Rudnici coal mine, a small-scale operation exploited only for local coal supply. Facilities here also include core logging and sampling areas, and core and sample storage.

Water for exploration needs is sourced from streams that flow into the Ibar River and the water table is encountered in drillholes at shallow depths, for example at 23m below surface in hole EVP2012-100. It is reported that the stream water is used by the local people for drinking water. A 35 kV electricity lines run across the licence and supply power to Baljevac.

There is good mobile phone reception throughout the Jarandol Basin and the Project area.

5.3 Climate

The climate in the Project area is typical of Eastern Europe with four seasons of approximately equal length; spring, summer, autumn and winter. According to Foreca (an international company providing digital weather forecast data), the temperatures range from between 11 ºC and 28 ºC during the summer months of June to September, to between -3 ºC and 5 ºC during the middle winter months, December and January. Rainfall is highest in the months of May to September with the monthly average of between 49 and 62 mm. The months of January and February have the minimum amount of precipitation (about 30 mm), falling mostly as snow. Exploration activities can continue throughout the year with minimal inconvenience during the winter months.

5.4 Physiography

The Jarandol Basin lies at an elevation of between 375 and 400 m above mean sea level (amsl), is elongated in an east-northeast - west-southwest direction and drains towards the north via the Ibar River which lies just outside the western boundary of the licence area. The terrain rises to approximately 750 m amsl to the west of the valley, outside of the Exploration Licence area, and to over 1,200 m immediately east of the Project area.

Minor tributaries to the Ibar River extend through the Exploration Licence area; the Kuricki to the North of the deposit and the Korlacki to the South. Between them is the Radic, an ephemeral water feature which is dry for most of the year.

The flood plains in the central part of the basin and the low angled valley sides are cultivated for crops and fruit, with the steeper terrain above 500 amsl generally covered by sparse deciduous woodland, (Figure 5-1).

Access to the Project through Surface Rights is covered in Section 4.4. The Surface Rights over the Piskanja site are held by private individuals and or local/state governments. Land access has to therefore be negotiated with the individual landowners, for which they are reimbursed according to a payment scheme approved by the State. As stated in Section 4.4, Erin does not currently have any Surface Rights in the Project area and some private land owners have refused Erin access for drilling. This has not affected the exploration programme to date although further infill drilling programmes will require access to land where the owners have previously refused Erin access. SRK understand that Erin intend to acquire the surface rights for a piece of industrial land currently owned by the State-owned Ibarski Rudnici Coal Company. Erin, intend to use this ground for mining operations and construction in the future.

Figure 5-1: Examples of the terrain and agricultural land use typical of the licence area. Top - Before drilling hole EVP2011-103 (left) and after drilling and remediation (right). Bottom - before drilling hole EVP2011-105 (left) and after drilling and remediation. All photos are taken facing approximately north.

6 HISTORY

Serbia's mining history dates back to the Middle Ages with the extraction of gold, silver and lead. The mining industry in Serbia represents the country's industrial base as well as the foundation for its entire economy. At present there are many mineral deposits and major occurrences distributed throughout the country, with copper, lead, zinc and bauxite contributing to the majority of metallic minerals currently being mined. Serbia also has a rich history of coal mining and lignite coal fed power stations currently provide 62% of the country's electrical requirements.

The first record of boron mineralisation in the Jarandol Basin relates to a hand-sized sample containing howlite found in a tributary of the Ibar river in 1967 during State-organised geological prospecting, (Stojanovich, 1967). Following this, geological mapping at a scale of 1:10,000 was performed and the Pobrdje occurrence was identified some 2.6 km northwest of the present Erin licence.

The geochemical investigation of boron in the Jarandol Basin began in 1979 with the first identification of colemanite in a structural borehole (no. 127) occurring later in 1987. Between 1987 and 1992, the Yugoslavian state-owned company Ibar Mines completed a number of soil and stream sediment sampling programmes, followed by 21 diamond core holes totalling 6,508 m of drilling to an average hole depth of 300 m. Total core recovery was reportedly very good (90-100%) in shale, marl, sandstone and tuff horizons, but less so (60-75%) in volcanic breccia, breccia-conglomerate, conglomerate and borate mineralisation. A total of 89 core samples averaging 1 m in length were collected from 11 boreholes which intersected mineralisation and were analysed for boron. The core is no longer available, however, pulp rejects, assay results and some lithological columns are available (Ilic and Eric, 2009, Podunavac and Vukicevic, 2011). Mineralisation was identified in two horizons with an average thickness of 4.5 m for the upper bed and 3.5 m for the lower bed, lying between 50 m and 260 m depth.

Erin first obtained the Project in 1997 as part of a 50% Joint venture with Elektroprevreda doo (Serbia). The JV company, known as Ras Borati doo, completed 10 reverse circulation (RC) holes, totalling 2,304 m. These holes were drilled by subcontractor Midnight Sun Drilling Co. Ltd, Canada, using a T685H Schramm drilling rig. A total of 206 chip samples were collected from 8 RC holes. The samples were prepared and analysed at the Geozavod-Nemetali laboratory in Belgrade using wet chemistry analysis.

Following the resolution of international conflicts and a change of the governing party in 2006, Rio Tinto acquired the Piskanja Project as part of its regional investigation of borate potential in Tertiary basins across the Balkan region. An initial phase of diamond core drilling in 2006 to twin existing holes was followed by the completion of further diamond holes on a wide spacing aimed at targeting a mineralised body of world-class size. A total of 6,074 m of drilling was completed by Rio Tinto and 817 samples prepared at the ITMNS laboratory in Belgrade and assayed by SGS in Lakefield, Canada, using potassium fusion ICP-AES as the primary method for determination of boron content.

Mineralogical investigations included 69 X-ray Diffraction (XRD) tests, petrographic determinations and a number of Scanning Electron Microprobe (SEM) analyses conducted by SGS, the Department of Mineralogy and Petrology, University of Belgrade (Serbia) and Spectrum Petrographics Inc of Vancouver, Canada. The main boron-bearing minerals in the Project were identified as colemanite (CaB3O4(OH)3•(H2O)) and ulexite (NaCaB5O6(OH)6•5(H2O)) with minor howlite (Ca2B5SiO9(OH)5) and probertite (NaCaB5O9.5H2O).

Rio Tinto also completed a magnetotelluric (MT) survey to assess the conductivity variation within the Jarandol Basin and to map the extent and thickness of the fine-grained sedimentary sequence. A low resistivity zone representing hydrous mineralisation was expected to be encountered. The results of this survey, completed by Geosystem srl in 2006, however were inconclusive with respect to identifying conductivity variation in the shallow (<500m) below surface that might be related to borate mineralisation within the Jarandol Basin sediments. No known historical resource estimates were completed by Rio Tinto before the licence was returned to the Serbian Ministry of Mining and Energy in 2009.

Erin reacquired the exploration licence for the Project in August 2010 through its wholly owned subsidiary company, Balkan Gold doo. This new licence covered an area of historic exploration southeast of Baljevac where historic drilling had identified borate mineralisation.

This licence has been renewed and extended twice, as detailed in Section 4.2. All exploration activities undertaken by Erin since 2010 are detailed in Section 9 onwards.

According to information in a report by Podunavac and Vukicevic (2011) the Mining Institute of Belgrade prepared a mineral resource estimation report for Piskanja in 1992. This appears to have been an unofficial resource estimate as defined by Serbian law, created using interpolation of cross sections between 200x300 m spaced and 100x100 m spaced drill holes. It does not appear to have been included in the State mineral balance. The mineral resource was said to be approximately 6.5 Mt of boric oxide (B_2O_3) in the C1+C2 categories as defined by Serbian mining regulations, though no records of grades have been located. SRK has not verified these numbers and does not know what data have been used to arrive at this estimate.

In the technical documentation related to the Public Tender of the Piskanja Project (Public Tender, 2005) the Ministry of Mining and Energy of the Government of the Republic of Serbia stated that "potential reserves of boron ore in Piskanja deposit are estimated to be 7,500,000 tonnes with an average grade of 36.39% B_2O_3".

Although this figure may be compliant with Serbian resource and reserve classifications, no economic parameters were used in the assessment and SRK does not consider this estimate to be compliant with CIM Guidelines.

SRK understands that there has been no production of borate from the property in the past.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

The geology of Serbia is controlled by a complex tectonic history represented principally in the South Eastern European and Alpine Orogenic Belts that can be divided into five geotectonic units as seen in Figure 7-1 and summarised below:

1. The Pannonian Basin – this basin occupies the northern portion of Serbia and extends across much of Central Europe. It is understood to be the largest extensional basin in the Central Alps that formed in a back arc setting during the Cenozoic era (66 Ma to present). The typical sedimentary succession of the southern Pannonian Basin consists of continental, alluvial and lacustrine sediments which unconformably overlie a strongly tectonised basement.
2. During the Neogene period (23 Ma to 2.5 Ma) of the Cenozoic era, numerous intra-mountain basins developed within the Balkan peninsular. As a result, northeast to east striking grabens formed from back arc extension of the European plate creating the Pannonian Basin (Tolijic et al., 2012, and Marović et al., 2002). It was in this later tectonic extensional period that the Jarandol Basin which hosts the mineralisation at Piskanja was formed.

3. The Dinaric Alps – the Dinaric Alps occupy the western part of Central Serbia, trending almost northwest-southeast. These are composed of Mesozoic (some 230 Ma to 66 Ma) sediments, the most significant of which are thick deposits of karstic Triassic (about 250 Ma to 200 Ma) limestones and dolomites together with Jurassic (approximately 200 Ma to 145 Ma) ophiolitic melange and Cretaceous (some 145 Ma to 66 Ma) flysch deposits (sequences of shales rhythmically interbedded with fine sandstones deposited in deep marine environments).

4. The Vardar Zone - this zone lies east of the Dinaric Alps, continuing into central Macedonia. It consists of three ultrabasic blocks separated by fractured ophiolites that characterise Early Mesozoic (Triassic-Jurassic) ophiolitic paleo-rifts. The Western Vardar ophiolitic unit represents a suture zone between the continental Adriatic plate (Dinarides of the Western Serbia) and the European plate (Carpatho-Balkanides and Macedonian Massif of Eastern Serbia).

5. The Jarandol Basin (Piskanja Project) is found within an Upper Jurassic ophiolite formation in the Vardar Zone. The ophiolite formation is composed of ultra-mafic rocks which include serpentinites, serpentinised dunites, harzburgites and Iherzoilite as well as minor fragments of layered gabbro-peridotite complexes. Also included in the assemblage is a mélange formation composed of an upper ophiolite unit with some sedimentary chert, carbonate and sandstone units.

6. The Serbo-Macedonian Massif – this massif trends north-south and generally comprises Tertiary (approximately 66 Ma to 2.5 Ma) volcano-plutonic complexes.

7. The Carpatho-Balkan Arc - covering Eastern Serbia is an extension of the Carpathian Range that joins the western parts of the Balkan Mountains. The Carpatho-Balkanides were formed in the Mesozoic as a carbonate platform and separated from the Dinaric Alps by the Serbo-Macedonian Massif.

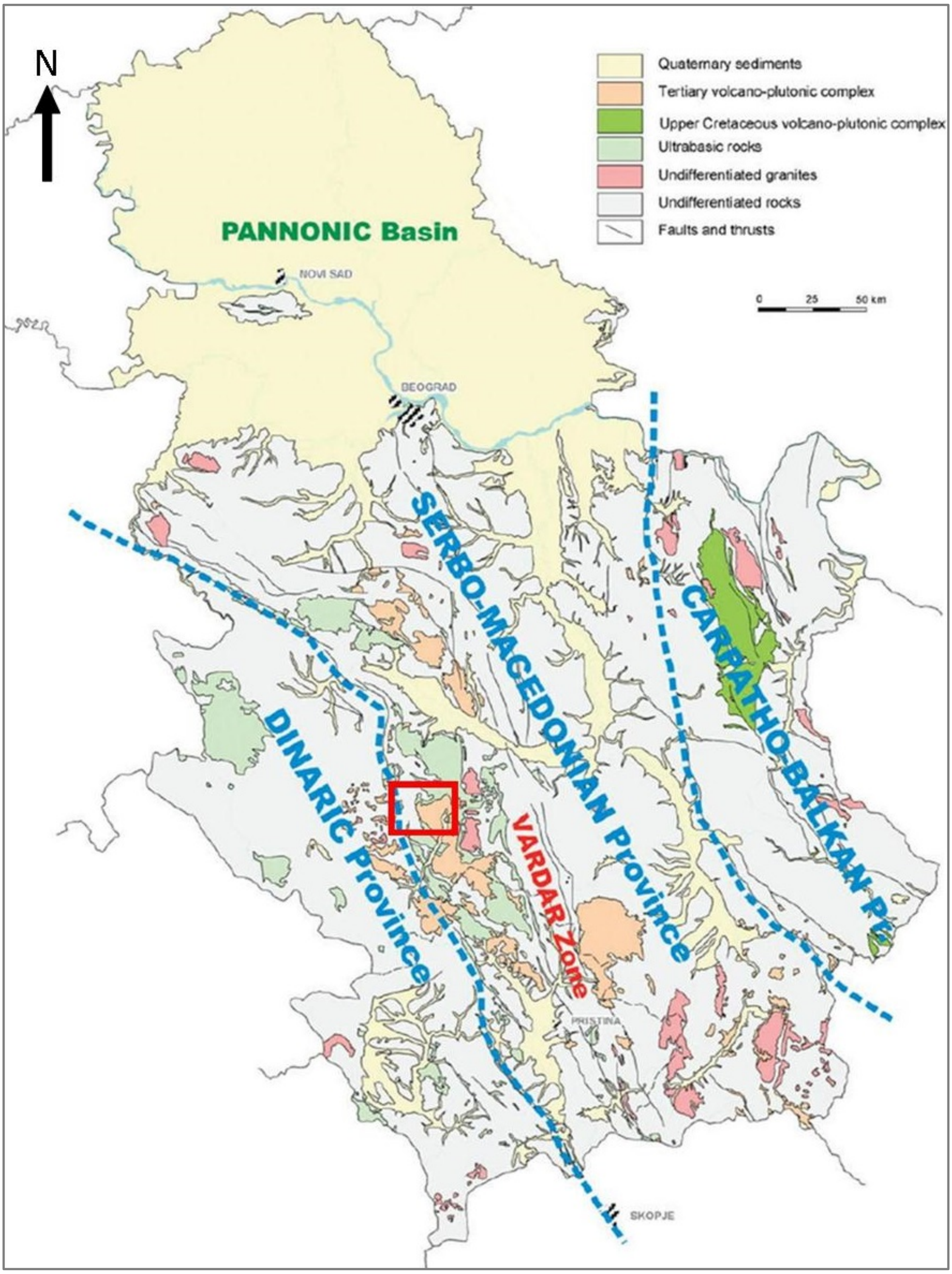


Figure 7-1: Simplified geological map of Serbia (Republic of Serbia, Ministry of Mining and Energy) Location of Piskanja Project in red

7.2 Local Geology

Piskanja is situated within the Jarandol Basin, which formed the eastern part of the larger Gradac-Baljevac graben and is part of the Varder Zone, see section 7.1. The basins associated with the Gradac-Baljevac graben formed during rifting in the Miocene (23Ma to 5 Ma) which affected the ophiolite basement of Upper Jurassic age (Figure 7-2). Prior to the formation of the Gradac-Baljevac graben and the Jarandol Basin intense magmatism in the Oligocene (between 34 Ma and 23 Ma) introduced andesitic, dacitic volcanic and pyroclastic flows which extended over the Ibar and Raška River valleys for some 40 km. A number of granite stocks were also intruded some 5-10 km west of the licence area during this period.

During the Miocene, the basins in the Vardar Zone were filled by sediments associated with various facies typical of continental basins including alluvial, lacustrine and swamp settings and the transitional environments associated with them. Fluctuations in water level and sediment input gave rise to alternating units of mudstone, shale, sandstone and lignite seams, tuffaceous material was also deposited related to on-going volcanism related to extension. This rift-related volcanism was accompanied by hydrothermal activity beneath the basins, circulating fluids through the basement and sediments. It was this high thermal and tectonic activity that led to borate mobilisation and deposition within the Jarandol Basin sediments. The age of sediments within the Jarandol Basin has been determined based on sparse fossils as Lower and Middle Miocene, however, Upper Miocene sedimentation is also possible (Urošević et al., 1970).

The Jarandol Basin extends for some 25 km in an east-west direction with a width of up to 12.5 km, covering an area more than 200 km^2. It was originally much larger, but has since been reduced due to uplift and erosion of sediments shortly after deposition in the northern and southern parts of the basin.

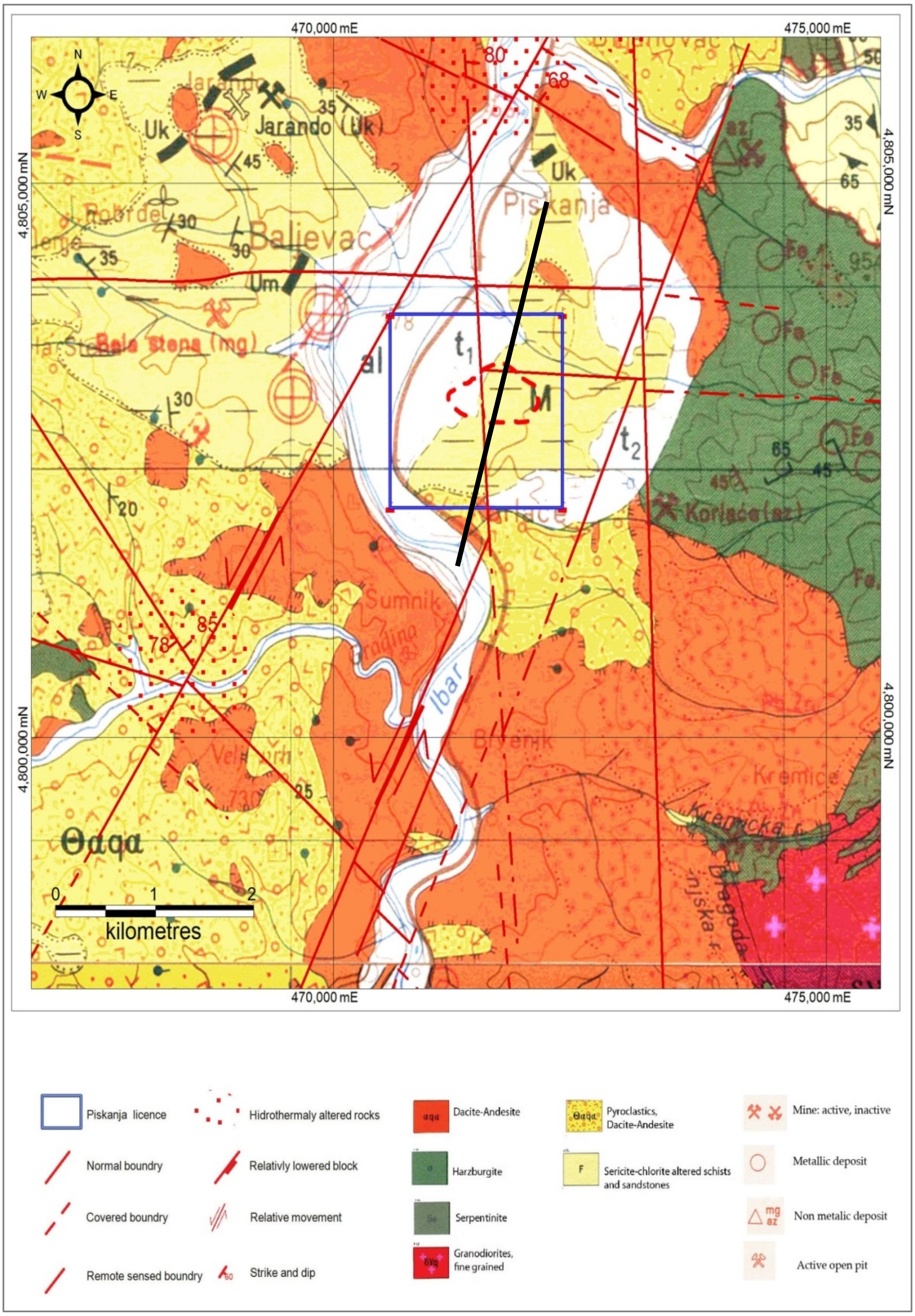


Figure 7-2: Geological map of the Jarandol Basin and location of the Erin exploration licence (blue square). Black line indicates location of cross section in Figure 7-3 (Edited from the Federal Geological Institute, Belgrade, 1970)

7.3 Licence Geology

Exploration Licence #1934 covers an area of some 305.7 hectares in the eastern part of the Jarandol Basin on the eastern bank of the Ibar River (Figure 7-2). There are three principal basin-fill sedimentary units recognised in the Project area that reach a combined thickness of almost 560 m in places. These units are described below as TcP_1 to TcP_3 and their general distribution within the basin is shown schematically in the approximately north-south section in Figure 7-3. Erin has completed 1:5,000 scale geological mapping over the Piskanja Project as shown in Figure 7-4.

TcP_1 The Lower conglomerate and sandstone unit is characterised by a dominance of coarse clastic sediments with a few thin interlayers of carbonate rocks. The thickness of individual layers of sedimentary breccias and conglomerates typically vary from 0.1 m to 10 m but can reach 25 m in the upper part of the unit. This thick conglomerate unit is used as a marker horizon in the basin stratigraphy. The thickness of sandstone horizons in this unit varies between 0.3 m and 0.5 m. There is a general tendency for grain size to fine upwards in the sandstone beds. Clasts and pebbles are represented by quartz, feldspar and rock fragments including andesite, schist, monzodiorite, granite, peridotite and serpentinite. The thickness of such (clasts and pebble beds) varies from 90 m to more than 130 m.

TcP_2 The claystone and carbonate unit is characterised by thin millimetre-scale laminations of claystone, silty claystone, tuff, travertine, dolomite, dolomitic limestone with claystone and rarely sandstone, breccia and conglomerate. The thickness of this unit can be up to 330 m, increasing in areas of greater basin depth. Horizons of borate mineralisation are associated with the carbonate sediments. These horizons are concordant with bedding and have a transitional zone between the high grade mineralisation and host rock, generally less than 1 m thick and characterised by inclusions, veins and impregnations of borate minerals in the dolomitic rocks.

TcP_3 The upper claystone and sandstone unit is characterised by interbedding of sandstone and claystone horizons between 1.0 m and 1.6m thick and dolomitic rocks between 2 m and 10 m thick. Beds of claystone and sandstone have no lamination and generally possess a massive texture. The TcP_3 unit as a whole varies from between 20m to more than 90 m in thickness.

Quaternary sediments cover some 65% of the Project licence area and are represented by delluvial-colluvial and alluvial sediments including rounded and semi-rounded pebbles and boulders mixed with fine and coarse sand. These outcrop in the peripheral parts of the licence. The thickness of the quaternary cover is up to 25 m, with greatest thickness covering the central and western parts of the licence area. The rest of the licence area is covered by Lower and Middle Miocene sediments.

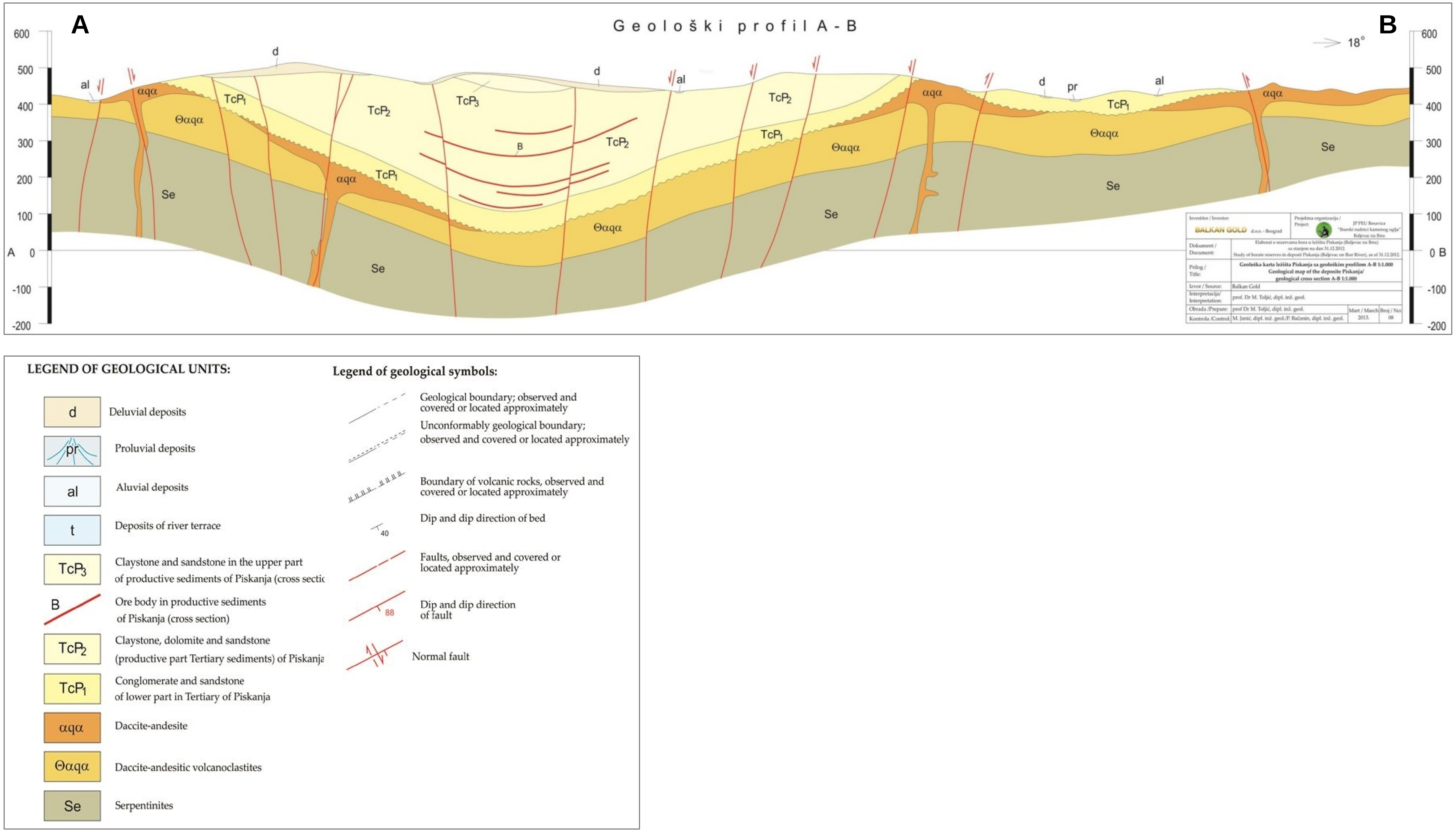


Figure 7-3: Schematic cross section of the Jarandol Basin (Erin, 2013). See Figure 7-2 and Figure 7-4 for section location.

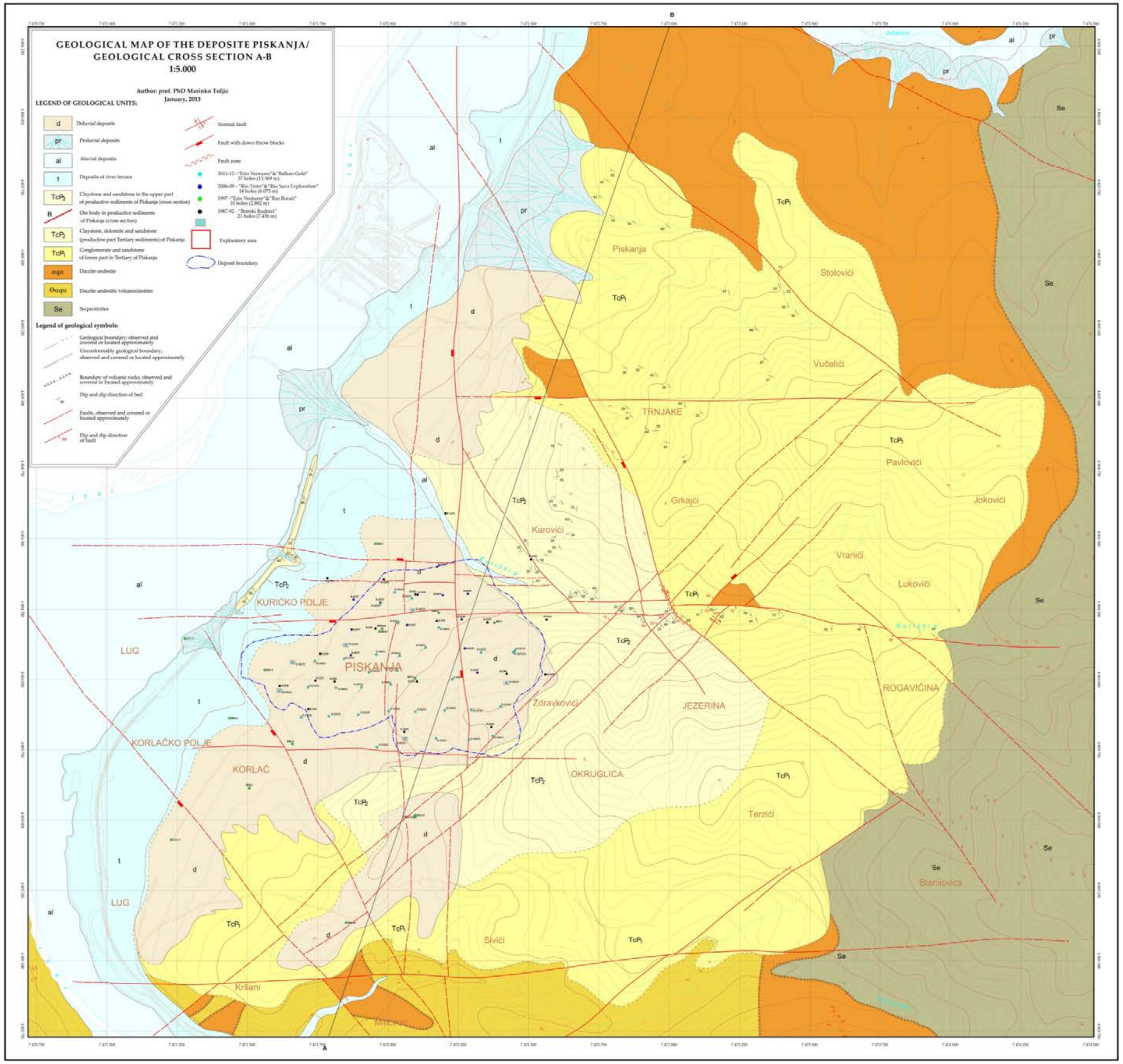


Figure 7-4: 1:5,000 Geological Map of the Piskanja Project, (Erin Ventures 2013)

7.3.1 Structure

A series of northwest-southeast and north-south striking local and regional faults as well as intrabasinal syndepositional microfaults are clearly visible in outcrops and in drill core.

Faults are accompanied by breccia and clay shear zones, steep to vertical bedding of the host sediments, microfolds and slumping. Figure 7-5 demonstrates faulting in outcrop on the slopes of the Ibar valley in andesite with intense shearing and clay alteration. Numerous hand-scale local tectonic structures including; convolute bedding (slumping), broken layers, clastic dykes, higher content of coarse sediments above the syndepositional faults and very small veins of re-deposited borate mineralisation are seen in the core resulting from syndepositional tectonics due to stress or extension conditions and solution metamorphism, Figure 7-6 and Figure 7-7..

Erin's structural analysis of the core and field observations suggest that a number of regional and local-scale faults cross the basin but have only a limited amplitude of vertical movement, and indeed the relatively small variation in the elevations of mineralisation between the holes may be simply explained mainly by synclinal folding and a southward dip of the beds in the basin.

Figure 7-5: Hydrothermally altered Oligocene andesite of the Piskanja basin. Clay alteration is developed along the fault zones, road #22 north (left) and south (right) of the village of Baljevac. (photos provided by Erin).

Figure 7-6: Steep angled laminations related to slumping in boreholes EVP-2011-109 and EVP-2011-110

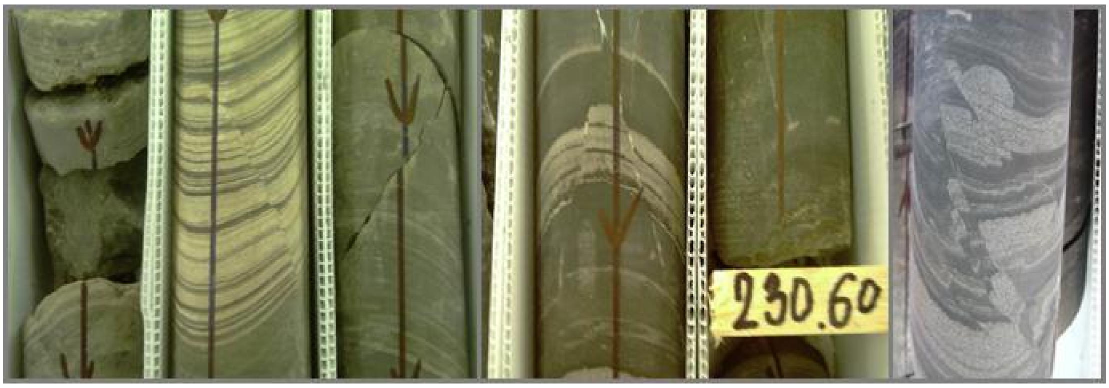


Figure 7-7: Example of syndepositional extension and stress in sandstone and mudstone within core from the Piskanja Project

7.4 Mineralisation

There are a variety of minerals which are found in borate deposits, although grade is always quoted in percent B_2O_3 terms (Table 7-1). According to mineralogical studies conducted by Erin, the boron bearing minerals found at Piskanja include major colemanite and ulexite with minor hydroboracite, howlite, probertite, pandermite, nobleite, meyerhofferite, inyoite, studenicite, rashite, jarandolite and tincalconite. Most minerals are considered as syn-sedimentary primary minerals. It is thought that howlite, a boron-silicate may occur as primary and diagenetic mineralisation and hydroboracite is considered to have formed during diagenesis.

Table 7-1: Composition of the main borate minerals found at Piskanja in order of decreasing abundance, (www.webmineral.com and www.mindat.org)

Mineral	Formula	B_2O_3 content, %	Density, g/cm^3
Colemanite	$Ca_2B_6O_{11}\cdot5H_2O$	50.8	2.42
Ulexite	$NaCaB_5O_9\cdot8H_2O$	43.0	1.95
Hydroboracite	$CaMgB_6O_{11}\cdot11H_2O$	52.5	2.00
Probertite	$NaCaB_5O_9\cdot5H_2O$	49.6	2.14
Jarandolite	$CaB_3O_4(OH)_3$	55.6	2.54
Studenitsite	$NaCa_2[B_9O_{14}(OH)_4]\cdot2H_2O$.	59.3	2.31
Howlite	$Ca_4Si_2B_{10}O_{23}\cdot5H_2O$	44.5	2.58
Veatchite	$Sr_2B_{11}O_{16}(OH)_5\cdot H_2O$	58.6	2.62

Ten continuous borate horizons have been recognised to date, including three major horizons (Zones 1-3) and seven minor horizons (Zones 4-10). The mineralised bodies are continuous stratiform and semi-parallel layers of irregular shape which are slightly folded and dip at approximately 18° to the southwest. The lateral extent of the mineralised bodies varies from approximately 50 m by 360 m (Zone 9) to 415m by 950m (Zone 2), with longest dimensions orientated approximately north-south. Variations of the thickness and averaged grades of B_2O_3 are presented in Table 7-2.

The depth of mineralisation at Piskanja varies significantly depending on the stratigraphic position of the mineralised horizon and location within the basin. The shallowest mineralisation was intersected in the northeast part of the basin, whereas the deepest horizons are intersected in the southwest part of the basin. Zone 1, for example, varies in depth from 234 m to 447 m over a lateral distance of some 880m moving from northeast to southwest.

Table 7-2: Thickness and weighted average intercept B_2O_3 grades of the mineralised bodies found within the Piskanja deposit, as defined in the drilling database (from top to bottom).

ZONE	Thickness, m			Grade B_2O_3, %		
	Min	Max	Average	Min	Max	Average
1	0.5	9.3	4.8	15	50.2	32.9
2	1	15.0	3.4	2.8	55.1	31.4
3	0.4	22.3	5.1	0.8	48.1	32.4
4	0.5	15.1	4.3	31	50.4	38.5
5	0.3	1.8	1.0	6.4	41.5	28.3
6	0.4	15.6	2.5	5.7	45.1	16.1
7	1.1	8.7	3.7	7.3	28.8	18.6
8	0.6	2.3	1.6	9.8	32.1	18.4
9	0.6	2.7	1.6	17.7	53.1	32
10	0.5	3.0	1.6	10.7	23.9	16.4
TOTAL	**0.3**	**22.3**	**2.9**	**0.8**	**55.1**	**30.4**

*- Including up to 1.5m thick intercalations of less mineralised rocks

The borate mineralisation forms clearly visible white-cream colour horizons which contrast with the hangingwall and footwall laminated claystone and carbonate lithologies of formation TcP2. Moving away from the mineralised borate horizons, the amount of dolomitic rock usually decreases and claystone increases. The following succession is common in the drill core seen at Piskanja whereby following a seam of borate mineralisation, dolomitic rock, dolomite with thin intercollations of claystone, claystone with thin dolomite laminas and finally claystone can be observed. Contacts between the borate mineralisation and claystone are rarely observed. Carbonate rocks below and above the borate seams may also contain thin impregnation, lenses, spherical nodules or aggregates of borate mineralisation which do not have economic importance (usually <1% B_2O_3 over 1 m intervals).

Borate mineralisation in the central part of the deposit is seen to be massive and homogenous with gradually increasing numbers of intercalations and interlayers of claystone and dolomitic rocks towards the periphery of the mineralised area. Figure 7-8 and 7-9 show examples of the types of mineralisation found at Piskanja.

The presence of laminated dolomitic rocks and claystone in association with borate mineralisation indicates its formation in the deeper part of a lacustrine environment. The main borate horizons and surrounding sediments contain digenetic howlite and hydroboracite, cavities and veins of borate mineralisation indicating solution metamorphism, however is it considered that dissolution or digenetic re-distribution of mineralisation did not play significant role.

According to multi-element geochemical assay results from the Erin exploration programme, borate mineralisation contains elevated strontium (up to 1.0% Sr), which may be due to the presence of veatchite or celestine, or incorporation of Sr into other borate minerals such as colemanite, which can contain 0.36-1.51% Sr (Garrett, 1998).

Figure 7-8: Massive borate mineralisation in hole EVP2012-111 from 310.30 m to 313.20 m. Mineralisation comprises colemanite and ulexite (grey) and howlite (white) at the contact between shale and dolomite units, (Technical Report, 2012)

Figure 7-9: Interbedding of borate mineralisation (grey and white) with laminated dolomitic and shale lithologies. Hole EVP2012-106 from 291.90 m to 294.90 m depth, (Technical Report, 2012)

8 DEPOSIT TYPE

The Piskanja borate deposit is of continental lacustrine type, typical of many global boron deposits, and is considered to have formed within a closed basin with abnormally high salinity. The boron mineralisation is most likely to have been sourced from local volcanic rocks, from which it has been leached by hydrothermal fluids. Boron minerals were deposited in sedimentary successions in lacustrine conditions through the processes of evaporation and chemical precipitation. The presence of laminated dolomitic rocks and claystone in association with borate mineralisation indicates sedimentation in the deeper parts of a lake.

Most borate minerals are highly soluble in water which restricts the areas in which they form, and more importantly, are preserved. The majority of known global borate deposits have formed in lacustrine or playa lake environments in closed basins that opened up in active extensional setting near subductive plate boundaries. Rock types associated with the deposits generally include calc-alkaline extrusive rocks, tuff, limestone, marl, claystone, gypsum, continental silts and sands. The source of boron is not always the same and can be derived variously from leached marine sediments, magmatic fluids from subducted crust or from volcanic material (tuff).

The boron deposits in the USA and Turkey (which together account for some 80% of world production), are associated with continental sediments and show a continuum between hydrothermal spring, playa lake and lake deposits. Borate minerals precipitate once they become saturated in the fluids circulating these basins, either through evaporation of the basinal waters or addition of borate rich fluids from hydrothermal springs and circulating meteoric waters. Different borate minerals form at different levels of acidity; for example, borax (sodium borate) precipitates at a higher pH than ulexite, and in comparison colemanite forms at a lower pH and in warmer fluids. Due to cycles of basin refill and sediment input, there may be numerous layers of borate mineralisation interbedded with barren sedimentary horizons.

Borate deposits, due to their process of formation, are generally found as stratiform layers within basins, typically of Tertiary (Neogene) age and proximal to areas of volcanic activity of a similar age. Deposits showing these characteristics have already been identified and exploited in western Turkey at Kirka, Bigadiç and Kestelek among others. The origin of the borates within these deposits is related to mixing of borate-rich solutions within lacustrine basins controlled by evaporation (Helvaci and Alonso, 2000).

The Turkish deposits of Kirka, Bigadiç and Kestelek are owned and operated by Eti Maden. According to Eti Maden's website, (http://en.etimaden.gov.tr/) the Kirka deposit reportedly produces some 2.5 million tonnes per annum (tpa) of sodium borate ore at a mean grade of 26% B_2O_3. The Bigadiç deposit is reported to produce some 800,000 tpa of ulexite and colemanite ore at between 29% and 31% B_2O_3. The Kestelek deposit produces some 200,000 tpa of colemanite ore with a mean grade of 29% B_2O_3 from an open pit.

The Rio Tinto owned Jadar project in northwest Serbia is a unique lithium borate deposit at a prefeasibility stage with an Inferred Mineral Resource of 118Mt containing 1.8% LiO_2 and 16.2Mt B_2O_3 in the lower of three mineralised zones, (Rio Tinto 2012 Annual Report). Finally, Pan Global Resources has a number of joint venture properties in central Serbia at an early stage of exploration which it is exploring for deposits analogous to the Jadar deposit.

To assist with further exploration, Erin is using the deposit model of syn and post depositional borate mineralisation in discrete horizons within well stratified lacustrine sediments to guide its exploration which SRK considers appropriate.

9 EXPLORATION

Apart from drilling, the exploration undertaken by Erin since 2010 has comprised the collection and analysis of all available historical data relating to the Jarandol Basin and its lithologies, tectonic structures and mineralisation. An assessment of the quality of this data and its reliability was also completed internally to determine the suitability of the data for use in further studies and to form the basis of a MRE. This included the review of the results from historical drilling and geophysical investigations conducted by Rio Tinto as mentioned previously in Section 6. From this work it was determined by the geophysical contractor, Geosystem Srl, that the magnetotelluric geophysical data was not of sufficient detail or resolution at depths of <500m to provide insight on borate mineralisation or sedimentary sequences.

Publically available documents have also been considered by Erin, including scientific publications regarding the regional geological setting and evolution of the Miocene Jarandol Basin and the analysis of aerial photographs.

All available historical drill core has been re-logged by Erin, with particular focus on lithologies that might be identified as "marker horizons" that could be used to correlate the position of mineralisation across holes. This included the creation of historical drilling database containing all available data from multiple historic drilling programmes.

The Piskanja Exploration Licence and surrounding area was geologically mapped at a scale of 1:5,000 by Erin in 2012 (Figure 7-4). This indicated the presence of a number of normal faults which may affect continuity of mineralisation in the deposit. Due to the limited outcrop across the licence area, however, it has not been possible to collect structural measurements of fault orientations across the deposit or to undertake any surface (soil or rock chip) sampling, surface trenching or pitting.

Through the Faculty of Mining and Geology at the University of Belgrade, Erin has conducted mineralogical studies on 47 singular and composite mineralised samples taken from a selection of their drill core, (December 2012). The length of the tested intervals ranges from 0.45 m to 8.40 m and were selected by visual estimation of samples containing either massive borate mineralisation or disseminated borate mineralisation in laminated claystone, dolomite and intercalated calcite.

The studies involved petrographic, x-ray diffraction (XRD) and scanning electron microscope with energy dispersive x-ray spectroscopy (SEM-EDS) analysis of the main mineral phases and are summarised in the report titled "*Testing samples from the Piskanja borate deposit, Baljevac na Ibru – Drillholes 101,103, 104, 106, 107, 111, 120, 121 and 126*". SRK has reviewed this report which concludes that the borate minerals are dominated by colemanite, ulexite and less commonly hydroboracite or jarandolite.

In spring 2012 Erin contracted MWH UK LTD, a UK based water management, engineering and monitoring company, to undertake an ongoing hydrogeological study for the Project. This work has included a review of pre-existing hydrological, meteorological and hydrogeological studies covering the Jarandol Basin (reported in an MWH Technical Memo dated June 2012), hydrogeological mapping, an initial survey of domestic wells and water supply, geophysical logging of Erin's drill holes, and design of a preliminary hydrogeological conceptual model.

Erin has undertaken a density study on samples from mineralised intervals and host rocks in accordance with the requirements of the Mineral Resource Code of Serbia. A total of 101 samples, each 9 to 25 cm in length totalling 15.64 m of core, were collected from the core stored in Erin's Baljevac storage facility. Samples taken from host rock lithologies were all whole core samples, whereas samples from the mineralised intervals were ¼ (quarter) core samples. The samples were sent to the Faculty of Ore Geology, Department of Geomechanics at the University of Belgrade for analysis.

The analysis included determination of unit weight (UW) using core and specific density (SD) using rock powder. Table 10-3 shows the results of this determination for the main lithologies found in the Piskanja deposit.

Table 9-1: Summary of unit weight test results for core samples of different lithologies within the Piskanja deposit. Figures in brackets are number of individual samples tested

Rock type	Specific Density (kN/m^3)	Unit Weight (kN/m^3)	Unit Weight (t/m^3)
Siltstone	25.76	25.12	
Borate mineralisation	24.57 (8)	22.53 (36)	2.287 (36)
Breccia	26.70	25.31	
Claystone	25.59 (17)	24.52 (21)	2.48 (4)
Conglomerate	27.57 (1)	25.6 (1)	
Dolomite	24.12 (3)	23.32 (3)	
Sandstone	23.79 (3)	22.78 (3)	

*-Unit Weight (t/m^3) was calculated from kN/m^3 by dividing by 9.81
Numbers in brackets are the number of samples tested from each lithology

SRK has not visited this facility to observe any of the analyses, however, review of the reports written by the University indicate the methods used were as follows.

Unit weight (also known as specific weight) was determined using the water immersion method where natural state (non-dried) samples were weighted in air, coated in paraffin and weighed in water. Specific density (also known as specific gravity) measurements were conducted on powdered, dried samples using a pycnometer.

There is significant variation in unit weight measurements for natural state (undried) borate mineralisation samples, between 1.914 t/m^3 and 2.537 t/m^3. This is most likely caused by variations in the dominant borate minerals in each sample, (colemanite has an SG of 2.42 t/m^3, whereas ulexite has an SG of 1.95 t/m^3) but may also be due to intercalations of clay and dolomitic rock, if mineralised samples selected were not all composed solely borate minerals. It is also likely that variable water content and presence of cavities in the natural state samples also add to the inconsistencies in SG.

SRK has advised Erin that in future all SG measurements are undertaken on dried core samples using the water immersion method.

Finally, Erin has an agreement with Ibarski Rudnici, the operators of the coal mine in the Jarandol Basin, for the provision of geological services and exploration support to the Project. As part of the contract between the two companies, Ibarski Rudnici prepared and submitted the 2013 mineral resource report to the Ministry of Mining and Environmental and Spatial Planning on behalf of Erin and according to Serbian Guidelines.

10 DRILLING

As part of its 2011/2012 exploration programme, Erin planned for approximately of 18,000 m of diamond core drilling, however only 13,569 m were drilled in order to complete the planned 38 holes (Table 10-1 and Table 10-2). When combined with the historical Ras Borati RC drilling completed in 1997 and Rio Tinto diamond holes in 2006, this new programme completes an approximately 100 m x 100 m spaced grid of drill collars across the Piskanja deposit. This was a requirement of the original exploration licence granted to Erin in 2010. During its exploration campaign, Rio Tinto had twinned two pre-1992 drillholes drilled by Ibar Mines, B6-90 and B8-91, the twins being IBM-9 and IBM-3 respectively. Erin has also twinned two pre-1992 drillholes drilled by Ibar Mines, B29-97 and B10-91, these twinned holes are known by the drillhole identifications as EVP2011-100 and EVP2011-102 respectively.

All of the drilling undertaken by Erin was completed between 11 July 2011 and 18 December 2012 and was performed on a 24 hour shift pattern. In order to complete this drilling quickly, Erin commissioned companies and rigs which were available in Serbia that could mobilise at short notice: Drilling contractor GeoMag d.o.o (Serbia) completed 27 DD holes totalling 9,881 m using an Atlas Copco - Christensen CS14, a Delta Makina - Delta Drill D-150 drill rig and a Diamant Boart DB-1200 drill rig; Silur d.o.o. (Serbia) completed 8 DD holes totalling 2,877m using Diamant Boart DB-1200 and Mustang A65 drilling rigs; three DD holes, totalling 810m, were drilled by Serbian contractor Geosonda d.o.o. using Diamant Boart DB-1200 and GEO 500 rigs. During SRK ES's visit to Piskanja in June 2012, drilling was observed at two of the GeoMag rigs (Christensen CS14 and Delta Drill D-150). It is SRK's opinion that drilling was conducted by experienced drilling crews using suitable rigs and to a high standard with due consideration of environmental and health and safety procedures.

All drillholes completed on behalf of Erin were of HQ diameter (64mm) and used double tube (Silur d.o.o) and triple tube (Geomag d.o.o) core barrels. Holes were all planned as vertical (with an azimuth of 000 and dip of -90) to intersect the mineralisation at 90°. Down-hole surveys of all drill holes were conducted by Geo-Log doo (Belgrade, Serbia) at 1m downhole intervals, shortly after the completion of each drillhole. The results of this work indicate that the maximum deviation is found in hole EVP2011-102 which deviated by a maximum 29.7m from the collared X-Y coordinates, measured at the end of hole (EOH) depth of 287.5m. The holes that do deviate a small amount do so in a west - northwest direction.

The depths of the drillholes are variable as the termination of a hole was determined by the on-site geologists during drilling. This was based on working cross sections of the deposit and the intersection of a marker conglomerate bed at the base of the Lower Conglomerate and Sandstone Unit, TcP_1.

The core recovery for individual zones is reported to be between 90.2 and 97.4% except Zone 4, which had on average 84.9% recovery. Overall average core recovery is 93.5% for mineralised intervals and 93.3% for host rocks throughout drilling and is considered by SRK to not materially affect the reliability or accuracy of sampling and assay results. As the borate mineralisation observed is concordant with the bedding and the strata with a gentle dip southwest and as the holes are drilled perpendicular to the bedding, SRK is satisfied that the difference between the drilled sample length and true thickness of mineralisation is not an issue and that true thickness is observed in the drill core.

Table 10-1 summarises the 38 Erin holes completed and Table 10-2 details these 38 holes. Figure 10-1 shows those holes drilled by Erin as well as the location of historical drill holes. All of the collars indicated in this figure fall within the Piskanja Exploration Licence (#1934). Drillhole EVP2012-127 was terminated at 91 m depth due methane gas release from a fault zone at 90.1m. Drillhole EVP2012-127A was therefore re-drilled 8.5 m to the southwest of the terminated hole to maintain the required drillhole spacing. Methane gas has not been noted in any other holes.

Table 10-1: Summary of Erin's 2011/2012 Piskanja Project diamond drilling programme

Programme state date	11 July 2011
Programme completion date	18 December 2012
Number of diamond core holes completed	38
Total meters drilled	13,569m
Minimum hole depth	91m (EVP2012-127)
Maximum hole depth	485.6m (EVP2012-132)
Mean average hole depth	357.1m

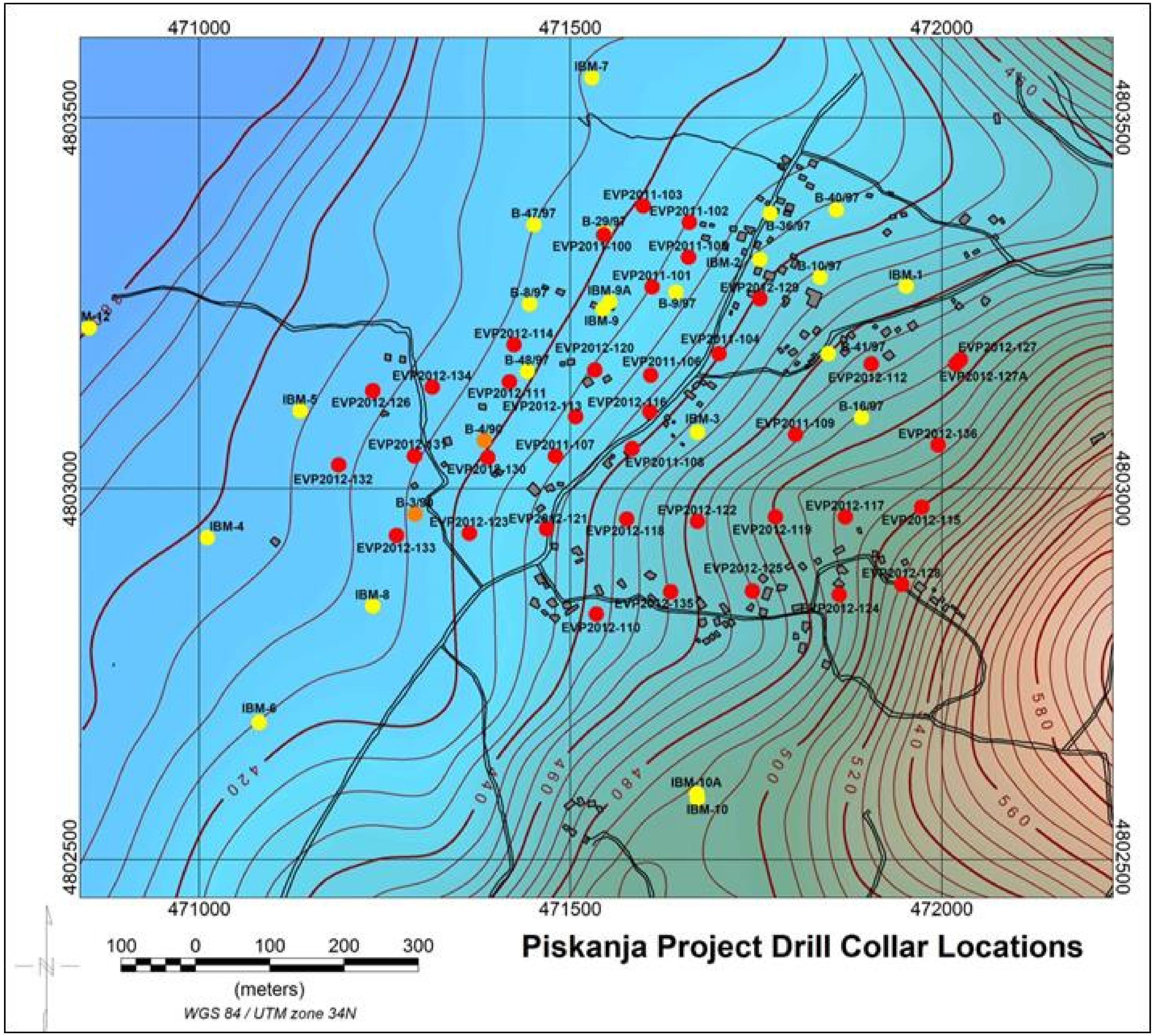


Figure 10-1: Location of drill collars for the Piskanaja Project overlaid on topography.

