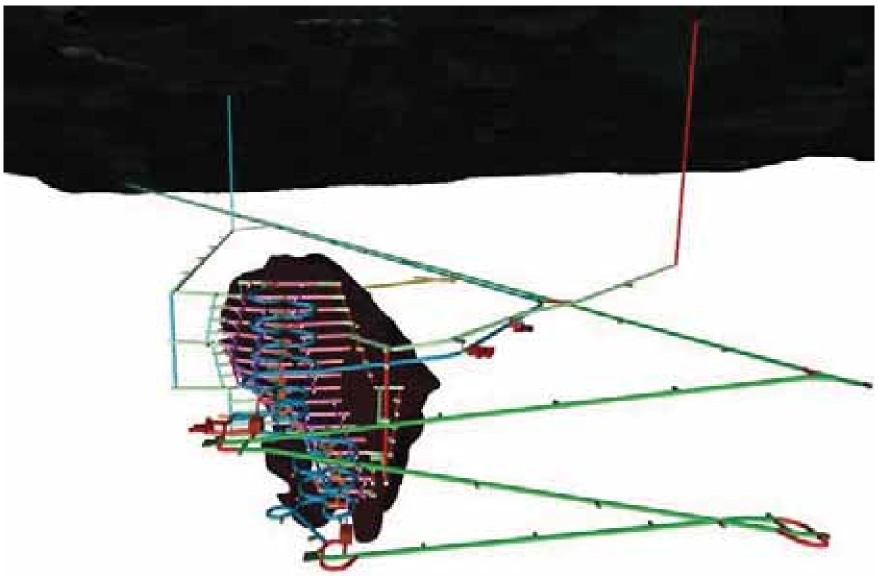
NI 43-101 Technical Report – Timok Copper-Gold Project, Serbia: Upper Zone Prefeasibility Study and Resource Estimate for the Lower Zone



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IMPORTANT NOTICE

This report was prepared by the qualified persons listed in Table 2.1. Each QP assumes responsibility for those sections or areas of this report that are referenced opposite their name in Table 2.1. None of the QPs, however, accepts any responsibility or liability for the sections or areas of this report that were prepared by other QPs.

This report was prepared to allow Nevsun Resources Ltd. (the "Owner") to reach informed decisions respecting the development of the Timok Project. Each QP has consented to the filing by the Owner of this report with Canadian Securities Administrators and its System for Electronic Document Analysis and Retrieval (SEDAR). Except for the purposes legislated under provincial securities laws, any use of this report by any third party is at that party's sole risk. This report is intended to be read as a whole, and sections should not be read or relied upon out of context.

This report contains estimates, projections and conclusions that are forward-looking information within the meaning of applicable securities laws. Forward-looking statements are based upon the responsible QP's opinion at the time that they are made but in most cases involve significant risk and uncertainty. Although each of the responsible QPs has attempted to identify factors that could cause actual events or results to differ materially from those described in this report, there may be other factors that cause events or results to not be as anticipated, estimated or projected. None of the QPs undertake any obligation to update the forward-looking information.

As permitted by Item 3 of Form 43-101F1, the QPs have, in the preparation of this report, relied upon certain reports, opinions and statements of certain experts. These reports, opinions and statements, the date, title and author of each such report, opinion or statement and the extent of reliance thereon is described in Section 3 of this report. Each of the QPs hereby disclaims liability for such reports, opinions and statement to the extent that they have been relied upon in the preparation of this report, as described in Section 3.

As permitted by Item 12 of Form 43-101F1, the QPs have, in the preparation of this Report, relied upon certain data provided to the QPs by the Owner and certain other parties. The relevant data and the extent of reliance upon such data is described in Section 3 of this Report.

Table of Contents

Table of Contents

List of Appendices

List of Tables

List of Figures

1. Summary

1.1 Introduction

This Preliminary Feasibility Study (PFS) update describes a future mine development on the Upper Zone and includes an initial resource estimate of the Lower Zone of the Čukaru Peki Copper/Gold massive sulphide deposit at Čukaru Peki, outside the town of Bor in Serbia.

The Upper Zone deposit at Čukaru Peki is 100% owned by Nevsun Resources Ltd. (Nevsun) through its wholly owned subsidiary Rakita Exploration d.o.o (Rakita). The Lower Zone is a joint venture between Nevsun and Freeport-McMoRan Exploration Corporation (Freeport). Nevsun currently has a 60.4% interest in the Lower Zone and Freeport a 39.6% interest. Upon completion of any feasibility study (on the Upper or Lower Zone), Nevsun will own 46% and Freeport 54% of the Lower Zone. Nevsun will continue to be a 100% owner of the Upper Zone.

The focus of the PFS (PFS, 2018) is an estimation of financial costs and revenues for the Upper Zone deposit only; assuming a normal project development cycle and using current technology to mine the deposit by underground means. Ore mined from the Upper Zone would be hauled to surface using a decline and conveyor and milled in a conventional flotation mill to produce a single sulphide copper gold concentrate which will be sold to smelters on normal terms.

This Technical Report follows on from the PFS published by Hatch Ltd. and others on May 11, 2018. Nevsun declared an initial Resource, including Measured, Indicated and Inferred tonnage and grade on the Upper Zone of the Čukaru Peki deposit in April 2017. The PFS Mineral Reserve Statement, by definition, contains less mineralization than the Resource statement, since a Reserve statement cannot contain Inferred mineralization.

The Lower Zone resource estimate is based on the data from a recently completed US$20 million drill program. This estimate has been prepared in accordance with the Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects and classified in accordance with Canadian Institute of Mining Metallurgy and Petroleum's "CIM Standards on Mineral Resources and Reserves Definitions and Guidelines".

This PFS will be followed by the generation of a Feasibility Study (FS) expected to be published in mid-2019. The FS will involve greater accuracy than the PFS and will also incorporate a number of Value Improvement (VIP) studies aimed at maximising the potential return from the deposit.

Throughout the years 2018 to 2020, Nevsun will continue to work to permit and develop the Čukaru Peki deposit and based on the PFS studies to-date, subject to permitting and finance, Nevsun is targeting first production from the Čukaru Peki Upper Zone in 2022.

1.2 Geology and Mineral Resource Estimates

The Čukaru Peki Upper Zone (UZ) is a copper-gold deposit located within the central zone (or Bor District) of the Timok Magmatic Complex, which represents one of the most highly endowed copper and gold districts in the world. The Timok Magmatic Complex is located within the central segment of the Late Cretaceous Apuseni-Banat-Timok-Srednogorie magmatic belt in the Carpatho-Balkan region of southern-eastern Europe. The Apuseni-Banat-Timok-Srednogorie belt forms part of the western segment of the Tethyan Magmatic and Metallogenic Belt, which lies along the southern Eurasian continental margin and extends over 1,000 km from Hungary, through the Apuseni Mountains of Romania, to Serbia and Bulgaria to the Black Sea.

The UZ deposit comprises two different styles of copper-gold mineralization - the Upper Zone (subject of this PFS) and the significantly larger Lower Zone (subject of the new Lower Zone resource estimate). Upper Zone high sulphidation (HS) epithermal mineralization occurs at depths from 450 to 850 m below surface. Lower Zone porphyry style mineralization is found from 700 to 2,200 m below surface. To date, the deepest drill hole intercepting Lower Zone mineralization terminated in mineralization at 2,268 m below surface.