Red dots = Erin 2011/2012 holes; Yellow = Rio Tinto 2006/2007 holes; Orange = Ibarmines pre-1997 holes.

Table 10-2: Location of Erin Ventures diamond holes drilled in 2011/2012 for the Piskanja Project, Serbia. Coordinates are stated in UTM WGS84

BHID	Start date	Finish date	Easting, m	Northing, m	Elevation, m	Dip	Azimuth	Final Depth, m	Drilling Company
EVP-2011-100	11/07/2011	14/08/2011	471545	4803342	420.15	-90	0	223.5	Geosonda
EVP-2011-101	10/08/2011	22/09/2011	471610	4803272	425.83	-90	0	407.0	Silur
EVP-2011-102	22/08/2012	08/10/2012	471660	4803359	426.12	-90	0	287.5	Geosonda
EVP-2011-103	07/10/2011	24/10/2011	471598	4803381	421.50	-90	0	309.3	Silur
EVP-2011-104	13/10/2011	07/12/2011	471700	4803182	440.40	-90	0	299.5	Geosonda
EVP-2011-105	01/11/2011	29/11/2011	471659	4803312	426.96	-90	0	270.8	Silur
EVP-2011-106	03/11/2011	29/11/2011	471608	4803153	433.97	-90	0	321.0	GeoMag
EVP-2011-107	02/12/2011	24/12/2011	471480	4803044	430.05	-90	0	373.3	GeoMag
EVP-2011-108	12/12/2011	21/12/2011	471583	4803054	439.80	-90	0	356.6	GeoMag
EVP-2011-109	27/12/2011	09/03/2012	471803	4803073	470.32	-90	0	362.0	Silur
EVP-2012-110	24/01/2012	12/03/2012	471535	4802831	437.19	-90	0	364.1	GeoMag
EVP-2012-111	23/02/2012	13/03/2012	471418	4803144	418.94	-90	0	380.4	GeoMag
EVP-2012-112	26/02/2012	19/03/2012	471905	4803168	468.26	-90	0	302.6	GeoMag
EVP-2012-113	16/03/2012	01/04/2012	471507	4803097	428.35	-90	0	389.1	GeoMag
EVP-2012-114	19/03/2012	10/04/2012	471424	4803194	417.57	-90	0	346.2	Silur
EVP-2012-115	20/03/2012	10/04/2012	471973	4802975	498.69	-90	0	340.3	GeoMag
EVP-2012-116	02/04/2012	21/04/2012	471607	4803104	437.46	-90	0	350.5	GeoMag
EVP-2012-117	17/04/2012	26/04/2012	471870	4802962	500.60	-90	0	371.6	GeoMag
EVP-2012-118	23/04/2012	06/05/2012	471576	4802959	449.53	-90	0	377.4	GeoMag
EVP-2012-119	28/04/2012	19/05/2012	471776	4802962	490.13	-90	0	386.5	GeoMag
EVP-2012-120	08/05/2012	07/06/2012	471533	4803160	426.22	-90	0	347.2	Silur
EVP-2012-121	09/05/2012	30/05/2012	471468	4802946	435.77	-90	0	403.8	GeoMag
EVP-2012-122	22/05/2012	02/06/2012	471671	4802956	468.02	-90	0	389.6	GeoMag
EVP-2012-123	01/06/2012	28/06/2012	471364	4802940	425.22	-90	0	424.2	GeoMag
EVP-2012-124	06/06/2012	16/06/2012	471862	4802857	481.81	-90	0	395.6	GeoMag
EVP-2012-125	18/06/2012	03/07/2012	471745	4802862	464.95	-90	0	348.9	GeoMag
EVP-2012-126	28/06/2012	08/08/2012	471234	4803132	410.57	-90	0	414.6	Silur
EVP-2012-127			472025	4803174	476.15	-90	0	91.0	GeoMag
EVP-2012-127A	05/09/2012	07/12/2012	472019	4803168	475.89	-90	0	320.2	GeoMag
EVP-2012-128	05/07/2012	20/07/2012	471946	4802871	505.09	-90	0	422.0	GeoMag
EVP-2012-129	08/07/2012	02/09/2012	471755	4803256	439.96	-90	0	281.6	GeoMag
EVP-2012-130	22/07/2012	03/08/2012	471389	4803042	422.16	-90	0	410.3	GeoMag
EVP-2012-131	27/07/2012	10/08/2012	471290	4803044	416.36	-90	0	430.6	GeoMag
EVP-2012-132	05/08/2012	26/08/2012	471188	4803032	411.75	-90	0	485.6	GeoMag
EVP-2012-133	18/08/2012	06/09/2012	471266	4802937	417.73	-90	0	451.7	GeoMag
EVP-2012-134	21/08/2012	10/12/2012	471314	4803137	413.18	-90	0	420.1	Silur
EVP-2012-135	28/11/2012	09/12/2012	471635	4802861	451.10	-90	0	368.2	GeoMag
EVP-2012-136	09/12/2012	18/12/2012	471995	4803059	485.75	-90	0	344.6	GeoMag

11 SAMPLE PREPARATION, ANALYSIS AND SECURITY

11.1 Sampling of Diamond Core

During its site visit in June 2012, SRK observed Erin's geologists undertaking logging and sampling procedures at the drill rig site and at Erin's office area in Baljevac. The geological team employed by Erin to carry out the sample procedures were in SRK's opinion following company procedures although these were not documented and so it was not clear if the same procedure was always being executed.

As such SRK recommended that protocols should be created for the team to follow and that these should include instructions to:

- Sample half core instead of quarter core;
- Sample intercalations of mineralised and host lithologies and the host rocks located in hanging and foot walls, and;
- Record logging and sampling procedures and protocols in written form.

During SRK's June 2013 visit, Erin provided SRK with the document; "Abstract of Exploration Methodology (Sampling and QA/QC Procedures)", relating to core logging, sampling, sample security and database management. These procedures have been in use since July 2012 for the sampling of holes EVP-2012-119 to EVP2012-136, equating to 61% of the total samples collected (including QAQC samples). Due to the timing of SRK's second site visit in June 2013 after the completion of drilling, these procedures were not observed in action by SRK. The procedure for logging and sampling requires that:

- Core is packed into plastic and metal core trays at the drill site, each tray containing up to a maximum of 3m of core (five sections of 60cm length). After the core has been washed, a down-hole direction line is drawn and the core box is marked with information about the borehole. A quick geological log is also prepared at this point. After each drilling shift the core boxes are transported to the Erin office in Baljevac.
- The core is geologically logged and basic geotechnical information (core recovery and RQD) recorded directly in to digital databases by Erin's geologist.
- Digital photographs are taken of both dry and wet core.
- Mineralised zones are defined visually. High grade, white-cream coloured borate mineralisation usually has clear and sharp lithological contacts and is marked for sampling at 0.3-1.0 m (maximum 1.45 m) intervals. Depending on the presence of disseminated mineralisation, up to two samples of host rocks from both above and below the mineralised borate horizons are sampled at intervals of 1 m to 3 m in thickness (minimal thickness is 0.45m). Thus each hole may have a number of mineralised sections sampled, each possibly including massive borate mineralisation, intercalated or disseminated mineralisation and barren rock from the hanging and foot walls. The core between horizons of massive mineralisation with no visually identified borate minerals is not sampled for assay.
- Marked core is cut lengthways using a diamond saw. Half core from the marked sample intervals is then selected, placed in cloth sample bags and numbered with the predefined sample number written on plastic tags.

- The remaining core is kept in a dry and secure storage at Erin's Baljevac facility, Figure 11-1).
- Following sampling by Erin's geologists in the Baljevac facility, individual samples in cloth bags are packed in to larger PVC sacks for transport to the SGS Bor preparation laboratory, Figure 11-3a). Samples are transported by Erin staff personally by car, or by courier if there was a sufficient volume of samples.
- Finally, all samples are checked-in at SGS Bor against the sample submission form by SGS Bor staff. All coarse and pulp reject material was returned to Erin's Baljevac facility in paper sample bags, as shown in Figure 11-3b). Samples for analysis at SGS Lakefield, Canada or ALS Romania, were shipped via DHL couriers.

Figure 11-1: Erin's Baljevac core logging (above) and storage facility (below) including Erin's 2011/2012 diamond core (lower, left) and historic Ibar Mines core (lower, right).

a) pvc sacks for sending samples to labs
or for rest of samples after crushing (from SGS lab)

b) paper bags for rest samples after pulverisation (from SGS lab)

Figure 11-2: Sample packaging types used by Erin and SGS Bor.

11.2 Sample Preparation

All half core mineralised samples were sent for preparation at the Dundee Precious Metals Ltd (SGS Bor) sample preparation laboratory located in the city of Bor, Serbia (managed by SGS). Samples were then analysed using the geochemical methods detailed in Section 11.3, predefined by Erin during logging depending on the expected grade of borate mineralisation. Assays to date have been conducted primarily at the SGS laboratory in Lakefield, Canada, with further analysis completed at SGS Bor and ALS in Romania.

Samples were submitted for preparation and assay in batches in sequential sample number, which corresponds with drillhole order.

The Bor laboratory was established by SGS to support Dundee's exploration programmes within Serbia and is now managed by SGS as a commercial analytical laboratory. The SGS Bor laboratory does not have international accreditation, but employs the quality assurance procedures and controls that all SGS laboratories use internationally. SRK has not visited this preparation laboratory. Erin uses SGS Bor's standard preparation procedure "PRP86", for preparing the drill core samples from the Piskanja Project. This procedure comprises:

- Drying of samples at 60°C for 8 hours (up to 24h if necessary for soils or weakly consolidated rocks);

- Crushing to 1-2mm using a jaw crusher and reduction of sample size to about 700 g using a Jones riffle splitter; and
- Pulverisation to 75µm using a Labtech Essa LM5 mill.

The drying procedure had initially been completed at a temperature of 105°C for the 240 samples prepared prior to July 2012. The remaining 61% of samples prepared following this date were dried at 60°C. Erin's justification for this reduced temperature was to prevent possible dehydration/decomposition of boron mineralisation and phase transition (Yilmaz et al., 2013). While it is SRK's opinion however that the main minerals found in the Piskanja Deposit will be stable during the "normal" drying procedure (105°C), the amendments to drying temperature should not have materially affected the estimation of boron grade in the samples.

Once prepared, approximately 250g of each pulped sample was sent by SGS Bor from Serbia to SGS Lakefield in Ontario, Canada for analysis. The remaining coarse and pulp rejects were returned to the Erin office in Baljevac for storage.

11.3 Sample Analysis

11.3.1 Analytical Methodology

Several analytical methods have been used to assay samples from the 2011/2012 resource drilling of the Piskanja deposit. Methods were assigned to samples based on the expected boron content estimated by Erin geologists during core logging. The number of analytical methods used were reduced in July 2012, mid-way through the drilling programme, as summarised in Table 11-1.

For clarity, all drilling, sampling and geochemical analysis conducted prior to July 2012 shall herein be termed Stage 1. The remaining drilling, sampling, analysis and amended protocols used after July 2012 shall be referred to as Stage 2. Stage 1 analyses applied to the 259 samples collected from holes EVP2011-100 to EVP2012-118 (sample ID 1065 to 1849) and Stage 2 applied to 401 samples collected from holes EVP2012-119 to EVP2012-136 (sample ID 1850 to 2251).

Table 11-1: Summary of Piskanja Project sample analyses carried out at primary and secondary laboratories (including QAQC samples)

Element analysed	Analytical Method	Laboratory	Laboratory Method Code	Stage 1 (Jul 2011 – Jun 2012)	Stage 2 (Jul 2012 – Jan 2013)	Total
PRIMARY LABORATORIES						
Total samples analysed				**259**	**401**	**660**
Boron	Volumetric Titration	SGS Lakefield	GC-CLA68V	223	198	418
	KOH-ICP-AES	SGS Lakefield	GC-ICP94V	143	1	144
B, As, Li, Sr	Aqua regia ICP-AES	SGS Lakefield	GC_ICP14C	61	0	61
Major oxides	XRF	SGS Lakefield	XRF76C	259	144	403
Sulphur	Combustion IR	SGS Lakefield	GC/GU/GP CS A06V	218	0	218
Multielement analysis	Aqua regia ICP-MS	SGS Bor	IMS14B	0	401	401
SECONDARY "UMPIRE" LABORATORY						
Total samples analysed				**0**	**30**	**30**
Boron	Na_2O_2 fusion ICP-AES	ALS Romania	ME-ICP41a	0	11	11
Major oxides	XRF	ALS Romania	ME-XRF26	0	11	11
Multielement analysis	Aqua regia ICP-AES	ALS Romania	B-ICP82a	0	30	30

The following methods were used to determine the boron and B_2O_3 content of samples;

Volumetric Titration

The sample is digested in an acidic solution and then filtered. Iron in the sample is oxidised by the addition of bromine water. Barium carbonate is added to react with the boric acid, forming soluble barium borate and precipitating hydroxides of interfering metals. The solution is filtered and complexed with sorbitol. It is then titrated with NaOH to determine the B_2O_3 content.

The titration method is good at producing repeatable results to a high level of accuracy for the total content of acid soluble B_2O_3. The method is designed for samples containing a high percentage of B_2O_3 and has a measurement uncertainty of 0.3 in the 10-50% B_2O_3 range.

Alkali fusion ICP-AES

Samples are fused at high temperature with an alkali flux, commonly potassium hydroxide (KOH) or sodium peroxide (Na_2O_2). Total boron (B) is determined by induction coupled plasma atomic emission spectroscopy (ICP-AES) and reported in ppm. This method is recognised as being suitable for ore-grade samples containing high percentages of boron. Conversion to equivalent B_2O_3 (%) requires multiplication of the ppm B grade by 3.2199×10^{-4}.

Aqua Regia ICP-AES or ICP-MS

Aqua Regia (AR) is the name given to a highly corrosive mixture of nitric acid and hydrochloric acid, optimally, but not always in the ratio of 1:3. Samples are dissolved by this acid at 90°C before determination of multiple elements is completed using induction coupled plasma atomic emission spectroscopy (ICP-AES) or induction coupled plasma mass spectrometry (ICP-MS) equipment.

Boron is easily volatilized from acid solutions, such as that used in aqua regia digestion, causing loss of boron during sample digestion, (Ryan and Langmuir, 1993, Nagaishi and Ishikawa, 2009). The inconsistent liberation of boron in acid digestion and low upper limit of detection used in aqua regia ICP-MS (~1% B) makes this method unsuitable for quantitative boron analysis.

SRK assessed the results of analysis obtained by different analytical methods. As a result, a discrepancy has been revealed between the aqua regia ICP-MS and volumetric titration method for the detection of the boron grade. Originally, the aqua regia ICP-MS method was used due to its relatively low cost compared with the other methods. This method was used to identify samples which exceeded 15% B_2O_3 grade, with those samples above this grade sent for titration analysis. Figure 11-2 below shows the poor correlation of aqua regia ICP-MS results with the titration grades for the same samples, thus confirming aqua regia ICP-MS as an inadequate method for high grade B_2O_3 analysis.

SRK is however satisfied that the alkali fusion ICP-AES methods and volumetric titration produce sufficiently similar results for boron content of samples, as seen in Figure 11-3 and Figure 11-4.

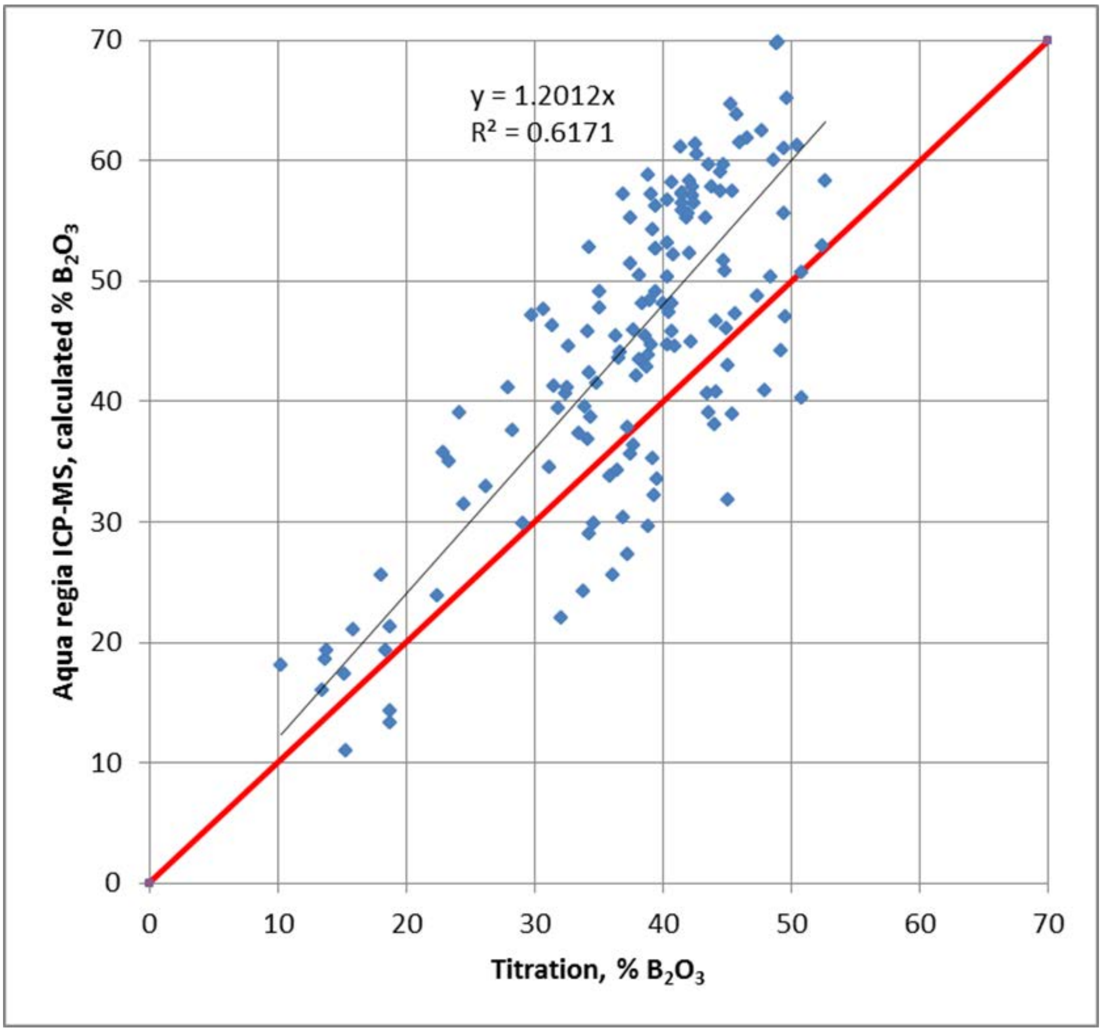


Figure 11-2: Aqua regia ICP-MS versus titration grade for B_2O_3 obtained during Erin exploration campaign

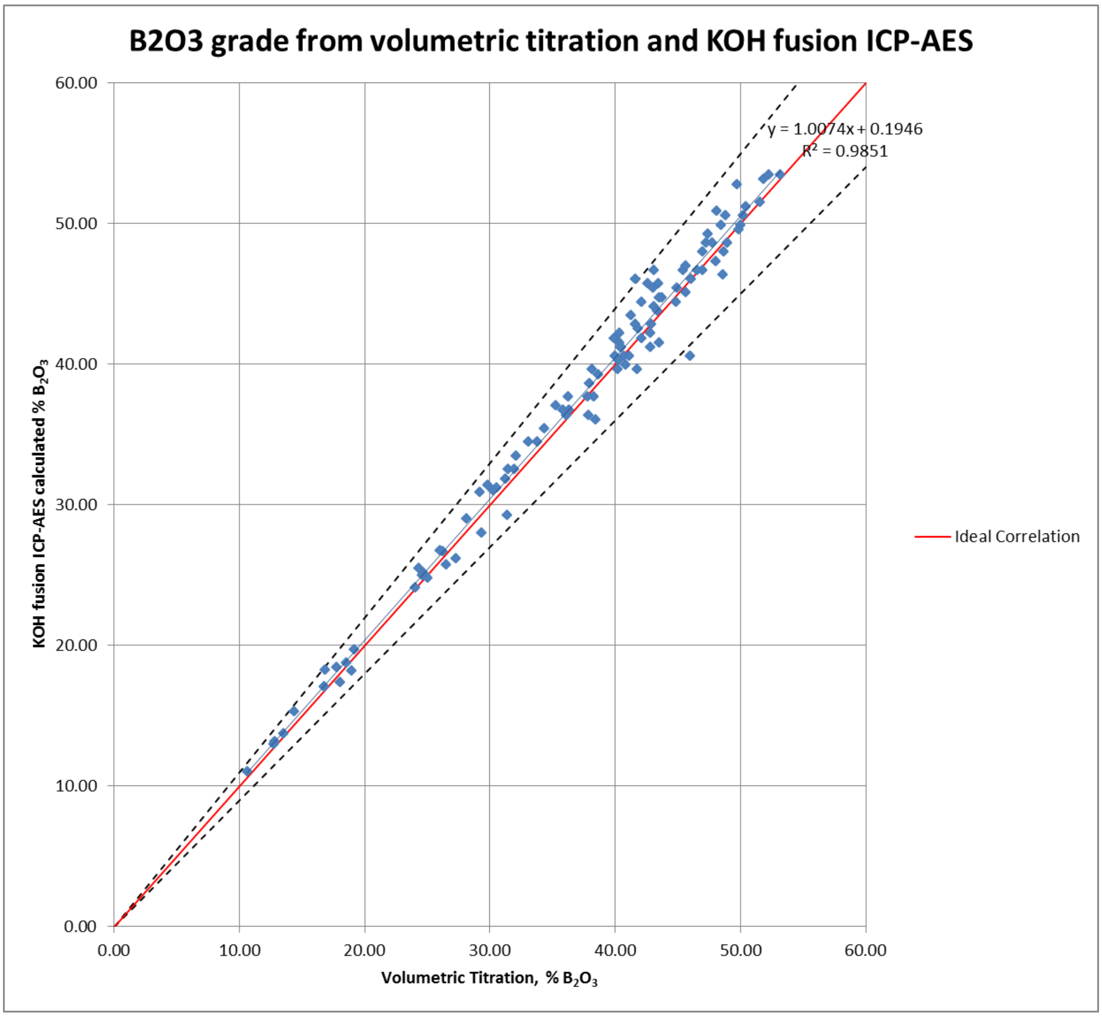


Figure 11-3: Correlation of results for samples analysed by both volumetric titration and KOH fusion ICP-AES during Phase 1

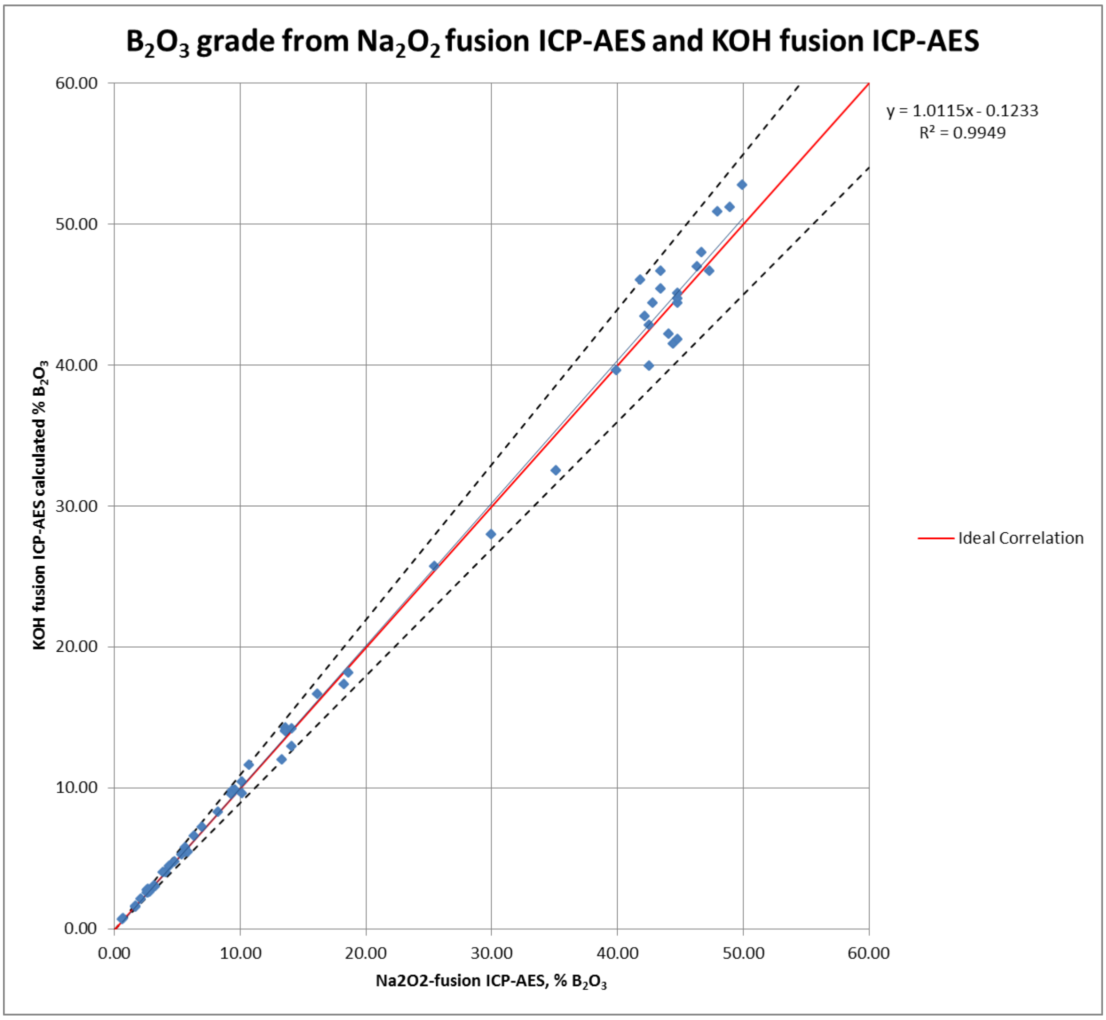


Figure 11-4: Correlation of results for samples analysed by both Na_2O_2 fusion ICP-AES and KOH fusion ICP-AES during Phase 1

11.3.2 Initial Analysis Programme (Stage 1)

During Stage 1 Erin conducted different analysis on the samples depending on whether they were deemed high grade or low grade. Erin determined this from observation of borate mineralisation in the core. In total, 259 samples were collected during Stage 1, taken from holes EVP2011-100 to EVP2012-118, Erin selected 223 samples which they deemed to be "high-grade" and sent them for volumetric titration to determine the B_2O_3 content. A total of 143 samples of the 259 samples, which were a combination of massive and disseminated/intercalated samples, were analysed for total boron by KOH fusion ICP-AES. A total of 107 of these samples had also previously been analysed using volumetric titration (Figure 11-3). A further 61 samples which were observed by Erin to contain intercalations or disseminations of borate minerals or samples which were deemed as "waste" material which were immediately adjacent to mineralised intervals in the core, were assayed for soluble boron by aqua regia ICP-AES. This final method also analysed for arsenic, lithium and strontium as these elements are known to be pathfinders for borate mineralisation in other deposits.

All 259 samples were analysed for whole rock oxide content with a metaborate fusion XRF method (x-ray fluorescence) and 218 samples were also assayed for sulphur by combustion infrared detection. All of the above analyses were completed by ISO 17025 accredited SGS Lakefield, Ontario, Canada. SRK has not visited this laboratory as part of this commission.

Stage 1 accounts for 39% of the total sampling programme resulting from the 2011/2012 drilling campaign and included very limited quality control procedures. This was, however, addressed during Stage 2.

11.3.3 Amended Analysis Programme (Stage 2)

Following amendment and simplification of the sample analysis and QAQC procedures in July 2012, the remaining high grade samples (198 of 401 Phase 2 samples) collected from drill core (holes EVP2012-119 to EVP2012-136), were all analysed for B_2O_3 by volumetric titration at SGS Lakefield. 144 samples were analysed for whole rock oxide by XRF, also at SGS Lakefield. In addition, all 401 samples were analysed by aqua regia ICP-MS for 53 elements at SGS Bor.

As detailed below, QAQC samples (duplicate, blanks and reference materials) were used routinely in Stage 2.

11.4 QAQC Procedures and Results

11.4.1 Background

Erin did not utilise any blanks or duplicates during Stage 1 and submitted only five lithium certified reference material (CRM) samples for analysis during this time and even these were only introduced at the end of Stage 1, in the final batch of 41 samples. The CRMs were sourced from independent preparation laboratory; Shea Clark Smith, Mineral Exploration & Environmental Geochemistry, Reno, USA. Shea Clark Smith was established in 1984 and although they do not have any international accreditations it is understood that they work closely with several analytical laboratories (http://sheaclarksmith.com/). The CRMs from Shea Clark Smith were, however, lithium standards with very low boron content; therefore they do not match well with the characteristics of mineralisation found at Piskanja, (Table 11-2) and it is SRK's opinion that these CRMs were not appropriate for use in this case.

Table 11-2: Shea Clark Smith Certified Reference Materials used during the analysis of the first 240 samples for the resource drilling at Pisknja in 2011/2012

CRM Name	B content (%)	Li content (ppm)
MEG-LI.10.14	0.2	807
MEG-Li.10.15	1.8	1611

As part of its internal QAQC procedures the SGS Lakefield laboratory inserted CRMs and blank samples into the sample batches. No results from these internal assays have, however, been provided to Erin.

This following sections of the report detail the QAQC measures that were implemented in July 2012 and used during the analysis of the remaining 61% of samples which were collected during Stage 2 of the programme.

11.4.2 Certified Reference Material

Erin selected material in order to create their own internal CRM. Specifically, SRK understands that Erin collected some 200 kg of borate mineralisation sourced from the JP PEU Resavica Pobrdje borate mine located some 2.6 km northwest of the Piskanja Project. Material was classified into three bulk samples; high, medium and low grade by Erin and sent to the independent preparation laboratory; Shea Clark Smith, Mineral Exploration & Environmental Geochemistry, Reno, USA, who prepared CRMs with three different borate grades.

SRK understands that the material was; dried in thermostatically controlled electric ovens; crushed in series through corrugated, flat plate and roll crushers; and milled to 95% passing a 150 mesh sieve (0.105mm). Initially the material was split in half for the milling, the milling was completed in a ceramic lined ball mill for 5 days (one half), 5 days (second half) and for a final 5 days with the halves combined. The sieved material was then packaged into 50g samples in envelopes. One in every 100 samples created was set aside for the round robin/homogeneity testing, the remaining 99% of envelopes were returned to Erin as reference materials. Low grade reference material is identified at “1X B”, medium grade material as “2X B” and high grade material as “3X B”.

In total, five samples of each CRM were sent to 8 geochemical laboratories (Table 11-3) for a round robin analysis to determine a grade for each standard. Samples were submitted with a randomised numbering system assigned by Shea Clark Smith and the labs were encouraged to analyse the submittals as a normal daily production run and as such, certain samples were repeated as part of the laboratories QAQC systems. The 8 laboratories analysed the samples using Na_2O_2 fusion preparation and either an ICP AES or ICP-MS finish.

Table 11-3: Accreditation of laboratories used in the round robin analysis used to determine certified reference materials for the Piskanja Project

Laboratory Name	Location	Accreditation	Analytical Method used
American Assay	Sparks, Nevada, USA	ISO/IEC 17025:2005	Na_2O_2-fusion ICP-AES
ACME Analytical Laboratories	Vancouver, Canada	ISO/IEC 17025:2005	Na_2O_2-fusion ICP-AES
Activation Laboratories	Ontario, Canada	ISO/IEC 17025 and CAN-P-1579	Na_2O_2-fusion ICP
ALS Chemex	Vancouver, Canada	ISO/IEC 17025:2005	Na_2O_2-fusion ICP-AES
Florin Analytical Services	Reno, Nevada, USA	Application pending (ISO/IEC 17025)	Na_2O_2-fusion ICP
Genalysis/Intertek	Perth, Australia	ISO/IEC 17025	Na_2O_2-fusion ICP-AES
Kalassay	Perth, Australia	Application pending (ISO/IEC 17025) Accredited by the National Association of Testing Authorities, Australia	Na_2O_2-fusion ICP-AES
Alex Stewart	Mendoza, Argentina	ISO 9001:2008 ISO 14001:2004 ISO 17025:2005 (only for Au by fire assay and Li and K liquid brine analysis)	Na_2O_2-fusion -fusion ICP
SGS Bor (Preparation lab and primary multielement analysis)	Bor, Serbia	No national/international accreditation. Utilises standard SGS internal QAQC procedures	AR ICP-MS
SGS Lakefield (Primary laboratory)	Lakefield, Canada	ISO/IEC 17025:	Volumetric Titration KOH-fusion ICP-AES AR ICP-AES XRF Combustion IR
ALS Romania (Umpire laboratory)	Rosia Montana, Romania	ISO/IEC 17025:2005	Na_2O_2-fusion ICP-AES XRF AR ICP-MS

Table 11-4 and Figure 11-5 to Figure 11-7 show the results of the round robin for the three reference materials, 1X B, 2X B and 3X B. Although the individual laboratories generally have good precession of results showing a close clustering of repeat measurements, there is not such a good correlation between laboratories. All low grade analyses fall within the 2SD envelope, but results for standards tested at ACME Analytical Laboratories and Alex Stewart report higher and lower than the 2SD envelope respectively for both medium and high grade materials.

The summary results of these round robin assays are presented in Table 11-4. The mean grades from the round robin and shown here are used as CRM grades in Erin's QAQC programme.

SRK has suggested to Erin that such variation may be due to minor differences in procedures for the fusion of samples and Na_2O_2 during preparation or due to inconsistencies in the calibration of ICP analysis equipment and that further investigation of sample fusion durations and concentrations should be made to identify the cause of the inter-laboratory variation seen in the round robin data to date.

Table 11-4: Summary results for determination of reference material grades

Reference Material Name	1X B	2X B	3X B
Total samples assayed	42	43	42
Minimum, (% B)	5.52	9.79	12.94
Maximum, (% B)	6.59	11.97	15.87
Number of outliers removed	0	13	2
Mean average, (% B)	6.10	11.19	14.35
Standard deviation, (% B)	0.30	0.33	0.60

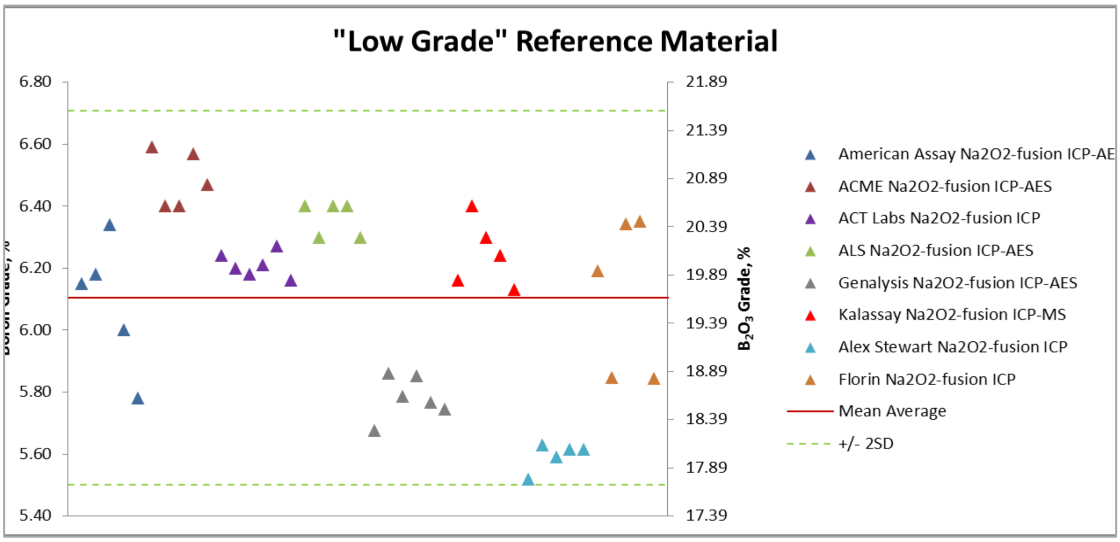


Figure 11-5: Round Robin results for Erin's "Low Grade" standard 1X B

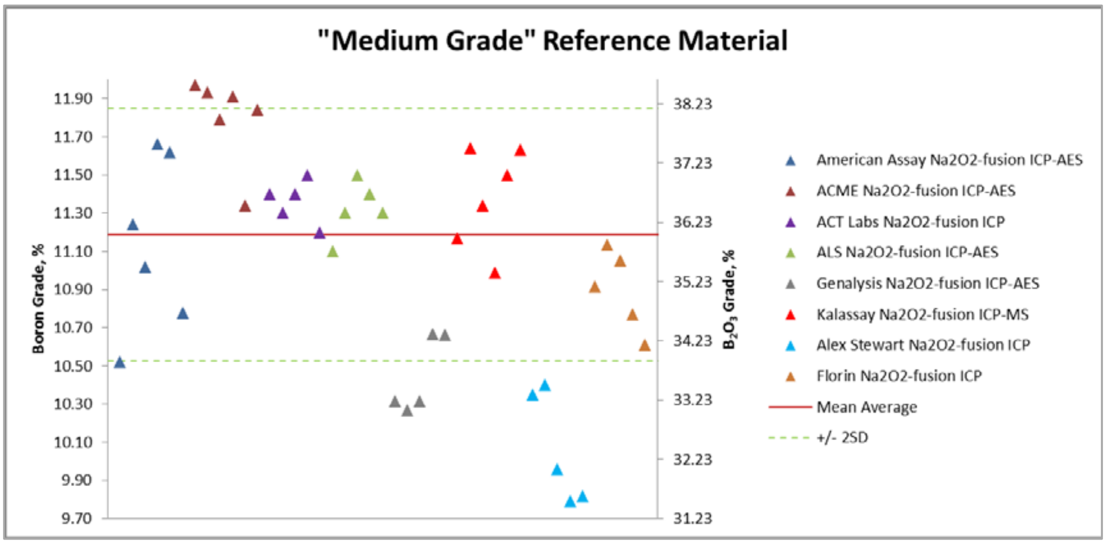


Figure 11-6: Round Robin results for Erin's "Medium Grade" standard 2X B

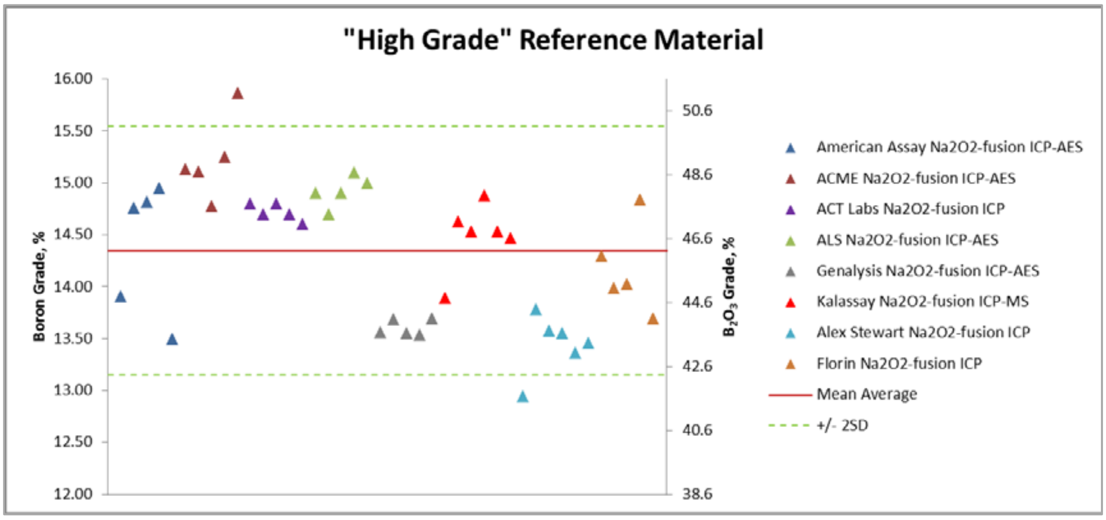


Figure 11-7: Round Robin results for Erin's "High Grade" standard 3X B

Results of Reference Material in the sample batches

As part of the analysis of the Stage 2 samples from the 2011/2012 Piskanja Project drilling, CRMs were inserted at a frequency of every 20th sample, alternating between “high” (3X B), “medium” (2X B) and “low” (1X B) grade standards.

A total of 19 CRMs were sent for analysis by volumetric titration (SGS Lakefield) and aqua-regia ICP-MS (SGS Bor) along with the primary samples. CRMs analysed with aqua regia ICP-MS have been excluded from this comparison due to the known inadequacy of this method for determination of boron.

18 CRMs were also submitted to a secondary “umpire” laboratory ALS Romania (Rosia Montana, Romania) where analysis was complete using the Na_2O_2 fusion ICP-AES method.

The results for all CRMs in the secondary lab fall well within the 2SD envelope for each standard, though consistently slightly above the standards’ respective means, indicating that the sodium fusion method is capable of providing repeatable boron analysis.

The results for the laboratory reference materials used as part of SGS' internal QAQC procedures have not routinely been reported. There is an inconsistency in reporting these internal QAQC results on the certificates produced by the SGS and so SRK is unable to fully evaluate the consistency and accuracy of the primary laboratory titration analysis results.

SRK recommend that more CRM analyses are required to reliably assess the repeatability of the analysis method.

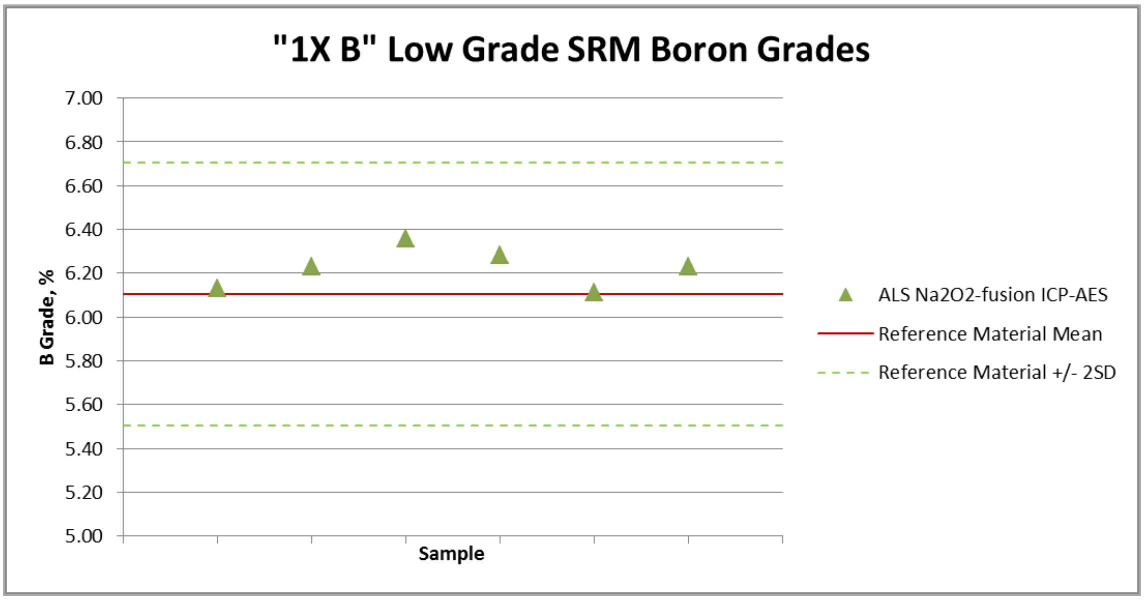


Figure 11-8: Plot of reference material 1X B grade by sodium peroxide fusion ICP-AES analysis method for ALS Romania (secondary laboratory)

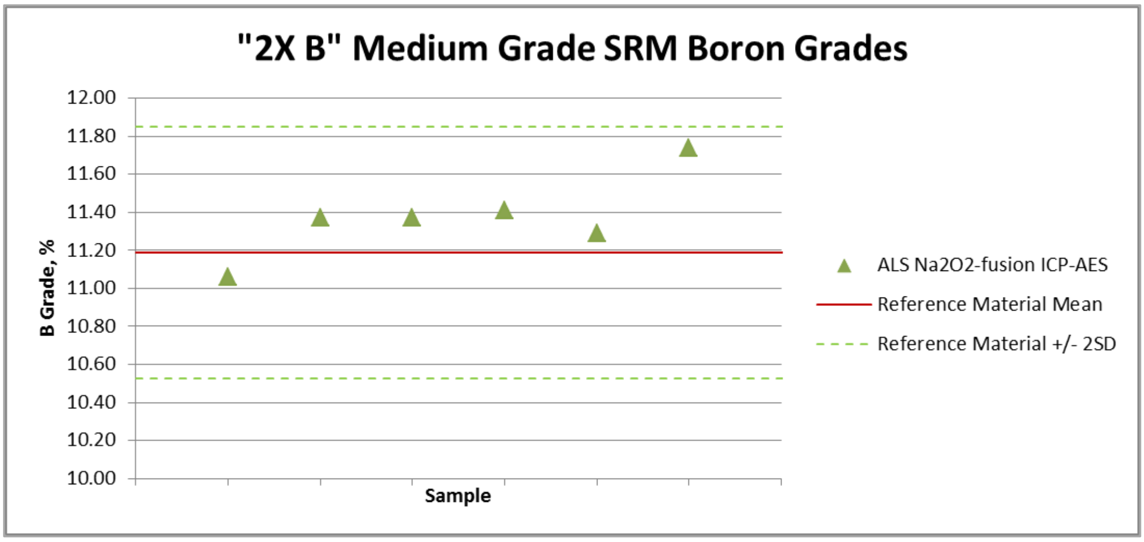


Figure 11-9: Plot of reference material 2X B grade by sodium peroxide fusion ICP-AES analysis method for ALS Romania (secondary laboratory)

Figure 11-10 : Plot of reference material 3X B grade by sodium peroxide fusion ICP-AES analysis method for ALS Romania (secondary laboratory)

11.4.3 Blank Samples

For Stage 2 of the resource drilling programme, Erin sourced material for blank QAQC samples from Vrh, near Studenica, approximately 40km by road northwest of Piskanja. Samples of quartzite, dolomite and three different marbles were tested at the SGS Bor laboratory by aqua regia ICP-MS. Erin selected one of the marbles to use as a QAQC blank material in the Piskanja sampling programme, based on the results of these ICP analyses.

Approximately 500kg of marble was collected, broken in to rock chips with hammers and between 2.3kg and 2.5kg of rock chips were put into sample bags. A blank sample was inserted into the sampling stream at the beginning of each new hole and following mineralised intervals. None of the 58 blanks created were assayed by volumetric titration or Na_2O_2 fusion ICP-AES, only by multi-element aqua regia ICP-MS.

The average content of boron in different types of carbonate rocks in the Earth's crust varies from 20ppm to 55ppm, while in clays the grades are higher, between 100ppm and 230ppm (Parker, 1967). The marble used for blanks in this QAQC programme possesses a higher boron content than expected for carbonate rocks (Figure 11-11). This elevated content of boron in the marble may be caused by the presence of disseminated boron mineralisation and contamination during the sample preparation and/or analysis.

SRK has recommended to Erin that further investigation of this is carried out. SRK also suggests an alternative blank material is sourced by Erin for future programmes with a lower boron content than currently used.

As the blank material has only been analysed using the aqua regia ICP-MS process at SGS Bor, the possibility of contamination of titration and alkali fusion ICP equipment in other laboratories used by Erin cannot be evaluated at this stage.

SRK has also recommended that the blank samples are submitted in the sample batches to both primary and secondary laboratories for analysis by the same methods as core samples and other QAQC samples.

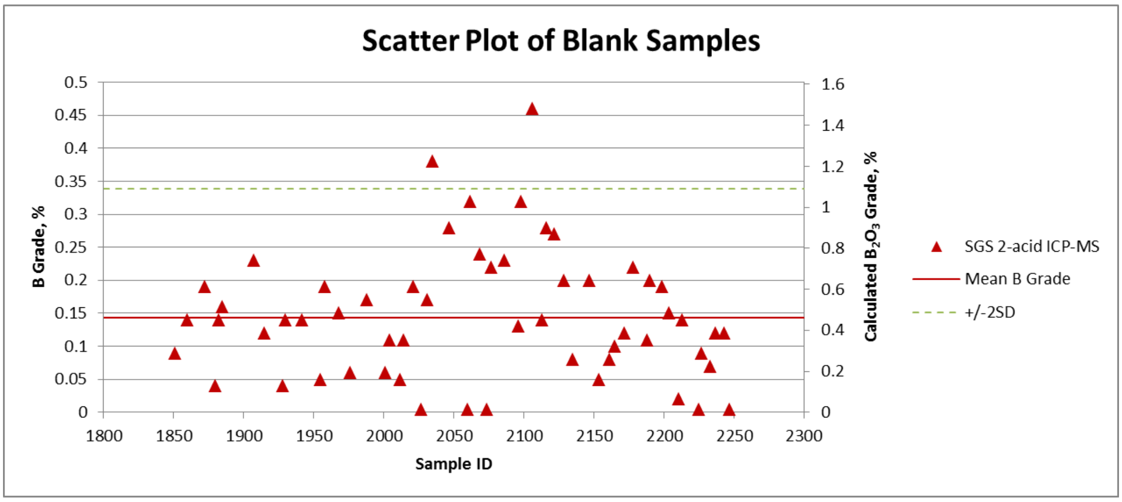


Figure 11-11: Scatter plot of blank marble samples submitted to SGS Bor

11.4.4 Duplicate Samples

After every 20th sample a field duplicate was taken by preparing a quarter (¼) core sample from the remaining half (½) core. The duplicate was assigned a sample number immediately following its original. Duplicates were assayed by the method selected by Erin for the original sample.

In total, 18 duplicates were created during Stage 2 and were analysed by aqua regia ICP-MS at SGS Bor. Only 8 of 18 samples were analysed by volumetric titration at SGS Lakefield (Figure 11-12). Aqua regia ICP-MS field duplicate results are not presented here due to the method's poor estimation of ore grade boron content. Volumetric titration field duplicate pairs show a good correlation, with only two pairs falling outside the +/-10% relative difference envelope. SRK has therefore suggested that at least 30 duplicate pairs are needed to statistically assess the repeatability of analysis by volumetric titration.

The variation seen in these duplicates can be explained by inhomogeneity in B_2O_3 grade or presence of voids between the two core samples. For this reason, SRK has recommended that henceforth pulp duplicates should be taken in addition of course field duplicates and all should be tested using the same analysis method.

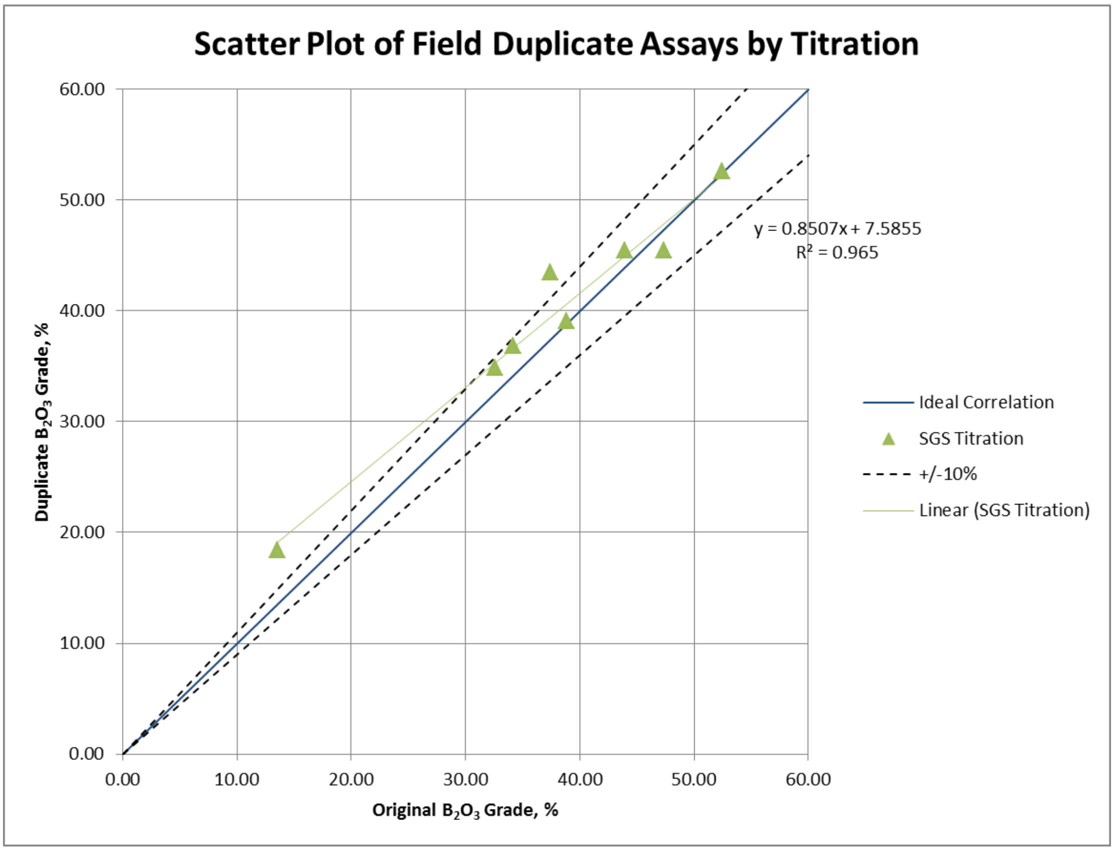


Figure 11-12: Scatter plot of B_2O_3 grade for 8 field coarse duplicates vs. original samples using volumetric titration

11.4.5 Umpire Laboratory

As of July 2012, Erin elected to use ALS Romania as an umpire laboratory for the remainder of the drilling and sampling programme (Stage 2). Duplicate samples were created by taking one pulp duplicate and one coarse duplicate from the reject material produced following sample preparation at SGS Bor, after every 25th primary laboratory sample (equating to 4% duplicates).