The original Upper Zone NI 43-101 Mineral Resource estimate prepared by SRK Consulting (UK) Limited ("SRK UK") in 2017, on which this PFS and subsequent NI 43-101 Reserve Statement is based, was reported using a resource net smelter return (RscNSR) cut-off value based on copper, gold and arsenic, using a copper price of $3.49/lb and gold price of $1,565/oz, derived from long-term consensus metal price forecasts, with a 20% uplift to ensure that the Mineral Resource includes all mineralization appropriate for assessing eventual economic potential of mineral resources. Assumed technical and economic parameters were based on the results of the 2017 PEA.

SRK UK considered that the blocks with a RscNSR value greater than an operating cost of $35/t had "reasonable prospects for eventual economic extraction" and could be reported as a Mineral Resource. Based on a review of average block values in 5-m horizontal slices, SRK UK determined a level in the block model, (-445 m amsl), below which the average RscNSR fell short of covering this cost. The 2017 reported Mineral Resource therefore comprised all blocks inside the mineralization model above this elevation including a small number of individual blocks with RscNSR values lower than 35; this approach also excluded isolated blocks with >$35/t RscNSR below -445 m amsl.

The new Lower Zone NI 43-101 Mineral Resource is based and reported on a US dollar per tonne cut-off of greater than $25 per tonne. Modeling, resource estimation and tabulation were completed by Dr. Gilles Arseneau of SRK Consulting (Canada) Inc. The deposit was modelled using Leapfrog and a 0.2% copper equivalent cut-off using 102 drill holes and 14,592 assays. Results are tabulated using a dollar equivalent using US$3.00 a pound for copper and US$1,400 an ounce for gold, with recoveries of 87% for copper and 69% for gold in the porphyry copper zone based on initial test work performed on representative samples. The mining method is assumed to be by block cave.

The mineral resources were estimated using ordinary kriging and multiple passes with increasing search radii from 75 m up to 250 m and required at least two drill holes within the search volume to estimate a block grade. A final pass was used to infill any un-estimated blocks that were within 50 m of holes within the 0.2% copper equivalent grade shell.

All estimated blocks were classified as inferred mineral resources due to wide-spaced drilling and the lack of any inclined drill holes.

The 2017 Mineral Resource statement for the Upper Zone of the Čukaru Peki deposit, which forms the basis for the 2018 PFS and Reserve statement is shown in Table 1.1.

Table 1.1: SRK Mineral Resource Statement as at April 24, 2017 for the Upper Zone of the Čukaru Peki Deposit

Category	Resource Domain	Quantity Mt	Grade			Metal	
			% Cu	g/t Au	% As	Cu Mt	Au Moz
Measured	UHG	0.44	18.7	11.70	0.29	0.082	0.17
	Massive Sulphide	1.70	6.0	4.10	0.29	0.10	0.23
Indicated	UHG	0.95	17.1	11.80	0.24	0.16	0.36
	Massive Sulphide	6.70	5.2	3.40	0.25	0.35	0.73
	Low grade covellite	19.00	1.9	1.10	0.17	0.36	0.70
Measured and Indicated	UHG	1.40	17.6	11.80	0.26	0.24	0.52
	Massive Sulphide	8.40	5.4	3.60	0.26	0.45	0.96
	Low grade covellite	19.00	1.9	1.10	0.17	0.36	0.70
Inferred	UHG	0.45	15.0	10.80	0.16	0.07	0.16
	Massive Sulphide	0.80	4.9	3.40	0.11	0.04	0.09
	Low grade covellite	12.70	1.0	0.44	0.05	0.12	0.18
Total-Measured		2.20	8.6	5.70	0.29	0.19	0.40
Total-Indicated		26.60	3.3	2.10	0.20	0.87	1.80
Total-Measured and Indicated		28.70	3.7	2.40	0.20	1.05	2.20
Total-Inferred		13.90	1.6	0.90	0.06	0.23	0.42

1. The RscNSR value used to report the estimate is $35/t.
2. All figures are rounded to reflect the relative accuracy of the estimate.
3. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
4. The Mineral Resource is reported on 100% basis, attributable to Rakita Exploration d.o.o.

The 2018 Mineral Resource statement for the Lower Zone of the Čukaru Peki deposit, which is not included in the 2018 PFS and Reserves statement is shown in Table 1.2.

Table 1.2: SRK Mineral Resource Statement as at June 19, 2018 for the Lower Zone of the Čukaru Peki Deposit

Category	Resource Domain	Quantity Mt	Grade			Metal Contained	
			% Cu	g/t Au	% As	Cu Mt	Au Moz
Inferred	Lower Zone Porphyry	1,659	0.86	0.18	0.01	14.3	9.6
Total-Inferred		1,659	0.86	0.18	0.01	14.3	9.6

1. The value used to report the estimate is $25/t.
2. All figures are rounded to reflect the relative accuracy of the estimate.
3. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
4. The Mineral Resource is reported on 100% basis, 60.4% is attributable to Nevsun.

1.3 Mineral Processing and Metallurgical Testing

1.3.1 Upper Zone

As part of the PEA published in 2016 by SRK UK, preliminary testing on samples from the Upper Zone, was conducted between November 2015 and March 2016 by SGS Canada Inc.

The flowsheet, developed during the 2016 PEA used conventional reagents achieved good separation of the copper minerals from pyrite and gangue into a sellable bulk copper concentrate, a pyrite concentrate and a gangue tails slurry stream. The pyrite concentrate, containing a significant portion of the gold in the deposit, was to be stored in a dedicated facility for possible later treatment, but for that study it was considered a waste stream.

During the course of the subsequent 2017 updated PEA study, a second test program was completed by SGS on samples from the Upper Zone between September 2016 and September 2017. The emphasis of the test work was to optimize the flotation conditions established in the 2016 PEA and to provide samples to evaluate processing options for a separate high arsenic copper concentrate, referred to as a complex copper concentrate, a low arsenic concentrate) and separate pyrite and waste gangue streams.

Later, in a third program, the scope was increased to include: mineralogy, comminution test work, process feed aging test work, bulk flotation test work, further flotation optimization and variability testing, solid-liquid separation testing, and environmental characterization.

A goal of testing for the 2017 study was to optimize the flowsheet selected in the 2016 PEA, which produced two copper concentrates, the low arsenic and a complex concentrate. However, during the variability testing program of this study, it was realized by the PEA process group that a proportion of the orebody was not likely to respond well to this two-concentrate production scenario. Testing, and analysis of a simplified flowsheet producing a single bulk concentrate, combining both the high and low arsenic concentrate into a single product, was then carried out and in conjunction with Nevsun's marketing consultants, a decision was made by the PEA team to change to this simpler, more robust single concentrate approach for the 2017 revised PEA.

During the 2018 PFS design and analysis process, it was also determined by the PFS team that there was no economically viable proven method for recovering the gold from the pyrite concentrate produced and stored as part of the 2017 PEA model. As a consequence for this 2018 PFS, no pyrite concentrate is separately produced or stored and all pyrite goes to a combined whole tailings stored in a single lined tailings storage facility. The optimized flowsheet, developed during the current PFS thus uses conventional reagents, applicable to all process feed types and achieves good separation of the copper minerals from pyrite and gangue into a bulk copper concentrate. Should an economic treatment method for recovery of the remaining gold in pyrite be proven at a later date, then the whole tailings, including the contained pyrite, would be retreated to recover the gold, potentially using the copper flotation circuit, which by that time would have ceased operation, to float the pyrite.

During the 2017 PEA Orway Mineral Consultants, in Mississauga, Ontario reviewed the initial grind optimization work and comminution test work and completed a process plant-sizing study. Their report and the SGS flotation and other test results were passed to Ausenco Engineering in Toronto to produce design criteria, flowsheets, layouts and capital and operating cost estimates for a grinding and flotation plant to treat plant feed from the Upper Zone of the Čukaru Peki deposit for the PEA design.

All test work results and study reports from the 2017 PEA were subsequently passed to Hatch for further optimization, cost estimation, execution planning and completion of this PFS report. XPS of Sudbury, Ontario, were also engaged in mid-2017 to provide a geo-metallurgical assessment of the Čukaru Peki test work and, later, to confirm the relative merits of single concentrate versus dual (low and high arsenic) concentrate production. This test work is ongoing and in February 2018 XPS released a draft report on their current test program in which they confirmed that rougher flotation would be sufficient to produce a single sellable final concentrate in the early years of production.

In addition, four trade-off studies (ToS’s, #1, 2, 3 and 4) were completed as part of the 2017 PEA to define options regarding:

1. Concentrate sale, bulk vs. separate high and low arsenic concentrates.
2. Process options for gold recovery from pyrite.
3. Process options for reduction of arsenic in the complex concentrate.
4. Definition of concentrate transportation considerations.

As part of ToS #2, samples of the pyrite concentrate were tested to determine the applicability of certain gold recovery processes, i.e. pyrite roasting (Outotec) and atmospheric oxidation following fine grinding (Albion). The results of the tests and preliminary reports from the respective process technology suppliers were reviewed by Ausenco Engineering of Brisbane who prepared preliminary scoping level, capital and operating cost estimates to assist in determining if the processes for recovery of gold from pyrite would be economic for future consideration. Preliminary indications were that these processes are not economic for Čukaru Peki pyrite grades at this time, but the reviews are ongoing. This work is outside the scope of this PFS.

As part of ToS #3, samples of complex (high As) copper concentrate were tested for various arsenic removal processes, i.e. partial reductive roasting (Outotec), ferric oxidation (FLSmidth ROL®) and caustic leaching (Toowong). Each process supplier compiled a preliminary report summarizing its potential application. This remains an option for further study.

1.3.2 Lower Zone

A limited amount of metallurgical test work has been performed on the Lower Zone mineralization. In 2016, Aminpro (Aminpro, 2016) performed tests on three types of mineralization found in the Lower Zone: DC1 Overprint, DC2 Mixed Zone and DC3 Primary. Primary, or porphyry copper mineralization is by far the most abundant and most important.

Three types of mineralization were treated from the Lower Zone (LZ). The received samples showed that the copper minerals are predominantly chalcopyrite and bornite with increasing chalcopyrite with depth.

The metallurgical test work done at Aminpro-Chile laboratory in Santiago was of conceptual level.

1.4 Mineral Reserve

Mineral Reserve statements are based on material classed as economically recoverable Measured and Indicated Mineral Resources with dilution and mining/processing recovery factors applied. Depletion has been included in these estimates. No Proven Mineral Reserves have been declared.

Factors which may affect the Mineral Reserve estimates include commodity prices and valuation assumptions; changes to the proposed sublevel cave design, geotechnical, mining, and processing plant recovery assumptions; appropriate dilution control; changes to capital and operating cost estimates.

The Mineral Reserve statement for the Čukaru Peki deposit is presented in Table 1.4.

Table 1.3: Mineral Reserve Statement, Čukaru Peki Deposit, Republic of Serbia, March 8, 2018

Description	Quantity (kt)	Grade			Contained Metal		
		(% Cu)	(g/t Au)	(% As)	(klbs Cu)	(kOz Au)	(kt As)
Proven	0	0.00	0.00	0.00	0	0	0
Probable	27,121	3.25	2.06	0.17	1,944,074	1,792	47
Total	**27,121**	**3.25**	**2.06**	**0.17**	**1,944,074**	**1,792**	**47**

Notes:

1. The Mineral Reserves and Resources in this news release were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
2. Metal prices used include US$3.00/lb Cu and US$1,300/oz Au.
3. A Reserve NSR cut-off of US$35/tonne was used to optimize the SLC Ring layout.
4. Contained metal figures and totals may differ due to rounding of figures.

1.5 Mine Design and Infrastructure

1.5.1 Mine Design

As highlighted in the 2017 PEA Study (SRK NA, 2017e), the Sublevel Caving (SLC) mining method has been selected for the Čukaru Peki deposit as it has better consideration of geotechnical controls and offers higher value for the project as compared to other mining methods. This mining method provides early access to significantly higher grades of mineralization and made it preferable compared to other mining methods.

A dual decline access with separate access decline and conveyor decline was selected as the preferred access and haulage method. The access decline will be used as the main entry to the mine and will provide access for personnel, equipment, materials, and will be utilized as a fresh air intake airway. Process Plant feed and waste will be hauled from the mine via conveyor up the conveyor decline. The conveyor decline will also act as a major exhaust airway and provide a secondary egress from the mine. Dual decline portal location, decline size and layout were selected during the exploration decline design (SRK NA, 2017d).

The ventilation system design consists of two separate air streams or splits. The first air stream ventilates the access and egress/conveyor declines, providing fresh air to the access decline that is exhausted from the mine via the conveyor decline. Additionally, production airflow is provided to the active levels via a system of fresh air raises (FAR) that provide air directly to each level in parallel. A series of return air raises (RAR) remove air directly from the levels and remove it from the mine without passing it over any other active areas.

The SLC production rate is a function of the deposit geometry and continuity, the prevailing ground conditions, the number of available mining areas, and the expected productivity for each stope. The production resources were scheduled to achieve a practical production output and taking into account these factors.

The expected drawpoint productivity has been limited to 260 tonnes per drawpoint per day, based on benchmarking comparable mines. This productivity rate results in annual targets of 3.00 Mtpa for the SLC operations and an additional 0.25 Mtpa from development, providing 3.25 Mtpa of feed to the processing plant.

In the final years of production, the annual mining rate decreases to account for the reduced number of extraction rings. Due to the nature of SLC, towards the end of mine life the waste dilution percentage in the extracted material increases to the point where the material has insufficient grade to sustain the operation. Life of mine dilution is 27%, of which 3% is measured and indicated material that was not included in the original mine design. The remaining dilution is waste and inferred material that has been assigned zero grade. This reduced production rate, increased dilution and lower grade experienced at the end of mine life has resulted in the 12-year mine life being truncated in the economic model to a 10-year mine life.

1.5.2 Mine Infrastructure

Infrastructure required to support the Čukaru Peki mine operations will consist of the following major equipment:

- Mine Portals.
- Conveyor system for transporting crushed ore (and waste rock) from the underground crusher to the crushed ore storage bins located near the process plant.
- Surface fan installations.
- Crusher stations.
- Refuge stations.
- Mine Dewatering pumping stations.
- Explosives storage.
- Maintenance facilities.
- Utilities including electrical power distribution system, communications system, etc.

1.6 Recovery Methods

To align with the life-of-mine plan, the process plant is designed to treat nominally 8,900 tonnes per day (equivalent to 3.25 million tonnes per year) and produce a single bulk copper concentrate.