Erin used an insertion rate of 1 in 3 for CRMs but it is SRK's opinion that this is unnecessarily high and SRK has recommended that a 1 in 20 frequency should be used for material sent to the umpire laboratory in future. Comparison of boron grades from aqua regia ICP-MS analysis are not discussed here due to the poor assessment of boron content using this method.

11.5 SRK Comments on Adequacy of Procedures

As already stated, the alkali fusion ICP methods (potassium hydroxide (KOH) or sodium peroxide (Na_2O_2)) are most suitable for the determination of boron in "ore grade" samples. Although volumetric titration uses an acid digest prior to titration, which may result in volatilisation of boron during digestion, the comparison of KOH fusion ICP-AES results with Na_2O_2 fusion ICP-AES and volumetric titration results indicates that there is a good correlation and regression between the three methods.

Overall, SRK is of the opinion that the sampling preparation, security and analytical procedures used by Erin have been significantly improved since SRK's initial site visit in June 2012. Adequate QAQC procedures have been undertaken and provide comfort that there is no significant variation between volumetric titration and alkali fusion ICP results. To further improve the analyses and QAQC procedures, SRK has made the following recommendations for the next phase of drilling:-

- That sodium peroxide fusion (Na_2O_2) with ICP AES finish should be used as the primary method for determination of all elements of interest (except sulphur) including high grade boron. This may be achieved, for example, by using a modification of SGS' analytical package ICP-90, which is Na_2O_2 fusion ICP-AES method designed for high grade samples.
- All mineralised samples and those from the adjacent hangingwall/footwall should be analysed with the same method. Sample preparation for volumetric titration uses an acid digestion which may result in minor loss of boron as boron hydride and it recovers only acid "soluble" boron. The Na_2O_2 fusion ICP-AES method provides recovery of total boron, and is likely to be a much cheaper analysis method than volumetric titration.
- All QAQC samples (blanks, CRMs and duplicates) should be inserted at the same frequency (1 in 20) into both initial and umpire laboratory sampling streams and analysed with the same method as the primary core samples.
- Pulp duplicates of primary samples should be used to verify the primary laboratory analysis.
- Duplicates sent to the umpire laboratory should be analysed with the same method as the primary lab samples.
- Variations in the certification of the CRM B_2O_3 grades should be investigated and further round robin analysis undertaken to reduce the standard deviation error of the certified grades for all three CRMs.
- An alternative blank material should be sourced that contains negligible amounts of boron as the current marble may not be sufficiently low grade to determine contamination in sample preparation or analysis.

12 DATA VERIFICATION

12.1 Introduction

The data for the Piskanja deposit has been validated for use in the MRE by SRK. Notably SRK compared the historical drilling results with more recent drilling results to assess if the former could be used in the MRE, and also reviewed checked the sampling protocols, QAQC procedures and results, and also the Microsoft Access database provided by the Client.

12.2 Comparison of Historical Drilling Data with Recent Drilling Data

The drilling database supplied to SRK contained the data obtained during four different drilling campaigns spanning from 1989 to 2012. In 1989, diamond drilling was performed by Ibar Mines a government owned company; in 1997 RC drilling was performed by Ras Borati; between 2006 and 2007 diamond drilling was performed by Rio Tinto; and more recently between 2011 and 2012 drilling has been performed by Erin. Geological reports detailing the drilling procedures used and information on the lithological logs is only available for the 2006 to 2007 period (Rio Tinto) and the 2011 to 2012 period (Erin). There is no available core or report detailing the exploration data and quality of the data for the 1980s and 1990s campaigns. There is also no QAQC data is available for the drilling and sampling completed during this period. Erin is in possession of the core from the Rio Tinto holes and it is securely stored on-site for re-examination and resampling if required.

An analysis by SRK of the of the data from the 1989 and 1997 drilling campaigns showed that the assays might have been carried out only for intervals where mineralisation was visually observed. It is possible that four sterilisation drillholes which were drilled by Ibar Mines and Ras Borati were also not assayed as no mineralisation was visible but with no records this is difficult to determine.

During the Rio Tinto campaign from 2006 to 2007, a number of the Ibar Mines drillholes were twinned. No twin drilling was performed between Rio Tinto and Ras Borati drillholes. SRK has reviewed this work. Erin has not twinned any historical holes drilled by Ibar Mines and Ras Borati; however SRK has compared the results for the Erin holes which were drilled approximately 20m from historical collars. The results from the onmi-directional semi variogram, discussed later in Section 14 which was created using the data from the Erin and Rio Tinto drilling campaigns shows a range of 100 m, and therefore it is SRK's opinion that comparing drillholes in a 20m range is appropriate.

SRK's analysis of the results of the twin drilling where Rio Tinto's IBM-9 drillhole is twinned with Ibar Mines B-6/90 drillholes is illustrated in Figure 12-1. Figure 12-2 below compares Ibar Mines drillhole B-4/90 with Erin's drillhole EVP2013. SRK notes that while the grade parameters from both holes are quite comparable, the Ibar Mines holes show problems with accurately identifying the depth of mineralisation. SRK has also identified that the error in depth determination increases with depth and suggest this may be due to incorrect depth measurements being recorded. The difference in the depth can yield up to 20 m for the deeper mineralised horizons. It is also clear that Ibar Mines only sampled certain intervals.

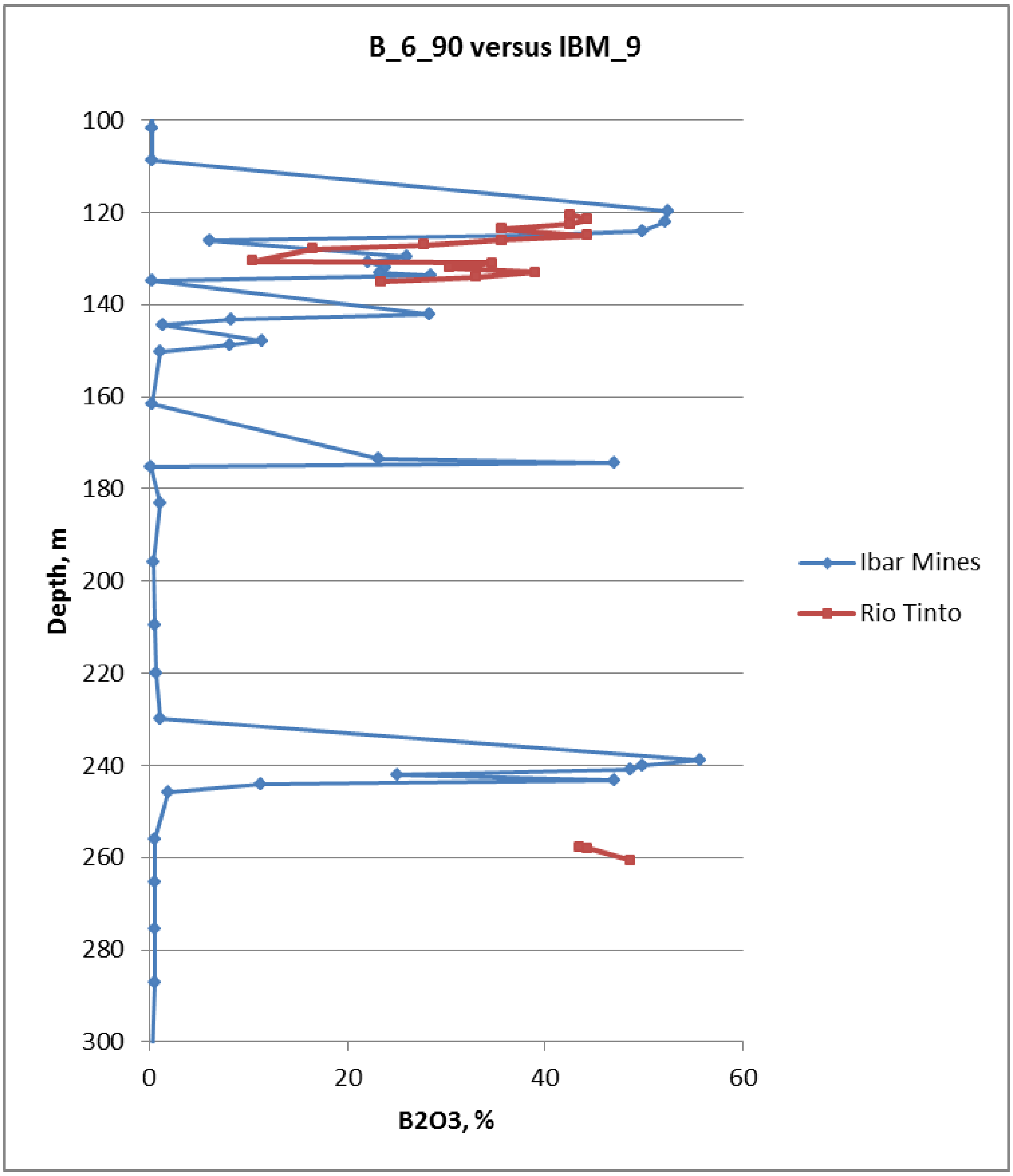


Figure 12-1: Comparison of the results for twin drilling between Ibar Mines and Rio Tinto campaigns

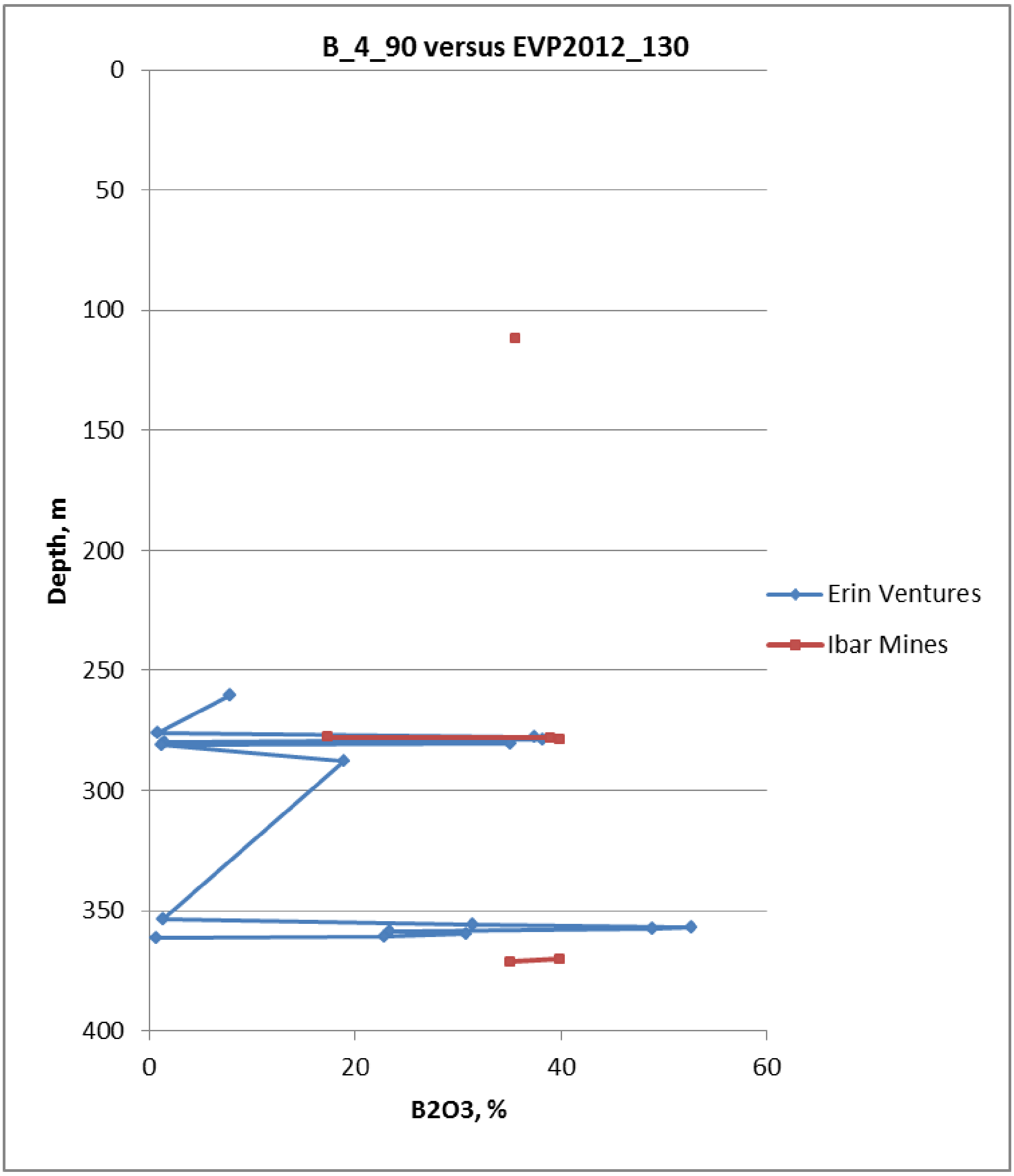


Figure 12-2: Comparison of downhole grade between Ibar Mines and Erin campaigns

By analysing the available database SRK has concluded that only the intervals with visual mineralisation were analysed during the Ibar Mines and Ras Borati periods. Generally, sampling of adjacent hangwall and footwall lithologies either side of the identified high-grade mineralisation was not systematically undertaken. It is SRK's opinion that such an approach could result in an underestimation of the thickness of mineralisation. There is also, in SRK's opinion, a possibility that depths were incorrectly recorded thus resulting in a difference in depth of mineralisation when compared to the more recent holes. Further, due to the fact that the core from these historical campaigns was not stored, there is no possibility of resampling these holes to verify the accuracy of the results.

In comparison the Rio Tinto drillholes were systematically sampled for the entire length of the core and yields comparable depths, grade and the thickness of mineralisation to the Erin holes.

Given the above, SRK determined that the results of the Ibar Mines and Ras Borati drilling campaigns are not sufficiently reliable to be used to derive resource estimates and this data was excluded from the database SRK used for this purpose.

12.3 Sampling and Assaying

During SRK's observations from the data sent prior to its first site visit and during the site visit it became apparent that the Erin drillholes were not being systematically sampled. Sampling by Erin was only being carried out within the intervals where mineralisation was observed and logged and was not always conducted in the footwall and hanging wall of each mineralised unit, nor win dilution zones and intercalations within the mineralised horizons.

During this site visit SRK undertook visual verification of the geology and sampling logs. Based on this analysis, SRK highlighted certain intervals to Erin and suggested that these were essential for better understanding the grade continuity and for SRK to make a detailed interpretation of the mineralisation.

Upon SRK's request, Erin sent 286 selected samples for sample preparation and analysis using Erin's approach; aqua regia ICP-MS for all samples and volumetric titration for mineralised intervals. Of these 286 analysed with aqua regia ICP-MS, seven samples showed a grade in excess of 5% B_2O_3. From this, SRK concludes that the Company's selection of mineralised samples is quite robust, however; care should be paid to the intervals adjacent to the mineralisation intersections.

The fact that the samples have been analysed using differing analytical analysis methods over time did however show varying precision in detection of B_2O_3 grade. The proportion of the samples which were used in the MRE and analysed by the less suitable aqua regia ICP-MS method constitute only 10% of the MRE database and SRK is of the opinion that this has minimum influence on the estimate overall.

As detailed in the previous section (Section 11), Erin has conducted QA/QC checks with a regular system of CRM's, duplicates and blanks being inserted into the sample stream. SRK's validation checks of this QAQC material suggest that the CRM's are broadly within acceptable reporting limits, the duplicate field samples show a strong correlation to the original sample, and blank samples were reported as showing a low B_2O_3 content.

12.4 Database

The database provided to SRK by Erin was in a Microsoft Access database format. The database contained the information on collar coordinates, downhole deviation data and assay data for 79 drillholes totalling 27,628 meters. Not all of the drillholes in the database contained assay information; four holes drilled in the 1980's and 1990's by Ibar Mines and Ras Borati were undertaken for sterilisation purposes at the flanks of the deposit and are missing assay data. As SRK has decided not to use the drilling data from the the Ibar Mines and Ras Borati campaigns this is not considered material. Based upon a detailed review of the core available from Rio Tinto and existing analytical results SRK found that the data from the holes drilled by Rio Tinto to be reliable and appropriate for the inclusion in geological modelling and the MRE.

The final MRE database included 53 drillholes, with a total length of 19,554.0 m, containing full information on collar position, survey, samples data and lithological data. From these holes, assay information is available for 1,473 samples with a total sampled length of 6,211.68 m. The data includes all Erin assays, comprising 224 samples analysed by the volumetric titration method, 544 samples analysed through the AR ICP-MS methodology and includes 705 historical assays from the Rio Tinto drilling campaigns.

SRK has validated 10% of the data used in the MRE selecting 5 holes from the database at random and checking their records against the original logs and assay results sent by the laboratory. Of the 168 assays from these holes, 14% were found to have incorrect B_2O_3 grades, mostly resulting from a transcription error and lack of conversion from assayed elemental B grade to equivalent B_2O_3 grade. These errors are predominantly in the low grade samples assayed by AR ICP-MS. SRK does not consider these errors to be material to the MRE, though recommends that full validation of the drilling database is conducted as part of any future MRE update.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Colemanite Production

The proposed mine production plan developed for the PEA and presented later in this report lists annual RoM grades ranging from 24.3% B_2O_3 to 32.9% B_2O_3, averaging 27.8% B_2O_3 over the life of mine.

SRK understands that Colemanite is typically marketed at a B_2O_3 content of 40%, with a tolerance on the B_2O_3 content of 5% absolute. SRK therefore believes that it will be necessary to upgrade the RoM ore to at least 35% B_2O_3, in order to meet the lower end of this typical specification range.

A sample of approximately 25 kg of ore from Piskanja, assaying approximately 30% B_2O_3, was sent to the laboratories of SGS in the UK in 2012 for upgrading testwork. The testwork consisted of magnetic and electrostatic separation tests, all conducted dry, with the aim of the testwork being to investigate the potential to both increase the B_2O_3 content, and to reduce the Fe content, into a concentrate. SRK understands that a typical Fe specification for Colemanite for glass production is 0.08% Fe max.

A “magnetic profiling” test was unsuccessful, in that no significant upgrading of the B_2O_3 content, or reduction of the Fe content, was achieved. Electrostatic separation produced the best grades – up to 34.5% B_2O_3 – but the mass yields and B_2O_3 recoveries to these fractions were very low (6-21 % B_2O_3 recovery). Fe was not assayed in the electrostatic separation tests.

The most successful test was a High Intensity Magnetic Separation (“HIMS”) test conducted on a sample crushed to -1 mm and with the -150 μm fraction removed. The results of this test are shown in Table 13-1.

Table 13-1: HIMS Test Results

Stream	Wt %	B_2O_3 Assay (%)	B_2O_3 Dist. (%)	Fe Assay (%)	Fe Dist. (%)
Feed	100	29.4	100	0.44	100
-150 μm Fraction	35.6	29.1	35.2	0.49	39.6
Magnetic Concentrate 1	6.43	5.28	1.15	3.15	46.0
Magnetic Concentrate 2	1.64	16.6	0.92	0.78	2.90
Combined Magnetic Concentrate	8.07	7.58	2.07	2.66	48.9
Non-Magnetics	56.3	32.8	62.7	0.09	11.5

This test produced a concentrate stream (the Non-Magnetics) with an Fe content very close to the target 0.08% specification, although this was accompanied by only a modest upgrading in B_2O_3 content; lower than the 35% B_2O_3 target level.

13.2 Boric Acid Production

A sample of high grade ore from Piskanja was sent to Societá Chimica Larderello ("SCL") in Italy to test its potential for Boric Acid production. The as-received sample assayed 42.3% B_2O_3.

As Colemanite is not water soluble, the production of Boric Acid from Colemanite requires leaching using sulphuric acid.

The test was reported as being successful, and the Boric Acid content of the product was reported as 100.9%.

A bulk (200 t) sample of ore from the Pobrdje mine, near to Piskanja, was sent to a potential off-taker / project partner, who tested it in their commercial Boric Acid plant. The Pobrdje material was reported to have behaved is a similar manner to the Colemanite imported from Turkey that this plant currently processes.

13.3 Conclusions and Recommendations

While the testwork conducted to date to test the suitability of the Piskanja ore to act as a feed for the production of Boric Acid has been positive, very little testwork has been conducted to determine the potential to upgrade the Piskanja ore for the production of a saleable Colemanite concentrate.

Significant further testwork is therefore required in order to develop a viable, and successful, process flowsheet for upgrading the Piskanja ore to a marketable Colemanite concentrate, both in terms of the B_2O_3 content and the Fe content. To this end, SRK notes that while the HIMS testwork reported by SGS, and shown in Table 13-1, reduced the Fe content to close to the target level, the level of B_2O_3 upgrading was insufficient, and just under one-third of the material could not be processed, as it was too fine for the selected unit process. Therefore, while this process showed some promise, it may not prove to be an appropriate basis on which to develop a technically and commercially viable process solution.

SRK understands that arsenic (As) is another key deleterious element in a potential Colemanite concentrate. The behaviour of As has not been reported in the testwork conducted to date; therefore the levels and deportment of As should be investigated in any future beneficiation testwork programmes.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The Mineral Resource Estimate (MRE) presented herein covers the whole of the Piskanja Project under Exploration Licence #1934 and is based on diamond drilling conducted by Erin and historical Rio Tinto drilling.

The database used for the Piskanja MRE was audited by SRK who is of the opinion that the drilling information provided by the Client is sufficiently reliable to support an MRE to be undertaken and reported in conformity with the CIM Guidelines.

This section of the report describes the methodology used to derive SRK's resource estimate, summarises the key assumptions made by SRK and presents the estimate itself.

14.2 Deposit modelling

14.2.1 Available Data

As already commented while the drilling database supplied by the Client to SRK contained the data obtained during four different drilling campaigns spanning from 1989 to 2012, SRK only used the data relating to the drilling completed by Rio Tinto and Erin in deriving the MRE presented here.

The final MRE database as used therefore included data for 53 drillholes, with a total length of 19,554 m, containing full information on collar position, survey, samples data and lithological data. From these holes, assay information is available for 1,473 samples with a total sampled length of 6,211.68 m. The data includes all Erin assays, comprising 224 samples analysed by the volumetric titration method, 544 samples analysed through the AR ICP-MS methodology as well as 705 historical assays from the Rio Tinto drilling campaigns.

Figure 14-1 and Figure 14-2 show a histogram and probability plot of all raw B_2O_3 assays in the assay database utilised for the MRE. As shown, the main population of data has an approximate mean grade of 42% B_2O_3 with the background mineralisation break being at approximately 5% B_2O_3.

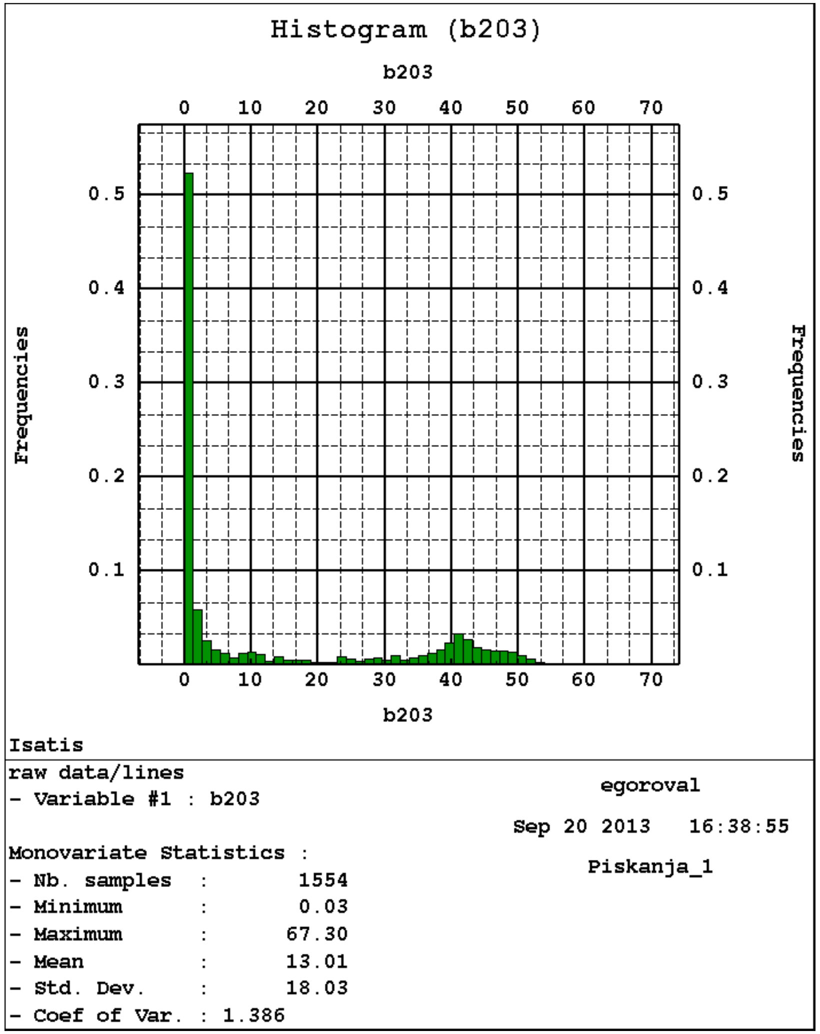


Figure 14-1: Histogram of raw Assays

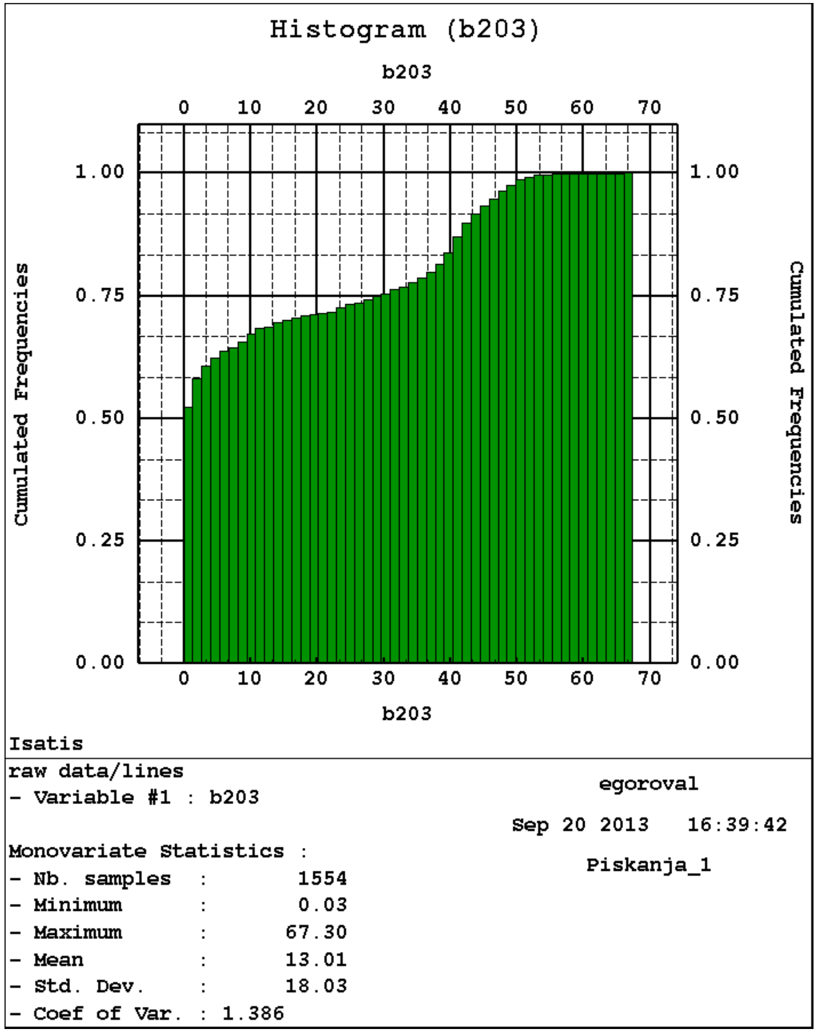


Figure 14-2: Probability plot of raw Assays

14.2.2 Mineralisation Zone Modelling

The mineralised domain wireframes created by SRK to constrain its MRE were based on lithology logs, assay results and knowledge of the relationship between adjacent mineralised zones. The georeferenced geological plans and cross sections that were provided by Erin geologists are shown in Figure 14-3 below.

SRK applied the following guidelines to model a series of separate sub-parallel gently dipping mineralisation domains:

- A grade cut off of 5.0% $B_2 0_3$ was applied to define the hangingwall and footwall contacts;
- The minimum domain thickness was set at 0.5 m; and
- The maximum thickness of barren rock interburden within a domain was set at 1.0 m.

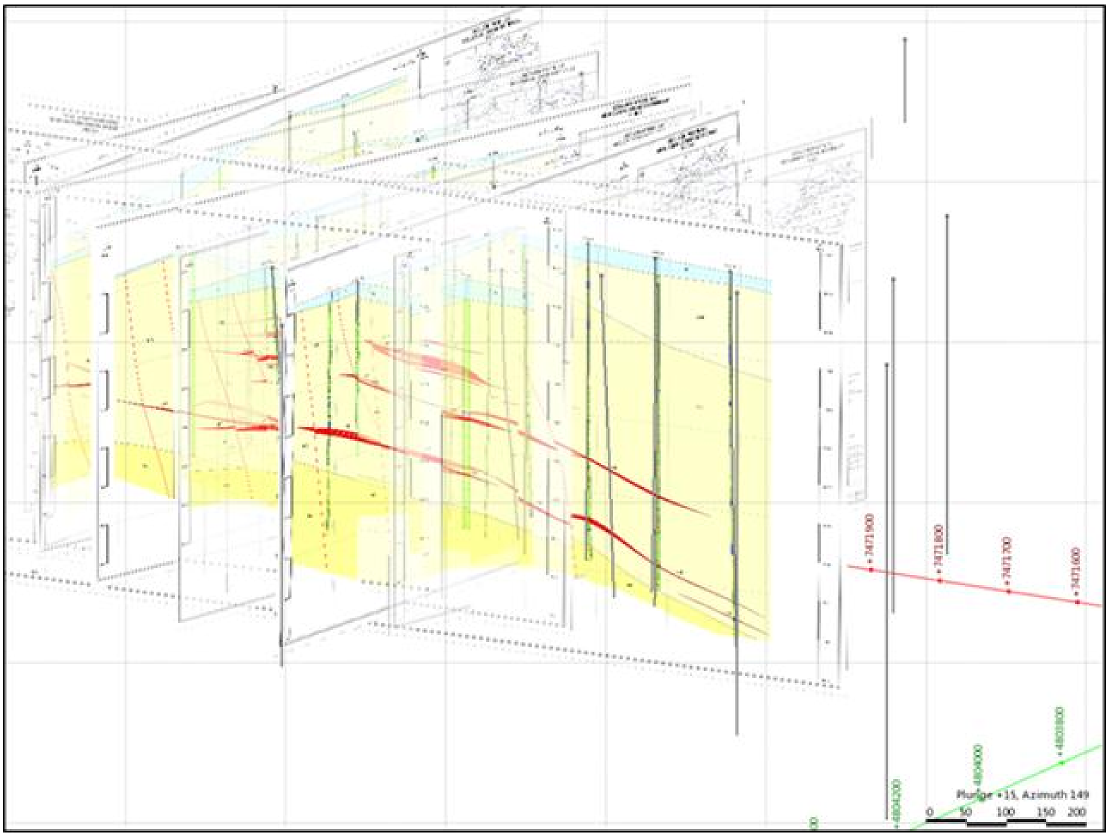


Figure 14-3: 3D view of Geological Cross-sections and along strike Cross-sections (Leapfrog, view along the azimuth 149)

To assist with the modelling process, the spatial distribution of B_2O_3 grade was assessed in Leapfrog software using a structural trend determined from the cross sections and through a statistical analysis which showed that a cut off of 5% B_2O_3 provided continuous zones of mineralisation. To maintain continuity to the mineralisation, it was required on occasion to include non-mineralised intersections, up to a maximum thickness of 1 m within the modelled zones. Figure 14-4 shows leapfrog shells generated at 5% and 10% B_2O_3, with greater continuity to the mineralised shell being clear when applying a cut off to 5% B_2O_3.

To create the final domain wireframes, shown in Figure 14-5, SRK combined the mineralisation shells with the lithological hanging wall and footwall surfaces. The resulting 3D mineralised zone wireframes were reviewed by SRK and Erin geological staff and subsequently amended (where required) and approved as providing an appropriate representation of the mineralisation.

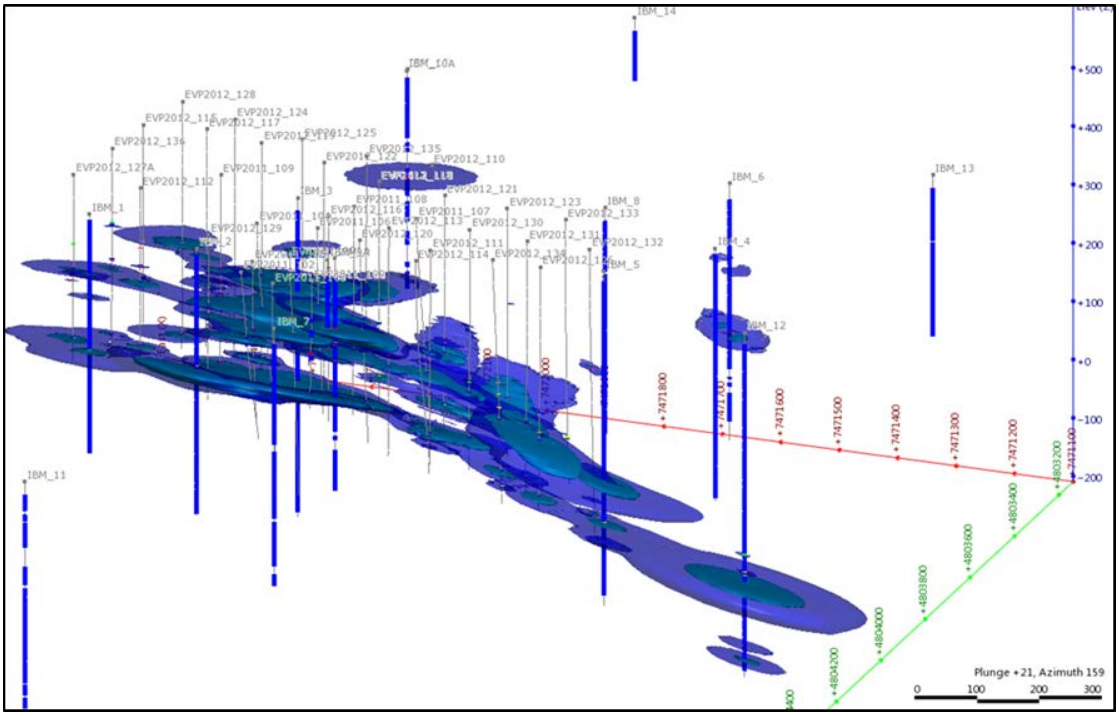


Figure 14-4: Shells that correspond to 5% B_2O_3 grade cut-off (blue) and 10% B_2O_3 grade cut-off (green) (Leapfrog, view along the azimuth 159)

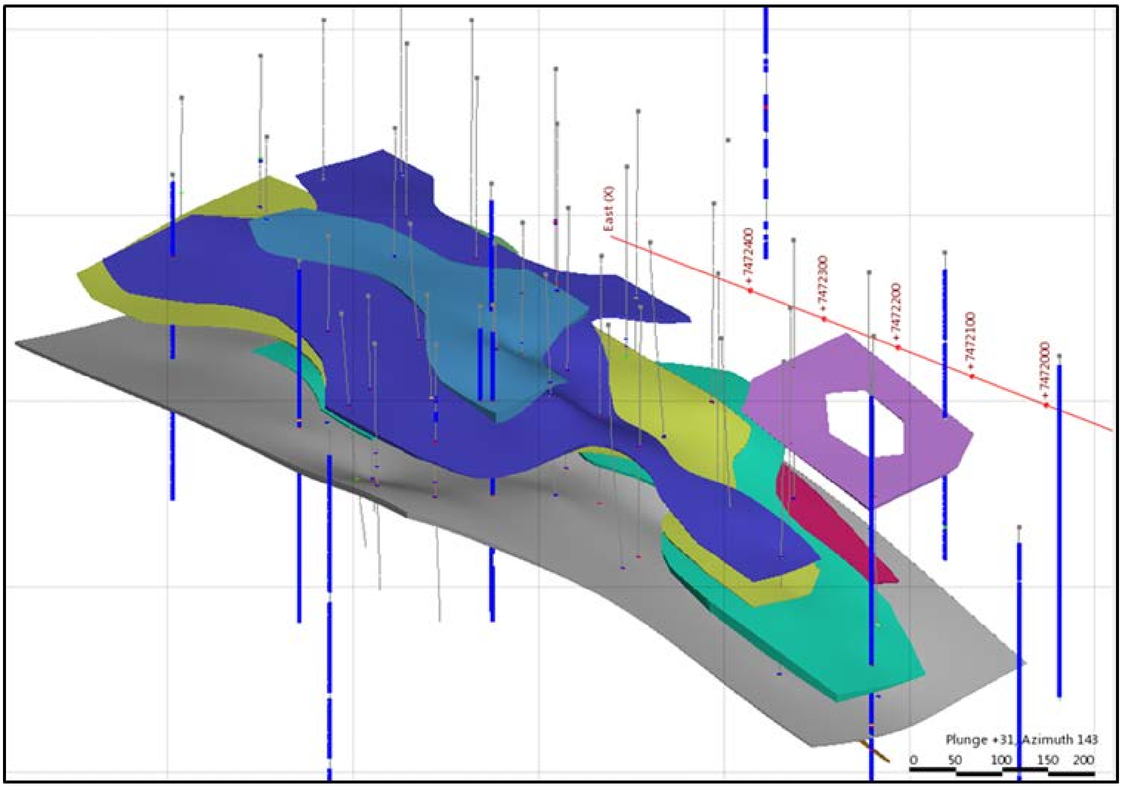


Figure 14-5: General view of the mineralisation zones of Piskanja (Leapfrog, view along the azimuth 143, dip 31)

In total, SRK modelled 10 mineralisation zones with Zone 1 being the deepest zone and Zone 10 being the uppermost zone. The mineralised units dip to the southwest at an angle of approximately 18º. Table 14-1 summarises the key parameters for the individual zones including the average thickness, average B_2O_3 grade and volumes.

The average thickness of the mineralised zones ranges from 0.3 to 22.3 m, with an overall average thickness of 2.9 m. The average B_2O_3 grade is 30.4% for all the mineralised zones with a total volume of 5.37 Mm^3. As shown, 72% of the volume is located within Zones 1, 2 and 3.

Table 14-1: Summary of mineralised zone properties

ZONE	Thickness, m			Grade B_2O_3, %			Volume, m³
	Min	Max	Average	Min	Max	Average	
1	0.5	9.3	4.8	15	50.2	32.9	2 010 900
2	1	15.0	3.4	2.8	55.1	31.4	762 750
3	0.4	22.3	5.1	0.8	48.1	32.4	1 077 000
4	0.5	15.1	4.3	31	50.4	38.5	271 120
5	0.3	1.8	1.0	6.4	41.5	28.3	86 589
6	0.4	15.6	2.5	5.7	45.1	16.1	685 600
7	1.1	8.7	3.7	7.3	28.8	18.6	212 200
8	0.6	2.3	1.6	9.8	32.1	18.4	76 211
9	0.6	2.7	1.6	17.7	53.1	32	129 540
10	0.5	3.0	1.6	10.7	23.9	16.4	59 866
TOTAL	**0.3**	**22.3**	**2.9**	**0.8**	**55.1**	**30.4**	**5 371 776**

14.2.3 Structural Assessment

As part of the modelling process, SRK analysed the Piskanja structural interpretation assumed by the geological cross-sections and maps provided by Erin. In addition, SRK carried out a visual check of all the core photos obtained during the 2011-2012 drilling programme. Notwithstanding the fact that faulting was observed in the core and that there may be some faulting present in practice, the existing drilling grid is not sufficient for the purpose of constructing a robust tectonic model of the deposit and it was decided for the purpose of this estimate to interpret all the mineralised zones as consistent bodies with slightly varying dips.

14.3 Compositing

Data compositing is typically undertaken to reduce the inherent variability that exists within the sample population and to generate sample lengths more appropriate to the scale of the mining operation envisaged. It is also a necessary part of the estimation process, as the techniques used assume that all samples are of equal weighting, and they should therefore be of equal length.

SRK therefore analysed the average length of the drillhole samples in order to determine appropriate composite length. The histogram presented in Figure 14-6 shows that the most common sample length is 1.0m.

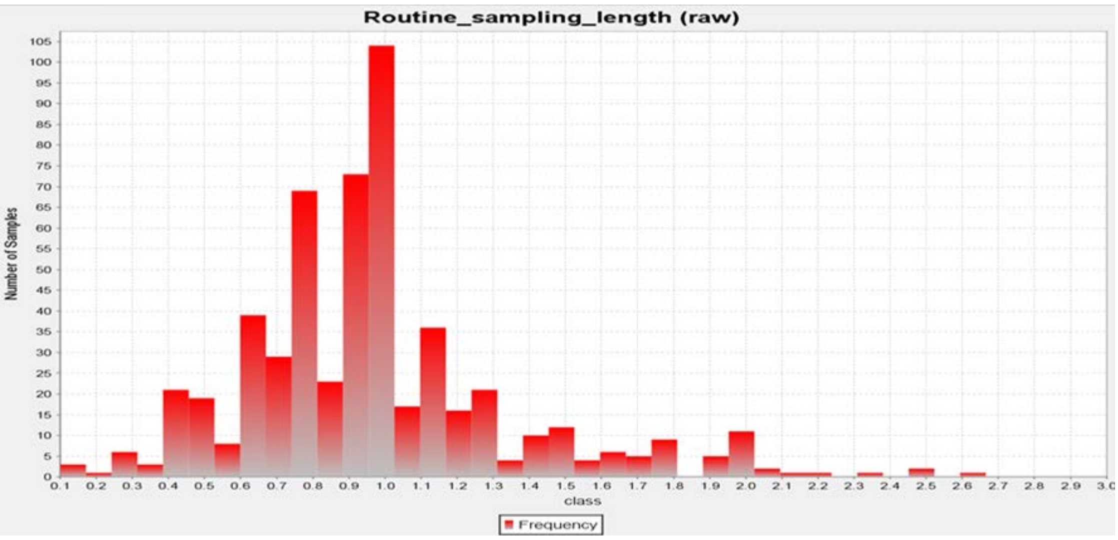


Figure 14-6: Ordinary sample length distribution histogram

SRK tested a minimum composite length of 0.5 m and a maximum composite length of 2 m. Table 14-2 shows the summary statistics for the composite lengths evaluated. Given the similarities observed in the mean grades of the various composite lengths tested, a 1 m composite length was selected.

It is normal practice to then discard or ignore short remnant composites that are generated in the downhole compositing process at the edges of the mineralisation to avoid a bias in the estimation and in this case it was decided that all samples less than 0.5 m should be discarded from the composite drillhole file and should not be used in the grade interpolation for this reason.

Table 14-2: Summary statistics of raw samples versus composite samples

Variable B_2O_3, %	Raw Samples	COMPOSITE LENGTH			
		0.5 m	0.8 m	1.0 m	2.0 m
Number of samples	537	1153	743	606	519
Minimum value	0.1	0.1	0.1	0.1	0.1
Maximum value	67.3	65.4	64.2	65.4	64.0
Mean	29.5	28.9	28.7	28.7	28.7
Coefficient of variation	0.5	0.6	0.5	0.5	0.5

14.4 Statistical Analysis

Table 14-3 summarises the 1 m composite domain statistics for B_2O_3 grade within the ten modelled mineralised zones while Figure 14-7 shows the normal and log-normal histograms themselves. The summary statistics show that the coefficient of variation (CoV) for most of the zones is generally low (i.e. <1.0), which indicates a reasonably low degree of variability within each domain. However, it is clear that the mineralised zones also show a scattered population, being neither normal nor log-normal in distribution. This is in part due to the inclusion of low grade intervals to improve the grade continuity and the cut off of 5% B_2O_3 being used. From a geological perspective, the scattered populations could be explained by the existence of vein-type and massive mineralisation.

SRK investigated the possibility of generating separate high grade domains but found this problematic due to the erratic distribution.

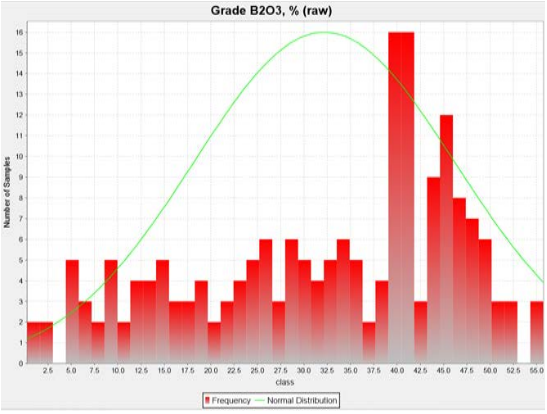
No grade capping was applied to the drillhole database as it was not considered necessary following the statistical study.

Table 14-3: Summary Statistics per mineralisation zone (1m composites)

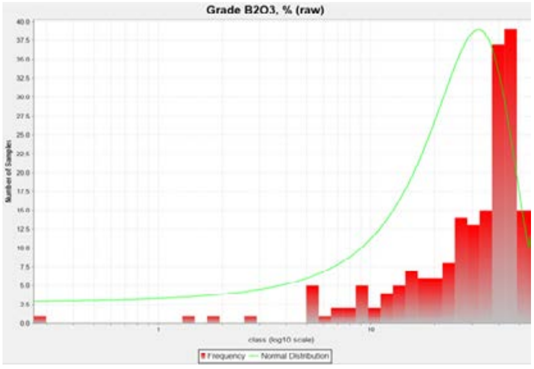
Mineralised Zone	Composites 1.0 m									
	1	2	3	4	5	6	7	8	9	10
Variable	**B_2O_3 %**	**B_2O_3%**	**B_2O_3, %**	**B_2O_3, %**	**B_2O_3, %**	**B_2O_3, %**	**B_2O_3, %**	**B_2O_3, %**	**B_2O_3, %**	**B_2O_3, %**
Number of drillholes	37	21	25	9	8	26	5	5	3	4
Number of samples	175	72	133	46	8	73	21	8	8	8
Minimum value	0.3	0.3	0.1	0.1	6.4	0.8	4.8	5.5	8.8	10.7
Maximum value	55.7	60.6	52.9	52.5	40.8	44.8	39.9	29.8	48.8	23.9
Mean	32.9	30.8	32.3	36.3	26.4	15.6	16.9	18.0	26.7	16.6
Median	35.8	36.8	39.9	40.3	27.3	11.6	10.3	12.8	25.5	14.3
Geometric Mean	27.7	23.9	26.1	29.5	21.5	12.1	13.4	15.8	21.7	16.0
Variance	195.1	227.4	209.4	190.9	168.9	136.8	135.7	69.2	235.7	21.0
Standard Deviation	14.0	15.1	14.5	13.8	13.0	11.7	11.6	8.3	15.4	4.6
Coefficient of variation	0.4	0.5	0.4	0.4	0.5	0.8	0.7	0.5	0.6	0.3
Skewness	-0.6	-0.6	-0.7	-0.9	-0.6	1.4	0.8	0.1	0.2	0.1
Kurtosis	-0.8	-1.0	-0.9	0.0	-1.3	0.7	-0.8	-1.4	-1.5	-1.4

Mineralisation Zone 1

Normal Distribution

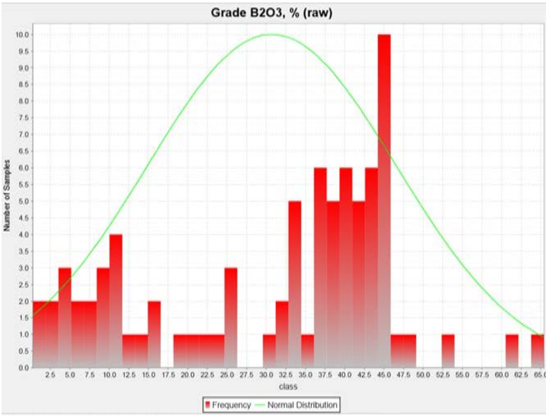


Lognormal Distribution

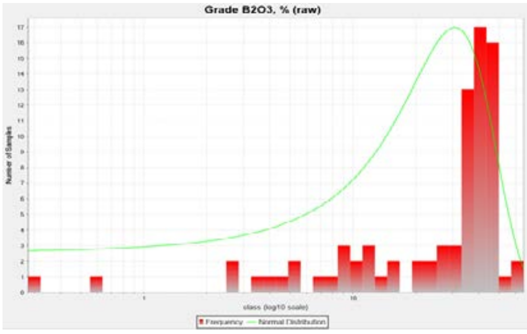


Mineralisation Zone 2

Normal Distribution

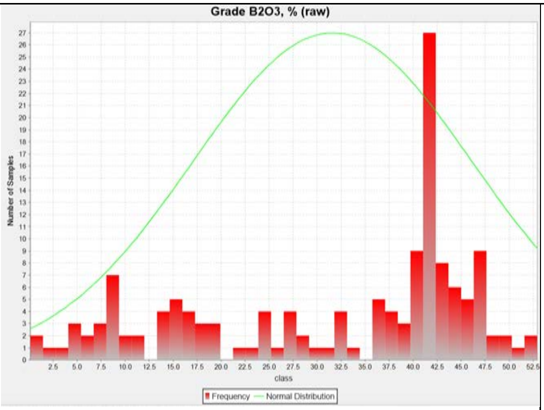


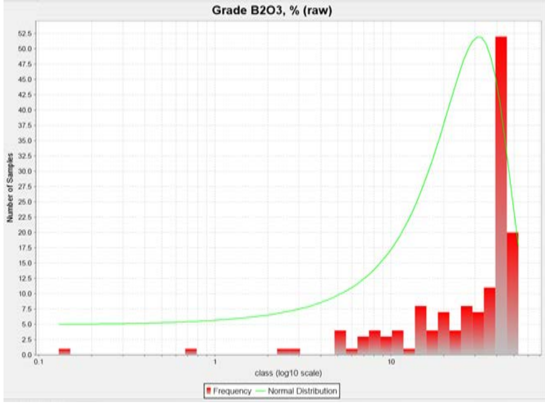
Lognormal Distribution



Mineralisation Zone 3

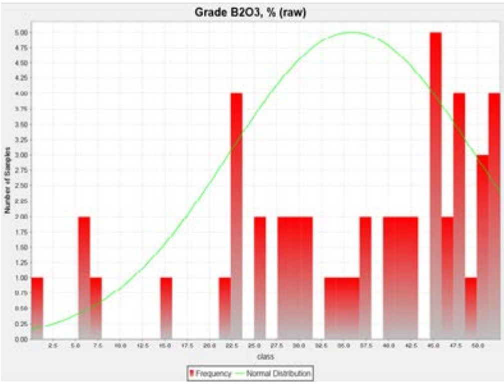
Normal Distribution	Lognormal Distribution



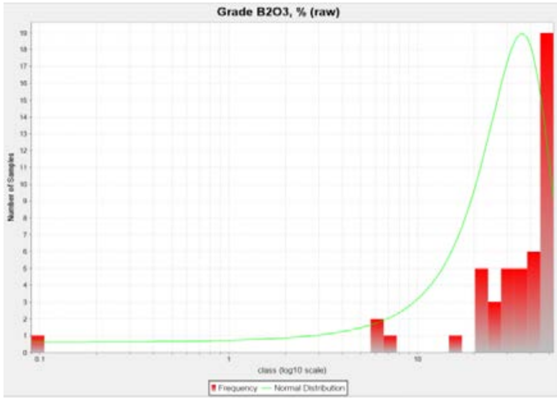


Mineralisation Zone 4

Normal Distribution

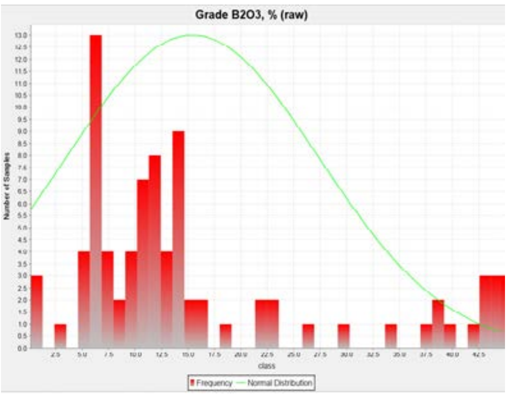


Lognormal Distribution



Mineralisation Zone 6

Normal Distribution



Lognormal Distribution

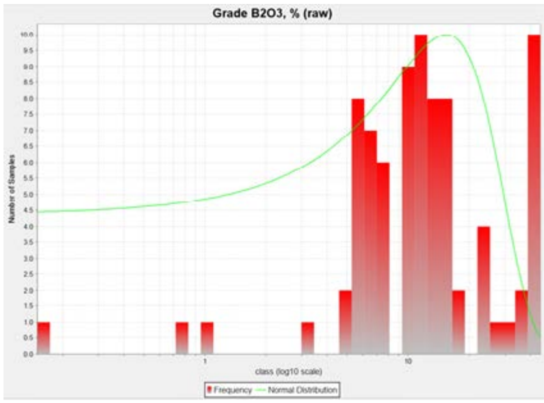


Figure 14-7: Normal (left side) and log-normal (right side) B_2O_3 histograms for major mineralised zones

14.5 Geostatistical Analysis

14.5.1 Semi-variogram Modelling

For the purpose of the geostatistical analysis, all of the composited samples within all of the mineralised zones were combined into a single zone. Combining the zones improved the variography due to limited samples present within the individual zones.

A downhole experimental semi-variogram was produced for B_2O_3 using a 1 m lag to allow the nugget to be determined (Figure 14-8). An omnidirectional variogram was then produced using a 100 m lag and the nugget fixed using the downhole variogram (Figure 14-9). An angular tolerance of 10° was used to improve the quality of the variogram and the assay data was masked to include only those samples over 20% B_2O_3.

The variogram produced (Figure 14-9) showed reasonable structure, allowing a reliable variogram model to be produced. The nugget and ranges were easily generated, providing an appropriate level of confidence in terms of both the short scale and longer range grade continuity.

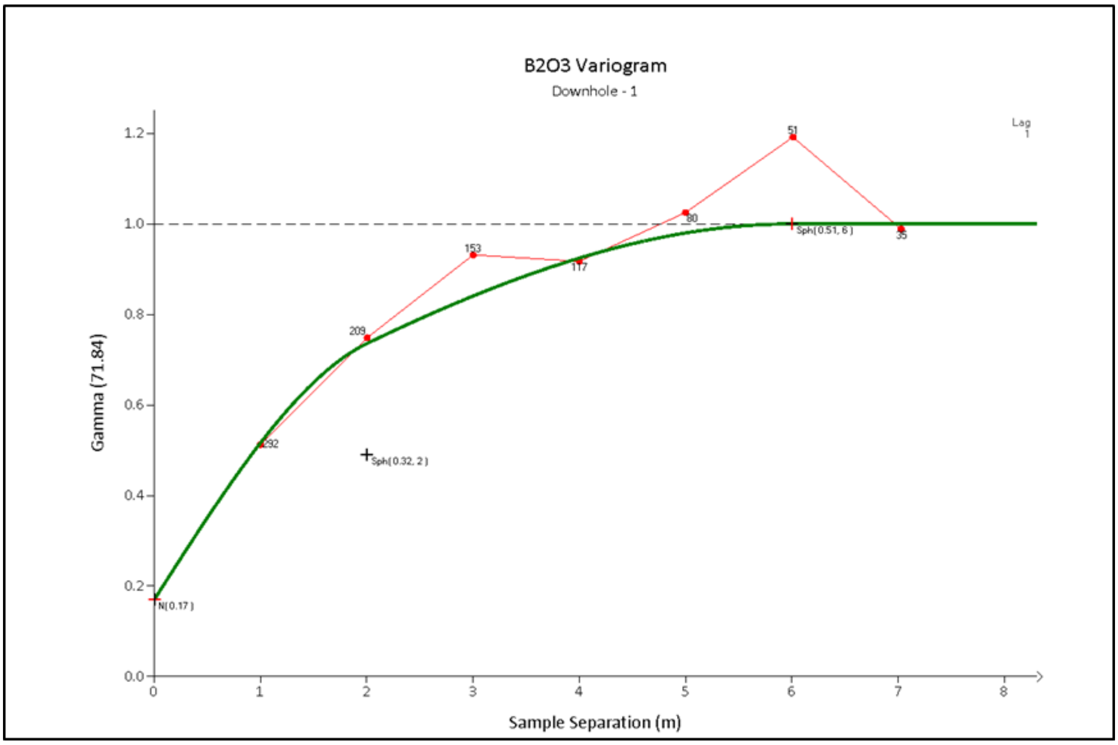


Figure 14-8: Piskanja B_2O_3 downhole Semi-variogram

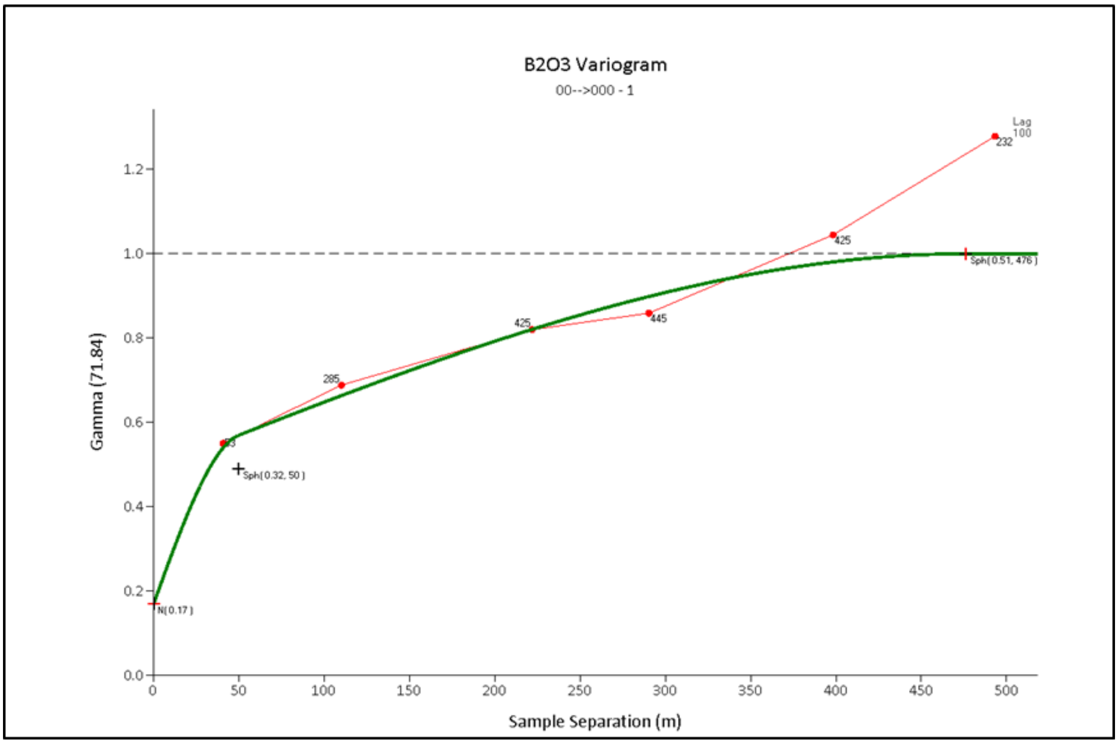


Figure 14-9: Piskanja B_2O_3 Omni directional Semi-variogram

The semi-variogram parameters derived were as follows:-

Nugget:	0.17
1st Structure Range:	50 m
1st Structure variance:	0.32
2nd Structure Range:	476 m
2nd Structure Variance:	0.51

14.5.2 Quantitative Kriging Neighbourhood Analysis (QKNA)

To supplement the semivariogram analysis and better define the ideal search parameters used in the interpolation, Quantitative Kriging Neighbourhood Analysis ("QKNA") was undertaken on the data set.

QKNA, as presented by Vann et al (2003), is used to refine the search parameters in the interpolation process to help ensure 'conditional unbiasedness' in the resulting estimates. 'Conditional unbiasedness' is defined by David (1977) as "…on average, all blocks Z which are estimated to have a grade equal to Zo will have that grade". The criteria considered when evaluating a search area through QKNA, in order of priority, are (Vann et al 2003):

- the slope of regression of the 'true' block grade on the 'estimated' block grade;
- the weight of the mean for a simple kriging;
- the distribution of kriging weights, and proportion of negative weights; and
- the kriging variance.

Under the assumption that the variogram is valid, and the regression is linear, the regression between the 'true' and 'estimated' blocks can be calculated. The actual scatter plot can never be demonstrated, as the 'true' grades are never known, but the covariance between 'true' and 'estimated' blocks can be calculated. The slope of regression should be as close to one as possible, implying conditional unbiasedness. If the slope of regression equals one, the estimated block grade will approximately equate to the unknown 'true' block grades (Vann et al 2003).

During Ordinary Kriging ("OK"), the sum of the kriging weights is equal to one. When Simple Kriging ("SK") is used, the sum of kriging weights is not constrained to add up to one, with the remaining kriging weight being allocated to the mean grade of the input data. Therefore, not only the data within the search area is used to krige the block grade, but the mean grade of the input data also influences the final block grade. The kriging weight assigned to the input data mean grade is termed "the weight of the mean". The weight of the mean of a SK is a good indication of the search area as it shows the influence of the Screen Effect. A sample is 'screened' if another sample lies between it and the point being estimated, causing the weight of the screened sample to be reduced. The Screen Effect is stronger when there are high levels of continuity denoted by the variogram. A high nugget effect (low continuity) will allow weights to be spread far from a block in order to reduce bias (Vann et al 2003).

The weight of the mean for a SK demonstrates the strength of the Screen Effect the larger the weight of the mean, the weaker the Screen Effect will be. The general rule is that the weight of the mean should be as close to zero as possible. QKNA is a balancing act between maximising the slope of regression, and minimising the weight of the mean for a SK (Vann et al 2003). The margins of an optimised search will contain samples with very small or slightly negative weights. Visual checks of the search area should be made in order to verify this. The proportion of negative weights in the search area should be less than 5% (Vann et al 2003).

QKNA provides a useful technique that uses mathematically sound tools to optimise a search area. It is an invaluable step in determining the correct search area for any estimation or simulation exercise.

Interpolation Parameters

Multiple neighbourhood scenarios were run on the zones (search distances, number of samples, etc) with the scenarios being tested by running the estimation in CAE Datamine Studio 3 software on the specific zones modelled and by utilisation of the 'KNA' function in the Supervisor geostatistical software package. All zones used a search ellipse generated from the results of the variography. The number of blocks filled in the neighbourhood run was checked to ensure that an adequate number of blocks were filled ensuring that meaningful results were generated.

The QKNA process was run to generate the slope of regression results using the chosen search parameters.

Table 14-4 shows the final search ellipse dimensions and sample numbers used for the first pass interpolation, that were selected based on the QKNA studies. Traditional dips and dip directions of the ellipse are not shown due to the use of dynamic anisotropy in the interpolation.

Table 14-4: Post QKNA Interpolation Parameters

Zone	Ellipse Ranges			Max Samples per drillhole	Min Samples	Max Samples
	Y	X	Z			
Mineralised bodies	300	300	12	4	8	20

Figure 14-10 shows the probability plot of the slope of regression results using the interpolation parameters selected. In summary, a proportion of the estimated blocks show a slope of regression value in excess of approximately 0.95 with a gradual decrease in slope. The areas above 0.95 represent areas of the estimated block model where the data is of a sufficient spacing to provide a reasonable level of confidence in the quality of the estimation results due to the blocks being relatively well informed with available data.

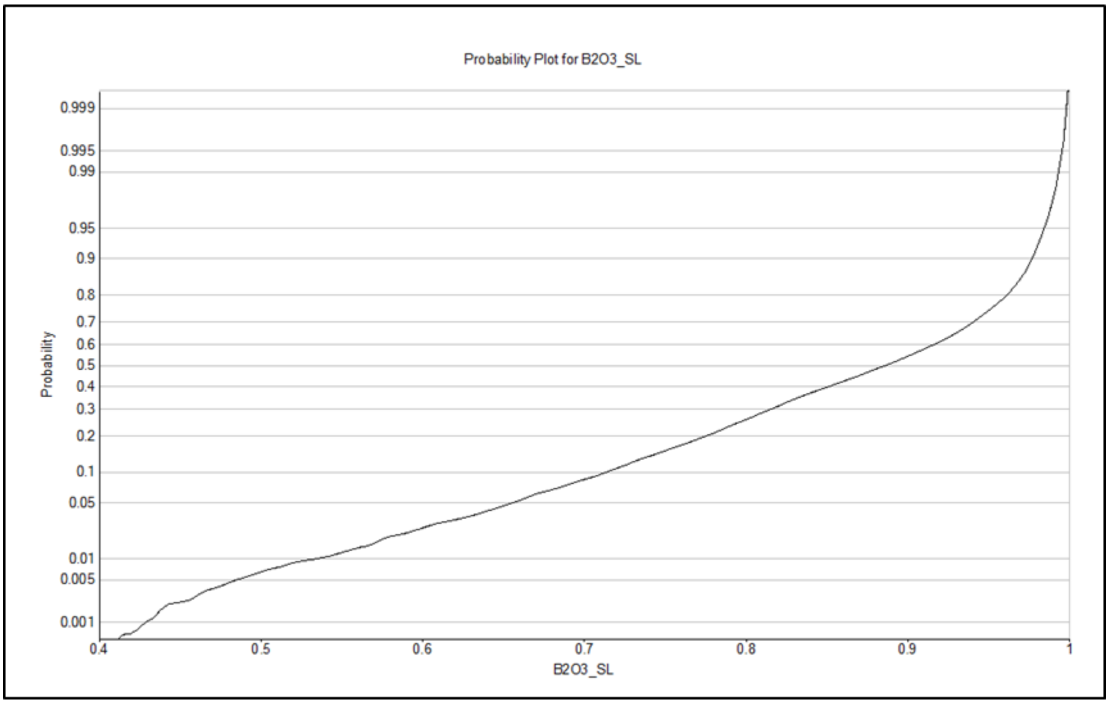


Figure 14-10: Probability Plot of Slope of Regression Values

Figure 14-11 illustrates the slope of regression values for Zone 1 (for ease of viewing) coloured by the splits identified in the probability plot in Figure 14-10 and in relation to the Zone 1 drillholes, whereby high slope of regression values are evident in areas of well-informed data. As the estimated blocks become more distant from the available sample data, the slope of regression values are seen to decrease. This is due to the lack of sample support available to the block.

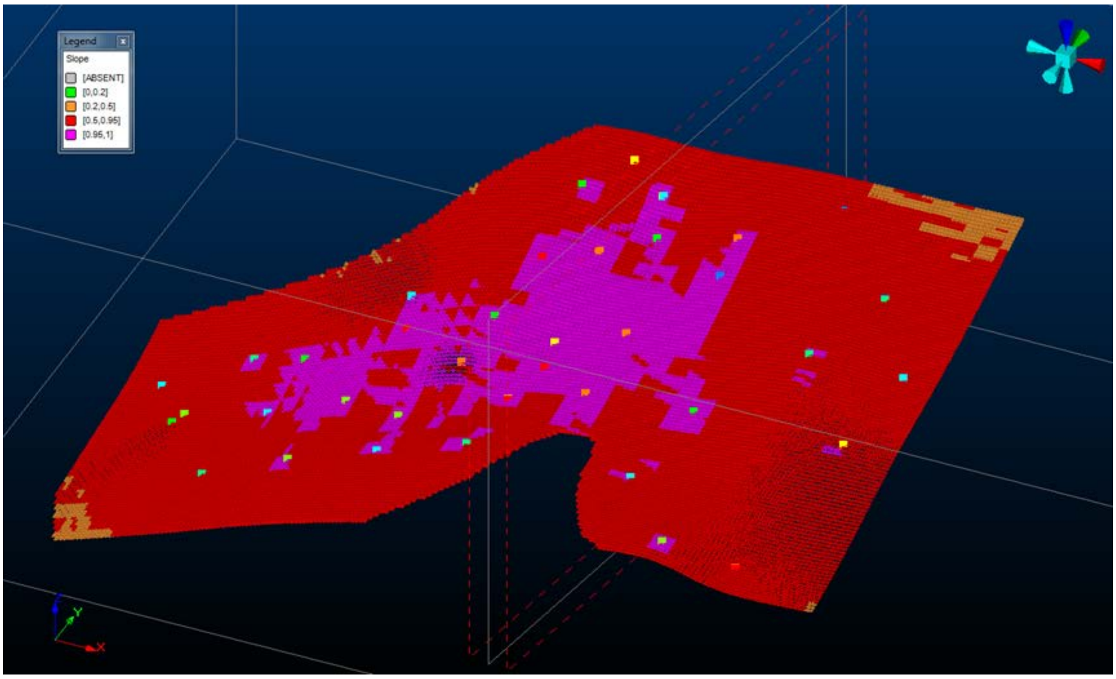


Figure 14-11: Slope of Regression Distribution around Well Informed Blocks at Piskaja, Looking North; Pink > 0.95, Red 0.5 to 0.95, Orange 0.2 to 0.5, Green < 0.2 (SRK, 2013)

14.6 Grade Interpolation

A single, non-rotated block model was created encompassing all of the modelled zones using a block size of 25 m (Y) by 25 m (X) by 2 m (Z). Table 14-5 summarises the block model parameters.

Table 14-5: Block model framework

ORIGIN		NUMBER OF BLOCKS		BLOCK SIZE (m)	
X	7471400	X	60	X	25
Y	4803700	Y	36	Y	25
Z	-90	Z	245	Z	2

B_2O_3 grade was interpolated into the block model using OK and the interpolation parameters as finalised following the QKNA assessment. The same parameters were used for each zone but only samples that fall within the given mineralised zone were used in the interpolation of that particular zone.

Based on the density data for the mineralised zones and host rocks obtained by Erin and commented upon earlier in this report, SRK assigned a density of 2.287 t/m^3 to all the blocks inside the mineralised zones.

The search ellipse parameters were determined through the QKNA tests undertaken. The dip and rotation of the ellipse mirrors the overall dip and strike of the individual domains. That said, in order to provide a continuous estimation and honour the geological structure and gentle along strike changes in strike orientation observed, it was decided to use dynamic anisotropy in the estimation process. Dynamic anisotropy uses angle data generated from the mineralisation wireframe to assign dip and dip direction to every block in the model. The search ellipse is rotated upon estimation of the block by honouring the associated dip and dip direction of that block.

Three interpolation runs were undertaken. The first pass used the parameters determined through the QKNA testwork. The second run doubled the search ellipse and decreased the number of samples required for all domains. The final run multiplied the first pass search by 10 and reduced the minimum number of samples required. The final pass was designed to estimate any blocks not estimated in the first two passes.

Prior to the interpolation, the ellipse was visualised in Datamine Studio 3 with the dip and dip direction being controlled by the dynamic anisotropy process. The effect of using dynamic anisotropy on the rotation of the search ellipse is shown in Figure 14-12.

Table 14-6 shows the number of blocks allocated grades during each estimation pass for all mineralised domains. As shown, a high percentage of blocks in all zones have been estimated in run one, this being the interpolation based on the most appropriate set of estimation parameters and in line with the results of the geostatistical studies undertaken.

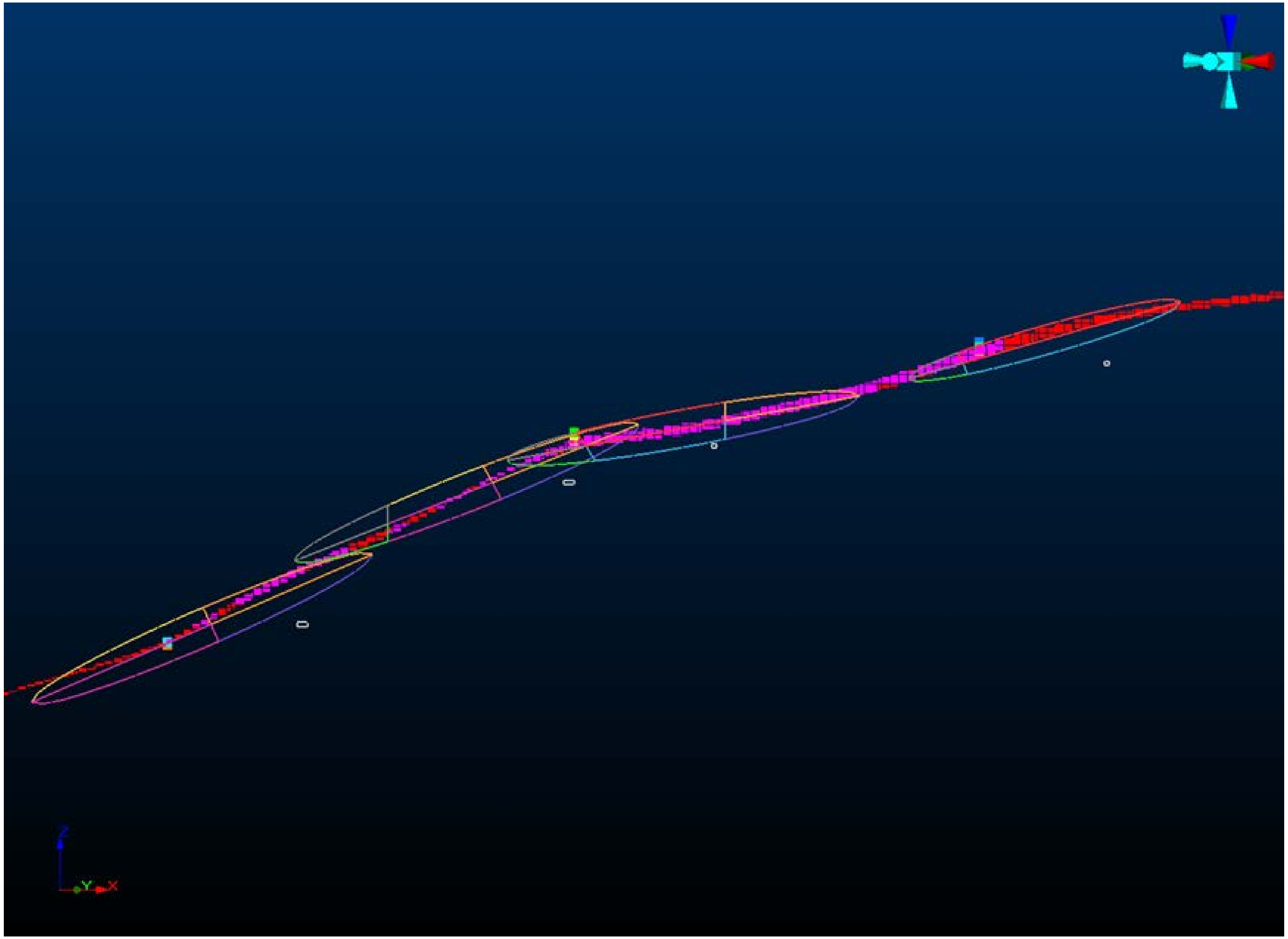


Figure 14-12: Visual Validation of search ellipses through the application of dynamic anisotropy, Zone 1,Looking NW (SRK, 2013)

Table 14-6: Blocks filled in each estimation run

ZONE	Estimation Pass	Number of Samples	% Blocks Filled	Total
1	**1**	**18**	**96**	**100%**
	2	19	4	
	3	20	0	
2	1	17	94	100%
	2	16	6	
	3	20	0	
3	1	19	95	100%
	2	16	5	
	3	20	0	
4	1	16	89	100%
	2	19	11	
5	2	7	100	100%
	3	8	0	
6	1	18	95	100%
	2	19	5	
7	1	11	99	100%
	2	13	1	
8	2	7	1	100%
9	2	8	97	100%
	3	8	3	
10	1	8	96	100%
	2	8	3	
	3	8	1	

14.7 Validation

The block model has been validated using the following techniques:

- visual inspection of block grades in plan and section and comparison with drillhole grades;
- comparison of global mean block grades and sample grades within mineralised domains;
- the generation of SWATH plots; and
- the use an Inverse Distance Weighting interpolation to generate alternative block grades to compare with the OK interpolated grades

Figure 14-13 shows an example of the visual validation checks and highlights the correspondence between the block B_2O_3 grades and the sample B_2O_3 grades. The image also shows that dynamic anisotropy has worked effectively with the grade profile following the general dip of the mineralised zones.

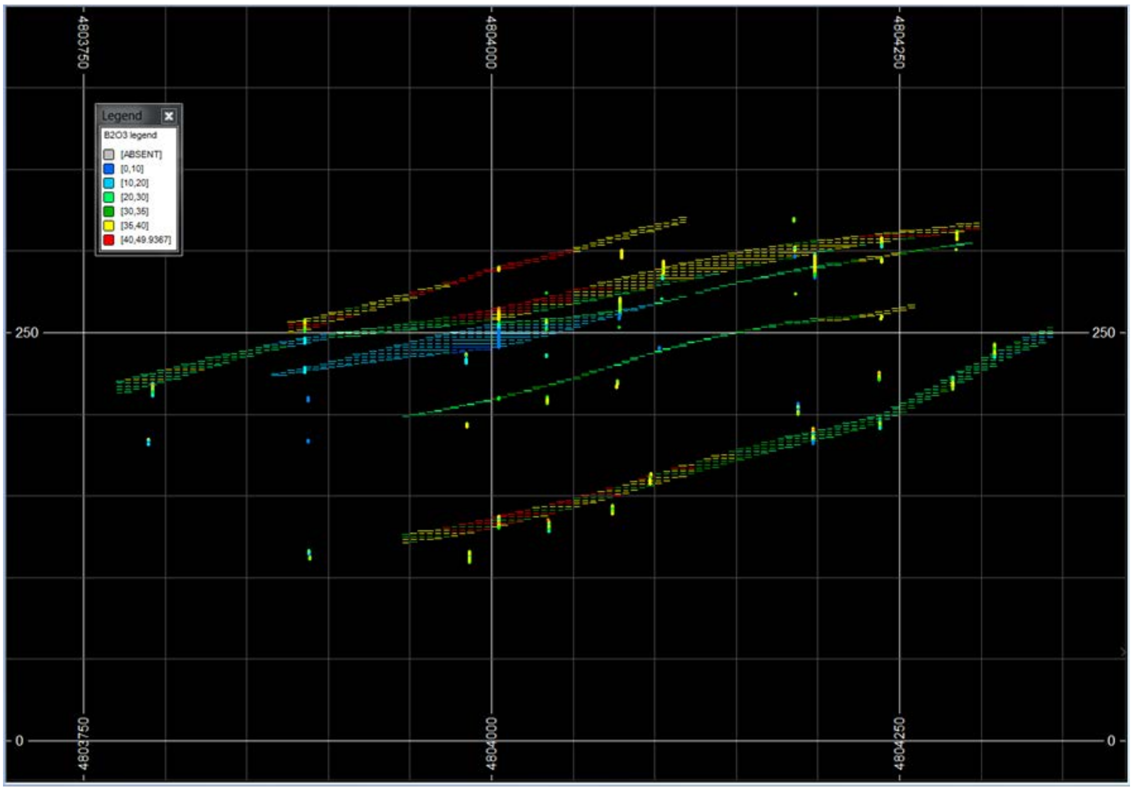


Figure 14-13: Cross section showing visual validation of block grades and sample grades – looking West (SRK, 2013)

Global mean grade comparison

For the interpolated domains, the global block means have been compared with the sample means as shown in Table 14-7.

Overall, SRK is confident that the interpolated grades are a reasonable reflection of the available sample data.

Table 14-7: Comparison of block and sample mean grades

Zone	Field	Composite Mean Grade (%)	Block Mean grade (%)	Difference (absolute)
1	B_2O_3	32.2	31.7	0.5
2	B_2O_3	30.8	30.3	0.5
3	B_2O_3	32.3	32.2	0.1
4	B_2O_3	36.3	37.3	-1.0
5	B_2O_3	26.4	30.2	-3.8
6	B_2O_3	15.5	18.4	-2.9
7	B_2O_3	16.9	15.4	1.5
8	B_2O_3	18.0	18.8	-0.8
9	B_2O_3	26.7	26.9	-0.2
10	B_2O_3	16.6	17.1	-0.5

SWATH Plots

This method comprised the comparison of the input composite samples with the block model grades within a series of coordinates. The results were then displayed on graphs to check for visual discrepancies between grades. Figure 14-14 to Figure 14-16 shows the results for the grades interpolated using different methods of interpolation.

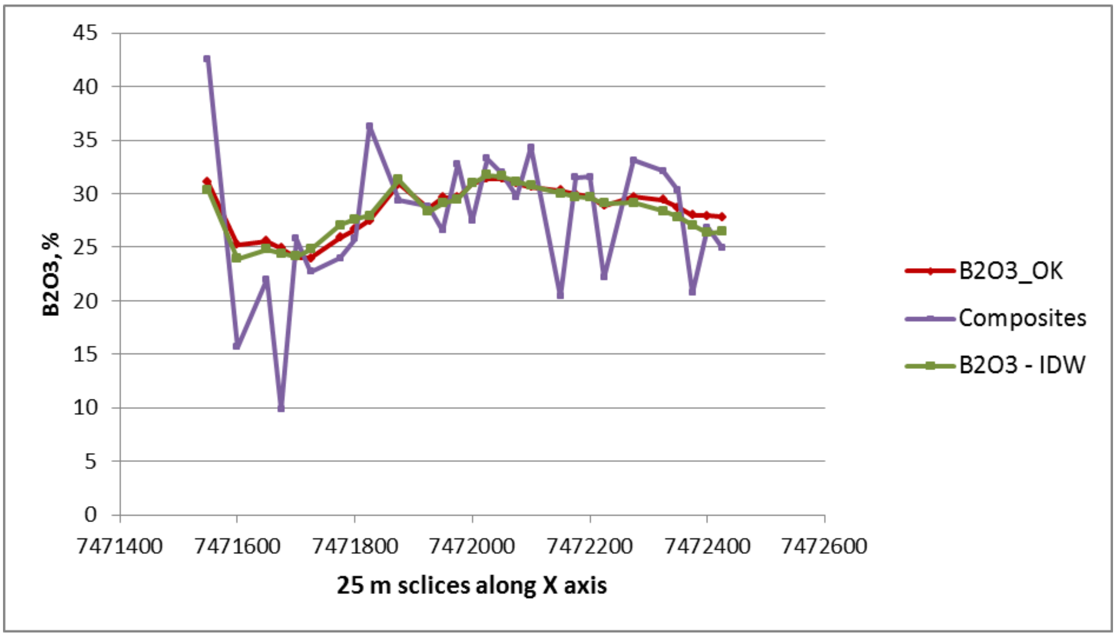


Figure 14-14: B_2O_3 sectional comparison in the X direction

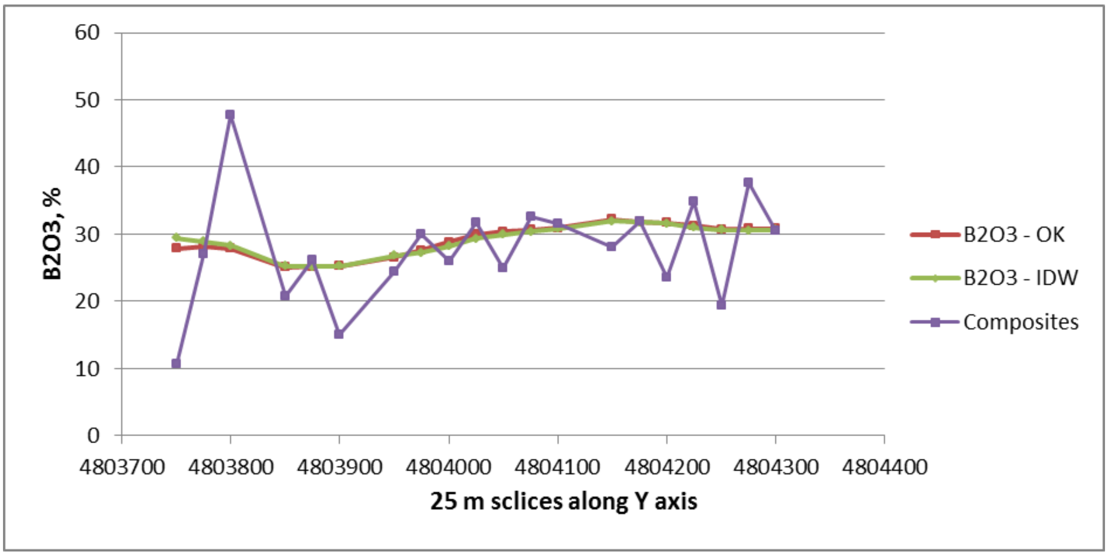


Figure 14-15: B_2O_3 sectional comparison in the Y direction

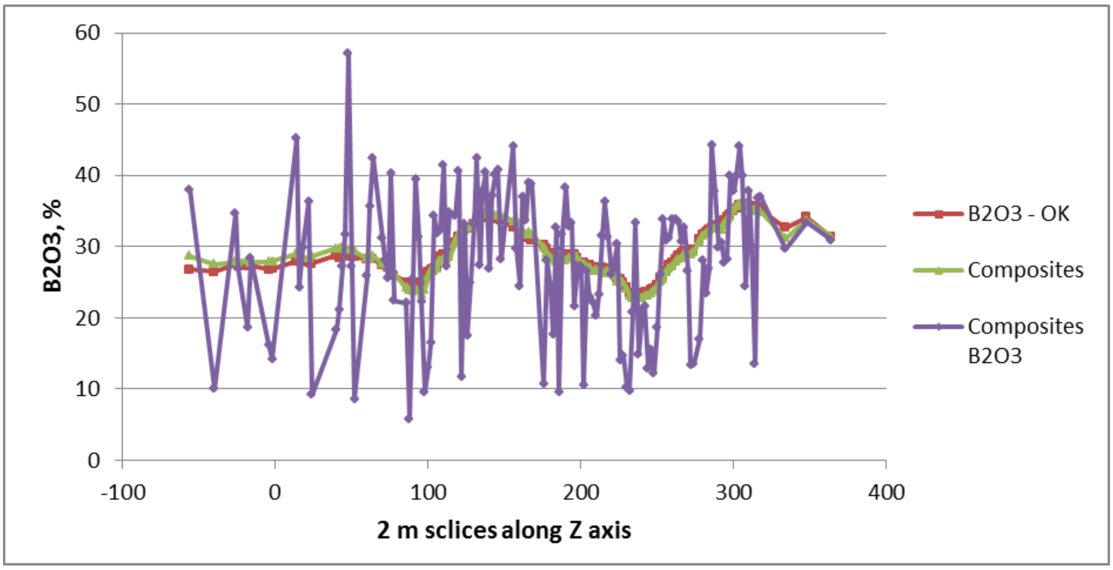


Figure 14-16: B_2O_3 sectional comparison in the Z direction

The results of the analysis in general showed satisfactory correlation and acceptable levels of smoothing using all methods of interpolation. Overall, SRK is satisfied that the current estimates in this region are representative based on visual comparisons between the samples and the block model.

IDW estimate

The resulting OK generated block grades were then compared with block grades interpolated using an IDW algorithm. Table 14-8 below presents the mean block grades produced for each Zone using each technique. As can be seen these are very similar.

Table 14-8: Comparison of Block OK and Block IDW Grades

Zone	Field	Block Mean OK Grade (%)	Block Mean IDW grade (%)	Difference (absolute)
1	B_2O_3	31.7	31.8	-0.1
2	B_2O_3	30.3	30.4	-0.1
3	B_2O_3	32.2	32.3	-0.2
4	B_2O_3	37.3	36.7	0.6
5	B_2O_3	30.2	30.5	-0.3
6	B_2O_3	18.4	16.6	1.8
7	B_2O_3	15.4	16.4	-1.0
8	B_2O_3	18.8	17.5	1.3
9	B_2O_3	26.9	27.6	-0.6
10	B_2O_3	17.1	16.3	0.8

14.8 Mineral Resource Classification

14.8.1 CIM Definitions

The CIM Guidelines define Mineral Resources and the different Mineral Resource categories as follows:-

Mineral Resource

A Mineral Resource is a concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals 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.

The term Mineral Resource covers mineralisation and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase ‘reasonable prospects for economic extraction’ implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralisation that under realistically assumed and justifiable technical and economic conditions might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

Inferred Mineral Resource

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 drill holes.

Due to the uncertainty that may be attached to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

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 drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Mineralisation may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralisation. The Qualified Person must recognise the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A 'Measured Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, and 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 drill holes that are spaced closely enough to confirm both geological and grade continuity.

Mineralisation or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralisation can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

14.8.2 Piskanja MRE Classification

Introduction

SRK has made an assessment of the following key indicators to classify its MRE:

- geological complexity;
- quality of the data used in the estimation:
  - QA/QC data;
  - results of the geostatistical analysis, namely the variography and QKNA results; and
- quality of the estimated block model.

Geological Complexity

The deposit has been modelled as numerous bodies of borate mineralisation. In total 10 mineralised bodies were delineated at different elevation levels. All of these zones are slightly folded in a similar manner. Zones 1, 2, 3, 4 and 6 show higher continuity and confidence in interpretation compared to the rest of the zones for which interpretation was usually only based on 4-5 holes.

While the numerous mineralised bodies show good continuity at 5% cut-off grade, it should be noted that with an increase in the cut-off grade the ore bodies become fragmented.

Overall, it appears that the zones identified are of a reasonably high geological complexity due to the difference of separating the low and high grade populations. As such, and based on the current level of data supporting the geological model, the associated risk relating to the geological continuation is considered at a medium level.

It is also noted that the zones may be more affected by faulting than currently considered; however, this will only be determined following further data collection and structural interpretation.

Quality of the data used in the estimation

Erin has introduced what is considered to be industry best practice in relation to the QA/QC checks with a regular system of standards, duplicates and blanks being inserted into the sample stream.

Validation checks of standards are broadly within acceptable reporting limits and duplicate field samples show a strong correlation to the original sample. Blank samples were reported as showing a low B_2O_3 content.

Different sample analysis methods did however show varying precision in detection of B_2O_3 grade. The proportion of the samples which were used in the MRE and analysed by the less suitable aqua regia ICP-MS method constitute only 10% of the MRE database and have a minimum influence on the estimate.

Core recovery is good and exceeds 90%.

Results of the geostatistical analysis

The data used in the geostatistical analysis resulted in a suitably reliable downhole and omni-directional variogram using the combined data set and a lag spacing of 1 m and 100 m respectively.

The variography allowed the determination of reasonable interpolation parameters to be tested in a QKNA process. The QKNA was undertaken in Datamine and Supervisor software with the slope of regression showing well supported blocks in areas drilled on the dominant drill spacing. The slope of regression values are clearly seen tailing off towards the extremities of the mineralised zones where sample support reduces.

At the same time, the existence of two grade populations in the raw data does decrease the applicability of the OK method or other linear methods.

Quality of the estimated block model

The validation tools utilised show that the input data used to estimate the model is replicated in the estimation. The block model grades are smoothed compared to the input composites as would be expected but the mean grades of the block model and composites are comparable for all interpolated domains.

The grade distribution of the block model shows that the two grade populations could still be distinguished, however the result is much smoothed compared to the raw data. The result shows that extra wireframing detail with regards to data collection and modelling will be needed in the future to be able separate the high and low grade populations.

Classification Guidelines Applied

Given the above comments, the resulting MRE has been reported as comprising material in a combination of Indicated and Inferred categories.

As a guideline to determine the contact between Indicated and Inferred, SRK calculated a Condition Bias Slope (CBS) parameter for every block of the model and created a wireframe around the Indicated areas a CBS of 0.9 (Figure 14-17).

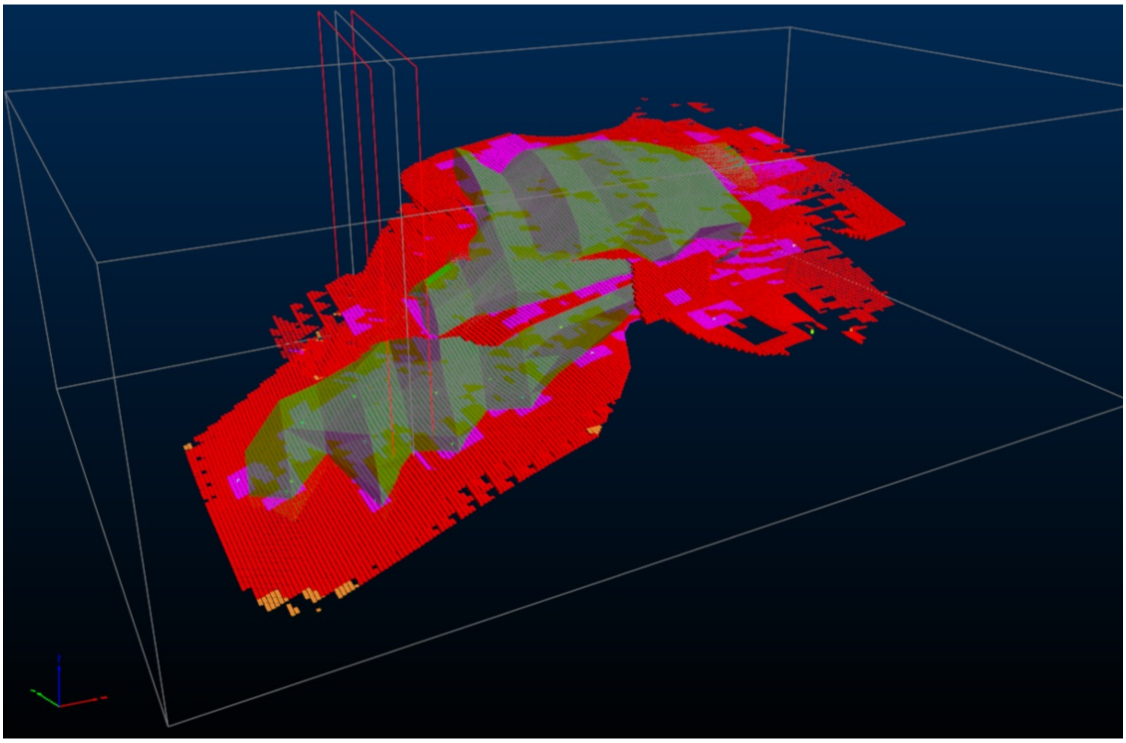


Figure 14-17: Wireframe of the Indicated Mineral Resource (in dark green)

14.9 Mineral Resource Statement

The Mineral Resource Statement generated by SRK has been restricted to material above a marginal cut-off grade of 12% B_2O_3 and a minimum mining height of 1 m so as to constrain the estimate to material which SRK considers has reasonable prospect for eventual economic extraction. This assumes that the mineralisation will be mined by underground methods.

Table 14-9 tabulates the resulting Mineral Resource Statement while Figure 14-18 shows the outlines of the different resource categories. In summary, SRK's estimate comprises an Indicated Mineral Resource of 5.6 Mt with a mean grade of 30.8% B_2O_3 and an Inferred Mineral Resource of 6.2 Mt with a mean grade of 28.8% B_2O_3.

Mineral Resources are not Mineral Reserves as they have no demonstrated economic viability. SRK and Erin are not aware of any factors (environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors) that have materially affected the Mineral Resource Estimate.

The quantity and grade of reported Indicated and Inferred Mineral Resources in this estimation are uncertain in nature. There has been insufficient exploration to report these Mineral Resources in the Measured category and it is uncertain if further exploration will result in upgrading a part of these to this category in due course or if further technical work will enable them to reported as Mineral Reserves.

It should be noted that this estimate was prepared for SRK's 2013 Technical Report and has not been updated for the purpose of this report as no additional drilling has been completed since this time and SRK's geological interpretation remains as developed at that time.

Table 14-9: SRK Mineral Resource Statement

Mineral Resource Category	Tonnage, Mt	B_2O_3 Grade, %	Contained B_2O_3, Mt
Indicated	5.6	30.8	1.73
Inferred	6.2	28.8	1.80

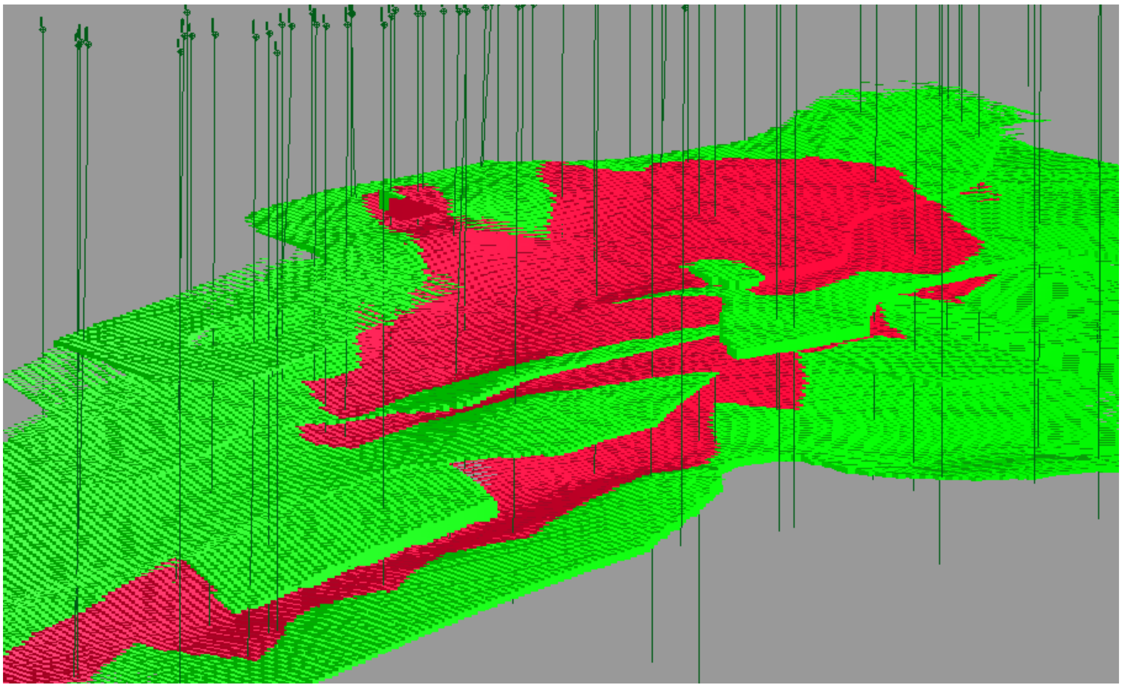


Figure 14-18: Mineral Resource Classification for the Piskanja Project (red = indicated, green = inferred)

14.10 Grade Tonnage Data

The results of a grade sensitivity analysis completed for all the zones combined are presented in Table 14-10 and Figure 14-19. This analysis has been undertaken to show the continuity of the resource at various cut-off grades and the sensitivity of the Mineral Resource to changes in cut-off. The tonnages and grades in these figures and tables should not however be interpreted as Mineral Resources.

Table 14-10: Grade/Tonnage Sensitivity to changes in cut-off

Cut-off grade (B_2O_3), %	Tonnage calculated, Mt	Average grade (B_2O_3), %	Contained B_2O_3, Mt
OK			
7	12.3	29.0	3.57
8	12.3	29.1	3.57
9	12.2	29.2	3.56
10	12.1	29.3	3.55
11	12.0	29.5	3.54
12	11.8	29.8	3.52
13	11.7	29.9	3.51
14	11.6	30.1	3.49
15	11.4	30.4	3.46
16	11.2	30.6	3.43
17	11.0	30.9	3.40
18	10.8	31.2	3.36
19	10.6	31.4	3.33
20	10.4	29.0	3.28

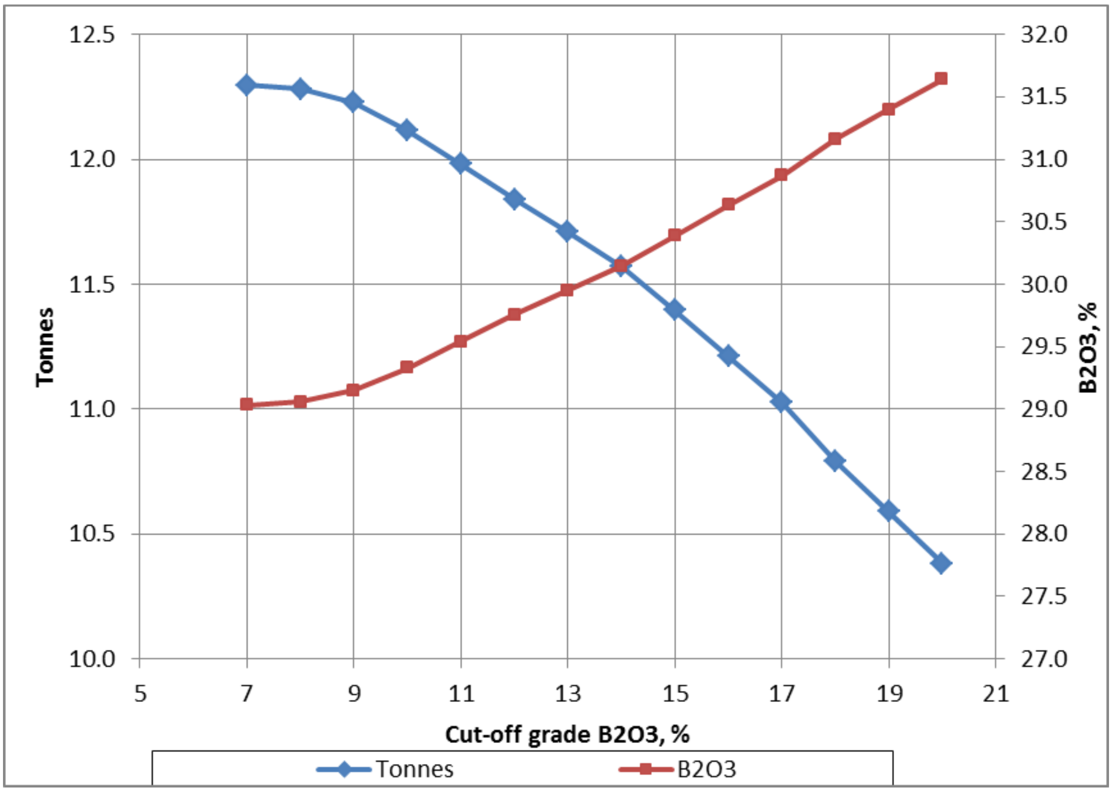


Figure 14-19: Total Grade – Tonnage curves

14.11 Comparison to Previous Mineral Resource Estimates

This is the first MRE derived for the project that meets NI 43-101 reporting standards.

15 MINERAL RESERVE ESTIMATES

No Mineral Reserve estimates have yet been produced for the Project.

16 MINING METHODS

16.1 Introduction

The geometry and depth of the mineralisation identified at Piskanja lends itself to an underground mining method. This section of the report presents the result of work completed by SRK to date to determine how the ore will be most appropriately worked and extracted. While further more detailed analysis is required and will be undertaken in due course it is the assumptions presented here that form the basis of the PEA presented later in this report.

It is envisaged that mining will be by a combination of room and pillar and cut and fill methods and that the key underground infrastructure will comprise:

- an access decline from surface to the lowest level,
- an underground spiral ramp for accessing the upper levels,
- a footwall drive located below the RT1 seam horizon,
- a ventilation shaft and ventilation connections from the FW drive and the spiral ramp to the south west of the orebody

16.2 Mine Access

16.2.1 Currently Proposed Access Location

The proposed access to the underground mine, and that assumed for the purpose of the PEA presented later in this report, is shown in Figure 16-1. Notably, the portal is located within a disused coal yard adjacent to the river.

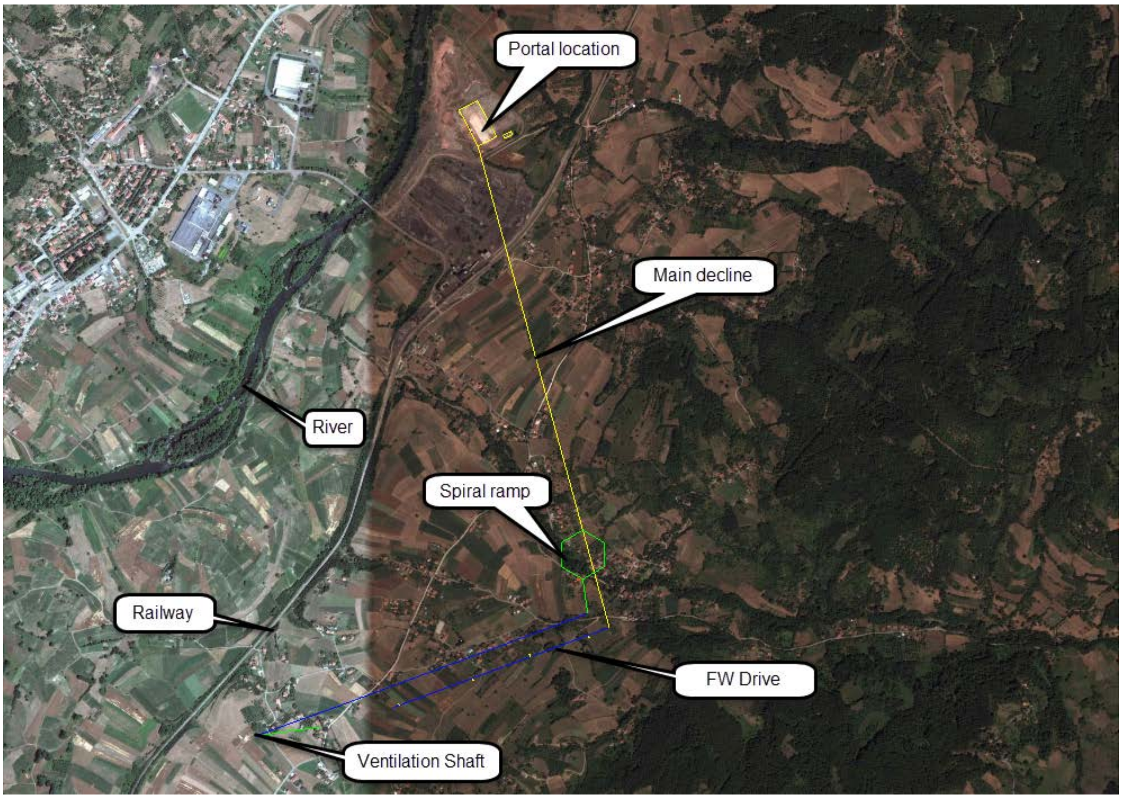


Figure 16-1: Current Piskanja Site

While this may be within the river flood plain, SRK understands that the reasons for choosing this location are because:

- The yard is located to the east of a railway line which runs nominally north-south. In order to cross the railway line a bridge or culvert will be required which will have capital cost implications.
- The land is available and has recent industrial use, so zoning for industrial use is likely to be in place.
- There is potential for third party funding to assist with the remediation of the site.The site is sufficiently large to allow for the construction of a processing plant.
- Though some refurbishment is required, it can make use of the rail loading infrastructure currently in place to allow for rail transport of the material from site to market. The coal loading facility currently has a number of sidings which allow for loading to take place away from the main line.