The flowsheet was designed in earlier studies with the ability to generate two copper concentrates; one with a low (<0.5% As) arsenic content and one with an elevated (>0.5% As) arsenic content. This same general flowsheet, utilized in a more simplified manner, has the capacity to produce a single bulk concentrate. Based on marketing considerations the production of a single bulk copper concentrate is the basis of this study. However, if future studies suggest reverting back to the two-product concept, it will be a relatively simple adjustment in terms of additional unit processes required at the back end of the flotation circuit.

The process adopted for this PFS specifies a moderate primary grind P_{80} of 108 µm using a SAG/Ball mill circuit, fed from a primary crusher underground. During the initial years, when the plant feed grade is high in copper, a rougher flotation circuit only produces a saleable copper concentrate. As the feed grade declines in later years a concentrate regrind followed by cleaner flotation stages are introduced to improve the copper grade.

The predicted metallurgical performance averaged over five nominal grade ranges are shown in Table 1.5.

Table 1.4: Predicted Metallurgical Performance

Ore type	Flowsheet and products	Concentrate Grade % Cu	Recovery Cu %	Recovery As %	Recovery Au %
High grade (>7% Cu)	Single rougher concentrate	19.0	96	92	61
High grade (5 to 7% Cu)	Single rougher concentrate	343	94	90	29
Medium high grade (4 to 5% Cu)	Single first cleaner concentrate	32	93	91	249
Medium low grade (2 to 4% Cu)	Single first cleaner concentrate	22	92	95	25
Low grade (<2% Cu)	Single second cleaner concentrate	21	91	94	22

1.7 Tailings and Waste Rock Management

1.7.1 TSF and Waste Rock Storage Location

A scoping level alternatives assessment determined the preferred location for the TSF in a valley east of the ore body. The assessment also concluded that conventional slurry tailings storage was the most appropriate tailings technology for the project. The preferred site was determined using a Multiple Accounts Assessment approach that considered technical, economic, environmental and socio-economic categories in the ranking and evaluation process.

1.7.2 Tailings and Waste Rock Storage Configuration

Mineral processing will generate tailings that will be deposited in the TSF. Waste rock will be hauled from the Plant Site and stockpiled in two designated waste rock storage sites within the TSF catchment.

Tailings will be deposited as a conventional slurry at a solids content of 31% by mass, with an estimated average settled dry density of 1.4 tonne/m^3. The TSF is designed to accommodate tailings, an operating pond and the Inflow Design Flood (IDF) for the facility.

The TSF embankment construction sequence involves a starter dam and impoundment, to store tailings for the first year of operations, along with ongoing expansions that utilize the downstream construction method.

Each of the embankment stages is constructed with 3H:1V slopes. There are 8 m benches at each of the stage raises. The final embankment has a minimum crest width of 10 m. There is a 40 m wide downstream buttress included for each of the stages. The overall slopes of the final constructed embankment are approximately 3.3H:1V for the upstream side and 3.5H:1V for the downstream side.

1.7.3 TSF Closure

TSF closure will be completed in a manner that will satisfy physical and chemical stability. The primary objective of closure and reclamation will be to return the TSF site to a self-sustaining condition consistent with the local landscape. The TSF will be capped with a geomembrane and low permeability soil layers, and contoured to transition the site to a landform. A closure spillway will be excavated into bedrock on the east side of the TSF embankment. The overall impoundment will be graded and revegetated. A swale will be incorporated in the final surface grading arrangement so that runoff may be routed to the closure spillway by gravity.

1.8 Surface Infrastructure

1.8.1 Site Infrastructure

Site infrastructure includes roads, drainage, security fencing, non-process buildings, power supply/distribution, water supply/distribution and various other utilities/services.

At the mill site, non-process buildings include:

- Gatehouses.
- Administration building.

- Fire and medical facilities.
- Mine dry.
- Maintenance and warehouse facilities.

Other site infrastructure at the mill site includes:

- Fuel and lubricant storage.
- Vehicle fueling facility.
- Long term core storage.
- Laydown space for spares.
- Truck wheel wash facility.
- Truck weigh scale.
- Rotainer storage area.
- Shotcrete and concrete batch plant.

Most of the site infrastructure is at the mill site area, but there are some facilities located at the mine portal area including, a gatehouse, fire and medical facilities, mine offices, lamp room, core handling building, and utilities.

Electrical power to the plant will be supplied from a purpose built 110/35 kV utility substation. The plant load is estimated at 35 MW. The new substation will be fed from the existing nearby 110 kV overhead transmission line "BOR 2 – ZAJECAR 2" through a double circuit transmission line. The 110/35 kV substation and connecting overhead line will be designed and supplied by the state owned Transmission System Operator (TSO).

For primary power distribution, the main 35 kV substation will be fed from a 110 kV utility substation by two 35 kV cable buses. A prefabricated modular building (E-House), with main 35 kV switchgear and all auxiliary equipment inside, will be located adjacent to the processing plant.

For secondary power distribution, the selected secondary distribution voltage levels for the process plant are 6 kV, 3 phase, 50 Hz for large drives and 400 V, 3 phase, 50 Hz for smaller drives.

Standby power for the process plant will be provided from diesel powered generators located at the main 35 kV substation.

1.8.2 Site Wide Water Balance

A site wide water balance was carried out under average climatic data, to assess the seasonal water volume variations over the life of mine and the performance of the site water management facilities. The balance includes pre-production and production period providing annual water inputs to the water treatment plant. Groundwater and subsidence infiltration water into the underground mine is a major input to the site wide water balance and acts as the main water source for the mill operation and for mine service water.

Water will be stored in the TSF to ensure sufficient water is available for mill start up and operations. In addition to tailings storage, during operations, the TSF will be used as a water storage reservoir to meet process plant demand. The TSF water inventory will be decreased as the mine closure period approaches and water will be discharged via a water treatment plant.

1.8.3 Effluent Treatment Plant

Although submergence of pyrite in the tailings storage is expected to prevent acidification, the tailings may generate acidity or leach sulfates and metals over time. Water from this pond along with other water in contact with mining materials (e.g., stockpiles, underground mine works, etc.) will be treated by the effluent treatment plant for re-use, and discharge when necessary.

The effluent treatment plant comprises a high-density sludge process followed by ultra-filtration prior to re-use. If discharge to the environment is required, reverse osmosis will be employed to remove dissolved solids to meet effluent objectives.

1.8.4 Off-Site Infrastructure

Off-site infrastructure includes tie-ins to power and water sources and facilities required for transportation of the concentrate product to an ocean port.

The project is favorably situated for inland and export logistics. Off-site infrastructure required for concentrate export includes tie-ins to road, rail and port facilities required for transportation of the concentrate to their final destinations. Rotainer storage areas will be required at the Bor train station and at Burgas port; as well as Rotainer handling equipment at the Bor train station and Burgas port.

1.9 Marketing

The Marketing of the concentrates produced by the Project will be a key economic driver. In developing its Marketing strategy, the Company is leveraging its network of smelter and trader relationships, gained in the marketing of copper concentrates from its existing Bisha mine. European and Asian-based smelting companies and concentrate trading companies with blending capabilities have expressed interest in procuring the Project's concentrate via long term contracts. With easy access to both European and Asian smelters, logistics costs were also considered in determining the most viable destinations for the concentrates. The realization costs in the PFS are based on these indicative discussions. Discussions with potential off takers will continue to advance in parallel with the feasibility study.

The Project will produce a single stream of copper concentrate with a life of mine (LoM) average grade of 26.2% copper, 5.7 grams per tonne gold and 1.4% arsenic. The estimated treatment and refining charges for the concentrate have been adjusted upwards and additional arsenic penalties have been added, to compensate prospective buyers for the higher arsenic content, particularly later in the mine life. Arsenic levels will be lower in the early years of production allowing for a more diverse customer profile and ease of product placement. Apart from elevated levels of arsenic, the Project's concentrate is not expected to have any elevated levels of other deleterious elements.

The copper concentrate market is expected to move into deficit over the next few years as the Project is developed. Final terms for the concentrates will be dependent on the relative supply and demand for the overall copper concentrate market and arsenic supply.

1.10 Capital Cost Estimate

The Project's capital costs are divided into two main categories:

- Pre- sanction date expenditures (US$114 million), which are not included in this financial evaluation. These are the costs the Company will incur to take the project to construction decision, including additional engineering studies, feasibility study work, maintenance of project staff in Serbia and Canada, land acquisition in Serbia, Serbian Permitting and development of the exploration decline complex.
- Post sanction date expenditures (US$574 million).

The Construction Decision Date (referred to as "Sanction Date") is defined as the date on which Nevsun's Board approves the construction of the Project and by which time the Project has all required permits and financing to allow construction of the mine, surface facilities and infrastructure to proceed to completion without constraint. A sanction date of Q3 2020 has been assumed.

A summary of the capital cost estimate according to major work breakdown structure (WBS) is presented in Table 1.6. The CAPEX was prepared in accordance with guidelines established by the Association for the Advancement of Cost Engineering (AACE) for a Class 4 (equipment factored) estimate. The anticipated level of accuracy is -20% to +25%.

The capital project is considered to include tasks up until the process plant/concentrator is ready to receive and start processing ore. Mine operating costs incurred prior to start up, were capitalized and included in the CAPEX. After mine/process plant start-up, costs were considered either operating costs (OPEX) or sustaining capital costs (SUSEX).

Table 1.5: Level 1 CAPEX Summary

WBS	Description	Estimated Cost (USD)
Pre Sanction Date Expenditures		
	Owner's Project Development Costs	70,928,000
	Decline Development (Exploration licence)	42,962,000
	Total Pre Sanction Date Expenditures	**113,890,000**
Post Sanction Date Expenditures		
	Direct Costs	
1000	Site Development	49,142,000
2000	Mining	159,798,000
3000	Concentrator	99,109,000
4000	Pre-Production Operating & Maintenance	17,194,000
5000	Tailings, Waste Rock and Reclaim Water Management	46,866,000
	Subtotal Direct Costs	**372,109,000**
9000	**Indirect Costs**	**105,882,000**
	Contingency	**95,598,000**
	Total Post Sanction Date Expenditures	**573,589,000**
Total Pre-Operational Expenditures		**687,479,000**

The capital cost estimate was prepared by mining, process and discipline engineers and cost estimators. Prefeasibility level documents used to generate the capital cost estimate include:

- Terrain (topographic) model: LiDAR survey data (with 1 m vertical contours) is available for the Project area.

- 3D model of the mine and mine infrastructure.
- 3D model of the tailings storage facilities.
- Process deliverables including metallurgical test work, block flow diagrams (BFDs), process flow diagrams (PFDs), process design criteria (PDC) and mass/energy balances.
- Preliminary site plans, area plot plans and general arrangement drawings of the process plant.
- Process Plant geotechnical conditions have been assumed since information is not available specifically in the proposed process plant location. Data from the mine portal and the TSF was considered for the assumed geotechnical conditions (to be confirmed during the feasibility study).
- Mechanical equipment list (MEL) including assumed electrical loads.
- Single line diagrams (SLD).
- Firm price quotes for the portal area site preparation, mine decline and budget quotations for the design/supply of major mechanical equipment.
- Material take-offs (MTOs) for mine development, mine infrastructure, site preparation and TSF bulk earthworks as well as process plant concrete, structural steel and architecture (i.e. building siding and roofing).
- Factored process plant utilities including electrical power distribution and controls/instrumentation.
- Unit pricing for bulk commodities (concrete, steel) and site labour based on in-house data for the Serbian market.
- Factored indirect cost estimates.
- A contingency equivalent to 20% of the direct and indirect cost estimates has been assigned to the project.

1.11 Operating Cost Estimate

The operating cost estimate is a joint effort by Nevsun, SRK Consulting, Knight Piésold (KP), Conveyor Dynamics INC.(CDI) and Hatch. The team worked together to coordinate their individual cost estimates and then reviewed the integrated document.

The operating cost structure developed for the project has four components: mining, ore re-handling on the surface and processing, water management and Tailings Storage Facility (TSF), and General and Administrative (G & A). The operating costs have been estimated on a quarterly basis and are linked to the mine production schedule. The labour component in each area of the operating costs has been developed separately from a detailed organization plan. The summary operating costs for the project used for the evaluation are shown in Table 1.7. Costs reported in Table 1.7 reflect plant operation with a Life Of Mine (LOM) milled tonnage of 27.1 Mt over 10 years of plant operation.

Table 1.6: Operating Cost Summary

Operating Cost Summary	LOM ($M)	Unit Costs ($/t)
OPEX - Mining	526	19.41
OPEX – Processing & Ore Re-handling	274	10.09
OPEX - Water Management, Effluent Treatment & TSF	27	1.01
OPEX - G&A	52	1.91
Total OPEX including Contingency	**879**	**32.42**

1.12 Project Execution Plan (PEP)

A PFS level project execution plan and level 2 schedule was developed for the Project. The methodology for developing the PEP and schedule is summarized below:

- A procurement packaging strategy was developed. This included lists of major equipment supply packages, bulk material supply (Project supply versus contractor supply) and site based contracts.
- Key permitting milestones were identified (by Nevsun/Rakita).
- Durations for the supply of critical equipment packages were estimated based on budget quotations or Hatch in-house data.
- The Hatch engineering manager, a construction manager and the project planner reviewed the site layout and general arrangement drawings to develop an integrated engineering, procurement, construction and commissioning schedule.
- A methodology for advancing engineering as per Serbian requirements was established including permitting and licensing requirements.
- Serbian vendor and contractor capabilities were discussed.

Refer to Table 1.8 for Project milestone dates:

Table 1.7: Key Milestone Dates

Milestone Description	Start	Finish
Sanction Date	Month 1	
Decline Construction Completed	Month 1	
Detailed Engineering	Month -2	Month 11
Procurement First Commitment	Month 1	
Construction First Commitment	Month 1	
Procurement – First Equipment on Site	Month 13	
Construction - Infrastructure	Month 11	Month 21
Construction – Process Facility	Month 8	Month 21
Construction - TSF	Month 4	Month 15
Water Collection - TSF	Month 13	Month 24
Construction – U/G	Month 13	Month 20
Process Plant – Pre-Commissioning	Month 18	Month 24
Mine Development	Month 1	Month 21
Stockpile Development	Month 13	Month 27
Ramp-Up	Month 24	
Full Production	Month 34	

The critical path activities (i.e. less than ten days of float in the schedule) are:

- The Sanction Date.
- Award of long lead packages for the Process Plant.
- Process plant construction.
- TSF Water collection.