16.2.2 Alternative Access options

SRK has accepted the proposed location for the access point for the purpose if the PEA, but has also assessed alternative locations to ensure that alternative options are available if required.

To ensure that it is clear of the floodplain, any alternative portal site would need to be located on the hillsides to the east of the railway line and the mineral would need to be transported overland from the portal to the loading area. This would necessitate a suitable rail crossing point to access the site. There are currently three points in the vicinity of the loading area which consist of a bridge and two culverts. These are shown in Figure 16-2.

SRK personnel visited the site and observed the bridge directly and were of the opinion that it would not be suitable for mining equipment without significant remedial work. The culverts were not observed so no comment can be made regarding their suitability. In the event that the rail loading facilities are not used, an alternative siding and loading facility would need to be constructed elsewhere along the rail line.

In addition, cognisance needs to be taken of private property in the area which consists of residential and farm buildings and agricultural land. The region appears to be reasonably well populated and buildings are located away from the fertile river valley and outside of the flood level. Any alternative portal location would need to take these community impacts into account.

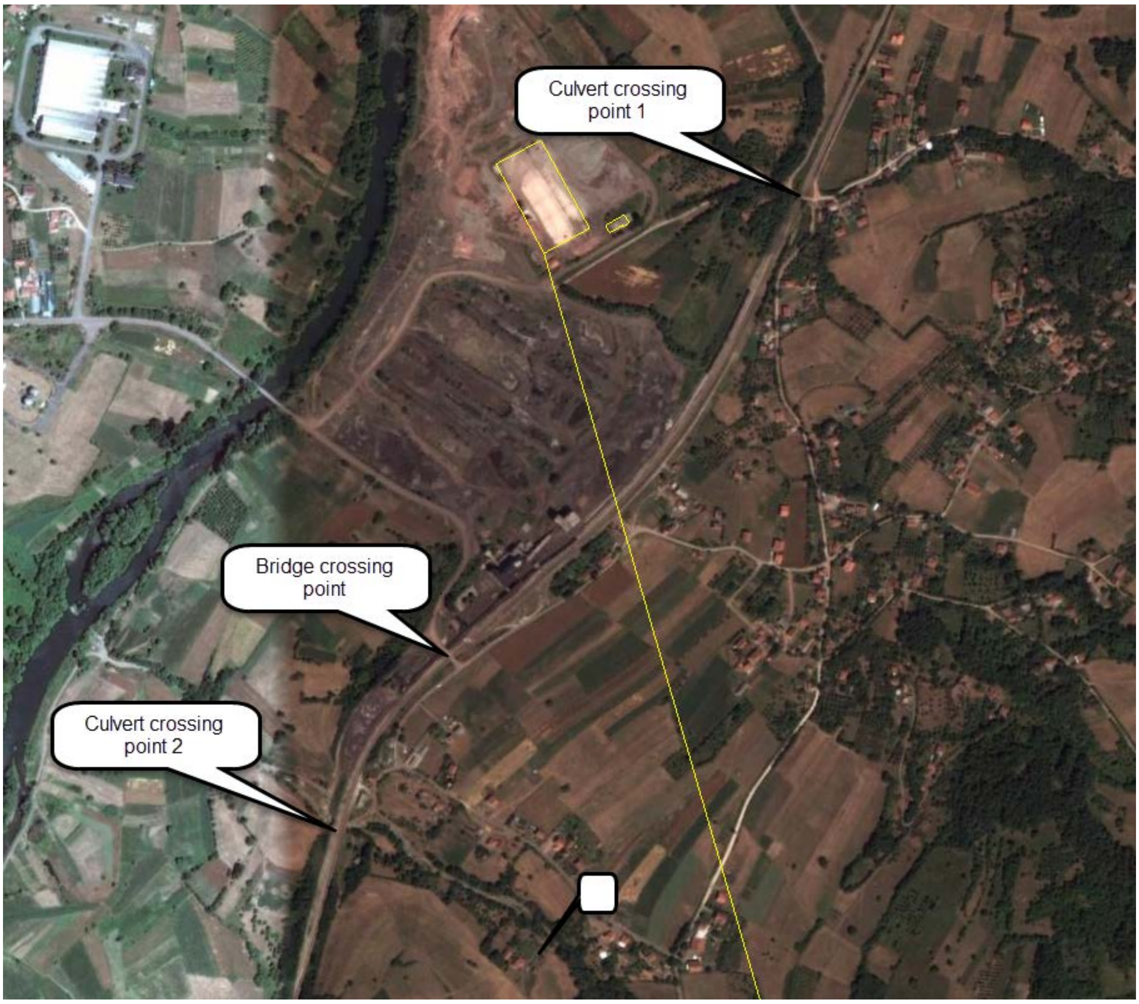


Figure 16-2: Existing Infrastructure

In the absence of hard data on the extent of the flood plain, it seems reasonable to assume that existing significant infrastructure is not located within the extents of flood plain. In addition, there is a lack of residential properties immediately adjacent to the river course. This is likely due to a historic flooding in the region. The boundary of the flood plain was therefore nominally assumed to be represented by the railway in the east and the road/ presence of properties in the west. This is shown in Figure 16-3.

There are a number of tributary river valleys that flow nominally east to west from the hills and feed into the river. Due to the number of properties in this region it seems plausible to also assume that there is limited flooding in these valleys.

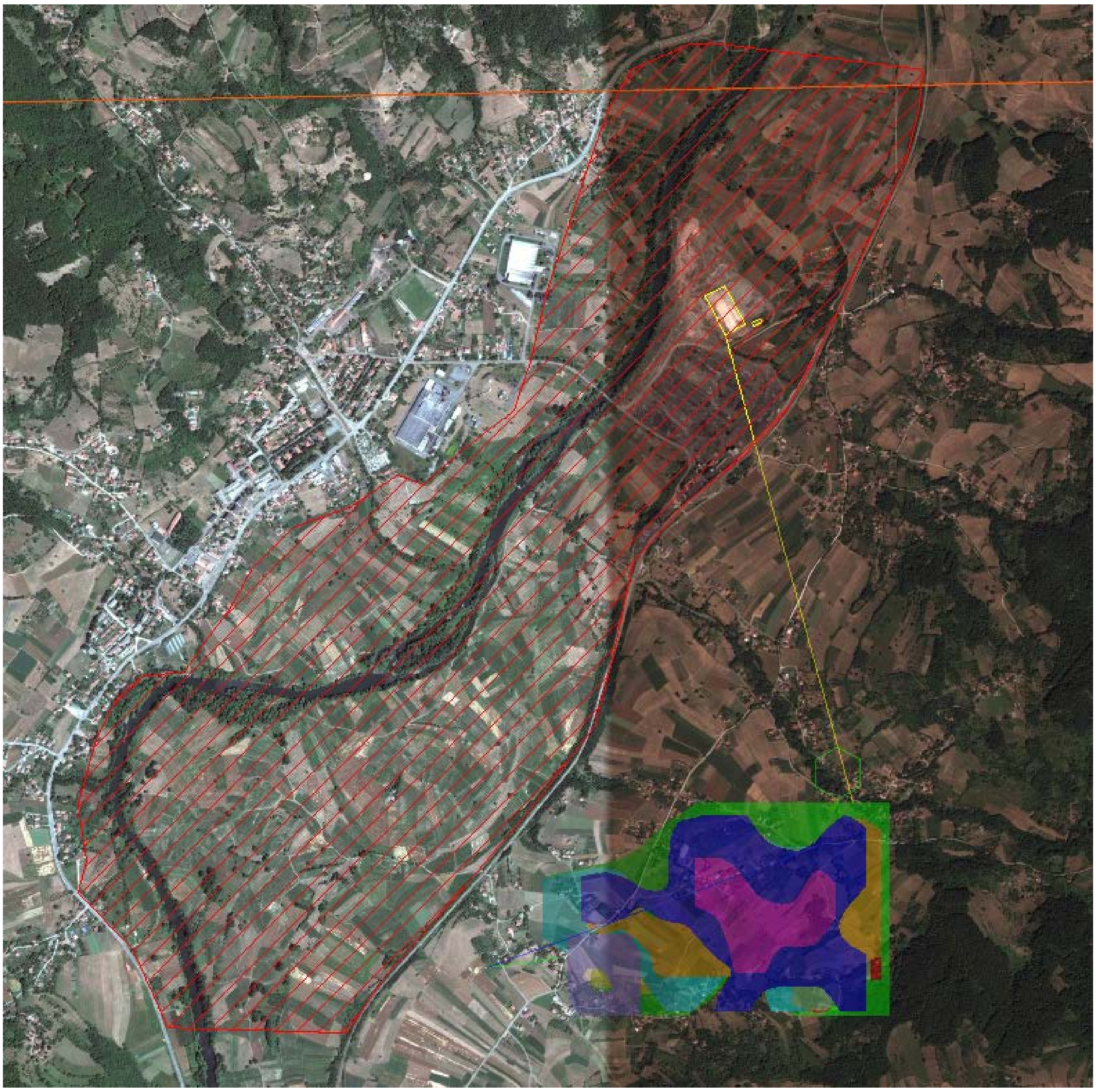


Figure 16-3: Assumed Flood Plain

In the current plan, conveyor belts will be used transport run of mine (ROM) material to surface. The implications of this base assumption are;

- The conveyor decline is straight to minimise transfer points
- The conveyor will be used to deliver the ROM straight to the plant stockpile which would require the use of overland conveyors to deliver product from the portal to the processing plant site

There are, however, several options for surface transfer of ore from the portal to the plant feed stockpile. SRK considers, for example, that the planned production rate could also be achieved using trucks provided that underground ore bins were constructed to manage the oreflow interface between continuous miners underground and a discontinuous trucking process. Trucks present the opportunity to deliver ore straight from underground to the process plant stockpile. There are two direct truck haulage options:

- Use underground mining trucks, as manufactured by Atlas Copco or Sandvik, which have the advantage of being larger and therefore require less truck movements to achieve the stated production rate. The disadvantage is that these are off-highway trucks and a dedicated haul road would be required from the portal to the plant site which would likely include a dedicated rail crossing point.
- Use road going trucks, such as a Scania P420 8x4 tipper. These trucks tend to be smaller so would require more of them to achieve the stated production rate, however they are rugged, readily available and, while these would need to be upgraded, could utilise existing roads between the mine and the plant, including the rail crossing points. The use of public roads for mineral delivery to the plant would result in increased traffic and could have possible negative implications on the community.

Stockpiling ore at the portal presents the following additional ore transport options:

- Ore could be rehandled by truck to the process plant by road trucks during approved working times.
- A dedicated overland conveyor could be used to transport ore to the process plant.
- By splitting the process plant and introducing the crushing and screening benefication process at the portal, it may be possible to pump plant feed to the process plant. This has the advantage of being the lowest impact method of transporting ore to the process plant. A waste stream from the screen will need to be re-handled and away from the site.

If trucks are used for haulage of ore to surface there are more options available for decline design than with conveyor haulage, for example the use of switch-backs. This presents opportunities for alternative portal locations.

Current best practice worldwide indicates that rubber tyred vehicles operate best in declines gradients of or less than 1:8 (12.5%/ 7.13°). Operations with steeper gradients typically experience more vehicle run-away type incidents, and high equipment maintenance costs due to higher than normal wear and tear. In the event that switchbacks feature in a design for truck haulage, these typically are at a centre line radius of 20m and a gradient of 1:33 (3%, 1.74°).

Conveyor ramps can be steeper, but tend to have their use limited to conveying, maintenance access and cleaning.

Skip hoisting could be a viable alternative. This would require locating a shaft collar nearer to the orebody than the existing portal location to remove the horizontal development and haulage component. A ramp access would still (likely) be recommended for equipment and personnel access.

SRK accepts that there are advantages and disadvantages with all of the above alternatives and suggests that all of these options be assessed as part of any pre feasibility study.

16.2.3 Alternative Access locations

Taking all of the above into account, five alternative possible sites for the portal have been identified which broadly met the following criteria;

- Outside the floodplain as indicated in Figure 16-3

- In close proximity to the identified mineralised zone but located outside the mineral extents so as not to sterilise known mineral resource.
- Orientated on hill sides in such a way as to make best advantage of the topography.

The sites are shown in Figure 16-4.

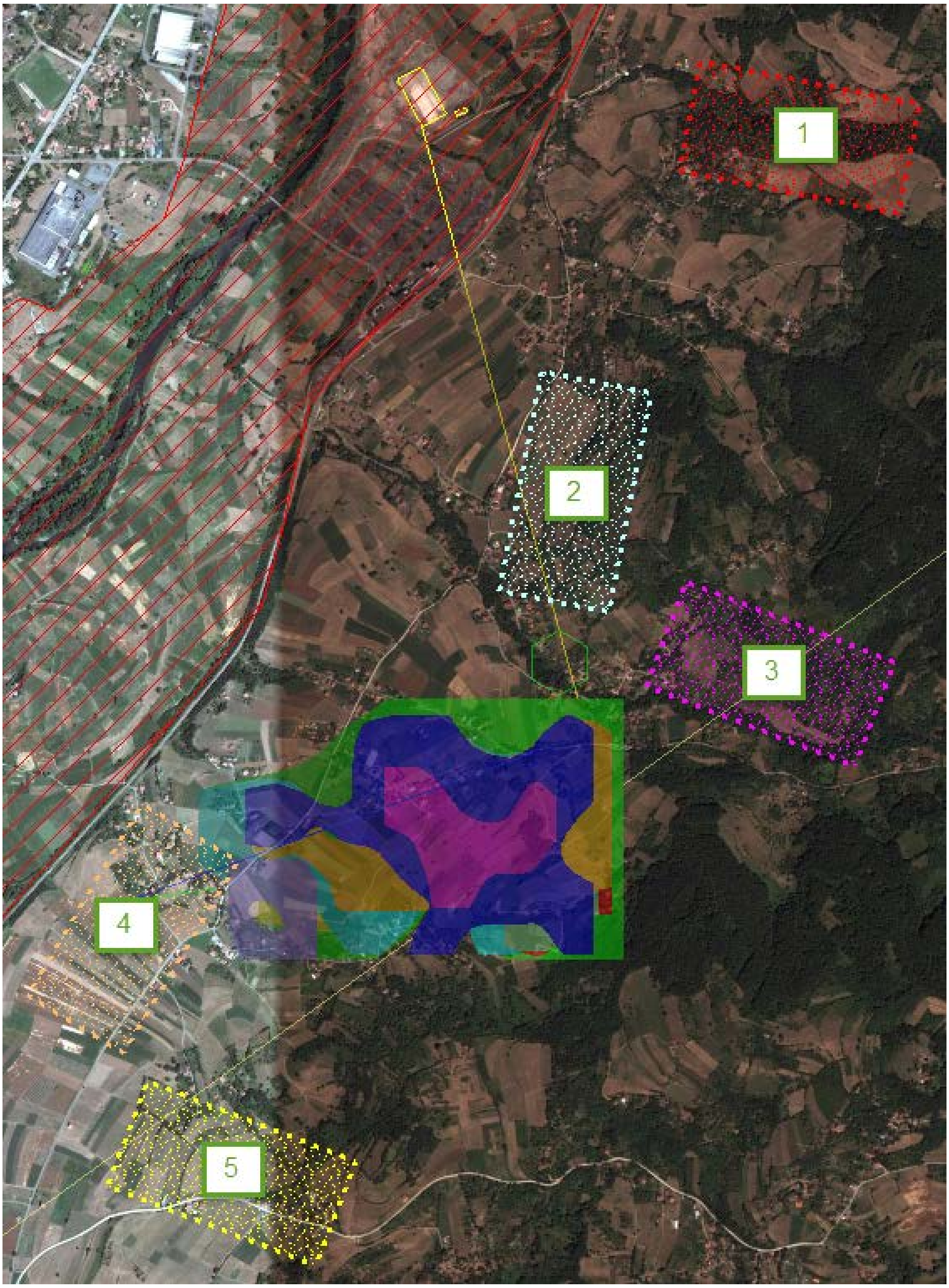


Figure 16-4: Alternative Site Options

Site 1

Site 1 is located east of the planned portal location. There are some indications that this site is located adjacent to the flood plain of the tributary river running to the north and this will need to be verified. The images indicate that the topography to the south of the site is steeply rising and that the portal could be developed into the hillside rather than into flat land. This may have some advantages with regards to reduced development costs.

An established road also links the site to culvert 1 under the railway and this could be used by haul trucks to deliver ROM mineral to the plant. In this instance, the haul would be approximately 670m each way.

The detailed contour elevation and topography surface does not extend to this site so a nominal elevation of +425m was used, which is 80m above the current site elevation.

Site 2 and Site 3

Sites 2 and 3 are orientated to make best use of the topography. A decline developed from these sites would likely initially head north-east into the south facing slopes of the hillside before switching back to head south towards the deposit.

Declines developed from sites on the north facing slopes of the valley would be too steep to reach the deposit or would require multiple switchbacks.

There are direct road links from both sites to culvert 2 allowing for ROM mineral to be transported via road-going trucks. In addition, there is open land which would facilitate the creation of a haul road for off-highway tracks. This would be limited by the lack of a dedicated rail crossing for these vehicles.

Haulage distances to the plant site are c.1.6km and 2.0km from Sites 2 and 3 respectively.

Site 2 is just outside the boundaries of the detailed topography and so the elevation of the surface was taken to be representative. Site 3 is within the confines of the detailed information so it taken to be correct.

Site 4 and Site 5

Site 4 has potential for vertical shaft developments to the mineral horizons. The mine plan locates the ventilation shaft within Site 4.

There is potential to use the shaft to hoist mineral as well as for ventilation. In this scenario, transport of ore to the current plant site would require construction of ~2.5km of roads and/or overland conveyors. However, there may be a cost benefit in establishing the mine access here, via a shaft rather than a decline, and constructing the processing plant and load out facility in this area as a greenfield site. The cost saving/ trade-off for this option is the cost of rehabilitating the coal site which is likely to be significant.

Site 5 has the same infrastructure constraints as Site 4 but allows for declines to access the upper mineral zones from the southern edge of the deposit.

Site 4 and 5 are within the boundaries of the detailed contour elevations.

16.2.4 Decline development

A total of eight decline options were created, two options A and B for Sites 1, 2 and 3 and Site 5. Option A declines targeted the shallowest point below surface of the RT1 mineralised zone; and Option B declines targeted the existing footwall (FW) drive determined by the TEA. In both options the decline would tie into the existing designed infrastructure and therefore the development underground would be the same. The only variation would be the length of the decline itself.

The eight declines developed can be seen in Figure 16-5 compared to the original client information and the RT1 mineral zone which they are targeting.

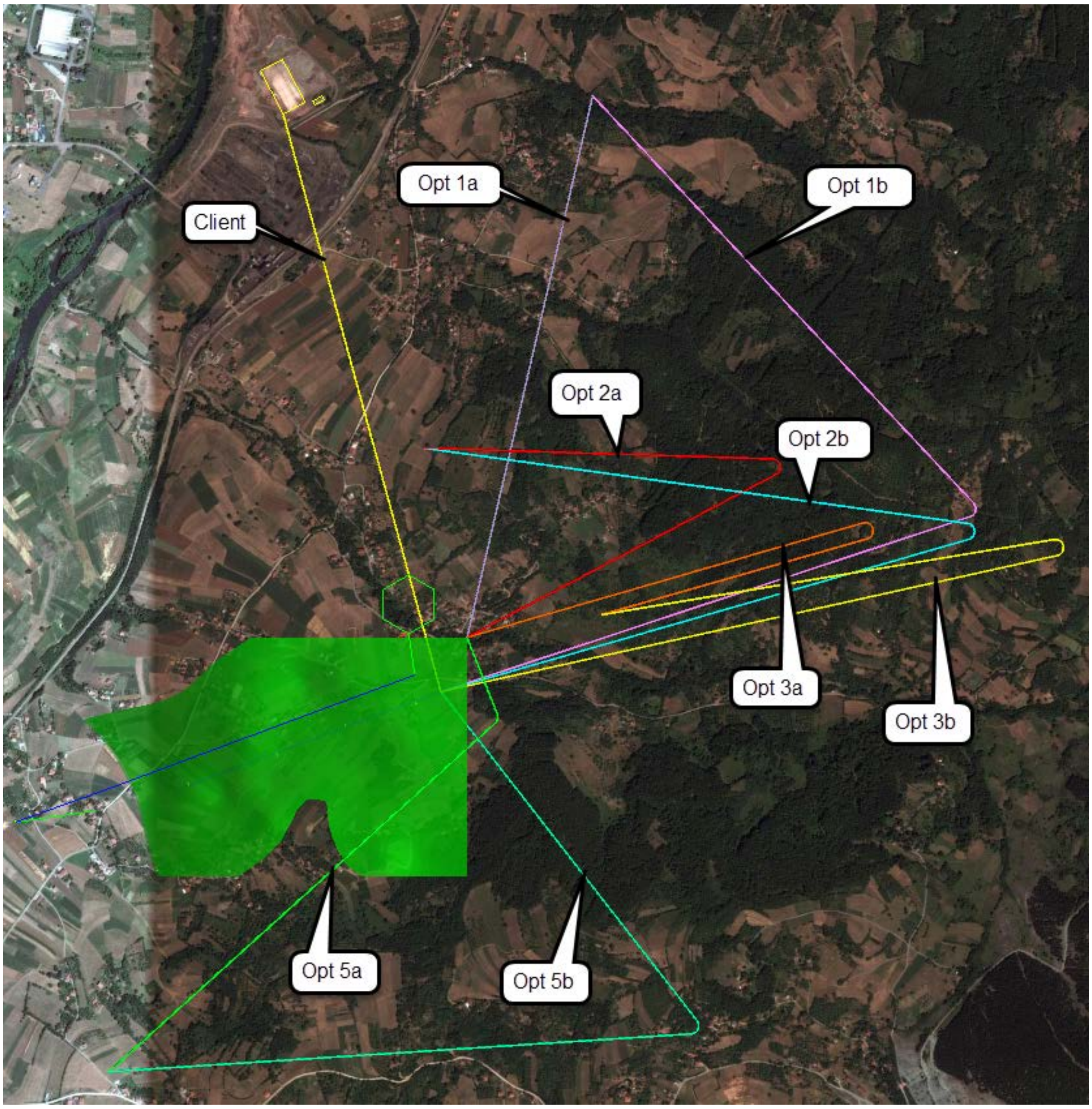


Figure 16-5: Decline Options

A comparison of the decline development lengths is shown in Table 16-1.

Table 16-1: Decline comparisons

Name	Decline length (m)	Decline in-situ volume, (m^3)	Variance from base case (m)	% of base case length
BASE CASE	1,484	32,216	--	--
Site 1 Option A	1,370	29,749	-114	92%
Site 1 Option B	2,786	60,485	1,302	188%
Site 2 Option A	1,776	38,545	292	120%
Site 2 Option B	2,765	60,020	1,281	186%
Site 3 Option A	1,761	38,230	277	119%
Site 3 Option B	2,751	59,710	1,267	185%
Site 5 Option A	1,511	32,808	27	102%
Site 5 Option B	2,513	54,545	1,029	169%

16.2.5 Mine Access Conclusions

Further work is required as part of any PFS to determine the most appropriate access point for the mine and while the base case location currently assumed for the PEA may be perfectly feasible, if not then there are several alternative options available which have been commented upon above. Notably:

- When decline meterage is compared with the base case, Options1A and Option 5A are broadly comparable to the base case assumption. However, decline lengths for Option1 are uncertain as the site's elevation is not known, and has been estimated.
- Sites 1, 2 and 3 have potential for further consideration if trucking is considered for mineral haulage.
- It is thought Sites 2 and 3 are likely to require more significant mitigating actions to be undertaken by the company in order to gain community acceptance.
- Site 4 could have potential if mineral hoisting is adopted. This site opens the possibility for consideration of an alternative mine head and processing plant site.
- Site 5 is a possible alternative which will also require consideration of the processing plant site and loading option, and the trade-off value this option has when compared to cost of rehabilitating the original site, and mitigating risk of project failure in the event the base case site is catastrophically flooded.

16.3 Geotechnical Considerations

16.3.1 Overview of Geotechnical Studies Completed

A geotechnical report by the University of Belgrade detailing the geotechnical conditions of the rock forming the Piskanja deposit was produced in support of the report entitled "Summary Elaborate of Resources and Reserves Piskanja Borate Deposit (Baljevac on Ibar River) on date 31.12.2012", (June, 2013), produced by the University of Belgrade (Technical Faculty in Bor) that details potential mining methods. This geotechnical report includes the results of detailed logging of a number of boreholes and laboratory testing to determine the strength characteristics of a number of lithologies present within the project area.

16.3.2 Geotechnical Characteristics

Sedimentary units in the form of claystones, conglomerates, siltstones, sandstones and carbonates make up the sequence of rocks within the mining horizons. Claystones make up the highest percentage of rock recovered by core drilling and these can often be intensely laminated and will exhibit bimodal strength distribution as a result of the laminations. The table below summarises the results of the material testing undertaken to date.

Table 16-2: Geotechnical material testing results

Lithology	Mean UCS (MPa)	Mean Tensile Strength (MPa)	Mean % Moisture Content	Mean Poisson's Ratio	Mean Young's Modulus (MPa)	Mean Internal Cohesion (MPa)	Mean Internal Friction Angle (°)
Alevrolite		6	3.3	0.32	28839		
Borate	14	2	2.4	0.33	29274	2.80	47
Brecia	11	1	2.8	0.30	32750		
Carbonate	18		2.7				
Claystone	23	3	2.8	0.31	10373	3.67	41
Colemanite	23		10.2				
Conglomerate	15		1.5				
Dolomite	24	2	2.7	0.29	31487	4.22	48
Sandstone	15	1	2.6	0.31	10914		

Lithologies tested have a similar uniaxial compressive strength of between 15MPa and 25MPa which would describe the rock as weak to moderately strong. Zones of intense fracturing (possibly drill related) and low RQD values were noted, as were a number of locations where faulting was clearly visible.

16.3.3 Geotechnical Conclusions

SRK has used the results of previous work along with its own opinions following the inspection of the core by an SRK Engineer to derive the preliminary room and pillar dimensions presented later in this section of the report which have been used in turn to determine extraction ratios for the purpose of the PEA. Clearly more work is needed to be done to derive such to a higher level of confidence and SRK has recommended the following to Erin:-

- Geotechnical logging of planned diamond cored boreholes. Resource/infill boreholes should be geotechnically logged to ensure maximum data recovery and to reduce the need for specific geotechnical boreholes at a later date.
- Development of a geotechnical database defined to collect parameters for input into internationally accepted rock mass classification schemes such as Q and RMR.
- Laboratory testing of selected samples to develop strength parameters for use in future stability modelling.
- Consideration of the need for a small number of specific geotechnical boreholes with the aim of targeting specific structures or achieving geotechnical coverage in the initial mining areas.

16.4 Hydrogeological Considerations

16.4.1 Hydrogeological Characterisation

A hydrogeological report was prepared in 2013 by MWH UK Ltd (MWH) entitled "Interim Hydrogeological Report (Phase II), Piskanja boron, near Baljevac, Raška, Serbia." The following is a summary of the findings of this study.

Groundwater flow in the area of the deposit is in a north-westerly direction i.e. towards the valley hosting the Ibar River. The Quaternary deposits in the area, which are up to 28m in thickness, comprise a partially saturated perched aquifer with low to moderate permeability. The underlying Clayey Silt Deluvial Sediments (between 80m and 300m thickness) mainly comprise claystone and are considered a low permeability aquitard, although there is evidence for some disturbed/fault zones from loss of circulation recorded on occasions during drilling.

The Tertiary Claystone, Dolomite and Volcaniclastics unit (up to 320m) is a claystone-dominant formation with interbedded tuffs, dolomite etc. Occasional circulation loss during drilling in the borate and adjacent carbonate beds suggests higher permeability locally although generally this formation has a low to moderate permeability. It is considered to act as an aquitard with respect to local aquifers. Preliminary permeability testing in this formation indicated permeabilities ranging between 2E-6 and 2E-8m/s (i.e. low to moderate permeability). There is little data on the underlying Tertiary Sandstone, Conglomerate and Claystone formation. It is considered to have similar aquifer properties to the overlying formation.

There is a spring up the hill from the deposit with a chemical signature that suggests a deep groundwater source, likely to be via a deep fault structure. This reflects the potential for artesian conditions due to an upwards vertical flow from deep groundwater where pathways (i.e structural conduits) are available.

16.4.2 Mine Water Inflow and Dewatering Considerations

Average groundwater inflows to an open pit scenario were provisionally estimated by MWH to be in the range 5 to 50 litres/second. Inflows to an underground mine development would likely be in the lower range of this estimate.

Inflow rates of this magnitude are manageable by straightforward dewatering methods. The workings may intercept fault structures where sustained, localised inflows may occur due to upward flowing groundwater. The flow rates of such inflows cannot be predicted without further characterisation of these structures although the flow rate from the observed deep groundwater spring higher in the catchment has a modest flow rate of approximately 0.1 litres/second.

16.4.3 Hydrogeological Conclusions

Further work is required as part of any PFS to characterise site hydrogeological conditions and assess mine water inflows, in particular:

- Hydrogeological testing to constrain aquifer parameters and groundwater behaviour;
- Installation of additional piezometers to better constrain groundwater piezometry;

- Integrated structural/hydrogeological studies to better understand the role of geological structures as conduits for groundwater flow;
- Groundwater modelling to constrain predictions on mine water inflows;
- Mine dewatering infrastructure design and costing.

In addition, a mine site stormwater management study, including surface water hydrology characterisation and the development of a mine site water balance is recommended.

16.5 Mining Method

16.5.1 Introduction

Excavation is currently proposed by mechanical cutting using continuous miners and shuttle cars for transport of mineral from the working area to panel conveyor.

The panel conveyor would feed the mines ore handling system that is yet to be determined, but may comprise conveyor haulage to surface, or a perhaps a combination of conveyor haulage to a central point underground from where mineral would be either hoisted to surface via a hoisting shaft or trucked to surface.

Two mining methods have been identified as having potential for extraction; room and pillar and drift and fill. Both would require backfill. The application of room and pillar is limited by the orebody geometry and notably the fact that it is comprised of a series tabular lenses that vary in width between 0.4 m and 15.0 m and which dip at around 18°.

16.5.2 Previous Proposals

The mining method proposed in the report titled “Summary Elaborate of Resources and Reserves Piskanja Borate Deposit (Baljevac on Ibar River) on date 31.12.2012”, (June, 2013) was room and pillar using continuous miners that can excavate to around 1.0 m thickness. This report also proposed that by backfilling rooms, pillars could be spilt and resource recovery increased from around 60% to 75%.

This report also proposed the drifting of preparation declines/ramps through the ore body ensuring that maximum inclination does not exceed 12°. Additional preparation work proposed comprised the division of the deposit into mining blocks 160 m long and wide separated by 20 m wide protective pillars.

The proximity of Ibar and the village excluded the application of caving methods and bearing in mind the Erin’s preference for mechanized mining without drill and blast operations, and the structure and the size of the individual ore bodies, the only solution considered was room and pillar mining. The calculation of room and pillar dimensions was performed using tributary theory and *BasRock Room and pillar optimizer* software. The value of 6 m was accepted for room width and the pillar dimensions were defined according to this width and the depth of the deposit with the minimum safety factor of 1.5. Due to the low compressive strength of the rock and large depths, the pillars in the deepest parts of the deposits were 12x12 m for the room height of 1.65 m.

Since ore is left in the pillars the recovery was in the range of 55 % to 67 % depending on the depth and pillar size while overall recovery was 60 %. In order to increase recovery a possibility of recovering a part of the pillar after the rooms have been backfilled was considered. By creating a 3.3 m wide room inside a 12x12 m pillar the recovery was increased to 68 % and when a full width room (6 m) was assumed, the recovery increased to 78 % in the deeper parts of the deposit and to even 84 % in the shallower points, providing an overall recovery of 75 %.

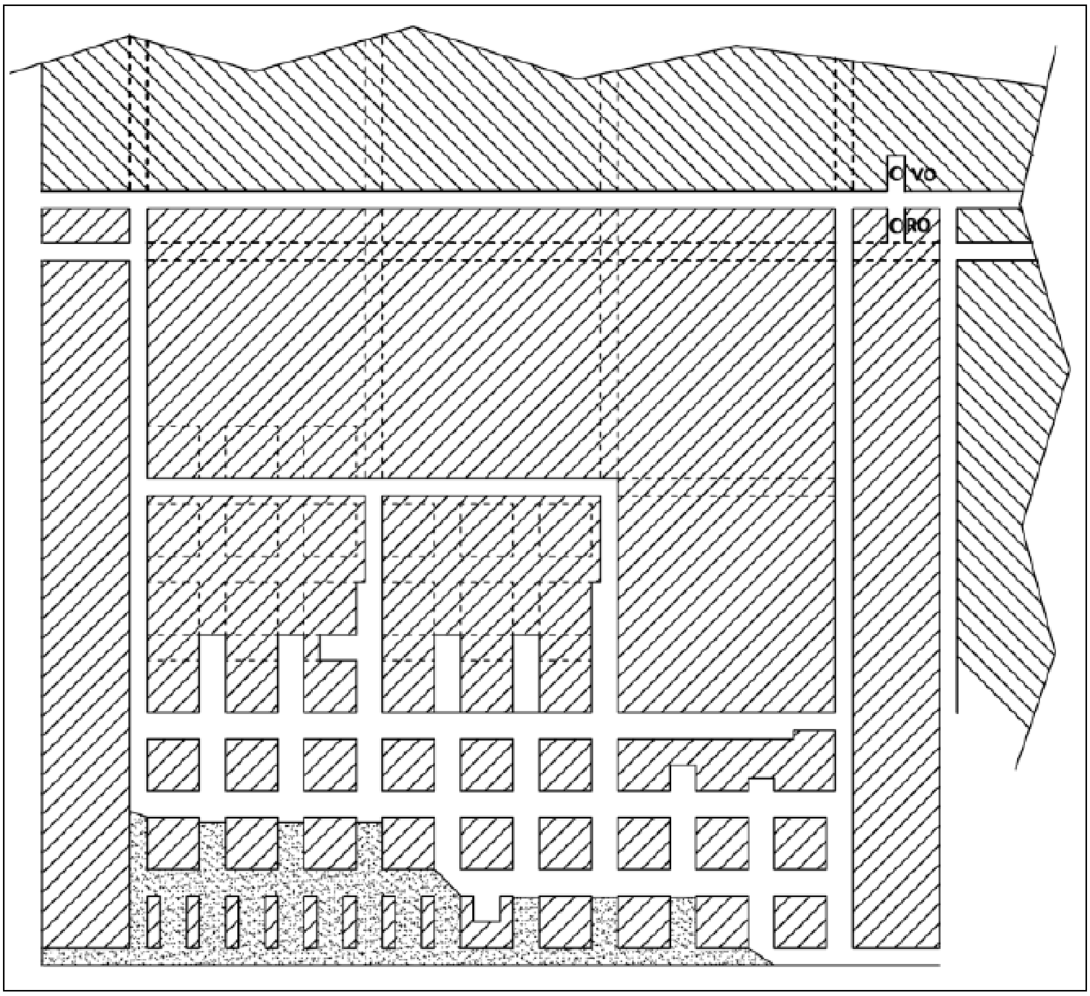


Figure 16-6: The construction of mining blocks and division to mining pillars with room and pillar details[1]

16.5.3 SRK Proposals

In SRK's opinion, the orebody dip will prevent mining on dip, and so excavation by continuous miners on strike, or an apparent dip is more likely. This will introduce dilution and ore loss at the footwall and hangingwall.

[1] Figure from "Summary Elaborate of Resources and Reserves Piskanja Borate Deposit (Baljevac on Ibar River) on date 31.12.2012. June, 2013", p.77.

Drift and Fill is an alternative approach to extraction that might be more suitable to use of backfill, and variable lens width and orebody dips expected. The method has potential for higher proportions of orebody extraction than room and pillar method, in theory up to 100%. However, there may be a need for regional pillars to be left in-situ for mine stability which would reduce orebody extraction.

In order to achieve the overall recovery of 75 % and ensure the stability of excavated spaces it will be necessary to apply solidifying material for a backfill and certainly further geotechnical assessment inclusive of an assessment of the geometry, rock strength, and backfill characteristics will be required.

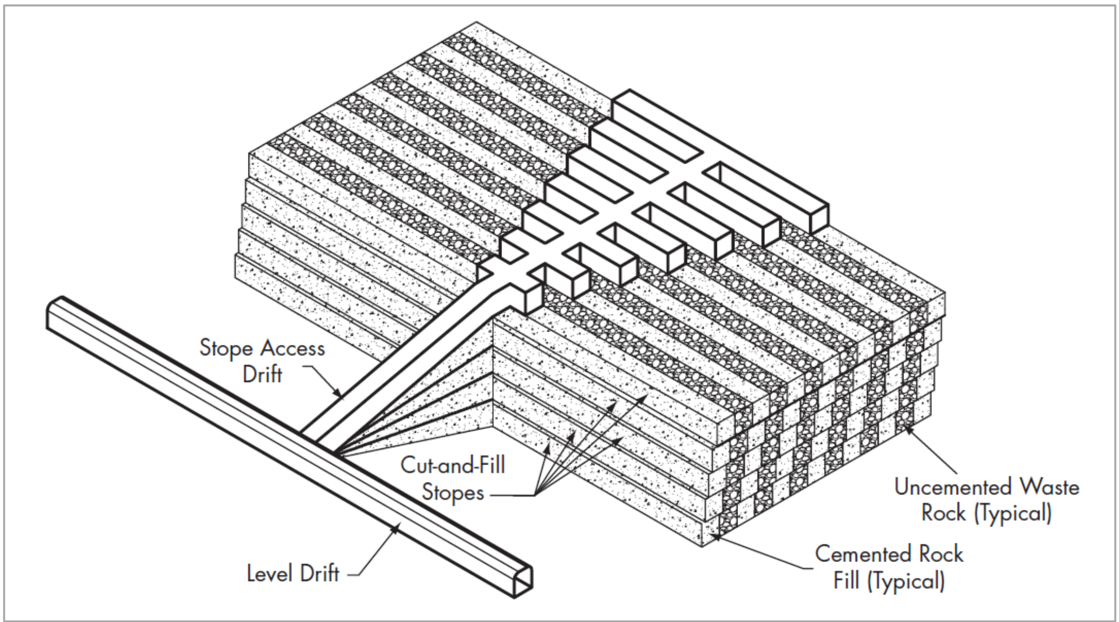


Figure 16-7: Typical Overhand Drift and Fill (SME handbook).

Backfill design and materials selection will be subject of future investigation into the availability and characterisation of materials. The waste material excavated and processed and subsequently available as tailings are expected to comprise fine grained sandstones and mudstones. It is unlikely this material alone will be suitable to develop backfill appropriate for the mining method, and additional materials will be required to achieve design characteristics.

Additional backfill materials would be expected to be sourced locally from sand and gravel pits and may also include imported cement or pozzolonic materials. There is potential that the gypsum waste resulting from the boric acid plant process could be incorporated into the backfill design, and if the gypsum is calcined it might be used to provide binding qualities to the backfill.

The backfill will be required to have strength to as a working platform to support equipment in thicker orebody sections.

16.6 Mining Tonnage

16.6.1 Introduction

Notwithstanding the fact that more work is required to be done to confirm the most appropriate mining method, for the purpose of the PEA SRK has determined the potential tonnage available for mining by applying the following factors to the Mineral Resource:

- A minimum mining width of 1m.
- 75% orebody extraction with 25% of orebody left in-situ as pillars for ground stability.
- Ore loss introduced on the hangingwall and footwall based on the hangingwall dip.
- A requirement for the mined B_2O_3 grade to be over 20%.

16.6.2 Minimum Mining Width

While the selected mining equipment can excavate to a minimum width of 1.0m but the shuttle car needs a minimum of 1.2 m height in which to operate, and this is considered a minimum practical mining height utilising mechanical equipment.

It has been assumed therefore that around 10% of the footprint of each orebody lens is between 1.0 m and 1.2 m thick, with an average thickness of 1.1 m. The tonnage associated with this area has been excluded from Mineral Resources available to mine.

16.6.3 In-situ pillars

The orebody has variable dip and in some flatter zones room and pillar mining may be applicable and more appropriate than drift and fill (Figure 16-8).

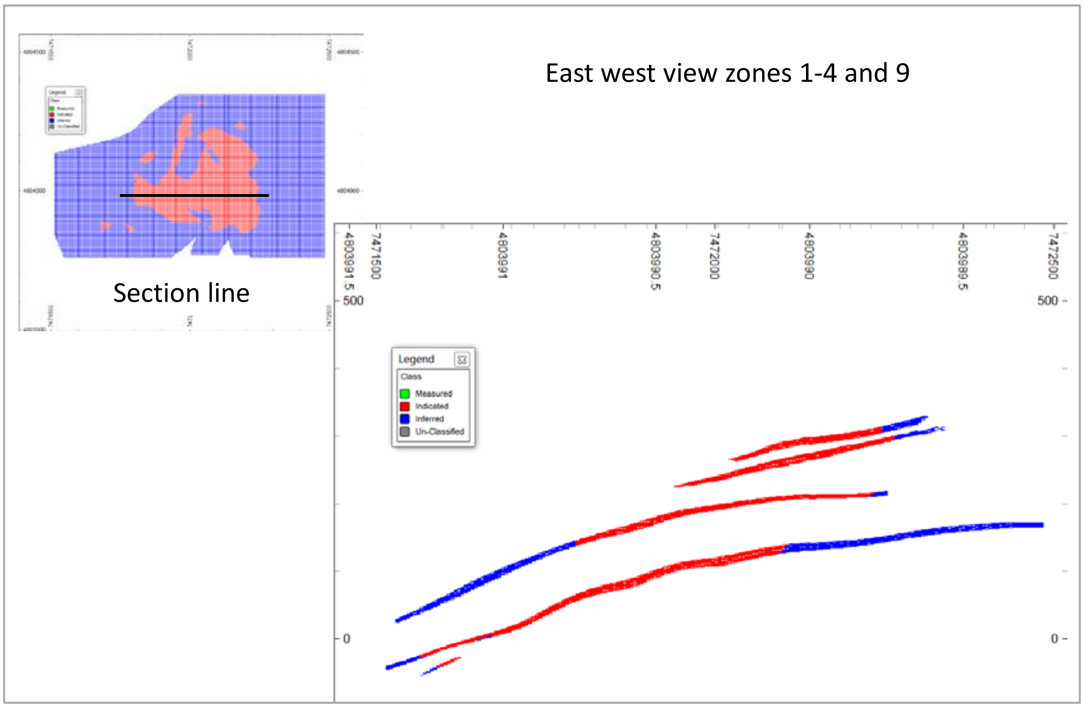


Figure 16-8: East West View of Zones 1, 2, 3, 4 and 9.

For the purposes of this PEA, it is assumed that either room and pillar or drift and fill mining methods will be used to extract the mineral. With the application of backfill it has been assumed that an ore extraction ratio of 75% can be achieved.

16.6.4 Ore Loss and Dilution Due to Orebody Dip

An average orebody dip of 12° has been selected to calculate possible loss of ore from, and introduction of dilution into mine production. This is based on room widths of 8 m and has been applied to 75% of the ore lens area.

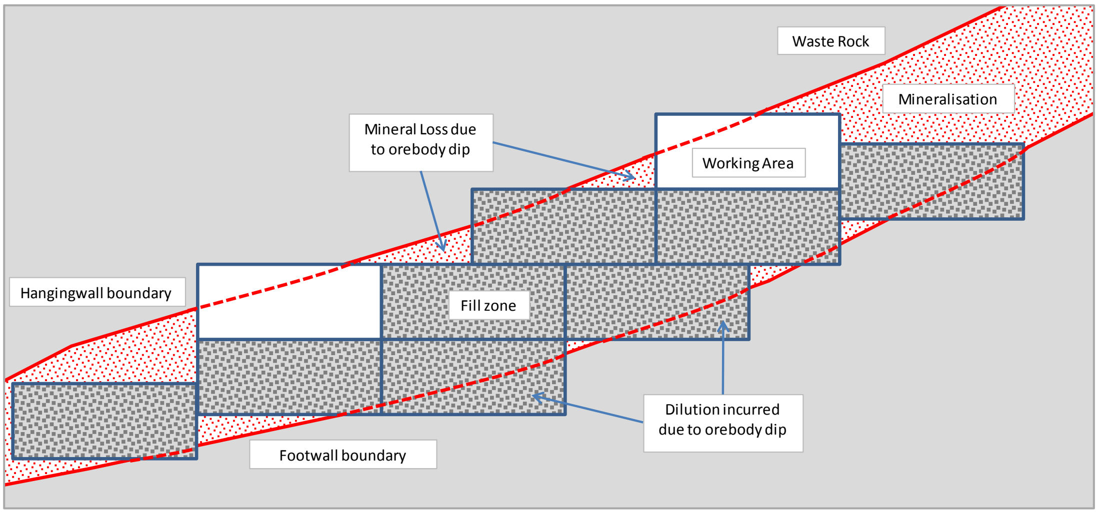


Figure 16-9: Schematic Section showing ore loss and Dilution in a drift and fill layout.

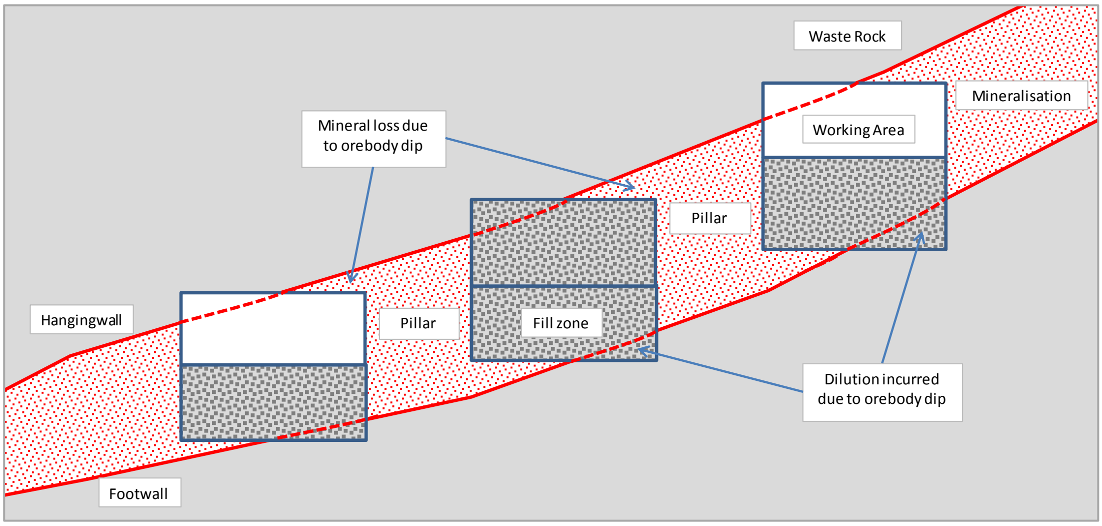


Figure 16-10: Schematic section showing ore loss and dilution in a room and pillar layout.

16.6.5 Production requirements

Two products are planned from the Project: 200,000 tonnes per annum (tpa) of colemanite grading of 35% B_2O_3 and 25,000 tpa boric acid.

On the basis of its experience, SRK has conceptualised a possible processing route in order to develop an understanding of possible mine production rates that would be required to support planned production. This is described in more detail in Section 17 of this report but in summary this assumes that:-

- The Colemanite product grade achievable is 35% B2O3.
- Boric Acid looses 20% of B2O3 to the tailings.
- The beneficiation plant losses 7.5% B_2O_3 to the tailings

On this basis, annual run of mine production requires a B_2O_3 content of 94,300 t (Table 16-3).

Table 16-3: Potential ROM production rate

Colemanite product grade		% B_2O_3	35%
Colemanite Sales @ 35% B_2O_3		tpa	200,000
B_2O_3 contained in Colemanite		tpa	70,000
B_2O_3 Requirement for Boric Acid			
BA requirement		tpa	25,000
B_2O_3 content		tonnes	14,074
Losses in processing		%	20%
B_2O_3 content in Plant Feed		tonnes	17,593
Total B_2O_3 requirement from beneficiation			87,593
Beneficiation plant: recovery of B_2O_3[(1)]			92.9%
B_2O_3 contained in ROM		tpa	94,264

[(1)] Based on an average Beneficiation Plant feed grade of 27.8% B_2O_3.

ROM production rate is dependent on the B_2O_3 grade of the ore being mined. The grade-recovery relationships for each process are not yet determined and are subject to further work. On the basis of grade defined by ore zone, and sequence of mining determined by grade; a life of mine schedule over 20 years has been developed.

16.6.6 Mine Production Modifying Factors

SRK has assessed the orebodies in terms of thickness, grade, and dip to develop a Run of Mine (RoM) plant feed tonnage from the Mineral Resource.

Specifically the ore zones have been ranked in order of contained B_2O_3; 79% of B_2O_3 is contained within ore zones 1, 2 and 3 (Table 16-4).

Table 16-4: Ore Zones ranked by contained B_2O_3.

Ore Zone	In-situ Ore Tonnage (Mineral Resource) (Mt)	B_2O_3 Grade (% B_2O_3)	B_2O_3 tonnage (t B_2O_3)	B_2O_3 tonnage (% of total)
Zone 1 INF	2.71	30.5	826	23%
Zone 1 IND	1.91	33.3	636	18%
Zone 3 IND	1.62	33.1	535	15%
Zone 2 INF	1.20	29.6	355	10%
Zone 3 INF	0.84	30.3	255	7%
Zone 2 IND	0.55	31.8	174	5%
Zone 6 IND	0.91	19.1	173	5%
Zone 4 IND	0.41	37.5	155	4%
Zone 6 INF	0.37	23.0	86	2%
Zone 4 INF	0.20	36.9	75	2%
Zone 9 INF	0.22	27.4	61	2%
Zone 7 INF	0.31	18.2	56	2%
Zone 5 INF	0.11	32.3	35	1%
Zone 5 IND	0.09	27.6	24	1%
Zone 8 INF	0.13	18.8	24	1%
Zone 10 INF	0.14	17.1	23	1%
Zone 9 IND	0.07	25.6	18	1%
Zone 8 IND	0.05	18.9	9	0%
Zone 7 IND	0.01	17.8	3	0%
	11.84	**29.8**	**3,523**	**100%**

Ore Zones Excluded from the Mine Schedule

Ore Zones 6, 7, 8, 9 and 10 have been excluded from the schedule on the basis of contained grade being too low to maintain planned plant feed rates at the nominal mine production rate established by mining the rest of the orebody.

Ore Zone 5 has been excluded on the basis of having limited tonnage (200 ktpa)combined with being too thin (average thickness of 1.0 m in the Indicated and 0.9 m in the Inferred) to provide practical mining efficiencies.

Orebody Thickness

Orebody grade is expected to graduate over short distances at the hanging and footwall boundaries. The Mineral Resource is defined >1.0 m thickness. The mining equipment selected can excavate mining widths of 1.0 m, but the shuttle car used to transport mineral requires a minimum operating height of 1.2 m.

The areal extent of each ore lens with thickness 1.0 m to 1.2 m is assumed to be in the order 10%, and this has been excluded from the mineable resource.

Orebody Dip

The adverse impact of ore loss is dependent on orebody dip and room width. The orebody has an average dip of 12°, with a maximum dip of 18°. A room width of 8 m has been applied for the calculation. Ore losses have been applied equally across all ore zones, at the resource extraction ratio of 75%.

Hangingwall and Footwall Dilution

In the same way ore is lost due to orebody dip, dilution is introduced into run of mine production. Volumes are the same as ore loss, but the greater density of waste rock (sg 2.48) results in greater tonnage.

Dilution grade has been applied at 0% B_2O_3.

Summary of Modifying Factors

A summary of the tonnage movement from Mineral Resource to Run of Mine is shown in Figure 16-11 and Table 16-6.