1.13 Economic Analysis

The Project economics for the Timok Pre-Feasibility Study were evaluated in a post-Sanction Date, real money, post-tax financial model. Only costs incurred after the proposed Sanction Date of Q3 2020 are considered in the model. The key financial results, project returns and cash flows are presented herein.

All economic assessments are calculated at the Timok Project level and therefore, do not include certain costs including corporate office, interest, financing and exploration expenses.

A summary of the financial results for the Timok Project is provided in Table 1.9:

Table 1.8: Summary of Key Financial Results

Parameter	Unit	Value
Cu Price	$/lb	3.15
Au Price	$/oz	1,300
Project CAPEX	$M	574
Sustaining Capital	$M	239
Closure Costs	$M	48
OPEX	$/t Ore	32.42
Total Cash Costs	$/t Ore	93.32
Concentrate Produced	Mt	3.16
C1 Costs	$/lb	0.92
After Tax NPV8%	**$M**	**1,816**
IRR	**%**	**80%**
Payback	years	0.9

The Timok Project has an initial estimated capital cost of $574M, an estimated sustaining capital of $239M and a total estimated operating expenditure of $879M (with a C1 cost of $0.92/lb Cu) over the life of the mine, generating an after-tax internal rate of return of 80% and an after-tax NPV8% of $1,816M.

Free cash flow is presented year on year in Figure 1.4.

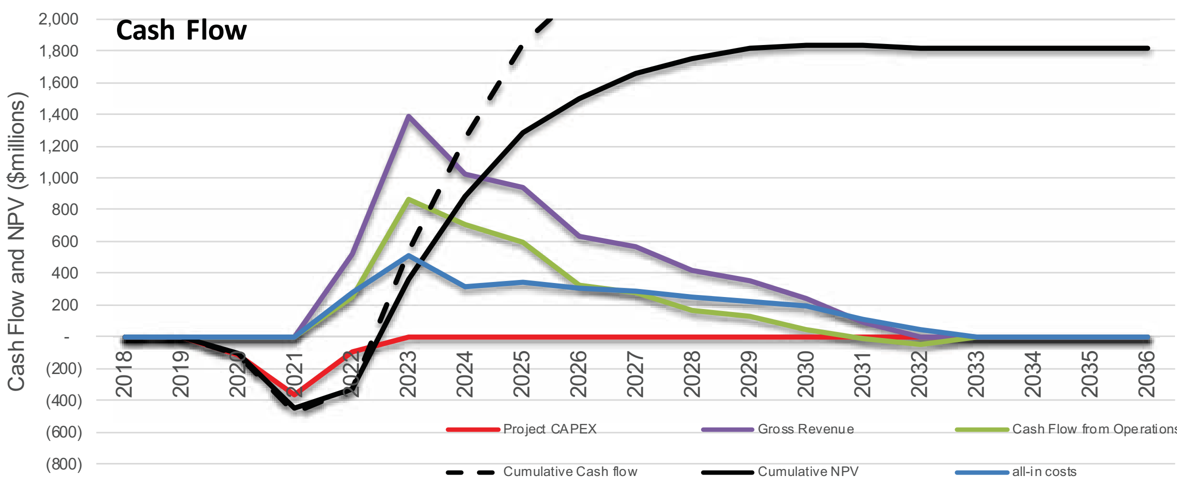


Figure 1.1: Net Cash Flows

1.14 Risks and Opportunities

Effective risk management is integral to the capital investment cycle, from evaluation of a business development opportunity, through basic and detailed engineering, project execution, operations and, ultimately, closure. A structured and thorough understanding of the key risks of the investment allows the project team to focus their attention and allocate resources effectively.

Aligned with the Nevsun Resources Ltd risk management guidelines and the Hatch risk management framework, the consideration of risk during the PFS encompassed all aspects of mining, geology, metallurgy, permits, land acquisition, cost and schedule, environment, communities, health and safety, human resources, project strategy and economics.

The primary focus of risk management was to update the project risk register developed in the previous project phases, to ensure that risk information was valid, accurate and complete, and all project risks (threats and opportunities), which could affect the achievement of the project and business objectives were identified and evaluated. The risk register update was undertaken with key project stakeholders from Nevsun, SRK, Knight Piésold and Hatch.

Figure 1.5 provides a summary of the project risk profile at end of PFS. In summary, ninety-six (96) risks remained open, of which 92 threats and four (4) opportunities.

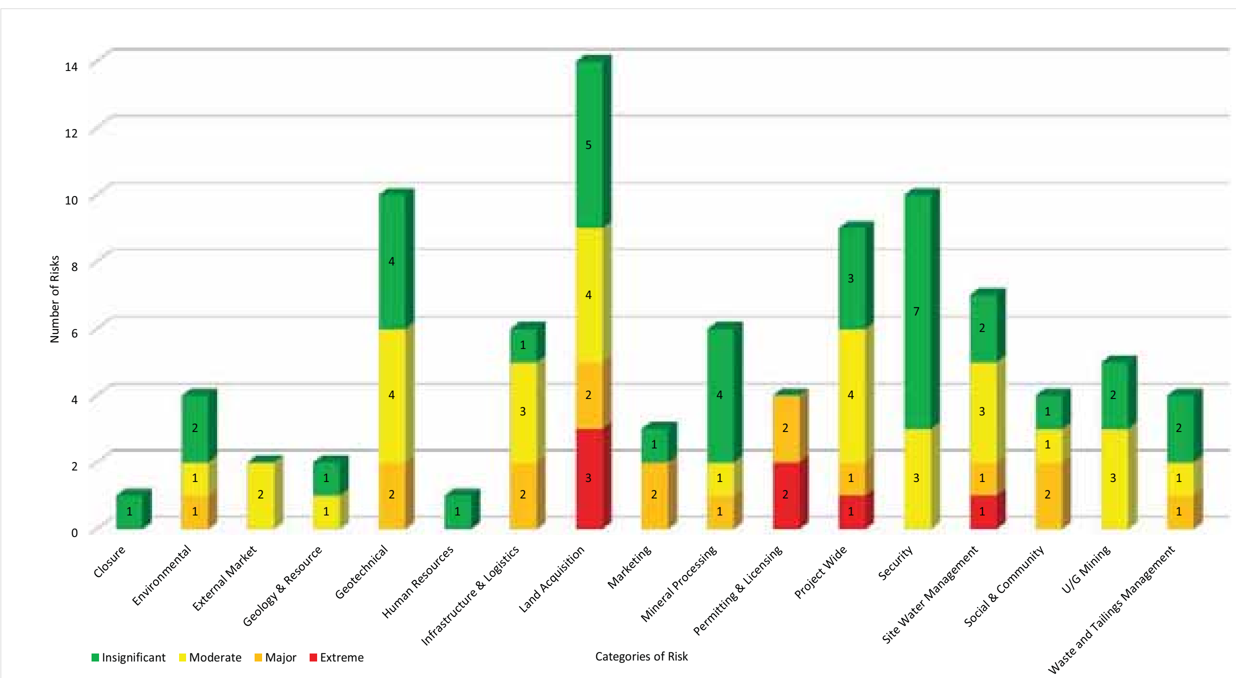


Figure 1.2: Timok PFS Project Risk Profile (Threats)

The most significant risks to the Project at end of the PFS are related to land acquisition, permitting and licensing, site water management and engineering.