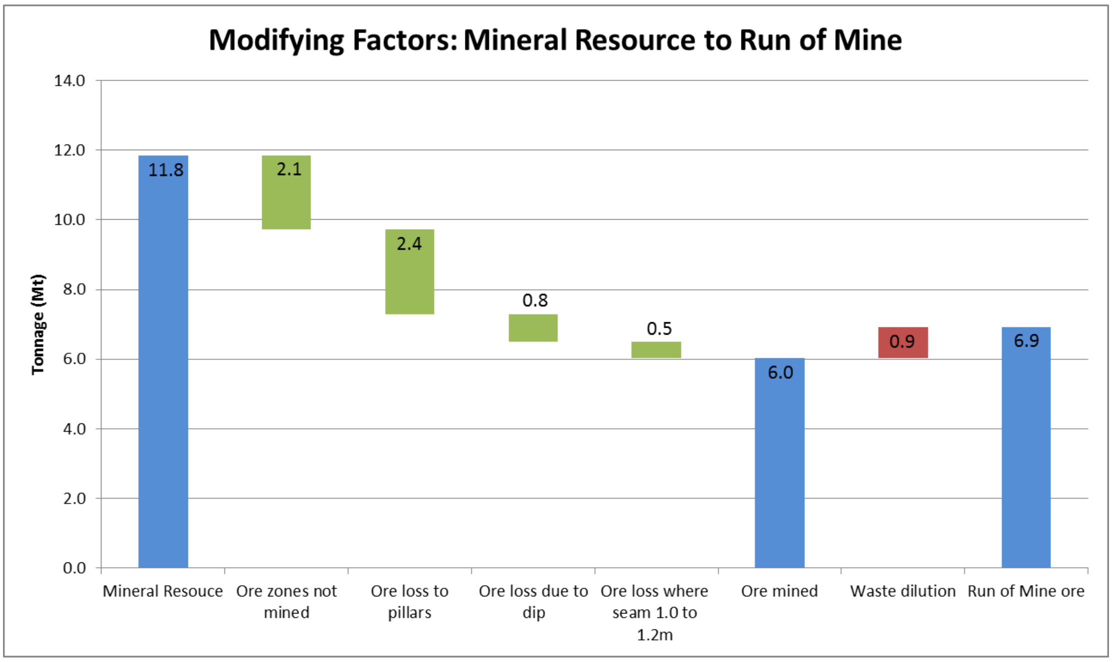


Figure 16-11: Mining Modifying Factors applied to Mineral Resource to Run of Mine

Table 16-5: Mineral Resource to Run of Mine

Ore Zone	In-situ Ore Tonnage (Mineral Resource)	B_2O_3 Grade	Orebody Zones scheduled	Ore Zones left in Situ	Zones to be mined	Orebody tonnage lost to pillars	Recoverable ore (less pillars)	Orebody tonnage lost due to orebody dip	Tonnage lost to orebody <1.2m thick	Ore Tonnage mined	Dilution Tonnage at 0% B_2O_3	Scheduled Tonnage	Mine Grade
	(Mt)	(%B_2O_3)		(Mt)	(Mt)	(Mt)	(Mt)	(Mt)	(Mt)	(Mt)	(Mt)	(Mt)	(%B_2O_3)
Zone 1 IND	1.91	33.3	1	-	1.91	0.48	1.43	0.15	0.09	1.19	0.17	1.36	29.2
Zone 1 INF	2.71	30.5	1	-	2.71	0.68	2.03	0.23	0.13	1.66	0.25	1.92	26.5
Zone 2 IND	0.55	31.8	1	-	0.55	0.14	0.41	0.05	0.03	0.33	0.06	0.38	27.1
Zone 2 INF	1.20	29.6	1	-	1.20	0.30	0.90	0.10	0.06	0.75	0.11	0.85	25.8
Zone 3 IND	1.62	33.1	1	-	1.62	0.40	1.21	0.11	0.07	1.03	0.12	1.16	29.6
Zone 3 INF	0.84	30.3	1	-	0.84	0.21	0.63	0.07	0.04	0.51	0.08	0.59	26.3
Zone 4 IND	0.41	37.5	1	-	0.41	0.10	0.31	0.03	0.02	0.26	0.04	0.29	32.9
Zone 4 INF	0.20	36.9	1	-	0.20	0.05	0.15	0.02	0.01	0.12	0.02	0.14	32.0
Zone 5 IND	0.09	27.6	0	0.09	-	-	-	-	-	-	-	-	-
Zone 5 INF	0.11	32.3	0	0.11	-	-	-	-	-	-	-	-	-
Zone 6 IND	0.91	19.1	0	0.91	-	-	-	-	-	-	-	-	-
Zone 6 INF	0.37	23.0	0	0.37	-	-	-	-	-	-	-	-	-
Zone 7 IND	0.01	17.8	0	0.01	-	-	-	-	-	-	-	-	-
Zone 7 INF	0.31	18.2	0	0.31	-	-	-	-	-	-	-	-	-
Zone 8 IND	0.05	18.9	0	0.05	-	-	-	-	-	-	-	-	-
Zone 8 INF	0.13	18.8	0	0.13	-	-	-	-	-	-	-	-	-
Zone 9 IND	0.07	25.6	1	-	0.07	0.02	0.05	0.01	0.00	0.04	0.01	0.05	22.3
Zone 9 INF	0.22	27.4	1	-	0.22	0.06	0.17	0.02	0.01	0.14	0.02	0.16	24.3
Zone 10 INF	0.14	17.1	0	0.14	-	-	-	-	-	-	-	-	-
	11.84	**29.8**		**2.11**	**9.73**	**2.43**	**7.30**	**0.80**	**0.46**	**6.04**	**0.87**	**6.91**	**27.8**

16.6.7 Mine Production Schedule

The schedule was developed to target the highest B_2O_3 grade and Indicated Mineral Resource classification first and each ore zone has been depleted in succession. The resulting mined tonnage varies by year according to its B_2O_3 grade such that the B_2O_3 content supplied to the process plant is at a constant rate of 94,250 tpa.

Ore zones included in the mining schedule

Ore zones 1, 2, 3, 4, and 9 are included in the mine schedule which represents 82% of the resource tonnage and contains 88% of the B_20_3 of the diluted ore available to mine (Table 16-6).

Table 16-6: Diluted tonnage and grade available for Mining by Ore Zone

		Indicated		Inferred		Total	
		(Mt)	(%B_2O_3)	(Mt)	(%B_2O_3)	(Mt)	(%B_2O_3)
Zone 1	Scheduled	1.36	29.2	1.92	26.5		
Zone 2		0.38	27.1	0.85	25.8		
Zone 3		1.16	29.6	0.59	26.3		
Zone 4		0.29	32.9	0.14	32.0		
Zone 5	Excluded from Schedule	0.06	21.1	0.07	24.3		
Zone 6		0.64	16.5	0.26	19.1		
Zone 7		0.01	15.3	0.22	16.1		
Zone 8		0.03	15.9	0.09	15.5		
Zone 9	Scheduled	0.05	22.3	0.16	24.3		
Zone 10	Excluded	0.00		0.09	14.2		
	Indicated and Inferred	**3.98**	**27.0**	**4.40**	**25.0**	**8.39**	**25.9**
	Scheduled	**3.24**	**29.3**	**3.67**	**26.4**	**6.91**	**27.8**

Mining Sequence

Some 20 years of mining is scheduled from zones 1, 2, 3, 4, and 9 at an average rate of 340,000 tpa at 28.7% B_2O_3.

Mining rates are lowest from the highest grade ore zone (Zone 4 Indicated), which accounts for the first year of production at 286 kt at 32.9% B_2O_3. The last full year of production (Zone 2 Inferred and Zone 9 Inferred) is at 366 kt at 25.8% B_2O_3.

The resulting mining schedule developed by SRK is shown in Figure 16-12 and Table 16-7.

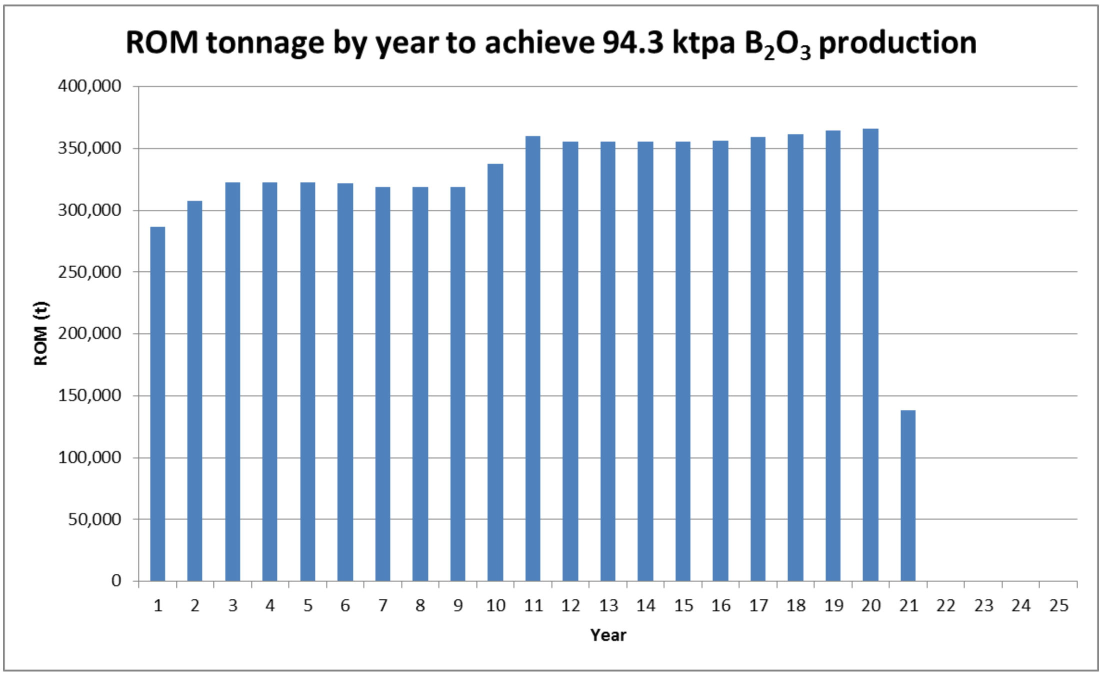


Figure 16-12: LOM Schedule to achieve 94 ktpa B_2O_3.

Table 16-7: PEA Production Schedule over proposed LOM.

Orebody	Mined Tonnage (kt)	Mined Grade (% B2O3)	B2O3 (kt)	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Yr11	Yr12	Yr13	Yr14	Yr15	Yr16	Yr17	Yr18	Yr19	Yr20	Yr21
Zone 4 IND	295	32.9	97	286	9																			
Zone 4 INF	144	32.0	46		144																			
Zone 1 IND	1,355	29.2	396		155	322	322	322	232															
Zone 3 IND	1,156	29.6	342						89	319	319	319	111											
Zone 2 IND	383	27.1	104										227	157										
Zone 9 IND	50	22.3	11											50										
Zone 1 INF	1,916	26.5	508											154	356	356	356	356	339					
Zone 3 INF	593	26.3	156																17	359	218			
Zone 2 INF	853	25.8	221																		144	365	345	
Zone 9 INF	160	24.3	39																				21	138
	6,905	27.8	1,919	286	308	322	322	322	321	319	319	319	337	360	356	356	356	356	356	359	361	365	366	138
ROM grade		(% B2O3)		32.9	30.6	29.2	29.2	29.2	29.3	29.6	29.6	29.6	27.9	26.2	26.5	26.5	26.5	26.5	26.5	26.3	26.1	25.8	25.8	24.3
Indicated	3,239		950	286	164	322	322	322	321	319	319	319	337	206										
	29.3			32.9	29.4	29.2	29.2	29.2	29.3	29.6	29.6	29.6	27.9	26.0										
Inferred	3,666		969		144									154	356	356	356	356	356	359	361	365	366	138
	26.4				32.0									26.5	26.5	26.5	26.5	26.5	26.5	26.3	26.1	25.8	25.8	24.3

16.7 Mine Operations and Construction

16.7.1 Ventilation

With consideration to the rules of thumb that:

- The total mine air requirement in mechanised mines not requiring heat removal: is 0.08 m^3/s/tonne ore for intensive mining with complex geometry; and
- A mechanized cut-and-fill mine with diesel equipment typically has an airflow ratio of 12 t of air per t of ore. A non-diesel mine has a ratio of 7:1.

On the basis of annual production rate of 340,000 t, over 240 working days, the planned daily production rate is 1,413 tonnes. The calculated airflow requirement is 113 m^3/s, which at an air density of 1.2 kg/m^3 (1 atm at 20°C) is 8.3 t_{air}/t_{ore}.

With a 30% contingency applied, 150 m^3/s ventilation rate is considered appropriate.

On the basis that the decline is the main air intake to the mine, a cross sectional area of 30 m^2 will be required to limit air speed in the travel way to the legislated maximum of 6 m/s.

16.7.2 Backfill

It has been assumed that 85% of the mined void would be filled. As discussed, backfill design and materials selection will be subject of future investigation into the availability and characterisation of materials from which a plan to provide backfill for the mining operation can be developed.

Given the relatively small mine production rate, a number of placement options are likely to available ranging from trucked backfill and mechanical placement (stowing) to borehole and pipe distribution and placement. This will need to be subject of further study.

For the purposes of the PEA, a stowed backfill system has been conceptualised comprising dry backfill materials with cement slurry addition prior to stowing.

Dry backfill material would be transported underground by truck and handled into working area by LHD and the section conveyor used during excavation to transport ore from the stoping area. The section conveyor run direction would be reversed to deliver backfill into the stope area where it would feed a slinger conveyor that places the material and allows stowage to the roof. Cement slurry blinder would be added to the backfill mix at the section conveyor discharge point at a regulated rate and mix concentration. The cement slurry system would be containerised, the arrangement including hopper, mixing tank and slurry pump, and would be located underground near to the area being backfilled. The equipment would be electrically powered and skid mounted to enable relocation in the mine.

16.7.3 Second egress

Given that ventilation requirements are not expected to require a second fresh air intake, it is possible to equip the return ventilation shaft with an emergency hoist to provide a second means of egress for the mine. This will be subject to regulatory approval.

16.7.4 Materials handling

Materials handling is considered to be by conveyor direct to surface and a surface storage facility adjacent to the mine portal. This will be located in the mine decline and will run parallel with other mine traffic.

A materials handling study will be required to identify the most appropriate way to deliver mined ore from stopes to surface.

16.7.5 Decline construction

As the only ventilation intake, a 30 m^2 decline heading will be required. This size of development will also help accommodate the materials handling conveyor use to take excavated ore from the mine.

The geotechnical conditions in which the decline will be constructed need investigation and a ground control plan created. However, it is expected that the siltstone / mudstone host rocks will predominated and these are weak, sedimentary rocks that are likely to be bedded. As a main travelway, the heading will need to be supported along its length; and in areas of poor and very poor ground conditions that might be expected around structural features additional ground support will be required.

This decline around 1,600 m long is required to access the orebody. For the purposes of this cost estimate 60% of the decline length is classified Poor, 10% Fair, and the remaining 30% as Very Poor. These classifications follow the Rock Classes used in the Q system support classification, from which the type of ground control that might be employed at Piskanja are shown (Figure 16-13).

In addition, 15% of decline length for development is included to account for pumping, materials handling, and ventilation requirements.

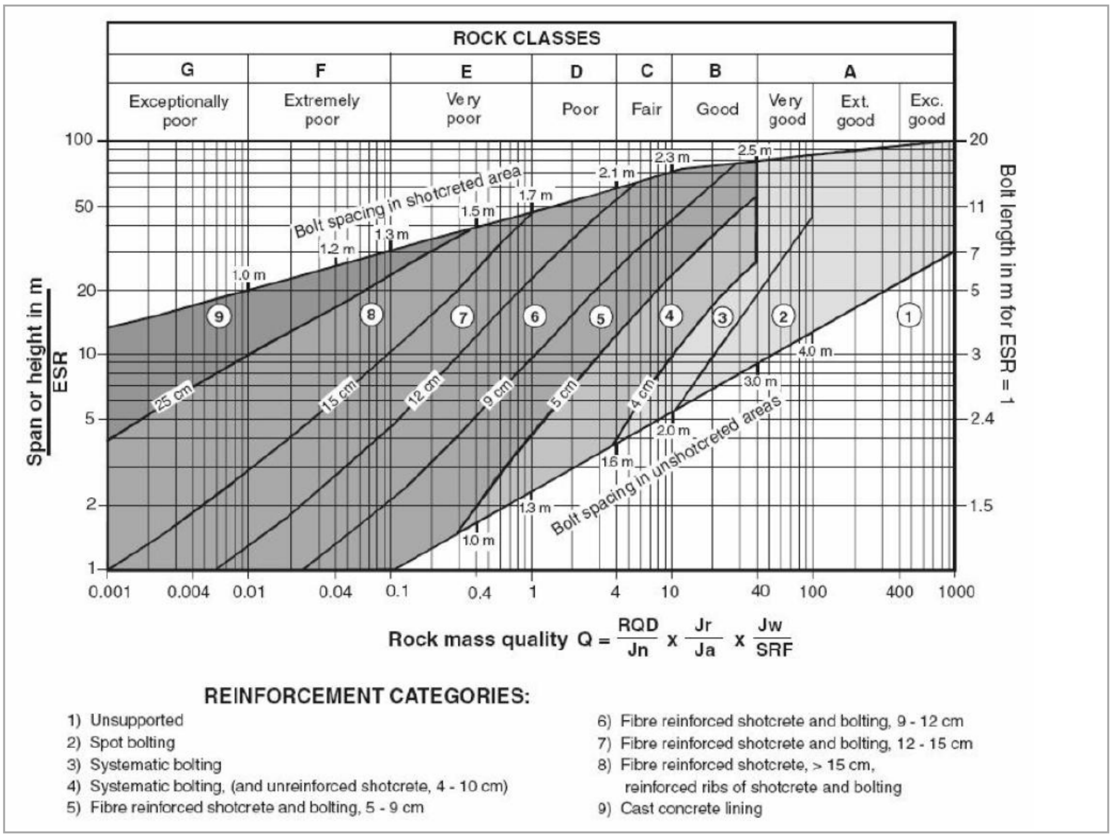


Figure 16-13: Barton (2002) support requirements.[2]

16.7.6 Mining equipment

Two continuous miners with a shuttle car each are envisaged as being required for mine production. A productivity study will be required once mine layouts are completed for the mine design to determine the fleet make. However, although it is considered that two continuous miners are likely to be under-utilised, two machines are likely to be required to ensure there are sufficient working places available to maintain production and to mitigate against the risk of production stoppages due to unavailability of equipment.

17 RECOVERY METHODS

17.1 Processing Assumptions

As noted in Section 13, no technically feasible process route has yet been demonstrated that can upgrade the Piskanja ore to what is considered to represent a minimum marketable Colemanite concentrate grade.

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

For the purpose of the PEA presented later in this report, SRK has assumed that a process can be developed to upgrade the ore to satisfactory levels of B_2O_3 and Fe, to the following criteria:

- A Concentrate grade of 35% B_2O_3, for the range of head grades in the proposed mine plan (24.3-32.9% B2O3); and
- A Tailings grade of 7.5% B_2O_3, the figure achieved for the magnetite fraction in the SGS HIMS testwork (see Table 13-1).

The production plan calls for the production of both Colemanite concentrate and Boric Acid, the latter at a rate of 25 ktpa, and the former at a rate of approximately 200 ktpa. SRK has modelled this production scenario according to the process route shown in block form in Figure 17-1.

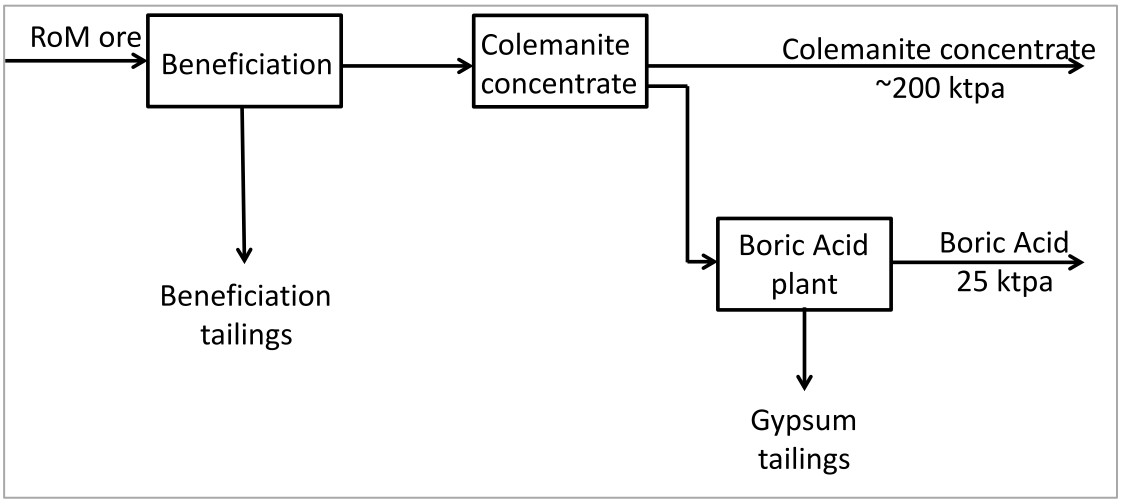


Figure 17-1: Conceptual Process Block Diagram

The mass yield and B_2O_3 recovery across the beneficiation stage are variable, a function of the variable RoM grade and the fixed concentrate and tails grades detailed above. For the Boric Acid plant, a fixed B_2O_3 recovery of 80% has been assumed as representing a typical industry value.

The quantity of beneficiation plant tailings is projected to range from 22 ktpa to 123 ktpa over the life of the project, averaging 88 ktpa. The quantity of tailings from the Boric Acid plant is expected to be 49 ktpa.

17.2 Tailings Management

17.2.1 Introduction

To support the PEA, SRK has undertaken a desktop study to determine the possible location for the tailings storage facility (TSF) to store 5% of 1.8 million tonnes (Mt) of colemanite and 5% of 1.0Mt of boric acid tailings material. The amount of tailings material to be stored within the tailings storage facility totals to 0.135Mt, i.e. 0.122Mm^3 (Table 17-1) at the dry density of 1.1t/m^3. The remaining 95% of boric acid and colemanite tailings will be utilised for backfill.

Table 17-1: Tailings Material Distribution

Tailings Material	Total Produced	95% to backfill	5% to TSF
Tonnage (t)			
From Colemanite Plant	1,810,707	1,720,171	86,009
From Boric Acid Plant	1,019,866	968,873	48,444
Total	**2,830,573**	**2,689,044**	**134,452**
Volume (m3)			
From Colemanite Plant	1,646,097	1,563,792	78,190
From Boric Acid Plant	927,151	880,793	44,040
Total	**2,573,248**	**2,444,586**	**122,229**

17.2.2 Site Selection Study

The site selection study was not carried out as the Client's preferred location was the site located immediately south of the proposed Plant. The Client stated that this site was previously used by the Ibar Mine to clarify water for the coal washing plant therefore was a perfect location for the tailings storage facility. Furthermore it was stated that in a primarily agricultural district this is a major advantage. SRK considered the advantage and agreed with the location selected.

The location of the site including the proposed plant and portal area is shown in Figure 17-1.

Figure 17-2: Tailings Storage Facility Location

17.2.3 TSF Design

A conceptual tailings storage facility design has been developed by SRK to store about 150,000m^3 of tailings material that occupies an area of about 6ha including the dam footprint area. SRK considered a downstream construction method for the proposed facility which will require the construction of full height dam walls using suitable material for construction acquired from local borrow areas. The dam wall will have slopes of 1V:2.5H on the downstream side and 1V:3H on the upstream side and a crest width of 5m. The maximum dam height at the lowest ground is about 4.4m, and the crest elevation at 381 metres above sea level (masl). A freeboard of 1m was added to the dam height.

Tailings waste will be disposed as slurry which will be delivered via a pipeline into the facilities. SRK did not consider dry stacking at this stage as there are many unknowns regarding the final tailings product. In order to evaluate the preferred tailings deposition method, SRK recommends that the geophysical, rheological and geochemical testing on tailings material is carried out.

SRK's conceptual design for the TSF is based on the following criteria:

- The need to accommodate 150,000m^3 of colemanite and boric acid tailings at a slurry density of 1.1t/m^3;
- 5% of 1.8Mt colemanite tailings production – about 86,000t;
- 5% of 0.9Mt boric acid tailings production – about 50,000t;
- 95% of total tailings production will be used for backfill;
- The dam configuration consists of 5m wide crest with the side slopes of 1V:2.5H on the downstream side and 1V:3H on the upstream side. The dam will be constructed out of mine waste or from the local borrow areas;
- The TSF will have decant system to decant/pump any excess water from the facilities. The drainage system will drain to a collector sump by gravity that will be located at the lowest point of the TSF. Supernatant water recovered from the facility will be pumped back to the Plant;
- An emergency spillway will be constructed to accommodate storm conditions;
- Knowing the nature of boric acid and colemanite tailings, SRK incorporated a double 2mm HDPE liner system. In addition a leakage detection system between the liners and below the bottom liner will be installed to detect any HDPE liner leakage.

17.3 Recommendations

As noted in Section 13, only a very limited amount of metallurgical testwork has been undertaken in support of the production of this PEA. This work consisted of some upgrading testwork, which while resulting in a reduction in the Fe content of the product to close to the assumed target level, did not result in any significant upgrading of the boron content, at least not to the assumed minimum figure of 35% B_2O_3 for a saleable Colemanite concentrate.

The metallurgical parameters developed for the PEA are therefore largely assumptions based on a limited amount of non-definitive data. In addition, virtually no specific engineering was conducted with regard to the process plant design, and the process plant capital and operating costs subsequently generated are high level estimates based on generic databases and parallel project data.

Further development of the Piskanja project will therefore require the execution of metallurgical testwork and plant engineering programs commensurate with the level of study being undertaken. Given the flowsheet proposed in this PEA, a future metallurgical testwork programme should focus on the technical feasibility of upgrading the ore to meet the B_2O_3 and Fe specifications, and the effect of head grade on both this potential for upgrading, and on the resulting recovery. Boric Acid production testwork should determine the effect on key parameters, such as the sulphuric acid consumption, of variations in the feed grade to Boric Acid production.

Samples selected for the metallurgical testwork programme should cover the expected range of potential variability within the Piskanja orebodies. The variability parameters should include grade and mineralogy (i.e. varying ratios of Colemanite to Howlite), as well as location within the orebody, such as lateral extent and depth.

On the basis of more specific process parameters developed from this testwork, a more detailed plant engineering study can be undertaken, again commensurate with the precision of the overall study.

The following studies are required at the next level of study to further define the project components and confirm the tailings management assumptions made within the PEA:

- A topographical or LIDAR survey of the proposed location to provide a topographical map with contours of 1m accuracy;
- Identification of borrow areas location ;
- Land access and acquisition to be confirmed;
- Geotechnical, geochemical and rheology testing of tailings material;
- Geotechnical and geochemical testing of borrow areas and the in-situ ground conditions within the TSF footprint;

18 PROJECT INFRASTRUCTURE

18.1 Introduction

This section of the report presents the results of preliminary studies undertaken by SRK to determine the infrastructure and services / utilities requirements needed to facilitate the mining and commercial export of the borate product(s), Colemanite concentrate and Boric Acid as presented in the preceeding sections.

18.2 Existing Project Area Infrastructure

18.2.1 Overview

The Project area is serviced by existing regional rail, road, power and water supply infrastructure. The town of Baljevac is located immediately to the west and a number of operational mines are located in the vicinity of the project as well as other manufacturing and industry.

18.2.2 Existing Operations

Operational and disused mines are located within the vicinity of the Project.

Ibar Coal Mines Company produces around 30,000 to 50,000 tonnes per year of coal from a number of open pits and underground operations both in in Baljevac and to the northwest of Baljevac. Coal is transported to a reception facility at Baljevac by aerial tramway where a transfer station directs to the Coal Preparation Plant (crushing, screening, and washing) on the east side of the River Ibar. At the Preparation Plant, washing and screening occurs and coarse material is loaded directly to ore wagons while the fines which contains high moisture, are settled and dried and loaded by front end loader (FEL) or truck.

Ibar Coal Mines Company also owns a number of packages of land for mining, processing and support facilities including:

- Coal mine and support areas;
- Coal preparation / processing buildings;
- Waste dump
- Settling area for fines and wet process products
- Stockyard area (also for Pobrdje borate); and
- Sidings / loading area.

Currently, Erin Ventures utilises existing office space and core store facilities currently owned by Ibar Coal Mines Company under a lease arrangement.

A producing Borate Mine is located at Pobrdje, 2.6 km to the west-northwest of the Project. The mine produces around 500 t per year of borate feed material. The run of mine material is transported by road to a sorting area at the Ibar Coal Mines Company (comprising an office and concrete laydown area) where it is sorted by hand, bagged and exported by road.

Historically, both Magnesite and Asbestos were mined in the area. Asbestos was mined 2.5 km east of the Korlace, which is located at the eastern edge of the Project. The Magnesite mine was located 2.5 km west. Other industry, and former industry, also occurs at Baljevac, including a former Fibreglass Factory located to the southwest of the Coal Preparation plant.

18.2.3 Road

Access to the Project is by paved road from Belgrade, a journey that takes approximately 4 hours and passes through the towns of Kragujevac and Kraljevo. The proposed mine site is in close proximity to the “No.22 regional road” via a small access road that crosses the River Ibar. The No.22 road is of bituminous construction and links the project area to the city of Kraljevo to the north (66 km) and to Kosovo border to the south (26 km).

18.2.4 Rail

Rail Infrastructure

A standard gauge single track railway passes through the licence area and connects to Belgrade via the towns of Kraljevo and Raška. Further connections to Thessaloniki in Greece, to the south through Pristina and Skopje of Kosovo and Macedonia are possible. The railway is used for both commuter and freight traffic.

Kraljevo possess a good regional and international rail networks enabling it to be linked to the following inland river port terminals at Belgrade and Smederevo, and Ports of Constanța (Romania, north-east), Thessaloniki (Greece, south) and the Port of Bar in Montenegro (west)

Rail infrastructure is owned and maintained by The Railways of Serbia Company (“Zeleznice Srbije”), a state owned entity. RSC also operate commuter and freight services within the country and can provide a range of rolling stock types. Independent freight service providers are also available within the country.

Current Usage

The existing coal operations utilise the railway for export of coal products. Three sidings are located at the facility running adjacent to the main line. The existing loading facility is in poor condition. The rail alignment comprises wooden sleepers; the sidings are in variable condition with one shown to be serviceable however the mainline appears maintained and in generally reasonable condition; track speeds, gradients, capacity and logistics have not been assessed.

It is understood that rail travel times within Serbia can be below average however, given the production rate, likely available capacity, and provided adequate planning and scheduling is undertaken, this isn’t anticipated to affect the Project.

18.2.5 Inland Waterways

The River Ibar runs adjacent to the project area but is not considered suitable for navigation. The nearest inland river port for consideration is in Belgrade from where products can be transported by barge along the Danube to the black sea or to Western Europe.

18.2.6 Power

Power generation in Serbia is reported to be 70 % coal fired and 30 % by renewable energy sources. The general project area is supplied by a 35 kV transmission line and a 10 kV distribution lines which were observed during the visit.

Local industry, including the Coal Preparation plant, the Coal mine and factories are supplied by a 6 kV distribution line from a 35 kV / 6 kV substation and transformer located at a former Fibreglass Factory site. SRK understands the Coal Preparation plant has an installed demand of 1.6 MW. To meet local demand the 10 kV line is stepped-down to 220v and 380v supplies.

18.2.7 Water

The Project area has a developed water supply network and both raw and potable water are supplied to the nearby mines and business. Historically the existing coal preparation plant extracted raw water from River Ibar for wet processing.

18.3 Production Scenario

The current envisaged mining rate is anticipated to be approximately 339 kt per annum ("ktpa") ROM to produce around 225 ktpa of saleable products. Saleable products comprise 200 ktpa of colemanite concentrate, crushed, screened and bagged for export. A waste stream will result from the crushing and screening. A boric acid plant will be located at the site to produce around 25 ktpa of Boric Acid. The productions scenarios have considered to inform the PEA are presented below (Table 18-1).

Table 18-1: Run of Mine production Scenarios

Product	Production Rate per annum	ROM	Export Scenarios
Colemaniteconcentrate	~200,000 t	Granular material (dry / wet) with a P80 of around 20 mm.	West Europe (various including Mediterranean and northwest Europe) China.
Boric Acid	~25,000 t	Acid (Powder)	Europe

Colemanite concentrate product will be sold "free on board ("FOB") mine site" with the FOB point the point of loading for export. Access road construction, transportation, load out area and mobile equipment is considered within the PEA costing. It is assumed the project can utilise the existing areas and sidings of the Ibar Coal Company with activities can be coordinated with the Ibar Coal Company.

A Boric Acid Plant (modular) will be located on site and will a) produce a granular boric acid product, b) require sulphur to be transported to site, and c) produce a waste stream for disposal. Based on this, the load-out area shall require a sulphuric acid reception tank and appropriate mobile equipment to transfer sulphuric acid to the Boric Acid Plant.

The project requires support infrastructure (administration, change house, welfare and canteen facilities), warehousing, laydown areas, and workshops. Loading facilities (or sufficient surface area) will be required for packaging and export.

18.4 Proposed Infrastructure

18.4.1 Overview

The Piskanja Mine will utilise existing industrial zoned land located to the east of Baljevac adjacent to the Ibar Coal Mines Company coal processing facility. A discussion of the risks, benefits of the location, and possible access alternatives, is included in Section 16.

An entrance "portal" will be constructed with mine support infrastructure, a crushing and screening operations area and the Boric Acid Plant at surface. The product load-out will be located around 200m away to the southeast adjacent to existing railway sidings which will be refurbished. Associated utility supplies and security will be provided and the site shall be accessed via an existing access road alignment which will require refurbishment.

The overall general arrangement is presented in Figure 18-1 below:

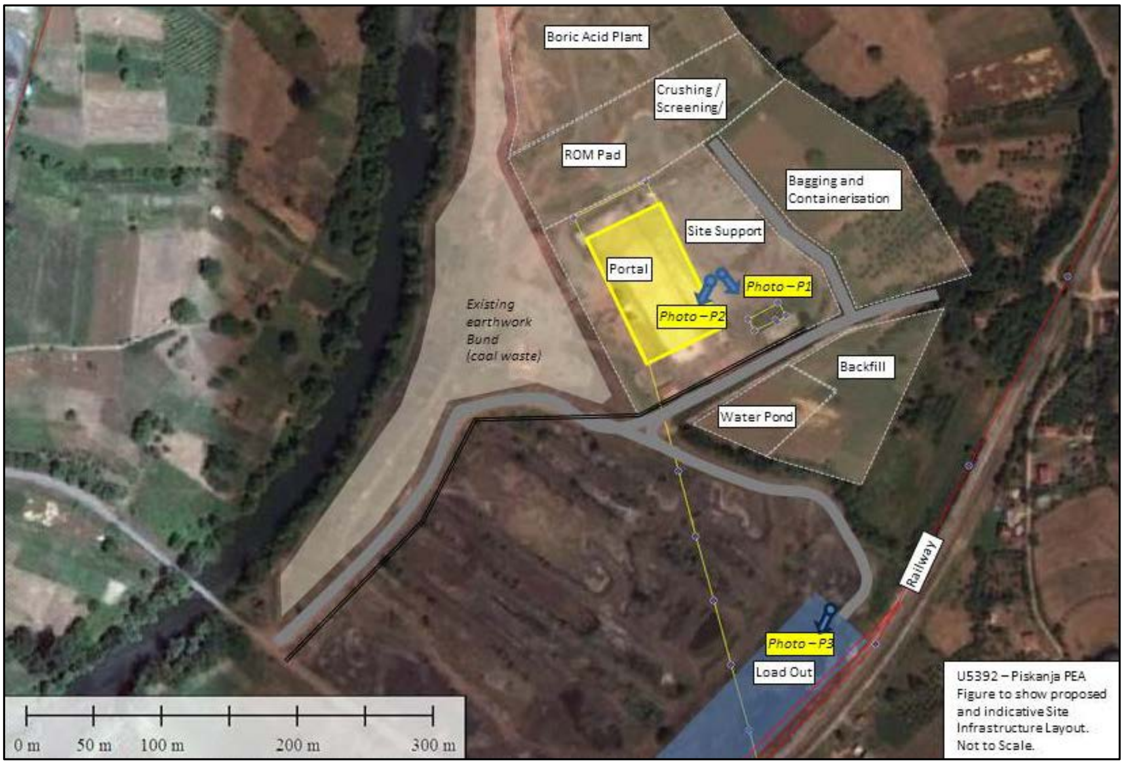


Figure 18-1: Piskanja Mine Infrastructure Layout

18.4.2 Site Support Infrastructure

The following support and operations infrastructure will be located to support mining and processing:

- Administration and planning building for mine planning and technical services, welfare / change-house, security and first aid (approximately 40 m by 30 m single storey);
- multi-purpose workshop, laydown area and warehouse (approximately 40 m by 20 m single storey); and
- Water supply and storage, power supply infrastructure.

The exact land requirements will be determined at a later stage of study. For the purpose of this study, a working area of approximately 150 m by 150 m has been allocated for this infrastructure around the portal entrance. There is additional land to the north and east should it be required subject to negotiation with the current land-owners.

Figure 18-2: (For location, see Figure 18-1 "Photo P1") Existing Industrial Land proposed for site infrastructure and Portal (photograph taken from proposed portal location looking south-southwest towards Ibar Coal Mines coal processing facility). Note existing power infrastructure. April, 2014.

Figure 18-3: (For location, see Figure 18-1 "Photo P2") Existing Industrial Land (photograph taken from proposed portal location looking southeast). April, 2014.

18.4.3 Crushing & Screening Area

Exact land and foundation requirements will be determined at a late stage of study. For the purpose of this study, a working area of approximately 50 m by 150 m has been allocated for crushing and screening plant in proximity to the portal entrance (Figure 18-1). A ROM stockpile and crushing and screening plant shall be situated on a concrete apron. The crushing and screening plant and product stockpiles may need to be covered and structures are allocated.

18.4.4 Product Handling

There are two possible options for product load-out of Colemanite concentrate:

- Product is bagged (e.g. into "1 tonne bulk bag") which is then containerised for load out onto flat-bed rail wagons (or road haulage trucks) using a reach stacker; or,
- Conveyor transport to a covered warehouse for direct / indirect feed to ore-hoppers. Indirect feed would using a front end loader ("FEL").

The PEA concept utilises containerisation of 20 tonnes into "twenty feet equivalent unit standard container units (TEU)" for load-out. Based on the assumptions presented in Table 18-2 below, approximately 45 TEU will be completed, handled and loaded per working day.

Table 18-2: Assumptions for Product Handling and Load-Out

Basis	Number	Unit
Production rate per year (total)	225,000	Tonnes
Tonnes per TEU	20	Tonnes
TEU per year	11,250	TEU
Shipment days per year (5 days per week, 50 weeks per year)	250	days
TEU shipped per day	45	TEU
TEU filled per day (7 days for 50 weeks)	32	TEU
TEU per consist (assumption)	45	TEU
Trains per day (assuming 30 TEUs per consist)	1	Day
Length of train (6.10 m per TEU) excluding locomotives	270	m

An approximate 75 by 150 m area has been allocated for the containerisation area. Within this area, empty TEUs will be stockpiled (15 m2 area per TEU) up to 2 TEUs height and TEUs will be loaded ready for transport. The containerisation area will be serviced by two "reach cranes" for manipulation of TEUs and transport to the load-out area.

An alternative option, dependant on the market destination, would be to transport TEUs using road haulage.

18.4.5 Load-Out Area

For product load out, the project assumes a parcel of land adjacent to existing sidings immediately to the southeast of the Portal. An area with dimensions of 300 m by 50 m has been allocated for the load-out area. The parcel of land currently forms part of a stockpile area owned by Ibar Coal Mining Company (Figure 18-4).The area can be access via an existing access road and earthwork which requires refurbishment or rebuilding. A cost for rebuilding has been considered in the estimate.

Figure 18-4: (For location, see Figure 18-1 label "Photo P3") Existing Industrial Land adjacent to mainline railway and sidings to be refurbished and utilised for load-out. Photo looking towards the south-southeast. April, 2014.

Based on the expected length and numbers of trains and accounting for return of empty TEUs and train marshalling, two 750 m lengths of existing rail siding require acquisition and refurbishment (renewal of rails, ballast and ties). Load-out will utilise mobile equipment from the containerisation area. An allowance for an open sided portal steel framed warehouse has been made at the load-out area.

An alternative location for load out exists to the south of the Ibar Coal Mining Company processing facility however this would result in increased transfer distances and use of public access roads.

18.4.6 Boric Acid Plant

It is proposed that the Boric Acid Plant will be located proximal to the mine entrance. The modular plant facility will be supplied, installed and commissioned by a third party and will include all civil, structural, electrical, piping, utilities, water and fuel storage and back-up power generation.

The exact land requirements will be determined at a late stage of study but for the purposes of the study, a working area of approximately 200 m by 200 m has been allocated for this infrastructure. A development pad assuming only ground preparation and levelling will be provided as well as perimeter fencing and dedicated access point.

18.4.7 Site Access Road

An approximate 1 km of site access road between the existing public road, the mine entrance and the load-out facility will require upgrading and refurbishment. The current alignment exists and but will require preparation and development of an unbound surface for access as well as drainage ditches on each side.

18.4.8 Earthworks

The Portal Entrance is located adjacent to the River Ibar on the floodplain area. A “box-cut” will be formed to access rockhead through a thickness of soft sediments which overlie. The box-cut at Piskanja will utilise a soil retaining structure (temporary or permanent) to minimise footprint and retain the groundwater. The entrance may need to be raised due to flood risk and would require additional earthworks however this would be assessed at a later stage of study together with the portal location itself. An area of 50 m by 100 m has been delineated for the final portal footprint.

General earthworks will be required at the access road entrance to the load-out. The existing ground levels rise by around 3-5 m in elevation to the existing vertical alignment of the railway. An earthwork constructed from imported placed and engineered fill will be required to support the access road.

18.4.9 Utilities / Security

Power will be supplied from the national grid. A 1 km overhead distribution line from the nearest appropriate point with substation and switchgear is be required.

Raw water will be abstracted from the Ibar River subject to the required permitting. A suitable pump and pipework will be required. A water settling pond is provided for mine water.

A 1.5 to 2.0 m chain link fence will be erected around the perimeter of the various infrastructure elements to protect the site from theft and to also segregate the general public from the mining and processing activities. A security cabin and barriers will be positioned by both the mine entrance and Boric Acid Plant.

18.5 Export Logistics

The intended point of sale for products is “FOB mine site” determined as the point of loading onto either rail wagons or road vehicles thus downstream costs for freight and rehandling costs at river ports are not considered. However, SRK has overviewed possible logistical solutions to confirm export routes to the anticipated markets exist (Table 18-3).

Table 18-3: Export Scenarios

Option	Description	Approximate Distance
1	Rail to the Port of Bar via Kraljevo. Bulk terminal facilities are available and destinations in Western Europe are accessible.	430 km by rail
2-A	Rail to the Port of Constantia via Kraljevo. Bulk terminal facilities are available and destinations in the Far East are accessible.	1042 km by rail
2-B	Rail direct to destinations in Western Europe.	1146 km by rail (Germany border)
2-B or 3-B	Rail to inland river ports at either Belgrade and Smederevo for barge transportation along the Danube into Western Europe or to the Port of Constanța	Rail to Belgrade or Smederevo: 250 km / 198 km Barge transport to Constanta: 861 km or Barge transport to Germany: 1150 km
5	Rail to the Port of Thessaloniki via Kosovo and Macedonia	444 km

For this PEA study, SRK proposes utilising the nearby rail infrastructure to reach inland destinations in Western Europe or international ports on the Mediterranean or Black Sea. Site observations suggests there to be available capacity on the adjacent rail.

Alternatives options may consider barge transportation from Belgrade to Western Europe or international ports on the Black Sea although this would result in an additional stage of double handling. However, the site is adequately accessed by national roads and considering the tonnages anticipated a road haulage option also exists.

A summary of the possible options is presented in Figure 18-5.

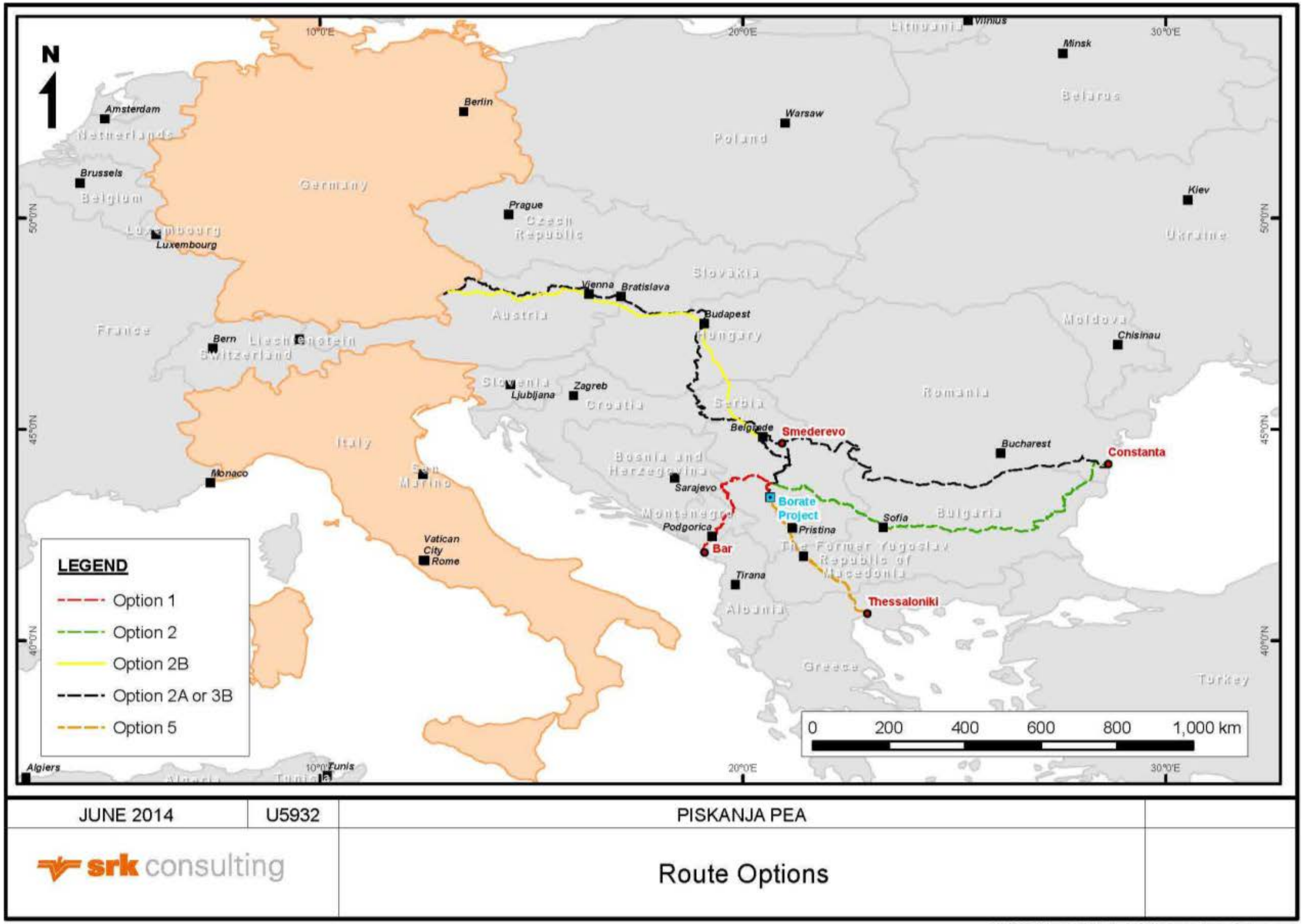


Figure 18-5: Possible Export Routes from the Project

18.6 Further Studies / Limitations

The following studies are required at the next level of study to further define the project components and confirm the assumptions made within the PEA:

- Portal entrance location to be confirmed.
- Land access and acquisition to be confirmed;
- The layout and interrelationships of infrastructure components to be defined for material flows and efficiencies;
- Logistics study is required to confirm rail capacity and a power study and engagement of the power provider(s) to confirm supply and capacity;
- Preliminary engineering design for portal development, standard foundation detail and structures and / or budget estimates for equipment, refurbishments and structures as appropriate; and
- There is a risk of poor ground conditions beneath the site and an intrusive ground investigation is required for the portal entrance, Boric Acid Plant location and other structures to properly define the risk.

19 MARKET STUDIES AND CONTRACTS

Neither the Company nor SRK has undertaken a market or contracts study for this report.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Environmental and Social Setting

The Project is located in south-central Serbia, approximately 160 km south of Belgrade, and approximately 17 km north of the Kosovo border. Administratively, the Project is in the Raška District and Municipality. The nearest villages are Brvenik, which is in the immediate vicinity of the deposit area, and Baljevac, which is approximately 1.5 km north-west of the project area. The regional capital, Raška, lies approximately 10 km to the south of the project area.

The project area is located on the lower north-western slopes of the Kopaonik Mountain Range; the largest mountain range in Serbia. Altitudes in the project area range between approximately 375 and 625 m above mean sea level and the climate is characterised as moderately continental. The average temperature in January is -1^{0}C, while that in July is around 19^{0}C. Winter temperatures are not as low as in other areas of Serbia, due to the southerly location. Average annual precipitation measured in the region of the site is 647 mm (Priboj Selo station), with the highest rainfall in the months of May to September (monthly average around 49-62 mm).

The Project is located within the Ibar River drainage basin, a trans-boundary river which flows through eastern Montenegro, Serbia and Kosovo prior to discharging into the Black Sea. The deposit area is drained by three streams; Korlaćki, Radić and Kurićki. All are tributaries of the Ibar River. Kurićki stream, located to the north of the deposit area, is approximately 5 km long and has a catchment area of approximately 5.5 km^2. There are a number of springs in the upper reaches maintaining a small perennial flow, which is augmented during runoff events. Korlaćki stream, located to the south of the deposit area, is approximately 5 km long, with a catchment area of approximately 6.5 km^2. Radić stream is located between the Kurićki and Korlaćki streams.

Groundwater is recharged within the slopes of Kopaonik Mountain. Localized spring discharge occurs where the slope decreases and at the bases of creek valleys. The dominant groundwater use is garden irrigation and small scale subsistence farming; use of groundwater as a source of potable drinking water occurs at a few properties.

The dominant land use is small-scale subsistence farming. The river flood plains and lower mountain slopes are cultivated for crops and fruit. Steeper slopes, above 500 mamsl are generally covered by sparse deciduous woodland.

Brvenik, Baljevac and Raška have populations of 67 (2002 census), 1,482 (2011 census) and 6,574 (2011 census), respectively. The population in the Raška municipality was 24,680 in the 2011 census. About 4,000 people have migrated out of the municipality in the last two decades. The negative population growth can be attributed to the stagnation of economic development of the region, which has led to population migration from the municipality to more developed parts of Serbia. The unemployment rate in the Raška municipality was 41.33% in 2011.

The deposit is located in the Kopaonik metallogenic district of Serbia; the district has seen production from many small deposits of lead, zinc, silver and iron since the Middle Ages. Exploration during the past 100 years has resulted in the discovery and development of asbestos, coal and magnetite properties. State-owned Ibarski Rudnici Coal Company operates a coal mine north of Baljevac.

20.2 Environmental and Social Approvals

Mining authorisations

As already commented, mining rights in Serbia are currently governed by the Law on mining and geological exploration (Official Gazette RS Number 88/2011); the responsible government agency is the Ministry of Energy and Mining. A new Law on Mining and Geological Survey, which is drafted and pending adoption by the Serbian Parliament, provides for a series of mining permits granting stepwise permission to develop a mine.

Prior to exploitation, Erin must obtain exploitation approval and approval of mining works. The key difference between these two approvals is the engineering document required to support the application – i.e. a Feasibility Study is required for the exploitation approval, and a Main Mining Project is required to support the application for approval of mining works. These approvals must be obtained prior to construction.

A further approval for use of mining facilities can only be obtained once the mining facilities have been constructed. The technical submissions required in support of applications for exploitation approval, approval of mining works and use of mining facilities is outlined in Table 20-1 below.

Table 20-1: Documents required for mining permit applications

Mining approval	Document required	Further detail on the evidence required
Approval of exploitation/ Approval of mining works	Map (1:25,000)	-
	Certificate of Mineral Resources and Reserves	-
	Feasibility Study / Main Mining Project	Feasibility Study - for the approval of exploitation. Main Mining Project - for approval of mining works.
	Evidence of compliance with urban planning	An act of municipal authorities in charge of urban planning with regard to exploitation compliance with the appropriate spatial and urban planning. A Spatial Plan for Special Purposes will have to be prepared to inform this agreement and has to be approved by regulatory authorities.
	Evidence of compliance with legislation on environmental protection	An act of the ministry competent for environmental protection on compliance of the exploitation with the environmental protection regulations.
	Evidence of compliance with cultural heritage legislation	An act of the ministry competent for protection of cultural heritage on compliance of the exploitation with the cultural heritage regulations.
	Evidence of compliance with water management legislation	An approval from the water management body, if mining operations affect the water regime.
	Evidence of surface rights	Proof of ownership or easement (usufruct) rights to the land designated for the construction of mining and mineral processing infrastructure.
Approval for the use of mining facilities	Evidence the mining facilities have been constructed in accordance with the Main Mining Project on the basis of which the approval for mining works was granted.	

Environmental authorisations

The Law of Environmental Protection (Official Journal RS, No. 135/O4, 29/10) oversees the management, support, restoration and preservation of natural resources and natural, cultural and historical heritage. It aims to prevent all forms of pollution, nuisance and deterioration of natural, social and cultural environments, and to preserve human, animal and plant health and to ensure the security of property and people. This law also deals with atmospheric emissions, waste disposal and the import, production or use of hazardous substances.

The Law on Environmental Impact Assessment (Official Journal RS, No. 135/O4, 36/09) requires that environmental impact assessment (EIA) licences are obtained for projects with potential to have significant impacts. Projects subject to an environmental impact assessment are outlined in the Decree on Determining the List of Projects Requiring Mandatory Environmental Impact Assessment and List of Projects Requiring Optional Environmental Impact Assessment (Official Journal RS, No. 114/08). The Law on Environmental Impact Assessment charts the procedure to obtain an EIA licence briefly, as outlined below.

- An application is submitted to the competent authority to determine the scope and content of the EIA. As part of determining the scope and content of the EIA, the competent authority will seek public opinion.
- The applicant should then prepare an EIA (applicants have one year to prepare the EIA following receipt of the required scope and content from the competent authority).
- On receipt of the EIA, the competent authority will open the EIA for public inspection and there will be a public hearing. The competent authority will notify the public of this (with an interval of at least 20 days between the public notice and public hearing).
- The EPA will submit the comments on the EIA, together with the EIA, to a Technical Commission that will provide comments (and a suggested decision) to the competent authority.
- The competent authority will make the decision whether to issue an EIA licence. An EIA licence will have a period of validity and contain conditions for the protection of the environment.
- The competent authority is required to inform stakeholders of the decision, including the contents of the decision, the justification for the decision, and the key measures to be implemented to prevent, mitigate or remediate impacts.

The EIA procedure is part of the permitting process for construction, integrated pollution prevention and control (IPPC) and waste management permits.