Note that subsequent to the Risk Assessment Workshop, Nevsun decided to advance the project Sanction Date from Q3 2021 to Q3 2020. The project risk profile presented in this document does not reflect this advanced Sanction Date. The impact of this strategy is an increase in the liklihood of a schedule delay, due to potential delays in the permitting process delaying the Sanction Date.

2. Introduction

2.1 Report Preparation

This Preliminary Feasibility Study (PFS) update describes a future mine development on the Upper Zone and includes an initial resource estimate of the Lower Zone of the Čukaru Peki Copper/Gold massive sulphide deposit at Čukaru Peki, outside the town of Bor in Serbia.

The Upper Zone deposit at Čukaru Peki is 100% owned by Nevsun Resources Ltd. (Nevsun) through its wholly owned subsidiary Rakita Exploration d.o.o. The Lower Zone is a joint venture between Nevsun and Freeport-McMoRan Exploration Corporation (Freeport). Nevsun currently has a 60.4% interest in the Lower Zone and Freeport a 39.6% interest. Upon completion of any feasibility study (on the Upper or Lower Zone), Nevsun will own 46% and Freeport 54% of the Lower Zone. Nevsun will continue to be a 100% owner of the Upper Zone.

The focus of this PFS is an estimation of financial costs and revenues for the Upper Zone deposit only; assuming a normal project development cycle and using current technology to mine the deposit by underground means. Ore mined from the Upper Zone would be hauled to surface using a decline and conveyor and milled in a conventional flotation mill to produce a single sulphide copper gold concentrate which will be sold to smelters on normal terms.

This Technical Report follows on from a PFS published by Hatch Ltd. and others on May 11, 2017. Nevsun declared an initial Resource, including Measured, Indicated and Inferred tonnage and grade on the Upper Zone of the Čukaru Peki deposit in April 2017. The PFS Mineral Reserve Statement, by definition, contains less mineralization than the Resource statement, since a Reserve statement cannot contain Inferred mineralization.

The Lower Zone resource estimate is based on the data from a recently completed US$20 million drill program. This estimate has been prepared in accordance with the Canadian National Instrument 43-101 – Standards of Disclosure for Mineral Projects and classified in accordance with Canadian Institute of Mining Metallurgy and Petroleum's "CIM Standards on Mineral Resources and Reserves Definitions and Guidelines".

Unless otherwise stated, when the Timok project or Project is referenced in this report, this refers to the development of the Upper Zone portion of the deposit only.

This Technical Report was prepared by Hatch, SRK and Knight Piésold for Nevsun Resources Ltd. A summary of the QP responsible for each section of the report and their respective company affiliation is provided in Table 2.1.

Table 2.1: Summary of Qualified Persons' Responsibilities

Responsible Person	Company	Primary Areas of Responsibility	Relevant Sections
Martin Pittuck C. Eng., FGS, MIMMM	SRK Consulting (UK) Limited	Geology and Mineral Resources – Upper Zone	Sections 1.2, 2.2.1.1, 2.3, 7.1, 7.2, 7.3.1, 7.4, 7.5, 7.6, 8, 10.1, 10.2, 10.3.1, 10.4, 10.5, 11.1, 11.3, 11.5, 12.1, 14.1, 14.3, 14.5, 14.7, 14.9, 14.11, 14.13, 14.15, 14.17, 14.19, 14.21, 14.23, 14.25, 14.27, 14.29, 14.31, 14.32, 25.1 and 26.1
Gilles Arseneau, P. Geo.	SRK Consulting (Canada) Inc.	Mineral Resources – Lower Zone, Mineral Processing, Metallurgical Testing and Recovery Methods – Lower Zone	Sections 1.2, 1.3.2, 2.2.1.2, 2.3, 7.3.2, 10.3.2, 10.5, 11.4, 11.5, 12.2, 13.3, 14.2, 14.4, 14.6, 14.8, 14.10, 14.12, 14.14, 14.16, 14.18, 14.20, 14.22, 14.24, 14.26, 14.28, 14.30, 25.1, 26.1, 26.3.2 and relevant content in section 1.14
Michael Bunyard C. Eng., FAusIMM	Hatch Ltd.	Mineral Processing and Metallurgical Testing – Upper Zone, Recovery Methods – Upper Zone, Operating Cost Estimate – Upper Zone	Sections 1.3.1, 1.6, 1.10, 1.11, 2.2.3, 13.1, 13.2, 17.1, 17.2, 17.3, 17.4, 25.3, 26.3, 26.3.1 and relevant content in sections 1.14 and 21
Jarek Jakubec C. Eng.	SRK Consulting (Canada) Inc.	Mine Geotechnical Engineering, Mine Design and Mineral Reserves – Upper Zone	Sections 1.4, 1.5.1, 2.2.2, 2.3, 15, 16.1, 16.2, 16.3, 16.4, 25.2, 26.2 and relevant content in sections 1.10, 1.11, 1.14 and 21
Mihajlo Samoukovic, P. Eng.	Knight Piésold	Tailings Management	Sections 1.7, 2.2.4, 2.3, 18.2, 25.4, 26.5 and relevant content in sections 1.10, 1.11, 1.14 and 21
Gary MacSporran P. Eng.	Nevsun Resources	Marketing, Environmental, Social and Community Impact, Permitting and Land Acquisition	Sections 1.9, 3, 19, 20.1, 20.2, 20.3, 20.4, 24.2, 25.5, 25.7, 26.6, 26.8 and relevant content in section 1.14
Peter Manojlovic P. Geo.	Nevsun Resources	Adjacent Properties	Sections 1.1, 2.1, 9, 11.2, 12.1.2, 12.2, 23, 24.3, 27, 28 and relevant content in section 1.14
Mark Sucharda, P. Eng	Hatch Ltd	Underground Mine and Surface Infrastructure, Capital Cost Estimate, Closure Plan, Project Execution Plan	Sections 1.5.2, 1.8, 1.12, 4, 5, 6, 16.5, 18.1, 18.3, 18.4, 18.5, 20.5, 24.1, 26.4 and relevant content in sections 1.10, 1.11, 1.14 and 21
Robert Duinker P. Eng, MBA	Hatch Ltd	Financial Analysis	Sections 1.13, 22, 25.6, 26.7 and relevant content in section 1.14

2.2 Terms of Reference

2.2.1 Geology and Resource

2.2.1.1 Upper Zone

This PFS is based on a previously reported Mineral Resource estimate (PFS, 2018), classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") classification system. Current report does not contain an update to the Mineral Resource estimates for the Upper Zone as there has been no material change to the data upon which it is based

2.2.1.2 Lower Zone

This Technical Report contains new information on the Lower Zone solely and includes an initial resource estimate prepared in accordance with the CIM classification system.

2.2.2 Mining

This Technical Report is a part of an ongoing development of the Timok project. Current mine plan is based on Measured and Indicated resource categories only and results in a reserve estimate, as opposed to the mine plan that was developed for the PEA which included an inferred resource material. Furthermore, in this PFS, a geotechnical model was used to produce the mine design.

2.2.3 Processing

This Technical Report does not contain any changes to the previously reported concentrate production strategy which has occurred since the May 2018 PFS (PFS, 2018) was issued. Additional marketing and economic evaluations conducted during the course of the PFS have suggested to:

- produce only a single copper concentrate rather than the two planned earlier, and
- eliminate the production of a pyrite concentrate.

2.2.4 Tailings Management

This Technical Report documents the tailings storage facility (TSF) design that includes a single storage cell for the combined tailings stream and does not include any changes to the TSF strategy disclosed under the previous Technical Report (PFS, 2018).