International legislation

Serbia officially applied for European Union membership in December 2009 and is aiming to achieve European Union accession within 5 to 7 years of this application. For this reason, the project should strive to comply with European Union Law (regulations, directives and decisions) as far as possible. The following European Directives are particularly important to mining projects:

- 85/337/EEC – The Environmental Impact Assessment Directive
- 2004/35/EC – The Environmental Liability Directive
- 2010/75/EU – The Industrial Emissions Directive
- 2006/21/EC – The Mine Waste Directive
- 2008/98/EC – The Waste Framework Directive
- 2000/60/EC – The Water Framework Directive
- 2008/50/EC – The Ambient Air Quality Directive (Pure Air for Europe)
- 1992/43/EC – The Habitats Directive (Natura 2000)

Voluntary international standards

There are a number of business charters, codes of conduct / ethics / practice and good-practice guidelines that have been developed by industry (often in partnership with key stakeholders). Those of particular importance to environmental management and sustainable development in the mining sector are:

- The International Council on Mining and Metals' Sustainable Development Framework (which comprises a set of ten principles, public reporting and independent assurance) and numerous best practice guidelines.
- The Voluntary Principles on Security and Human Rights.
- e3Plus – guidance on responsible exploration developed by the Prospectors and Developers Association of Canada (PDAC).
- Towards Sustainable Mining (TSM) – an initiative of The Mining Association of Canada.
- Enduring Value – the Australian minerals industry framework for sustainable development.

While these are largely voluntary, membership of certain industry associations requires compliance. At the same time, increasing numbers of stakeholders expect to see the environmental and social performance of individual companies aligned with these voluntary standards irrespective of membership of the relevant industry association.

20.3 Approach to Environmental and Social Management

Erin's responsibilities, though its subsidiary Balkan Gold as holder of the exploration licence, are described in the "Decision of the Ministry of Natural Resources, Mining and Spatial Planning" dated 05 November 2012. It is understood by SRK that this decree states Balkan Gold is committed to undertaking the activities outlined in the 2012-2015 Exploration Programme submitted to the Ministry at the time of licence application. The 2012-2015 Exploration Programme was approved by the Institute for Nature Conservation of Serbia and the Institute for Cultural Heritage and Preservation prior to it being submitted as part of the licence application. SRK understands the only obligation in the 2012-2015 Exploration Programme pertaining to environmental and social management is the requirement to conduct on-going hydrological and hydrogeological investigations in the area.

Erin intends to initiate an EIA in the subsequent project development phases. A preliminary hydrological and hydrogeological sampling and monitoring network was established by a third party in September 2012, with on-going sampling and monitoring undertaken by Erin. It is acknowledged by Erin that once project information is further defined the scope of these water resources studies may need to be expanded to address a wider study area and to focus on the key issues.

Erin recognises the importance of stakeholder engagement and communicates with government at national, provincial and district levels and with local communities on an on-going basis. Based on the information made available to SRK, there appears to be a good relationship between Erin and government and local-level stakeholders. Erin acknowledged that the on-going stakeholder engagement has not yet been formalised through a stakeholder mapping (identification) exercise to identify stakeholders interested in or affected by the project, development of a stakeholder engagement plan (SEP) and establishing a database of records of past stakeholder engagements.

With respect to the EIA, SRK recognises two types of consultation as outlined in Figure 20-2, Serbian EIA legislation requires stakeholder engagement as part of the EIA process (Section 20.2), however the responsibility for engagement is assigned to local authorities. International standards on environmental and social management promote a more active approach to community stakeholder engagement to ensure constructive relationships with stakeholders are developed and maintained. Active stakeholder engagement, beyond the immediate scope of the EIA, is also considered to be an important tool for identifying and managing environmental and social risks to the project during both development and into operations.

Table 20-2: EIA-specific stakeholder engagements

Type of engagement	Engagement events
Legally required EIA stakeholder consultation	Scoping consultations with project stakeholders (including regulatory authorities and governmental groups, non-government organisations, and local communities)
	Public hearings on the EIA, which are organised by the relevant regulatory authority in partnership with the company
Additional stakeholder engagement to improve the quality of EIA, baseline studies and environmental and social management plans	Environmental-permitting consultations with regulatory authorities, particularly where these pertain to the approach to the EIA, the content of the EIA and EMP and other detailed management plans and environmental approvals to be obtained
	Consultations with local communities and local authorities and community service providers during baseline studies as needed to provide information for these studies
	ESIA-related consultation with specialist interest groups during baseline studies, including non- governmental organisations (NGOs) to provide input on various matters, such as biodiversity matters

20.4 Environmental and Social Impacts and Risks

20.4.1 Anticipated Environmental and Social Impacts

This section provides an indication of the anticipated environmental and social impacts associated with the Piskanja Project and is based on information gathered during the site visit, review of available literature and the experience of the ESIA team on other similar projects.

The list of anticipated impacts in Table 20-3 are at a scoping level and therefore could change in terms of the type, nature and severity and additional impacts could emerge during the characterisation and assessment of environmental and social features of the site and the development of the mining and processing approaches and infrastructure service corridors for road, rail, water and power for the project. Table 20-3 is intended to provide an indication of the likely impacts (positive and negative) that could be expected from a mining development of this nature. These will be re-defined throughout the ESIA process, particularly following stakeholder consultation.

Table 20-3: Anticipated environmental and social impacts of the Piskanja Project

Aspect group	Aspect	Mechanism	Potential impacts
Land trans-formation	Surface disturbance and topographic change at the mine site	Site clearance within footprint of mine and associated infrastructure	• Disturbance of sites of archaeological, historic or cultural importance • Loss of land available to local communities • Changes to land capability • If the industrial land currently owned by the Ibarski Rudnici Coal Company is acquired, there could be positive impacts through rehabilitation of this area
Water resources	Water take	Water abstraction for supply Dewatering of workings	• Interference or reduced availability of water to other users and ecological receptors • Alteration of watercourse flow regimes, resulting in changes to flood patterns, fluvial processes, erosion, aquatic habitat, ecosystems and ecosystem services.
	Water diversion	Interruption of or changes to surface water channels from construction of mine infrastructure	
	Discharges from point and diffuse sources	Seepage from mine and mineral-processing residue disposal / dirty water holding facilities Uncontrolled discharges (such as during storm events, spills, leaks etc.) Wastewater discharges Runoff from exposed surfaces	• Deterioration of groundwater and surface water quality potentially used by communities and ecological systems, for example from increased turbidity from sediment laden runoff, leachate from mine facilities and nutrients from blasting or sewage treatment etc.
Air quality	Point emissions	Vehicle emissions Stack emissions Stationary sources (such as generators, crusher) Incinerators	• Increase in background concentrations of particulate matter (dust) leading to nuisance and health effects for nearby communities • Increase in background concentrations of gaseous pollutants (such as sulfur dioxide, nitrogen dioxide and carbon dioxide etc.) potentially causing health risks to nearby communities
	Diffuse emissions	Fugitive dust emissions from dry surfaces	• Increase in concentrations of particulate matter (dust) leading to nuisance and health effects for nearby communities
Noise and vibration	Equipment/vehicle operation Blasting	Noise emissions	• Increased disturbance to site workers and nearby sensitive receptors • Sensory disturbance resulting in animal displacement
Waste production (wastes other than mine wastes)	Domestic, construction and operational wastes	Litter Sewage Non-process related industrial wastes Hazardous wastes (such as waste oils, chemicals, spent packaging)	• Waste disposal sites resulting in creation of an attractive nuisance to scavenger animals • Contamination of soil and/or water • Degradation of land and health risks associated with the above impacts

Aspect group	Aspect	Mechanism	Potential impacts
Stimulation of economic growth	Job creation Procurement of services and supplies	Direct employment during construction and operation Indirect employment by service providers and suppliers	• Employment of local people • Skills acquisition through job training • Improved infrastructure and services • Potential for sustainable economic developments
	Payment of tax and levies	Tax on profits Duties on imports Payroll tax Value added tax	
	Community investment	Investment in social development initiatives	
Resettlement	Land acquisition within the project site	Economic displacement (loss of access to land used for agriculture, natural resources etc.)	• Loss of access to common property resources (such as wells, boreholes, etc.) • Loss of access to cultural resources
Closure	Retrenchment Cease of operations		• Inter-related potential impacts including: o Unemployment and loss of income o Closure of support and service businesses o Outward migration of skilled workers, leaving the elderly and the unskilled behind leading to the eradication of the consumer base o Psychological impacts on individuals manifesting as apathy, helplessness and a sense of inadequacy o Erosion of Governments' revenue base leading to a reduction in the allocation of funds to the area and subsequently deterioration in quality of life

20.4.2 Key Technical Environmental and Social Issues

Historical liabilities

The industrial area currently owned by Ibarski Rudnici Coal Company has historically been used for processing and waste disposal facilities. Sources of contamination are likely to include surface disturbance and degradation from land clearance, uncontrolled disposal of waste rock and fine coal tailings, and contamination of soil and water.

It appears that Ibarski Rudnici Coal Company is not currently obligated to undertake any investigation or remediation measures; however, there is a chance this may change in the future, either through a new permit application or as a result of changes to legislation, particularly following transposition of The Environmental Liability Directive (2004/35/EC).

The Environmental Liability Directive establishes a framework of environmental liability based on the “polluter pays” principle to prevent and remedy environmental damage. The principle of liability applies to environmental damage and imminent threat of damage resulting from occupational activities, where there is a causal link between the damage and the activity in question. Transposition of the Industrial Emissions Directive is also likely to result in new obligations being imposed on existing installations to identify and remediate contamination (during operation and closure) and monitoring the risks of contamination in the future.

Redevelopment of any brownfield sites by Erin therefore presents a liability risk; if surface rights are acquired, Erin could be obligated to remediate past environmental or social damage that has occurred or there could be complicated legal negotiations regarding liability for historic environmental contamination. A robust environmental liabilities assessment prior to acquisition (or shortly afterwards) to understand the extent of existing contamination and its impacts on the surrounding environment can assist in mitigating the uncertainties around this risk.

Water management

A preliminary water management study has been undertaken for the project. This study predicts that inflows into the mine workings could be in the order of between 5 and 50 l/s. The water quality of the inflows into the eventual mine may be of relatively high pH, with elevated boron. The implications of the mine operations on the quality of any eventual dewater and where this would be released to have not yet been investigated.

Potential water supply sources have not been investigated, but could include water abstracted from the Ibar River or groundwater from the mine dewatering operations.

Further work will be required during the PFS to define potential impacts and water management requirements.

Hazardous waste storage facilities

One of the waste streams resulting from the manufacture of boric acid is a solid waste known as boro-gypsum (it is an output from the reaction of colemanite and sulfuric acid). Boro-gypsum has a high content of boron oxide, which is water soluble and known to form complexes with heavy metals. Boro-gypsum is classified as a hazardous waste.

Risks associated with disposal of any hazardous waste should be thoroughly evaluated in the EIA and by project engineers and managed through the project design. The project should plan for appropriate characterisation of waste streams and the findings of these studies will need to be incorporated into the EIA to ensure impacts have been appropriately identified and adequate management is incorporated into project design.

High expectations of the positive socio-economic impacts

Communities generally have high expectations of socio-economic benefits derived from mining companies. This means that maintaining a social license to operate is linked to value perceived by host communities. There are high expectations in terms of reviving declining regional and local economies; promoting and stabilising a decreasing population; and contributing towards improvement of infrastructure.

Management of stakeholder expectations is likely to be an on-going challenge to maintaining the project's social licence to operate. Many socio-economic benefits will not be realised without the commitment and effort of both Erin and government. Tension and conflict could arise if these benefits are not realised.

Specific strategies/ plans should be developed to ensure the community expectations are addressed or managed, and that anticipated benefits are realised and maximised in favour of the local population. Responsibilities of other parties, such as government, for implementation of management measures should be clearly identified and communicated to local stakeholders.

20.5 Recommended next steps

SRK has recommended that Erin initiates the EIA process for the Project in accordance with Law on Environmental Impact Assessment (Official Journal RS, No. 135/O4, 36/09) and international guidelines. The EIA process comprises the elements summarised in Table 20-4 which outlines the overall objectives, activities, stakeholders likely to be involved and deliverables of each phase of the proposed EIA process, highlighting how these phases link to the overall project development phases – i.e. PEA, PFS, FS (as recognised by IRRS).

This work will also need to be undertaken for any "associated facilities" – that is any facility being developed as a direct result of the Project and upon which the Project is reliant on. This is particularly pertinent where Erin has committed to assisting with the permitting and licencing aspects of the boric acid plant.

It is recognised the terminology used for the overall project development phases differs between IRRS and Serbian legislation (i.e. Elaborat, Feasibility Study and Main Mining Project); therefore Erin needs to assess the required content of the technical studies required to support requirements of both IRRS and Serbian legislation to establish how these development phases align with each other.

The EIA process below includes recommendations on stakeholder engagement as input to the environmental permitting process but building a robust relationship with stakeholders requires engagement above and beyond the scope of the EIA. Erin may also want to maintain and/or initiate engagement to address the following issues:

- General maintenance of constructive relationships between the project proponent and government agencies, community leaders and communities as the project develops;

- Access to land for on-going exploration activities and intrusive engineering studies (such as geotechnical studies);
- Project grievance mechanism for receiving and responding to grievances of people from local communities that are/will be affected by project activities; and
- Obtaining other project approvals (such as approvals for infrastructure development, mining works and building approvals).

In addition to the EIA process, SRK recommends a comprehensive environmental liability assessment be undertaken for any brownfield sites likely to be redeveloped for the project. Although some of the sampling, data analysis and evaluation will be part of the bigger EIA project, more comprehensive sampling, particularly of soil, water and vegetation, in the vicinity of the brownfields sites may be needed to adequately characterise the extent of any historical liabilities so that legal responsibility can be more clearly defined.

Table 20-4: Overview of the ESIA process and linkages to project development

ESIA Phase	Project Development Phase	Objectives	Activities	Stakeholders involved	Documents produced by ESIA team
Screening	Engineering Scoping Study/ PEA	• Determine if ESIA required	• Review available secondary information on the project's social and environmental setting • Brief review of the proposed development and potential impacts • Discuss project with regulatory authorities	• Regulatory authorities	• Input to the Engineering Scoping Study
Environmental and Social Scoping	Pre-feasibility study (PFS)	• Identify the potential impacts requiring study • Identify project alternatives to be evaluated during the course of the ESIA process • Engage stakeholders • Identify law and standards applicable to the project (in particular, identify key environmental and social authorisations required and criteria that should be applied in the design of the project) • Identify environmental and social design criteria for the project engineers	• Undertake a review of environmental and social law and standards applicable to the project • Stakeholder identification and analysis (social scan) • Development of a stakeholder engagement plan (SEP) • Prepare a scoping report for submission to the competent authority who issue the ToR for the EIA (the content of the scoping report is outlined in OJ RS, No. 69/05) o Preliminary project description, preliminary evaluation of alternatives, preliminary identification of impacts etc. • The competent authority will notify stakeholders of the EIA process and provide information to facilitate their input into the decision on the ToR for the EIA	• Local communities, regulatory authorities, non-governmental organisations and other stake-holders that could have an interest in the project	• Write ups on relevant law and standards and environmental and social design criteria • Preliminary SEP • Stakeholder database • Background information document (BID) for stakeholders • Records of stakeholder consultations • Scoping report summarising the results of the scoping phase, including updated SEP, if necessary • Input to the PFS
Baseline characterization	Starts at PFS and continues into feasibility study (FS)	• Collect background information and describe the physical, biological, social and economic setting of the project • Establish pre-project conditions	• Baseline studies, where needed • Consultation with stakeholders as necessary to support baseline characterization	• Stakeholders who can provide input to baseline studies	• Interim and final baseline reports • Records of stakeholder consultations

ESIA Phase	Project Development Phase	Objectives	Activities	Stakeholders involved	Documents produced by ESIA team
Impact assessment and report compilation	Impact assessment occurs towards the end of the FS once a sufficiently defined project description is available	• Define and evaluate potential impacts (identified by stakeholders and project specialists) • Define measures for management of impacts • Determine the significance of potential impacts with and without management • Develop framework environmental and social management system (ESMS) • If necessary, continue to build capacity of stakeholders to participate in the ESIA process, where necessary • Record decisions on project alternatives and the environmental and social inputs to these decisions	• Review project information, stakeholder issues and baseline studies • Evaluate project alternatives from a technical, economic, environmental and social perspective • Eliminate or mitigate impacts through modification of the project design • Predictive modelling studies • Impact evaluation • Report compilation • Discuss specific procedural and/or substantive matters with stakeholders as required	• Stakeholders identified as requiring capacity building • Stakeholders who may need to input into project design	• Predictive modelling reports • ESIA report, which includes an ESMP • Records of stakeholder consultations
ESIA report review and decision making	Prior to construction, unless required by regulatory authorities as part of the approval process	• Government decision and conditions of approval • Feedback to stakeholders on progress with project planning, expected impacts and proposed mitigation • Acknowledge issues raised by stakeholders and explain how these will be addressed	• Review of ESIA report by regulatory authorities and other interested stakeholders • Notification and engagement of stakeholders • Feedback meetings, as determined in the SEP • Government public hearing/s (if prescribed by government) • Government decision and definition of the conditions of approval	• Stakeholders who participated in the ESIA process to date • Stakeholders responding to notices of feedback meetings (and any government public hearing/s)	• Notice from the regulatory authority • Advertisement/s in provincial newspapers • Records of stakeholder consultations • Record of hearing • Government record of decision and conditions of approval
Development of detailed management system and plans	Depending on regulatory authority requirements either occurs as part of FS or prior to construction	• Enable successful implementation of the management measures identified through the ESIA process during construction and operation	• Further develop the framework ESMS into a fully implementable ESMS • Develop detailed management plans, policies, protocols, procedures etc., to support the implementation of the ESMS	• Stakeholders potentially impacted by the project	• ESMS description • Policies, plans, procedures and protocols

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

This section summarises the capital and operating cost assumptions made by SRK in developing the PEA presented in Section 22 below.

21.2 Mining

21.2.1 Mining Capital Expenditure

Estimated expenditure for mining capital is USD 41.4M. A breakdown of this is shown in Table 21-1 which also shows the impact of the 20% contingency which has been applied to all capital costs derived for the Project for the purposes of the PEA presented in Section 22 of this report. The principal areas of mine capital expenditure are:

- Surface infrastructure including: roads; offices, workshops and stores facilities; ROM pad and loader; and security.
- Mine portal.
- Decline access, nominally 1,600 m in length.
- Ventilation shaft nominally 400 m deep, and primary ventilation fan.
- Emergency hoist to provide a second means of egress.
- Back fill equipment: a cement slurry plant and a truck and LHD for backfill handling.
- Materials handling system including: belt conveyors and feeder breakers.
- Mining equipment including: continuous miners, shuttle cars, roof bolters.
- Service equipment: light vehicles.
- Service infrastructure, including power supply, dewatering, secondary ventilation, and communications equipment.
- An allowance of 15% of mobile equipment costs has been made for freight; and 3% of mobile equipment costs for commissioning.

Table 21-1: Mining Capital Costs

		Qty	Unit Cost	Total
Surface Infrastructure				
Waste Dump site preparation	LS	1	50,000	50,000
Workshop equipping	LS	1	150,000	150,000
Back fill mix plant & delivery system				
Additional truck	LS	1	550,000	550,000
Backfill Loader / Scoop	LS	1	450,000	450,000
Ejector conveyor and belt cleaner	LS	1	150,000	150,000
Additional belt for managing B/F	metre	800	400	320,000
Containerised cement slurry mixer, hopper and pump	LS	1	500,000	500,000
Mine Portal	LS	1	350,000	350,000
Decline access				
Mobilisation and set up	LS	1	350,000	350,000
Decline excavation and ground support				
Fair Ground	metre	160	3,500	560,000
Poor Ground	metre	960	5,750	5,520,000
Very Poor Ground	metre	480	10,000	4,800,000
Off decline headings	%	15%	10,880,000	1,632,000
Level development	metre	750	3,500	2,625,000
Ventilation shaft				
Sinking	metre	400	20,000	8,000,000
Ventilation Fan	LS	1	403,000	403,000
Panel Ventilation Fans	each	4	9,125	36,500
Second Means of Egress				
Sinking	metre	0	20,000	0
Equipping vent shaft with emergency hoist	LS	1	2,500,000	2,500,000
Materials Handling System				
Conveyor	LS	1	2,503,000	2,503,000
Bridge conveyor	LS	1	75,300	75,300
Surface Storage System	LS	1	218,600	218,600
Mining Equipment				
Roadheader / Miners	each	2	1,733,400	3,466,800
Shuttle Cars	each	2	750,000	1,500,000
Roof Bolter	each	1	460,000	460,000
Section belts	metre	800	400	320,000
Scoop	each	1	500,000	500,000
Shuttle cars spare	each	1	750,000	750,000
Service Equipment				
Light vehicles	each	6	50,000	300,000
UG Service vehicle	LS	1	236,700	236,700
Miscellaneous	LS	1	100,000	100,000
ROM pad loader	LS	1	100,000	100,000
Service Infrastructure				
Mine Power System	LS	1	250,000	250,000
Mine Pumping System	LS	1	250,000	250,000
Mine Communications System	LS	1	100,000	100,000
Freight	%	15%	6,996,800	1,049,500
Commissioning	%	3%	6,996,800	209,900
Sub – total				41,400,000
Contingency			30%	12,500,000
Total				53,900,000

Backfill

Sources of fill materials to achieve backfill design characteristics still need to be determined. Infrastructure will be required for transport, stockpile and rehandle of feed materials, provision of services including power and water; mixing and transport of fill underground; and placement into the stopes. In total around USD1.97M has been allocated for these aspects.

Second Egress

A second means of egress is provided by equipping the return air shaft with an emergency hoist. USD2.5M has been allocated for this.

Decline construction

Three nominal rates have been applied for decline construction dependent on possible ground conditions: USD3500/m for Fair ground requiring systematic bolting; USD5750/m for Poor ground requiring systematic bolting and 50 to 80 mm thick application of shotcrete or fibre-reinforced shotcrete; and USD10,000/m for Very Poor ground which might require systematic bolting and 90 to 120 mm thick application of fibre-reinforced shotcrete. On this basis the nominal capital cost per metre of decline is USD6800/m.

21.2.2 Mining Operating Cost

Factored Mine Operating costs have been applied and a total mine operating cost of some USD37/$t_{(ROM)}$ estimated as shown in Table 21-2 below.

Table 21-2: Estimated Annual Operating Costs

		Unit Cost (USD/t)	Total (USD/a)
Total Labour Cost	11%	4.20	1,425,000
Total Maintenance Cost	10%	3.85	1,306,000
Total Backfill cost	30%	11.15	3,782,000
Total Power cost	6%	2.10	712,000
Total Diesel cost	9%	3.50	1,187,000
Excavation consumables	8%	2.80	950,000
Total Support cost - Roofbolts	2%	0.70	237,000
Overhead costs	3%	1.05	356,000
Electrical installation costs	7%	2.45	831,000
Consumables	2%	0.70	237,000
Sub total		32.50	11,025,000
Other	12%	4.55	1,543,000
Total Mining Operating Costs		37.05	12,568,000

Backfill

Backfill is the most significant operating cost. As all materials are expected to be imported to the mine site, there will be considerable logistic and purchase costs, definition of which would be subject to a backfill study.

It has been assumed that 85% of the mined void would be filled, and that a placement cost of around USD 30/ m^3 would apply.

Labour

A mining staff productivity of around 3,500 tonne per man per year has been assigned based on similar operations in the region.

For a production rate of around of between 300,000 and 340,000 tpa, mine department staff would be between 85 and 100. Average annual salaries for mining staff Serbia have been estimated to be in the region of USD15,000 per year.

Power

Power costs have been estimated on the basis of 2MW installed power based on equipment requirements as shown in Table 21-3.

Power consumption cost are assumed to be USD0.04 / kWh.

Table 21-3: Underground Power Demand

	No. Units	Load Factor	Load (kW)	Utilisation Factor	Energy/ month (kW hours)
Surface Plant – Main Shaft Area					
Shop equipment	1	70%	30	20%	3,024
Hot water heaters	1	100%	50	65%	23,400
Batch plant	1	80%	45	30%	7,776
Surface pumps	1	60%	30	50%	6,480
Lighting	1	90%	20	60%	7,776
Office, etc.	1	40%	9	40%	1,037
Surface Plant -Vent Shaft Area					
Main Ventilation Fans	1	95%	500	100%	342,000
Pumps	1	75%	15	67%	5,427
Lighting	1	90%	5	50%	1,620
Underground					
Main dewatering pumps	1	80%	300	80%	138,240
Sump and mud pumps	2	80%	30	50%	17,280
Definition diamond drill	1	90%	90	70%	40,824
Feeder Breaker	2	75%	40	50%	21,600
Conveyor Drive	1	80%	300	60%	103,680
Stope fans	3	70%	45	100%	68,040
Continuous miner	2	80%	500	60%	345,600
Shuttle car	4	80%	120	70%	193,536
Section conveyor	2	80%	60	60%	41,472
MacLean roof bolter	2	80%	120	70%	96,768
Lunch room	1	80%	12	20%	1,382
Underground lighting	1	90%	40	100%	25,920
Subtotals			2,361		1,492,882
Contingency			20%		20%
Total load (kW)			2,833		
Diversification factor			70%		
Maximum Demand (kW)			**1,983**		
Energy consumption - month	(kWh)				1,791,459
Energy consumption - day	(kWh)				**59,715**

21.3 Processing

21.3.1 Process Plant Capital Cost Estimate

SRK has estimated a capital cost for the proposed conceptual beneficiation plant based on information from a subscription database. The Boric Acid plant capital cost has been estimated based on two sources:

- A "general reference" cost breakdown provided to Erin by a senior representative of SCL; and
- The cost breakdown recently published (April 2014) by Orocobre Limited for a 25 ktpa Boric Acid plant to be located in Argentina.

The estimated capital costs for the process plants are as follows:

- Beneficiation plant: USD 2.5 million; and
- Boric Acid plant: USD 15 million.

These figures, which are estimates suitable for a conceptual / scoping level of study only, can be considered to be inclusive of indirect costs such as EPCM. A 20% Contingency has, however, been applied on top of the stated capital costs for the purposes of the PEA presented in Section 22.

21.3.2 Process Plant Operating Cost Estimate

The processing operating costs have been estimated using the same background data as used for the capital cost estimates.

The estimated operating cost for the process plant, suitable for a conceptual / scoping level of study only, is as follows:

- Beneficiation plant: USD 2 /t RoM ore; and
- Boric Acid plant: USD 150 /t Boric Acid plant feed.

21.4 Tailings Management

21.4.1 Capital Costs

A capital cost estimate has been prepared for the proposed TSF that includes direct earthworks and associated structures costs. The cost budget estimate is based on typical unit costs and experience of similar civil work projects. The unit rates assumed for the cost estimate are as follows:

- A rate of USD 14,000/km for access and service road construction;
- A rate of USD7/m^2 for site clearance;
- A rate of USD10/m^3 for dam construction;
- A lump sum of USD0.5M for drainage system including the pumps;
- Decant system – a lump sum of USD0.5M;
- Emergency spillway – a lump sum of USD0.1M;
- Supply and installation of the HDPE liner at the rate of USD7.5/m^2;

- A lump sum of USD 0.2M for leakage detection system.

The capital cost estimates derived for the design of the facilities per stages are summarised in Table 21-4 below. A 20% contingency has also been added to the cost which is considered to have an overall accuracy of ±50%.

Table 21-4: TSF Capital Costs

Item No.	Description	Unit	Rate	Quantity	Amount
3	**TSF Construction**				
3.1	Access road	km	14,000	0.5	7,000
3.2	Service road	km	14,000	1	14,000
3.3	Site clearance - dam footprint	m^2	7	15000	105,000
3.5	Site clearance basin for liner	m^2	7	60000	420,000
3.6	Site clearance pipeline route	m^2	7	2500	17,500
3.9	Dam building - excavation spread and compact incl 0.5m strippin	m^3	10	35000	350,000
4.2	Drainage including drainage pump station	Lsum	500,000	1	500,000
4.3	Decant system	Lsum	500,000	1	500,000
4.5	Spillway	Lsum	100,000	1	100,000
4.6	Distribution Pipelines	Lsum	500,000	1.0	500,000
4.7	HDPE Liner (2mm) - double liner	m^2	7.5	120000	900,000
4.8	Leakage detection system	Lsum	200,000	2	400,000
	Total Capex (USD)				**3,813,500**

21.4.2 Operating Costs

An operating cost (“OPEX”) for pumps and pipelines maintenance has been estimated as USD 0.2/t.

21.5 Infrastructure

21.5.1 Introduction

Infrastructure capital costs have been established based upon the following assumptions:

- Land and rail sidings will be made available by Ibar Coal Mines Company as proposed for the required infrastructure;
- surface infrastructure footprints and layouts have been defined for costing purposes and will be confirmed at a later stage of study;
- point of sale is FOB mine site determined as the point of loading onto rail or road vehicles and therefore, all downstream costs for freight and rehandling costs at river ports are not considered;
- power will be supplied from the national grid and there is sufficient capacity within the system;
- capital costs have been estimated through benchmarking against similar operations, however, where regional cost data was not available, costs were developed based on Western Europe / North American standards and location factor of applied to get to local country costs;
- The main access roads to require upgrading and yearly maintenance; and
- No accommodation is required as staff will reside locally.

21.5.2 Capital Costs

Table 21-5 presents the anticipated capital costs exclusive of the 20% contingency added for the purpose of the PEA.

Table 21-5: Capital Cost Summary

Capital Item	USD $M
Access Road	0.25
Portal Entrance Civil Works	2.50
Development (Earthworks)	1.75
Security Measures	0.15
Structures / Buildings	1.35
Water and Surface Water Management	0.40
Power	0.75
Load-Out Area (Development and Refurbishment)	1.10
Sub-Total	8.25

21.5.3 Operating Costs

Table 21-6 below details the operating costs derived by SRK. It should be noted that:-

- These include costs for product load-out and maintenance. Lighting and energy for site infrastructure buildings is considered within the General and Administration costs ("G&A") which are considered separately. Operating costs for crushing and screening plant (considered elsewhere) include for power and water supply; and
- Operating costs for bagging of product is included within the processing and / or crushing and screening costs.
- The cost for purchase and transport of sulphuric acid for the Boric Acid is within the operating cost for the Boric Acid Plant.

Annual operating costs ("OPEX") for product load out and road maintenance are estimated as USD 0.66 M (USD 3.00/t product).

Table 21-6: Operating Cost Summary per year

Operating Cost	USD $M
Containerisation	0.23
Transport	0.05
Loading / Unloading	0.23
Road Maintenance	0.15
Sub-Total	**0.66**

21.6 Closure requirements and cost

A closure plan has not yet been prepared for the project but will be included in the EIA submission. 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 the provisional ballpark estimate for the closure of this site has been estimated to be in the region of USD 15 million.

22 ECONOMIC ANALYSIS

22.1 Introduction

SRK has constructed an Excel based Technical Economic Model (TEM) to assess the Project reflecting the assumptions as set out in the previous sections of this report.

Notably, SRK has constructed a pre-finance and pre- and post- tax TEM on an annual basis and assumed that:

- The currency is USD in H1-2014 real terms;
- A base case discount rate of 10%;
- Construction starts in 2015 and continues over a two year period with processing of ore commencing in 2017;
- Working capital assumptions of:
  - o Debtor days – 30
  - o Creditor days – 30
  - o Stores days – 30 (based on 10% of all operating costs)
- Corporation tax of 15% of taxable profits following a 10 year tax 'holiday' (commencing from the start of construction);
- Depreciation of project and sustaining capital costs on a straight line basis over 10 and 5 years respectively.

No allowance has been made for VAT movements and as noted above, no financing assumptions are included.

22.2 Model Assumptions

22.2.1 Physical Mining and Processing Schedule

Figure 22-1 to Figure 22-6 illustrate, on an annual basis, the key physical assumptions reported from the mining and processing schedules derived by SRK:

- Figure 22-1 – Mined & Colemanite Plant feed tonnage by ore classification and overall mined/processed grade
- Figure 22-2 – Colemanite Plant mass yield and recovery percentage
- Figure 22-3 – Total Colemanite Plant production (split to Boric Acid plant and for direct sale)

- Figure 22-4 – Boric Acid Plant mass yield and recovery percentage
- Figure 22-5 – Boric Acid Plant production
- Figure 22-6 – Product sales split - Colemanite & Boric Acid

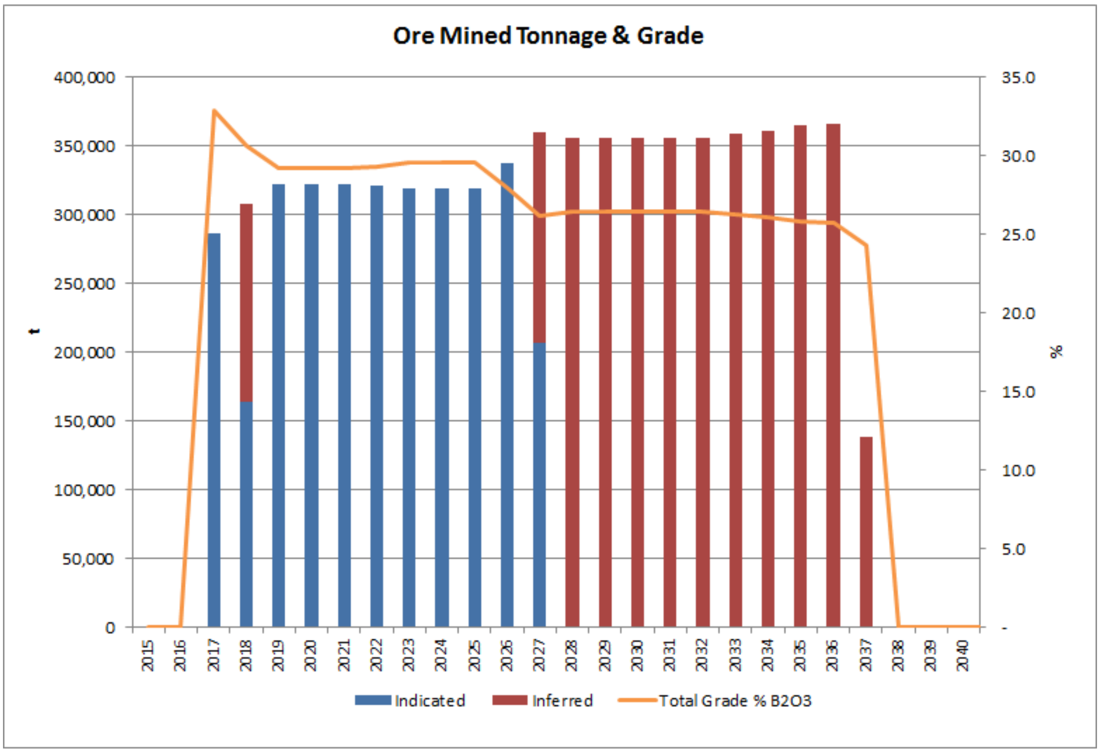


Figure 22-1: Mined Ore tonnage by classification and overall mined grade

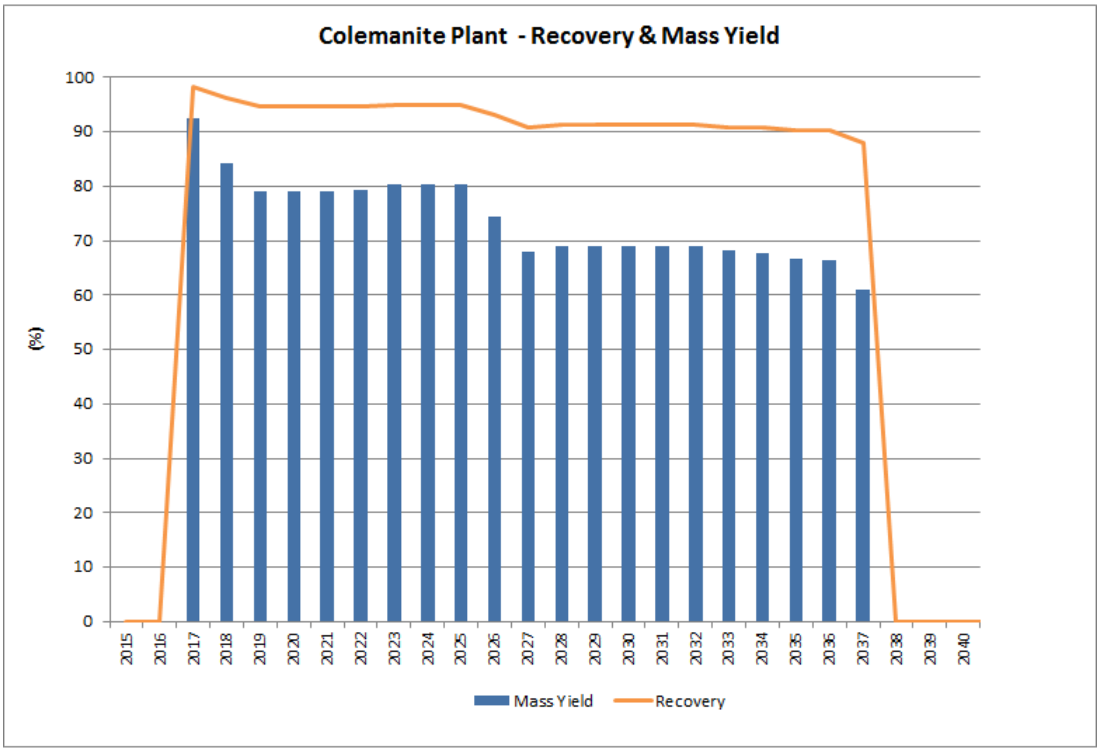


Figure 22-2: Colemanite Plant mass yield and recovery

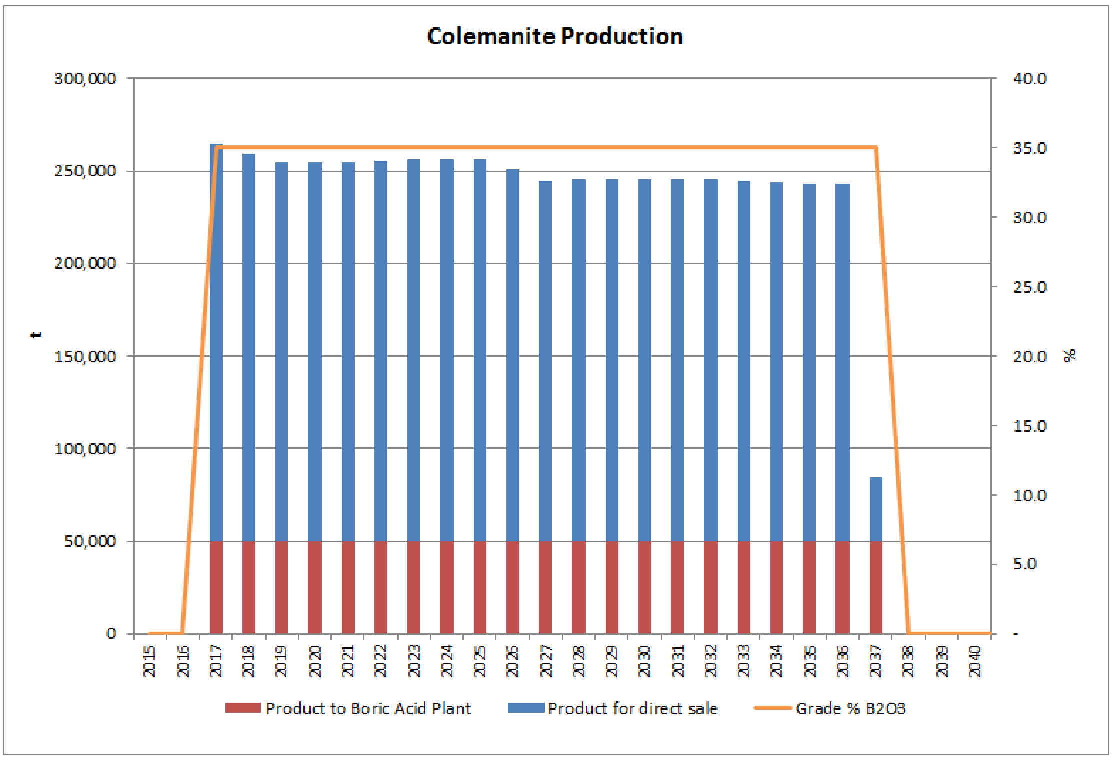


Figure 22-3: Total Colemanite production

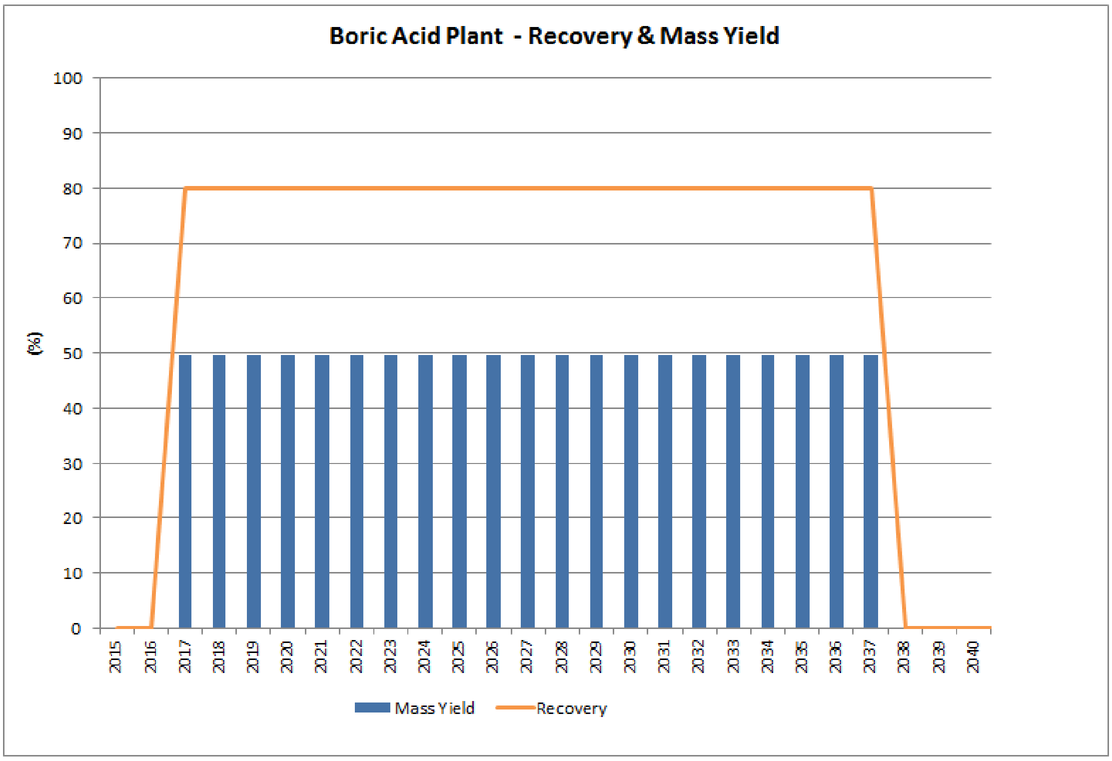


Figure 22-4: Boric Acid Plant yield and recovery

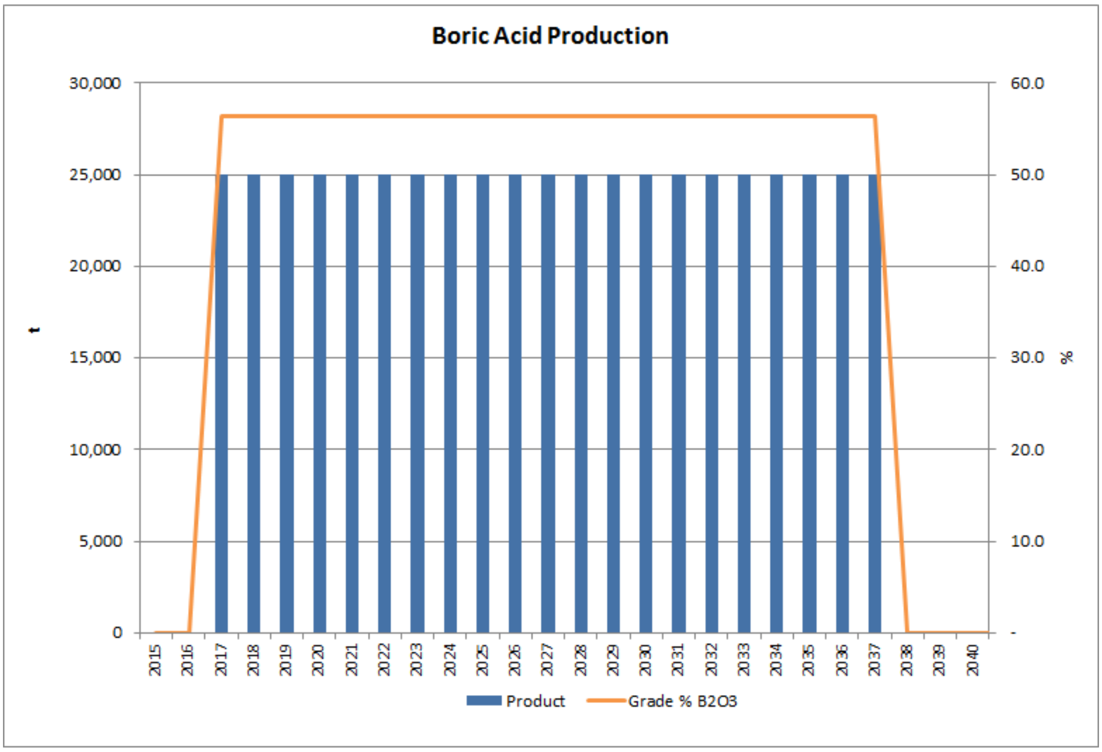


Figure 22-5: Boric Acid Plant production

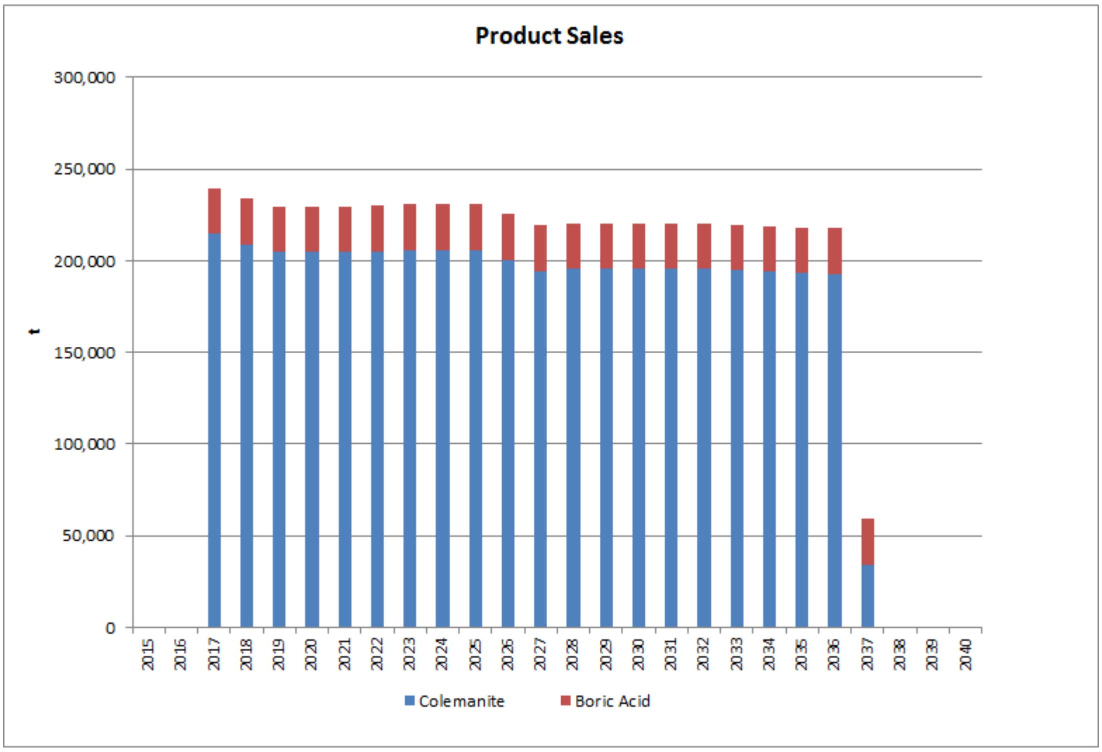


Figure 22-6: Product sales

In summary the following key physical assumptions are made to derive the production schedules presented in the TEM:

- The total tonnage of ore mined and fed to the Colemanite Plant on annual basis varies between some 138,000t and 366,000t averaging some 329,000tpa for a Life of Mine (LoM) total of some 6.9Mt of ore (total modified Indicated and Inferred Mineral Resources). This is comprised of some 3.2Mt of modified Indicated Mineral Resources and some 3.7Mt of modified Inferred Mineral Resources;
- Mined grades varying between 24.3% and 32.9% B_2O_3 and average 27.8% B_2O_3 over the LoM (combined modified Indicated and Inferred Mineral Resources);
- All of the Run of Mine (RoM) ore is fed to the Colemanite Plant for Colemanite production. A constant product grade of 35% B_2O_3 and tails grade of 7.5% B_2O_3 is assumed;
- The resulting mass yield of RoM ore to Colemanite product varies between 61.0% and 92.5% , averaging 73.8% over the LoM;
- The recovery of B_2O_3 from RoM ore to Colemanite product varies between 88.0% and 98.3% , averaging 92.9% over the LoM;
- Some 50,000tpa of Colemanite product (at 35% B_2O_3) is fed for subsequent processing to produce 25,000tpa of Boric Acid product with an assumed grade of 56.3% B_2O_3. The remaining Colemanite product not fed to the Boric Acid plant is sold;
- The mass yield of Colemanite to Boric Acid product averages 49.7% and it is assumed 80% of the B_2O_3 is recovered to the product.
- Over the LoM some 4.0Mt of Colemanite product at 35% B_2O_3 is assumed to be produced and sold, varying between some 34,000tpa and 214,000tpa and averaging 192,000tpa;
- Over the LoM some 525kt of Boric Acid product at 56.3% B_2O_3 is assumed to be produced and sold, at 25,000ktpa;

Table 22-1 presents a summary of the LoM physical assumptions.

Table 22-1: Life of Mine Physical Assumptions Summary

Mining	Units	Total
Life of Mine	(yrs)	21
Indicated Tonnage	(t)	3,239,327
Grade B_2O_3	(%)	29.33
Contained B_2O_3	(t)	950,203
Inferred Tonnage	(t)	3,665,817
Grade B_2O_3	(%)	26.42
Contained B_2O_3	(t)	968,653
Total ROM Tonnage	(t)	6,905,144
Grade B_2O_3	(%)	27.79
Contained B_2O_3	(t)	1,918,856
Processing - Colemanite Production	**Units**	**Total**
ROM Feed Tonnage	(t)	6,905,144
Grade B_2O_3	(%)	27.79
Contained B_2O_3	(t)	1,918,856
Mass Yield	(%)	73.78
Recovery	(%)	92.92
Colemanite Tonnage	(t)	5,094,437
Grade B_2O_3	(%)	35.00
Contained B_2O_3	(t)	1,783,053
Processing - Boric Acid Production	**Units**	**Total**
Colemanite Feed Tonnage	(t)	1,055,566
Grade B_2O_3	(%)	35.00
Contained B_2O_3	(t)	369,448
Mass Yield	(%)	49.74
Recovery	(%)	80.00
Boric Acid Tonnage	(t)	525,000
Grade B_2O_3	(%)	56.30
Contained B_2O_3	(t)	295,559
Product Sales	**Units**	**Total**
Colemanite (@35% B_2O_3)	(t)	4,038,871
Boric Acid (@56% B_2O_3)	(t)	525,000
Total Product	(t)	4,563,871

22.2.2 Commodity Prices and Revenue Deductions

Key revenue assumptions used in the TEM are as follows:

- Colemanite (at 35% B_2O_3) price of USD400/t product (flat lined)
- Boric Acid (at 56.3% B_2O_3) price of USD800/t product (flat lined)
- Royalty deduction of 5% on gross revenue
- Other sales and marketing costs of USD1.5/t product sold

Table 22-2 below shows a LoM summary of revenue and deductions, while Figure 22-7 shows the annual gross revenue split by the contribution from Colemanite and Boric Acid sales.

Table 22-2: LoM Revenue and Deductions

Revenue	Units	Total
Colemanite (@35% B_2O_3)	(USD'000)	1,615,548
Boric Acid (@56% B_2O_3)	(USD'000)	420,000
Gross Revenue	**(USD'000)**	**2,035,548**
Royalty	(USD'000)	101,777
Sales/Marketing	(USD'000)	6,846
Deductions	**(USD'000)**	**108,623**
Net Revenue	**(USD'000)**	**1,926,925**

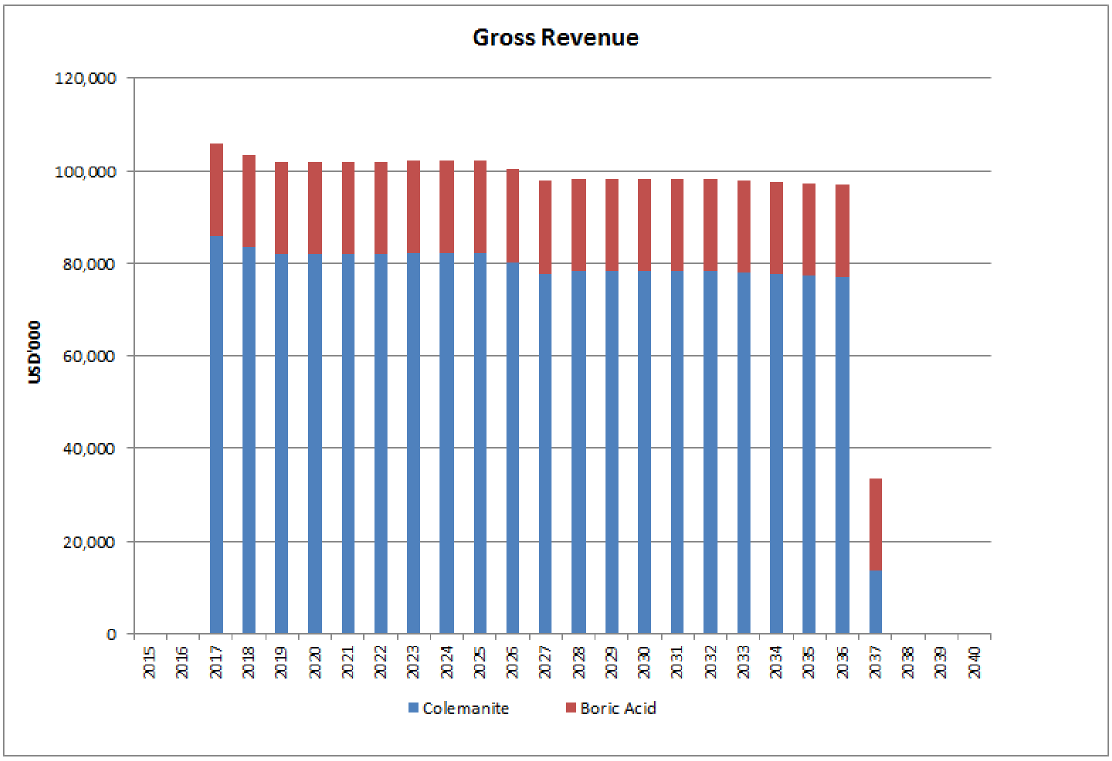


Figure 22-7: Gross Revenue

22.2.3 Operating Costs

Operating costs have been derived by SRK and are described in detail in Section 21 above. Table 22-3 below presents a summary of the base unit cost assumptions including a 15% contingency allowance to give the total unit costs assumed in the TEM.

Table 22-3: Unit Operating Costs

Operating Costs	Unit	Base Cost	Contingency (15%)	Total
Mining	(USD/t mined)	32.50	4.88	37.38
Processing – Colemanite	(USD/t plant feed)	2.00	0.30	2.30
Processing - BA Plant	(USD/t plant feed)	150.00	22.50	172.50
Tailings/Waste Disposal	(USD/t tailings placed)	0.20	0.03	0.23
Infrastructure	(USD/t product)	3.00	0.45	3.45
G&A	(USD/t product)	7.00	1.05	8.05

Table 22-4 below presents a summary of the LoM operating costs (including royalty, sales/marketing and corporation tax) and expressing the total costs as a unit cost per tonne of total product sold (Colemanite plus Boric Acid). Figure 22-8 and 22-9 show the operating costs over the LoM and unit cost per tonne of product sold respectively on an annual basis.

Table 22-4: LoM Operating Costs

Operating Costs	USD'000	USD/t total product
Mining	258,080	56.55
Processing – Colemanite	15,882	3.48
Processing - BA Plant	182,085	39.90
Tailings/Waste Disposal	651	0.14
Infrastructure	15,745	3.45
G&A	55,586	12.18
Royalty (5%)	101,777	22.30
Sales/Marketing	6,846	1.50
Corporation Tax	119,819	26.25
Total	**756,471**	**165.75**

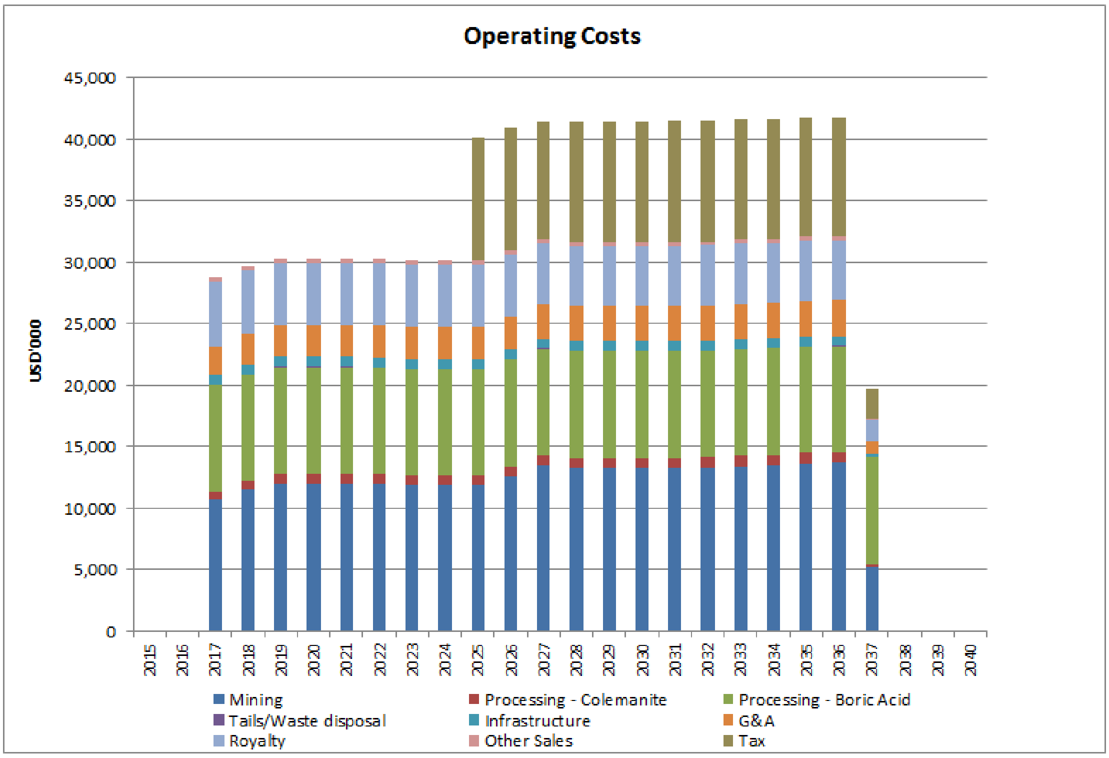


Figure 22-8: LoM Operating Costs

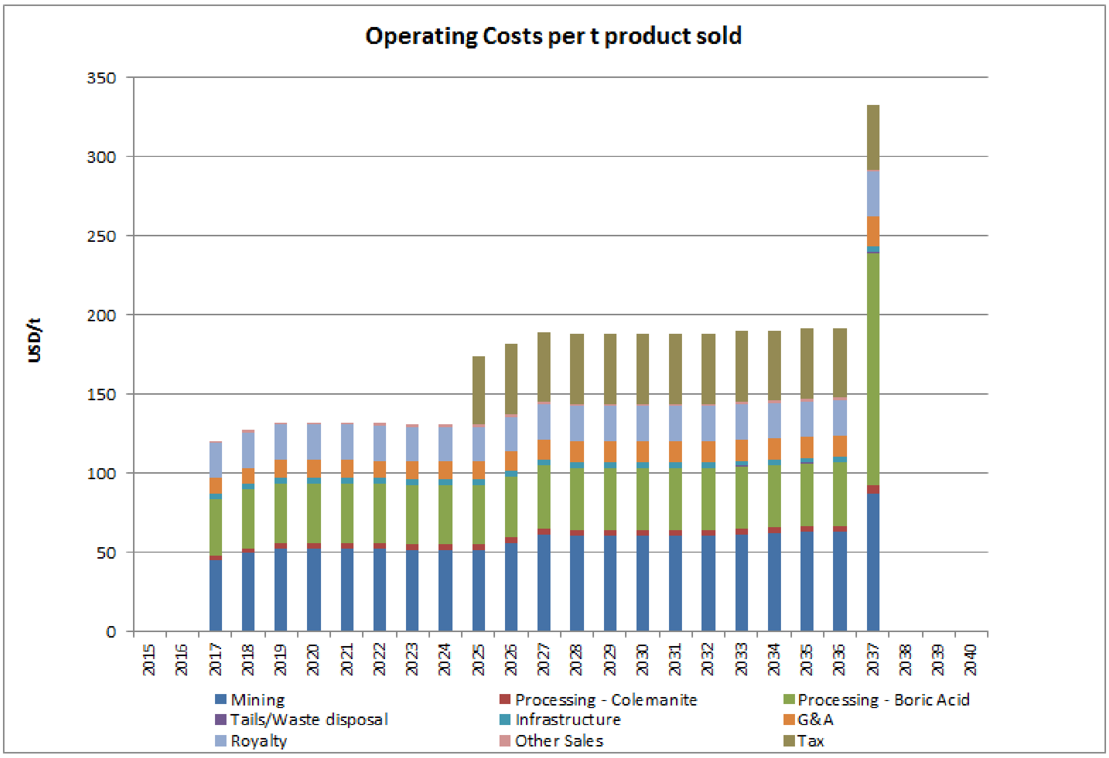


Figure 22-9: LoM Unit Operating Costs

22.2.4 Project Capital Costs

Project Capital costs have been derived by SRK and are described in detail in Section 21 above. Table 22-5 below presents a summary of the base cost assumptions and a 20% contingency allowance has been added to give the total costs assumed in the TEM.

It is assumed that construction of the project facilities will take place over a 2 year period and the capital expenditure has been spread equally over each year in the TEM.

Table 22-5 Project Capital Costs

Project Capital (USD'000)	Base Cost	Contingency	Total
Mining	41,400	8,280	49,680
Processing - Colemanite	2,000	400	2,400
Processing - Boric Acid	15,000	3,000	18,000
Infrastructure	8,250	1,650	9,900
Tailings	3,814	763	4,576
Total	**70,464**	**14,093**	**84,556**

22.2.5 Sustaining Capital

Sustaining Capital costs have been derived by SRK and are described in detail in Section 21 above. Table 22-6 below presents a summary of the LoM cost assumptions. A 20% contingency allowance has been included to give the total costs assumed below in the TEM.

Table 22-6: LoM Sustaining Capital Costs

Sustaining Capital	USD'000
Mining	15,300
Processing – Colemanite	-
Processing - Boric Acid	-
Infrastructure	3,366
Tailings	-
Total	**18,666**

Specific allowances have been made for mining related sustaining capital as summarised below in Table 22-7 and as illustrated in Figure 22-10. A general allowance for infrastructure has been estimated based on 2% of initial infrastructure capital costs to be incurred annually following 2 years of production and ceasing within 2 years of the end the LoM.

Table 22-7: LoM Mining Sustaining Capital Costs

Mining Sustaining Capital	USD'000
Backfill plant and delivery system	2,301
Decline access	1,248
Materials handling system	1,302
Mining equipment	9,304
Service equipment	2,128
Service infrastructure	293
Total	**16,575**

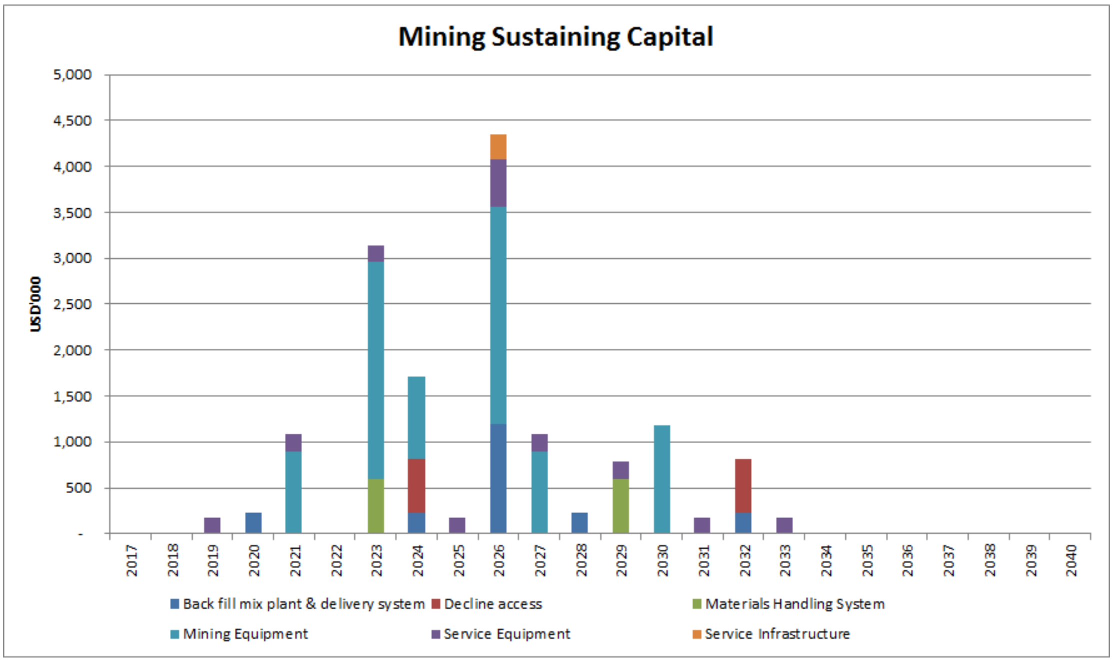


Figure 22-10: LoM Mining Sustaining Capital Costs

22.2.6 Closure Cost

An allowance of USD15M has been included for closure related costs and these have been incorporated at the end of the project life for cashflow purposes.

22.3 Project Economics

A summary of the TEM on an annual basis is show below in Table 22-8.

Table 22-8: Summary TEM

	Units	Total	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Year 18	Year 19	Year 20	Year 21	Year 22
Mining																										
Indicated																										
Tonnage	(t)	3,239,327	-	-	206,296	153,926	322,474	322,474	322,474	321,463	318,856	318,856	318,856	337,407	206,243	-	-	-	-	-	-	-	-	-	-	-
Grade B2O3	(%)	29.3	-	-	32.9	29.4	29.2	29.2	29.2	29.3	29.6	29.6	29.6	27.9	26.0	-	-	-	-	-	-	-	-	-	-	-
Contained B2O3	(t)	950,203	-	-	94,264	48,234	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	53,592	-	-	-	-	-	-	-	-	-	-	-
Inferred																										
Tonnage	(t)	3,665,817	-	-	-	153,732	-	-	-	-	-	-	-	-	163,627	355,822	355,822	355,822	355,822	355,971	359,037	361,279	364,727	366,011	138,243	-
Grade B2O3	(%)	26.4	-	-	-	32.0	-	-	-	-	-	-	-	-	26.5	26.5	26.5	26.5	26.5	26.5	26.3	26.1	25.8	25.8	24.3	-
Contained B2O3	(t)	968,653	-	-	-	46,030	-	-	-	-	-	-	-	-	40,672	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	33,574	-
Total																										
ROM Tonnage	(t)	6,905,144	-	-	206,296	307,658	322,474	322,474	322,474	321,463	318,856	318,856	318,856	337,407	369,771	355,822	355,822	355,822	355,822	355,971	359,037	361,279	364,727	366,011	138,243	-
Grade B2O3	(%)	27.8	-	-	32.9	30.6	29.2	29.2	29.2	29.3	29.6	29.6	29.6	27.9	26.2	26.5	26.5	26.5	26.5	26.5	26.3	26.1	25.8	25.8	24.3	-
Contained B2O3	(t)	1,918,856	-	-	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	33,574	-
Processing																										
Colemanite Production																										
ROM Feed																										
Tonnage	(t)	6,905,144	-	-	206,296	307,658	322,474	322,474	322,474	321,463	318,856	318,856	318,856	337,407	369,771	355,822	355,822	355,822	355,822	355,971	359,037	361,279	364,727	366,011	138,243	-
Grade B2O3	(%)	27.8	-	-	32.9	30.6	29.2	29.2	29.2	29.3	29.6	29.6	29.6	27.9	26.2	26.5	26.5	26.5	26.5	26.5	26.3	26.1	25.8	25.8	24.3	-
Contained B2O3	(t)	1,918,856	-	-	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	94,264	33,574	-
Colemanite Production																										
Tonnage	(t)	5,094,437	-	-	264,698	258,872	254,831	254,831	254,831	255,107	255,818	255,818	255,818	250,759	244,659	245,736	245,736	245,736	245,736	245,696	244,859	244,248	243,308	242,957	84,383	-
Grade B2O3	(%)	35.0	-	-	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	-
Contained B2O3	(t)	1,783,053	-	-	92,644	90,605	89,191	89,191	89,191	89,287	89,536	89,536	89,536	87,766	85,631	86,008	86,008	86,008	86,008	85,993	85,701	85,487	85,158	85,035	29,534	-
Mass Yield to Colemanite Product	(%)	73.8	-	-	92.5	84.1	79.0	79.0	79.0	79.4	80.2	80.2	80.2	74.3	68.0	69.1	69.1	69.1	69.1	69.0	68.2	67.6	66.7	66.4	61.0	-
B2O3 Recovery	(%)	92.9	-	-	98.3	96.1	94.6	94.6	94.6	94.7	95.0	95.0	95.0	93.1	90.8	91.2	91.2	91.2	91.2	91.2	90.9	90.7	90.3	90.2	88.0	-
Boric Acid (BA) Production																										
BA Plant Feed (with Colemanite Product)																										
Tonnage	(t)	1,055,566	-	-	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	50,265	-
Grade B2O3	(%)	35.0	-	-	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	-
Contained B2O3	(t)	369,448	-	-	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	17,593	-
B2O3 Recovery to Product	(%)	80.0	-	-	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	80.0	-
Mass Yield	(%)	49.7	-	-	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	49.7	-
BA Product																										
Tonnage	(t)	525,000	-	-	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	-
Grade B2O3	(%)	56.3	-	-	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	-
Contained B2O3	(t)	295,559	-	-	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	-
Product Sales																										
Colemanite Product for Sale																										
Tonnage	(t)	4,038,871	-	-	214,433	208,587	204,566	204,566	204,566	204,842	205,553	205,553	205,553	200,454	194,394	195,471	195,471	195,471	195,471	195,431	194,594	193,983	193,043	192,692	34,118	-
Grade B2O3	(%)	35.0	-	-	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	35.0	-
Contained B2O3	(t)	1,413,605	-	-	75,051	73,012	71,598	71,598	71,598	71,695	71,943	71,943	71,943	70,173	68,038	68,415	68,415	68,415	68,415	68,401	68,108	67,894	67,565	67,442	11,941	-
BA Product for Sale																										
Tonnage	(t)	525,000	-	-	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	25,000	-
Grade B2O3	(%)	56.3	-	-	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	56.3	-
Contained B2O3	(t)	295,559	-	-	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	14,074	-
Revenue																										
Commodity Price																										
Colemanite	(USD/t)	400	-	-	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	400	-
Boric Acid	(USD/t)	800	-	-	800	800	800	800	800	800	800	800	800	800	800	800	800	800	800	800	800	800	800	800	800	-
Gross Revenue																										
Colemanite	(USD'000)	1,615,548	-	-	85,773	83,443	81,826	81,826	81,826	81,937	82,221	82,221	82,221	80,157	77,758	78,188	78,188	78,188	78,188	78,172	77,838	77,593	77,217	77,077	13,647	-
Boric Acid	(USD'000)	420,000	-	-	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	20,000	-
Sub-total	(USD'000)	2,035,548	-	-	105,773	103,443	101,826	101,826	101,826	101,937	102,221	102,221	102,221	100,157	97,758	98,188	98,188	98,188	98,188	98,172	97,838	97,593	97,217	97,077	33,647	-
Deductions																										
Royalty	(USD'000)	101,777	-	-	5,289	5,172	5,091	5,091	5,091	5,097	5,111	5,111	5,111	5,010	4,888	4,909	4,909	4,909	4,909	4,909	4,892	4,880	4,861	4,854	1,682	-
Sales/Marketing	(USD'000)	6,846	-	-	359	350	344	344	344	345	346	346	346	338	329	331	331	331	331	331	329	328	327	327	89	-
Sub-total	(USD'000)	108,623	-	-	5,648	5,523	5,436	5,436	5,436	5,442	5,457	5,457	5,457	5,348	5,217	5,240	5,240	5,240	5,240	5,239	5,221	5,208	5,188	5,180	1,771	-
Net Revenue	(USD'000)	1,926,925	-	-	100,125	97,920	96,391	96,391	96,391	96,495	96,764	96,764	96,764	94,849	92,541	92,948	92,948	92,948	92,948	92,933	92,616	92,385	92,029	91,897	31,876	-
Operating Costs																										
Mining	(USD'000)	258,080	-	-	10,700	11,499	12,052	12,052	12,052	12,015	11,917	11,917	11,917	12,611	13,446	13,299	13,299	13,299	13,299	13,304	13,419	13,503	13,632	13,680	5,167	-
Processing - Colemanite	(USD'000)	15,882	-	-	658	788	742	742	742	739	733	733	733	776	827	818	818	818	818	819	826	831	839	842	318	-
Processing - BA Plant	(USD'000)	182,085	-	-	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	-
Tailings/Waste Disposal	(USD'000)	651	-	-	16	22	27	27	27	26	26	26	26	31	38	36	36	36	36	37	37	38	39	39	24	-
Infrastructure	(USD'000)	15,745	-	-	826	806	792	792	792	793	795	795	795	778	757	761	761	761	761	760	758	755	752	751	204	-
G&A	(USD'000)	55,586	-	-	2,305	2,477	2,596	2,596	2,596	2,588	2,567	2,567	2,567	2,716	2,896	2,864	2,864	2,864	2,864	2,866	2,890	2,908	2,936	2,946	1,113	-
Sub-total	(USD'000)	528,030	-	-	23,176	24,182	24,880	24,880	24,880	24,832	24,709	24,709	24,709	25,583	26,635	26,449	26,449	26,449	26,449	26,456	26,601	26,706	26,869	26,929	15,496	-
Capital Costs																										
Project Capital																										
Mining	(USD'000)	49,680	24,840	24,840	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing - Colemanite	(USD'000)	2,400	1,200	1,200	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing - Boric Acid	(USD'000)	18,000	9,000	9,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure	(USD'000)	9,900	4,950	4,950	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Tailings	(USD'000)	4,576	2,288	2,288	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Other	(USD'000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-total	(USD'000)	84,556	42,278	42,278	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital																										
Mining	(USD'000)	15,300	-	-	-	-	180	231	1,080	-	3,137	1,707	180	4,350	1,080	231	781	1,176	180	807	180	-	-	-	-	-
Processing - Colemanite	(USD'000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing - Boric Acid	(USD'000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure	(USD'000)	3,366	-	-	-	-	198	198	198	198	198	198	198	198	198	198	198	198	198	198	198	198	198	-	-	-
Tailings	(USD'000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Other	(USD'000)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sub-total	(USD'000)	18,666	-	-	-	-	378	429	1,278	198	3,335	1,905	378	4,548	1,278	429	979	1,374	378	1,005	378	198	198	-	-	-
Total Capital	(USD'000)	103,222	42,278	42,278	-	-	378	429	1,278	198	3,335	1,905	378	4,548	1,278	429	979	1,374	378	1,005	378	198	198	-	-	-
Closure																										
Closure Cost	(USD'000)	15,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	15,000	-
CASHFLOW																										
Gross Revenue	(USD'000)	2,035,548	-	-	105,773	103,443	101,826	101,826	101,826	101,937	102,221	102,221	102,221	100,157	97,758	98,188	98,188	98,188	98,188	98,172	97,838	97,593	97,217	97,077	33,647	-
Deductions	(USD'000)	108,623	-	-	5,648	5,523	5,436	5,436	5,436	5,442	5,457	5,457	5,457	5,348	5,217	5,240	5,240	5,240	5,240	5,239	5,221	5,208	5,188	5,180	1,771	-
Net Revenue	(USD'000)	1,926,925	-	-	100,125	97,920	96,391	96,391	96,391	96,495	96,764	96,764	96,764	94,849	92,541	92,948	92,948	92,948	92,948	92,933	92,616	92,385	92,029	91,897	31,876	-
Operating Costs																										
Mining	(USD'000)	258,080	-	-	10,700	11,499	12,052	12,052	12,052	12,015	11,917	11,917	11,917	12,611	13,446	13,299	13,299	13,299	13,299	13,304	13,419	13,503	13,632	13,680	5,167	-
Processing - Colemanite	(USD'000)	15,882	-	-	658	788	742	742	742	739	733	733	733	776	827	818	818	818	818	819	826	831	839	842	318	-
Processing - BA Plant	(USD'000)	182,085	-	-	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	8,671	-
Tailings/Waste Disposal	(USD'000)	651	-	-	16	22	27	27	27	26	26	26	26	31	38	36	36	36	36	37	37	38	39	39	24	-
Infrastructure	(USD'000)	15,745	-	-	826	806	792	792	792	793	795	795	795	778	757	761	761	761	761	760	758	755	752	751	204	-
G&A	(USD'000)	55,586	-	-	2,305	2,477	2,596	2,596	2,596	2,588	2,567	2,567	2,567	2,716	2,896	2,864	2,864	2,864	2,864	2,866	2,890	2,908	2,936	2,946	1,113	-
Sub-total	(USD'000)	528,030	-	-	23,176	24,182	24,880	24,880	24,880	24,832	24,709	24,709	24,709	25,583	26,635	26,449	26,449	26,449	26,449	26,456	26,601	26,706	26,869	26,929	15,496	-
Capital Costs																										
Project	(USD'000)	84,556	42,278	42,278	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining	(USD'000)	18,666	-	-	-	-	378	429	1,278	198	3,335	1,905	378	4,548	1,278	429	979	1,374	378	1,005	378	198	198	-	-	-
Closure	(USD'000)	15,000	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	15,000	-
Sub-total	(USD'000)	118,222	42,278	42,278	-	-	378	429	1,278	198	3,335	1,905	378	4,548	1,278	429	979	1,374	378	1,005	378	198	198	-	15,000	-
Net Project Cashflow	(USD'000)	1,280,674	-42,278	-42,278	76,949	73,738	71,133	71,082	70,233	71,465	68,720	70,150	71,677	64,719	64,627	66,070	65,520	65,125	66,121	65,472	65,638	65,481	64,962	64,967	1,380	-
Working Capital	(USD'000)	0	-	-	-6,134	272	189	16	-16	-13	-33	16	-16	236	285	-36	-14	-	-	16	25	29	44	30	3,886	1,219
Pre-Tax Project Cashflow	(USD'000)	1,280,674	-42,278	-42,278	70,815	74,010	71,322	71,098	70,218	71,452	68,687	70,166	71,661	64,955	64,912	66,034	65,506	65,125	66,121	65,488	65,662	65,509	65,006	64,998	5,266	1,219
Corporation Tax	(USD'000)	119,819	-	-	-	-	-	-	-	-	-	-	9,961	10,079	9,542	9,719	9,746	9,717	9,842	9,847	9,779	9,752	9,709	9,692	2,434	-
Post-Tax Project Cashflow	(USD'000)	1,160,855	-42,278	-42,278	70,815	74,010	71,322	71,098	70,218	71,452	68,687	70,166	61,700	54,876	55,370	56,315	55,759	55,408	56,279	55,641	55,883	55,757	55,297	55,306	2,832	1,219
Cumulative	(USD'000)		-42,278	-84,556	-13,741	60,269	131,591	202,689	272,906	344,359	413,046	483,211	544,911	599,787	655,157	711,472	767,232	822,640	878,919	934,560	990,444	1,046,201	1,101,498	1,156,804	1,159,636	1,160,855
COST ANALYSIS																										
Operating Costs per t product																										
Mining	(USD/t)	56.55	-	-	44.69	49.22	52.50	52.50	52.50	52.27	51.69	51.69	51.69	55.92	61.29	60.32	60.32	60.32	60.32	60.36	61.11	61.66	62.52	62.84	87.40	-
Processing - Colemanite	(USD/t)	3.48	-	-	2.75	3.03	3.23	3.23	3.23	3.22	3.18	3.18	3.18	3.44	3.77	3.71	3.71	3.71	3.71	3.71	3.76	3.79	3.85	3.87	5.38	-
Processing - BA Plant	(USD/t)	39.90	-	-	36.21	37.12	37.77	37.77	37.77	37.72	37.61	37.61	37.61	38.45	39.52	39.33	39.33	39.33	39.33	39.34	39.49	39.60	39.77	39.83	146.67	-
Tailings/Waste Disposal	(USD/t)	0.14	-	-	0.07	0.10	0.12	0.12	0.12	0.12	0.11	0.11	0.11	0.14	0.17	0.17	0.17	0.17	0.17	0.17	0.17	0.17	0.18	0.18	0.40	-
Infrastructure	(USD/t)	3.45	-	-	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	3.45	-
G&A	(USD/t)	12.18	-	-	9.63	10.60	11.31	11.31	11.31	11.26	11.13	11.13	11.13	12.05	13.20	12.99	12.99	12.99	12.99	13.00	13.16	13.28	13.47	13.53	18.82	-
Royalty	(USD/t)	22.30	-	-	22.09	22.14	22.18	22.18	22.18	22.18	22.17	22.17	22.17	22.22	22.28	22.27	22.27	22.27	22.27	22.27	22.28	22.28	22.29	22.30	28.46	-
Sales/Marketing	(USD/t)	1.50	-	-	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	1.50	-
Tax	(USD/t)	26.25	-	-	-	-	-	-	-	-	-	-	43.21	44.70	43.49	44.08	44.21	44.07	44.64	44.67	44.53	44.53	44.53	44.52	41.17	-
Total	(USD/t)	165.75	-	-	120.39	127.16	132.05	132.05	132.05	131.71	130.84	130.84	174.05	181.07	188.68	187.82	187.94	187.81	188.38	188.46	189.44	190.27	191.55	192.02	333.24	-
Weighted average revenue per t product	(USD/t)	446.01	-	-	441.77	442.81	443.56	443.56	443.56	443.51	443.37	443.37	443.37	444.35	445.58	445.36	445.36	445.36	445.36	445.37	445.54	445.67	445.86	445.94	569.15	-

In summary the Project has a LoM net project cashflow (pre-finance and post-tax) of some USD1,281M which returns a post-tax NPV (10%) of USD428M and an IRR of 64%. Table 22-9 presents the summary LoM cashflow resulting from the TEM.

It should be noted that this PEA is preliminary in nature, that it includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves and there is no certainty that the PEA will be realised.

Table 22-9: Summary Results

Project Cashflow	USD Millions
Gross Revenue	2,036
Deductions	109
Net Revenue	**1,927**
Operating Costs	528
Project Capital	85
Sustaining Capital	19
Closure	15
Project Cashflow	**1,281**
Working Capital	0
Corporation Tax	120
Net Project Cashflow	**1,161**

22.4 Sensitivities

22.4.1 Discount Rate

Table 22-10 shows the pre- and post-tax NPV's at varying discount rates.

Table 22-10: NPV at varying discount rates

NPV	Pre-Tax (USD Millions)	Post-Tax (USD Millions)
5%	734	678
8%	547	510
10%	456	428
12%	384	362
14%	326	309

In summary, at an 8% discount rate the post-tax NPV increases to some USD510M and in increases further to some USD678M at a 5% discount rate.

22.4.2 Commodity Prices

Table 22-11 below shows the impact on the post-tax NPV (10% discount rate) at specific commodity price scenarios.

Table 22-11: NPV at varying discount rates

Post Tax NPV at 10% discount rate		**USD Millions**
Commodity Price (USD/t)		
Colemanite	**Boric Acid**	
300	700	270
350	750	349
400	800	428
450	850	507
500	900	586

22.4.3 Single Parameter

Figure 22-11 shows the varying NPV for varying single parameter sensitivities at a 10% discount rate for commodity price, operating costs and capital costs.

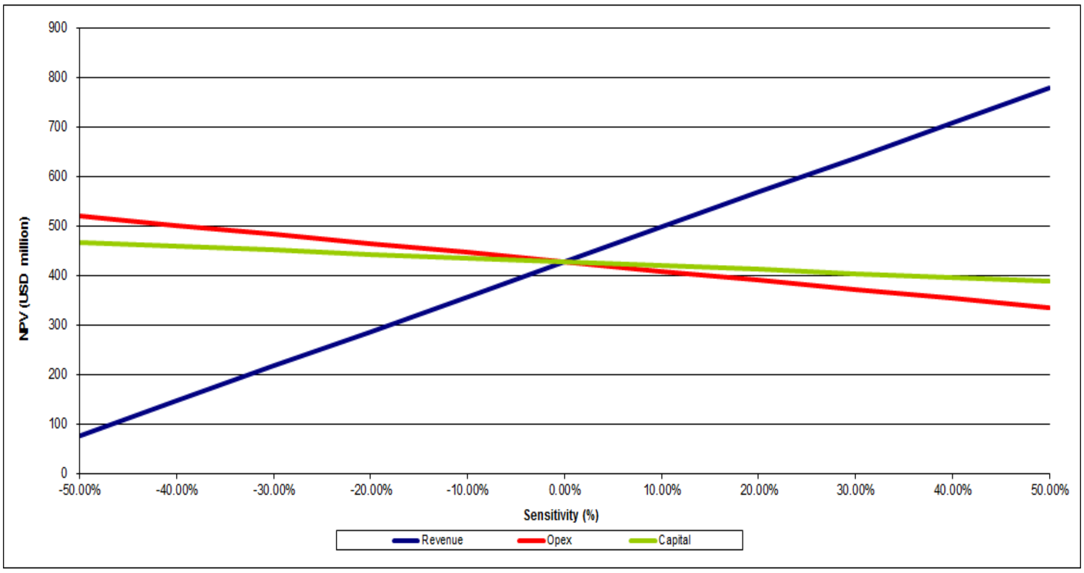


Figure 22-11: Single Parameter Sensitivity

22.4.4 Twin Parameter

Table 22-12 to Table 22-14 show the sensitivity of the Project, using a base case discount rate of 10%, to simultaneous changes in two parameters for revenue and operating costs, revenue and capital costs and operating costs and capital costs respectively.

Table 22-12: Twin Parameter Sensitivity, Revenue v Operating Costs

NPV (USD'000)		REVENUE										
	427,619	-50%	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%	50%
OPEX	-50%	169,370	239,530	309,690	379,849	450,009	520,169	590,329	660,488	730,648	800,808	870,968
	-40%	150,860	221,020	291,180	361,339	431,499	501,659	571,819	641,978	712,138	782,298	852,458
	-30%	132,350	202,510	272,670	342,829	412,989	483,149	553,309	623,468	693,628	763,788	833,948
	-20%	113,840	184,000	254,160	324,319	394,479	464,639	534,799	604,958	675,118	745,278	815,438
	-10%	95,330	165,490	235,650	305,809	375,969	446,129	516,289	586,448	656,608	726,768	796,928
	0%	76,820	146,980	217,140	287,299	357,459	427,619	497,779	567,938	638,098	708,258	778,418
	10%	58,286	128,470	198,630	268,789	338,949	409,109	479,269	549,428	619,588	689,748	759,908
	20%	39,747	109,960	180,119	250,279	320,439	390,599	460,759	530,918	601,078	671,238	741,398
	30%	21,209	91,427	161,609	231,769	301,929	372,089	442,248	512,408	582,568	652,728	722,888
	40%	2,670	72,888	143,099	213,259	283,419	353,579	423,738	493,898	564,058	634,218	704,378
	50%	(15,869)	54,350	124,568	194,749	264,909	335,069	405,228	475,388	545,548	615,708	685,867

Table 22-13: Twin Parameter Sensitivity, Revenue v Capital Costs

NPV (USD'000)		REVENUE										
	427,619	-50%	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%	50%
CAPEX	-50%	115,797	185,956	256,116	326,276	396,436	466,595	536,755	606,915	677,075	747,234	817,394
	-40%	108,001	178,161	248,321	318,481	388,640	458,800	528,960	599,120	669,279	739,439	809,599
	-30%	100,206	170,366	240,526	310,685	380,845	451,005	521,165	591,324	661,484	731,644	801,804
	-20%	92,411	162,570	232,730	302,890	373,050	443,209	513,369	583,529	653,689	723,848	794,008
	-10%	84,615	154,775	224,935	295,095	365,254	435,414	505,574	575,734	645,893	716,053	786,213
	0%	76,820	146,980	217,140	287,299	357,459	427,619	497,779	567,938	638,098	708,258	778,418
	10%	69,025	139,184	209,344	279,504	349,664	419,823	489,983	560,143	630,303	700,463	770,622
	20%	61,229	131,389	201,549	271,709	341,868	412,028	482,188	552,348	622,507	692,667	762,827
	30%	53,434	123,594	193,754	263,913	334,073	404,233	474,393	544,552	614,712	684,872	755,032
	40%	45,639	115,798	185,958	256,118	326,278	396,438	466,597	536,757	606,917	677,077	747,236
	50%	37,843	108,003	178,163	248,323	318,482	388,642	458,802	528,962	599,121	669,281	739,441

Table 22-14: Twin Parameter Sensitivity, Operating v Capital Costs

NPV (USD'000)		OPEX										
	427,619	-50%	-40%	-30%	-20%	-10%	0%	10%	20%	30%	40%	50%
CAPEX	-50%	559,146	540,636	522,126	503,615	485,105	466,595	448,085	429,575	411,065	392,555	374,045
	-40%	551,350	532,840	514,330	495,820	477,310	458,800	440,290	421,780	403,270	384,760	366,250
	-30%	543,555	525,045	506,535	488,025	469,515	451,005	432,495	413,985	395,475	376,965	358,455
	-20%	535,760	517,250	498,740	480,230	461,719	443,209	424,699	406,189	387,679	369,169	350,659
	-10%	527,964	509,454	490,944	472,434	453,924	435,414	416,904	398,394	379,884	361,374	342,864
	0%	520,169	501,659	483,149	464,639	446,129	427,619	409,109	390,599	372,089	353,579	335,069
	10%	512,374	493,864	475,354	456,844	438,334	419,823	401,313	382,803	364,293	345,783	327,273
	20%	504,578	486,068	467,558	449,048	430,538	412,028	393,518	375,008	356,498	337,988	319,478
	30%	496,783	478,273	459,763	441,253	422,743	404,233	385,723	367,213	348,703	330,193	311,683
	40%	488,988	470,478	451,968	433,458	414,948	396,438	377,927	359,417	340,907	322,397	303,887
	50%	481,192	462,682	444,172	425,662	407,152	388,642	370,132	351,622	333,112	314,602	296,092

22.4.5 Impact of Including Inferred Mineral Resources

As noted, the mine schedule and associated TEM as presented above includes modified Inferred Resources. Figure 22-1 illustrates the contribution of Inferred Resources to the mine schedule.

Excluding the modified Inferred Resources would reduce the mine life to 11 years. Table 22-15 illustrates the summary LoM cashflow resulting from excluding this material while Table 22-16 shows the resulting NPV at various discount rates.

Table 22-15: Summary Cashflow Excluding Inferred Material

Project Cashflow	USD Millions
Gross Revenue	1,028
Deductions	55
Net Revenue	**973**
Operating Costs	258
Project Capital	85
Sustaining Capital	8
Closure	15
Project Cashflow	607
Working Capital	-0
Corporation Tax	25
Net Project Cashflow	582

Table 22-16: NPV Excluding Inferred Material

NPV	Pre-Tax (USD Millions)	Post-Tax (USD Millions)
5%	417	402
8%	337	325
10%	293	284
12%	256	248
14%	223	217

As can be seen, in summary, excluding the Inferred material reduces the overall LoM net project cashflow by approximately 50% and the post tax NPV as a 10% discount rate to USD 284M.

23 ADJACENT PROPERTIES

On 10 December 2012, through Balkan Gold doo, Erin was granted exploration licence #2065 which covers an area of 35.22km^2, adjacent to, and largely surrounding the Piskanja #1934 licence, as shown in Figure 23-1, below. This larger licence is valid until 10 December 2015 and allows Erin to continue exploration for borates, lithium, sodium, strontium and potassium in the region surrounding the Piskanja Project but primarily to the west.

State-owned Ibarski Rudnici (JP PEU Resavica) has a boron mineral exploitation licence (cadastral number 470) for the Pobrdje mineral deposit located 2.6 km northeast of the Erin licence and on the western bank of the Ibar river, as also shown in Figure 23-1 below. Erin has reported that material was produced from this mineral deposit during 2011, but no official figures have been published. According to information sourced from the Ministry of Mining and Energy of the Republic of Serbia "ore reserves" of boron in categories A+B+C1 were estimated at 140,000 tonnes and the "estimated resources" (P category) at 60,000 tons of B_2O_3. SRK does not consider these figures to have been reported in compliance with NI 43-101 guidelines.

SRK visited the above ground buildings at the Pobrdje Mine on 14 June 2013 and verify that the mine is in operation; however no details regarding its ownership or production are known to SRK.

A number of other exploration and small scale exploitation licences have been issued in the region, predominantly for coal, gold, lead zinc and asbestos as shown below in Figure 23-1 and listed in Table 23-1.

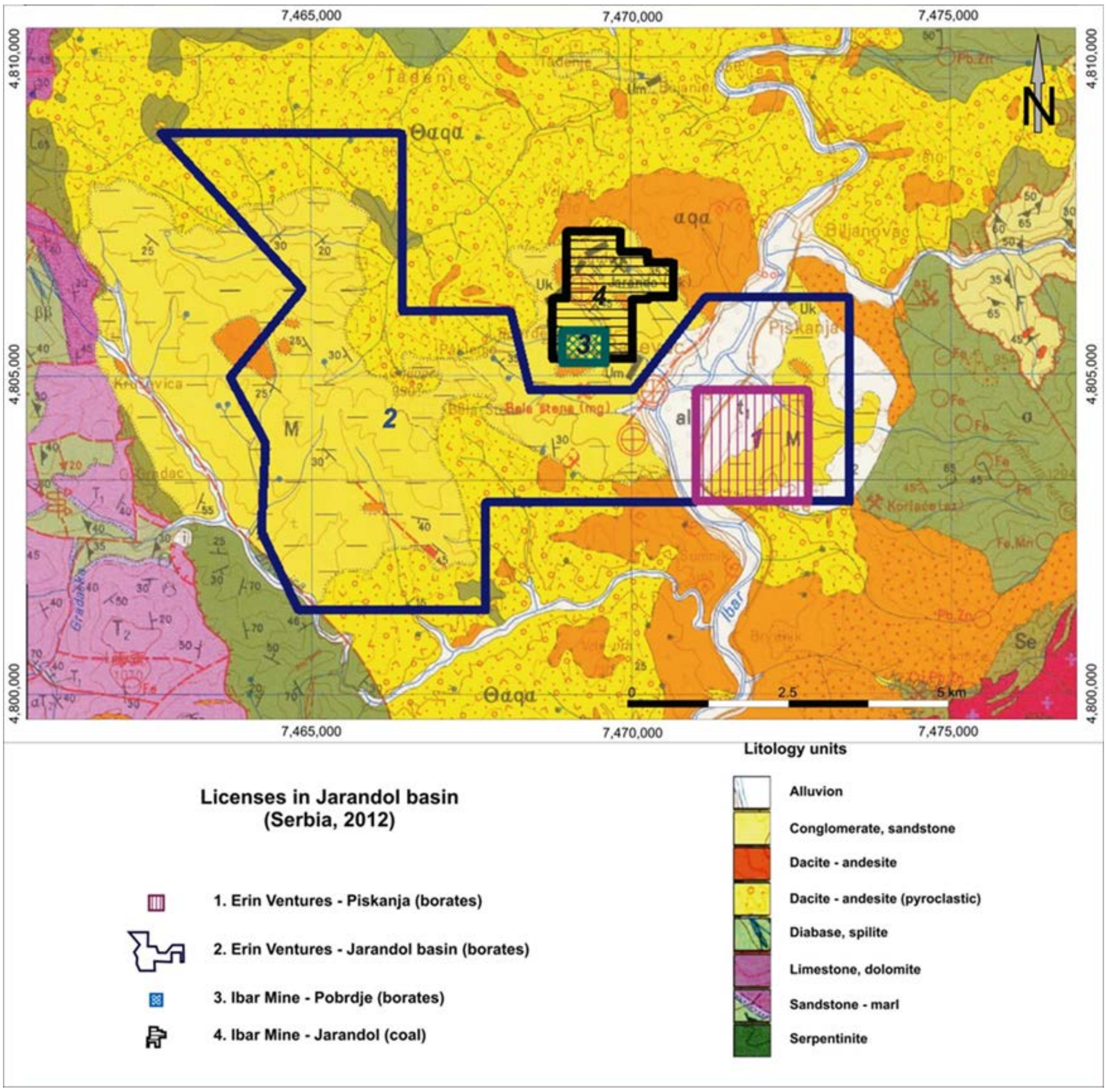


Figure 23 1: Exploration and mining licences immediately adjacent to the Piskanja Borate Project, (Erin press release, 8 January 2013)

Table 23-1: Mineral exploration and exploitation licences proximal to the Piskanja licence

Owner	Project name	Commodity	Licence number	Licence type	Distance from the Piskanja Project
Balkan Gold (Erin ventures)	Jarandol	Borates and associated elements	2065	Exploration	Immediately north and west
Preduzece Korlace	Korlace	Asbestos	98	Exploitation	2.5 km East
JP PEU Resavica	Pobrdje	Borates	470	Mining	2.6 km Northwest
Ibarski Rudnici (Ibar Mines)	Jarando	“Stone coal”	11	Mining	3km Northwest
Ibarski Rudnici (Ibar Mines)	Jarando	“Stone coal”	178	Mining	3km Northwest
Ibarski Rudnici (Ibar Mines)		“Stone coal”	177	Mining	8.5 km Northwest
JP PEV	Tadenje	Coal	485	Mining	5 km Northwest
Farmakom MB		Base metals and gold	1663	Exploration	2 km South
Farmakom MB	Kizevak	Lead and zinc	336B	Mining	5 km South
Balkan Exploration & Mining		Gold and silver	1969	Exploration	13 km South

24 OTHER RELEVANT DATA & INFORMATION

N/A

25 INTERPRETATION AND CONCLUSIONS

The exploration work undertaken by Erin to date in combination with work undertaken on the Project by previous workers has delineated a significant borate deposit which in SRK’s opinion justifies further exploration and assessment to determine whether or not it should be advanced to the development stage.

SRK has previously reported a Mineral Resource estimate for the Project comprising an Indicated Mineral Resource of 5.6 Mt with a mean grade of 30.8% B_2O_3 and an Inferred Mineral Resource of 6.2 Mt with a mean grade of 28.8% B_2O_3.

In this report SRK has now presented a PEA for the Project which has demonstrated the potential of the project and notably a post-tax NPV for the Project at a 10% discount rate of USD428M and an IRR of 64% which reduces to USD284M if based solely on Indicated Mineral Resources but increases to USD510M if the NPV is calculated using an 8% discount rate.

It should be noted that this PEA is preliminary in nature, that the NPV of USD428M includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as Mineral Reserves and there is no certainty that the PEA will be realised.

The Project is still at an early stage of assessment and much technical work remains to be completed and many risks removed before a decision could be made on putting a mine into production. This report highlights the work the authors consider needed to address these gaps and risks which notably includes further drilling to confirm the continuity and structure of the deposit (which is still uncertain), geotechnical testwork to help develop appropriate mine design parameters, further metallurgical testwork to confirm the potential to produce a saleable colemanite concentrate (which has not yet been demonstrated) as well as additional analysis in most areas to determine the infrastructure and service requirements of the project, the potential impacts of the Project on the environment and the measures needed to be put in place to mitigate these and also the likely construction costs for the Project (which remains preliminary at this stage).

SRK has agreed a work programme and costs for this work with Erin which culminates in the preparation of a PFS following which a decision will be able to made on whether or not to complete a feasibility study and if so the technical assumptions that should form the basis of this.

26 RECOMMENDATIONS

SRK has discussed with Erin the work required to be done to advance the Project towards the development stage, much of which is highlighted in this report, and based on this Erin has developed the budget given in Table 24-1 below. In summary the work comprises further data collection followed by the completion of a PFS all to be completed by end-2015 and the total budget developed for this inclusive of Erin management costs is USD5.1 million. The aim of the PFS will be enable the various options for the development of the Project as outlined in this report to be assessed so that a feasibility study can be commenced focussed on a single mining and processing option and the justification for such a study determined. Further funds though would then need to be raised to complete this study.

The exploration drilling includes both infill and extension drilling plus specific drilling to assess the presence of faulting; the bulk sampling will also be done via wide diameter drilling while the environmental and hydrological work will be commenced in tandem with this.

SRK is confident that the work proposed is justified by the potential of the project and that the budget allowed is reasonable given the work planned and recommends that this work is carried out as planned.

Table 26-1: Planned Expenditure

Item	USD 000
Exploration/Resource Drilling	2,100
Bulk Sampling/Metallurgical Testwork	800
Decline Drilling	450
Environmental Studies	150
Geotechnical Testwork	60
Hydrological and Hydrogeological Analysis	150
PFS Study	550
Office Costs	540
Contingency	300
Total	5,100

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For and on behalf of SRK Consulting (UK) Limited

Dr Mike Armitage, *BSc, MIMMM, FGS, CEng*
Corporate Consultant & Chairman
SRK Consulting (UK) Limited
15th September 2014

APPENDIX

A CERTIFICATES

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CERTIFICATE

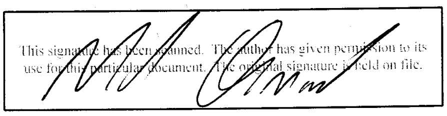
To Accompany the report entitled: TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE PISKANJA BORATE PROJECT, SERBIA Effective date 1st September 2014.

I, Mike Armitage, BSc, MIMMM, CEng, residing at Maesaeson House, Peterston-Super-Ely, Vale of Glamorgan CF5 6NE, Wales, UK, do hereby certify that.

1. I am Group Chairman and Corporate Consultant (Mining Geology) with the firm of SRK Consulting (UK) Ltd ("SRK") with an office at 5th Floor, Churchill House, Churchill Way, Cardiff, UK;
2. I am a graduate from the University of Wales, College Cardiff with an BSc. Honours Degree in Mineral Exploitation, (Specializing in Mining Geology) awarded in 1983 and also have a PhD from Bristol University in Structural and Resource Geology awarded in 1993. I have practised my profession continuously since 1983.
3. I am a Member of the Institution of Materials Mining and Metallurgy and I am a Chartered Engineer and a Fellow of the Geological Society.
4. I have not visited the Project site.
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 am one of the authors of this report and accept professional responsibility for this technical report as a whole;
7. As a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
8. I have had no prior involvement with the subject property;
9. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10. SRK was retained by Erin Ventures Inc. to prepare a technical audit of the Piskanja. The report is based on site visits, a review of project files and discussions with Erin Ventures personnel;
11. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Piskanja or securities of Erin Ventures Inc.;

12. 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; and

13. 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.



Dr Mike Armitage, *FGS, CGeol, MIMMM, CEng*
Group Chairman & Corporate Consultant
(Mining Geology), SRK (UK) Ltd.
Cardiff, UK, 15th September 2012

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CERTIFICATE

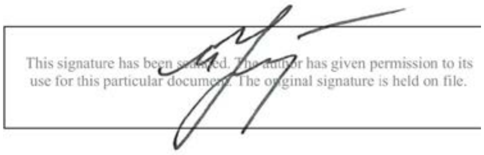
To accompany the report entitled: TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE PISKANJA BORATE PROJECT, SERBIA. Effective Date 1st September 2014.

I, Dr. Mikhail Tsypukov, residing at apt.54, Svobody Str. 20, Moscow, Russian Federation, do hereby certify that:

1. I am a Principal Exploration Geologist with the firm of SRK Exploration Services Ltd ("SRK ES") with an office at 12 St Andrew's Crescent, Cardiff, United Kingdom, CF10 3DD;
2. I am a graduate of the Irkutsk Polytechnic Institute, Russia in 1985. I obtained a Master of Science (Mineral Deposits and Applied Geochemistry) degree followed by a PhD in 1994 (Geochemistry and Economic Geology) from the A. P. Vinogradov Institute of Geochemistry, Russian Academy of Science, Irkutsk. I have practiced my profession continuously since 1985.
3. I am a Fellow of the Institute of Materials, Minerals and Mining (IMMM), membership number 459707;
4. I have personally inspected the subject project during the period 10 June 2012 to 15 June 2012 and again during the period 12 June 2013 and 14 June 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 fulfill the requirements to be a "qualified person" for the purposes of National Instrument 43-101;
6. I am one of the authors of this report and accept professional responsibility for the geology, mineralisation and data quality aspects and sections of this technical report.
7. I, as a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
8. I have had previous involvement with the subject property as co-author of the National Instrument 43-101 technical reports entitled Technical Report on the Piskanja Project, 16 July 2012 and Technical Report and Mineral Resource Estimate for the Piskanja Borate Project, Serbia, 23 November, 2013;
9. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10. SRK was retained by Erin Ventures Inc. to prepare a technical audit of the Piskanja Project. The report is based on site visits, a review of project files and discussions with Erin Ventures personnel;
11. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Piskanja Project or securities of Erin Ventures Inc;

12. 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; and

13. 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.



Moscow, Russia

15 September 2014

Dr Mikhail Tsypukov, PhD., FIMMM

Principal Exploration Geologist

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CERTIFICATE

To Accompany the report entitled: TECHNICAL REPORT AND PRELIMINARY ECONOMIC ASSESSMENT FOR THE PISKANJA BORATE PROJECT, SERBIA

I, Mr Mark Campodonic, residing at 50 Llanfair Road, Cardiff do hereby certify that:

1. I am a Principal Resource Geologist with the firm of SRK Consulting (UK) Ltd ("SRK UK") with an office at 5th Floor, Churchill House, 17 Churchill Way, Cardiff, CF10 2HH, Wales, UK;
2. I am a graduate of Cardiff University, University of Wales, UK in 2000. I obtained a Bachelor of Science with First Class Honours degree (Exploration Geology) in 1999, followed by a Masters with Distinction (Mineral Resource Evaluation) in 2000, both from Cardiff University. I have practiced my profession continuously since 2000
3. I am a Chartered Professional member of the AusIMM, membership number CP # 225925
4. I have not personally inspected the subject project
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 fulfill the requirements to be a "qualified person" for the purposes of National Instrument 43-101;
6. I am one of the authors of this report and accept professional responsibility for the Mineral Resource Estimation aspects and sections of this technical report.
7. I, as a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
8. I have had previous involvement with the subject property as co
Instrument 43
the Piskanja Project, Serbia, 23 November, 2013;
9. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10. SRK was retained by Erin Ventures Inc. to prepare a technical audit of the Piskanja Project. The report is based on site visits, a review of project files and discussions with Erin Ventures personnel;
11. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Piskanja Project or securities of Erin Ventures Inc;
12. 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; and

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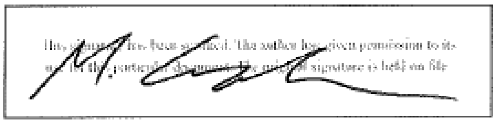

Mr Mark Campodonic BSc, MSc,
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Cardiff, UK 15th September 201