2.3 Personal Inspection of Timok Property

In accordance with NI 43-101 guidelines, project qualified persons visited the Timok copper-gold project to inspect the site and review geology and exploration protocols. The most recent site visits conducted by the qualified persons are provided in Table 2.2.

Table 2.2: Qualified Persons Site Visits

Qualified Person	Company	Visit Date
Martin Pittuck	SRK Consulting (UK)	March 2017
Jarek Jakubec	SRK Consulting (Canada)	June 2017
Mihajlo Samoukovic	Knight Piésold	February 2017
Gilles Arseneau	SRK Consulting (Canada)	May 2018

3. Reliance on Other Experts

3.1 General

This report has been prepared by the QPs referred to in Table 2.1 for Nevsun. The information, conclusions, opinions and estimates contained herein relating to tax, legal, political, environment and commodity pricing are based on:

i) Information available to the QPs at the time of preparation of this report, including the 2017 Preliminary Economic Assessment.

ii) Assumptions, conditions and qualifications as set forth in this report.

iii) Data, reports and other information supplied by Nevsun.

For the purpose of this report, the QPs have relied on property ownership information provided by Nevsun. Hatch has not researched property title or mineral rights for the Timok property and expresses no opinion as to the ownership status of the property.

A draft copy of the Report has been reviewed for factual errors by Nevsun Resources. Any statements and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are not false and misleading at the date of this Report.

3.2 Marketing Expertise

Nevsun continues to contracted the services of Bluequest Resources AG in preparing marketing studies for the concentrates likely to be produced by the Project (Bluequest, 2017 and updated in 2018). The results of these reports along with an independent review by Ocean Partners UK Limited (Nevsun Resources – Timok Copper Marketing/Sales Allocation Strategy – April 2018). Nevsun has also relied on it’s developing in-house marketing team for their experience and knowledge which is supplemented by the Bluequest and Ocean Partners reports. This information is used in Sections 19 of this report and relevant conclusions and recommendations.

3.3 Serbian Permitting & Environmental Expertise

Nevsun continues to contract the services of ENVICO Environmental Consulting d.o.o. (Belgrade) in assisting with the Serbian Permitting Procedure for Mine Development. An initial report was established in July 2016 to commence the permitting process and ENVICO has continued to review and provide guidance on the procedure. Nevsun relies on Rakita’s site team for their experience and knowledge which is supplemented by the ENVICO reports and guidance. This information is used in Sections 20 of this report and relevant conclusions and recommendations.

4. Property Description and Location

The Project is located in eastern Serbia, 5 km from the Bor mining complex and approximately 250 km southeast of Belgrade. It lies within the central zone of the Timok Magmatic Complex (TMC), in the Serbian section of the East European Carpathian-Balkan Arc. The TMC has one of the highest concentrations of copper enrichment in the Tethyan Belt.

Figure 4.1 shows the location of the Brestovać-Metovnica exploration licence in which the Project is located. The Brestovać-Metovnica exploration licence is one of four exploration licences held by Rakita Exploration d.o.o., a Serbian joint venture between Nevsun and Freeport-McMoRan Exploration Corporation as shown on Figure 4.2.

Figure 4.1: Location of the Bor Project and Associated Licences

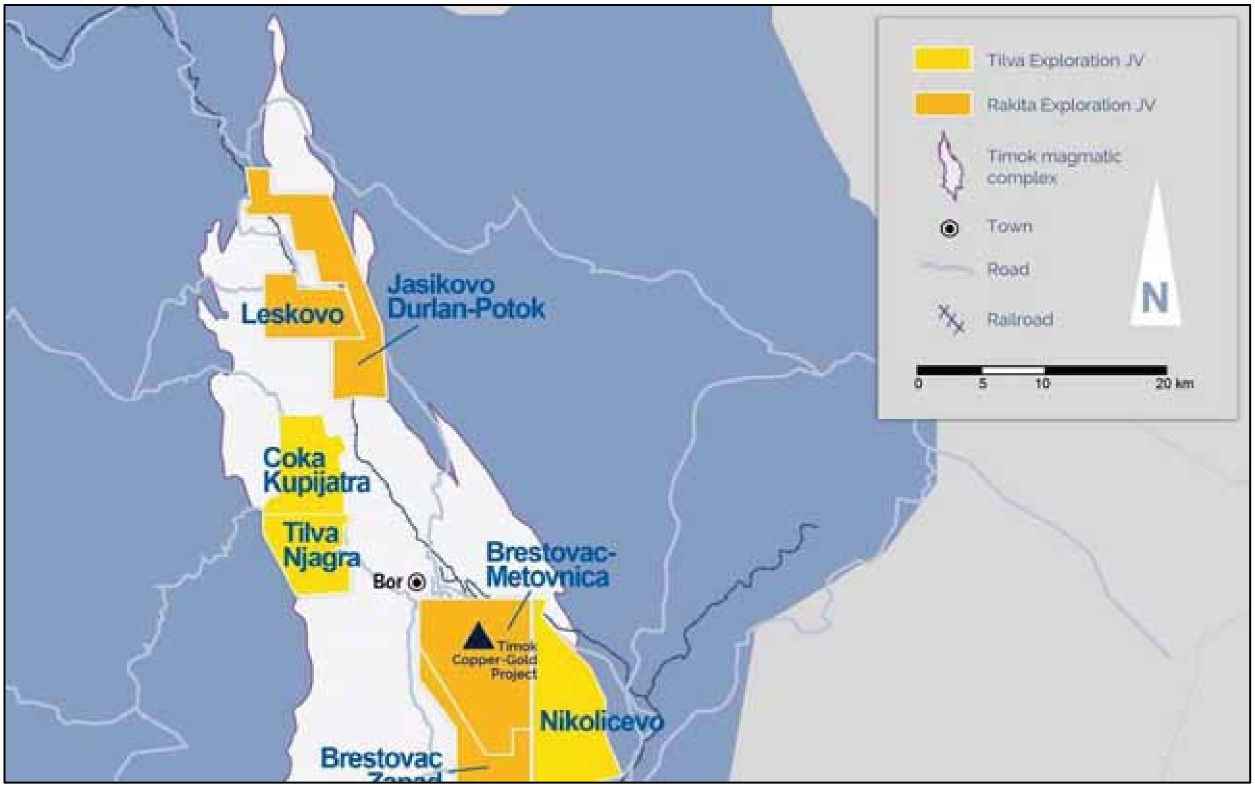


Figure 4.2: Project Exploration Licence Location Map

5. Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

The nearby municipality of Bor is connected to the capital, Belgrade, by the A1 motorway (part of the European E75 and Pan-European Corridor X route) and the international E-road E761, from Paraćin to Zaječar. Travel time from Belgrade to Bor and the Project by road is about three and a half hours.

Locally, the Project is situated 5 km south of Bor, on the south side of state road IB n°37. There are numerous small agricultural and forestry tracks within the licence area that are suitable for four-wheel drive vehicles.

A regional bus service connects Bor with Belgrade and other cities and towns. Bor is integrated into the Serbian railway system and connects to Belgrade and the main lines. The line from Bor is primarily for freight, but there are regular passenger services to Belgrade. The site is also favourably situated for export freight logistics.

5.2 Climate

The regional climate for the Project area is moderate-continental with local variations. Average annual air temperature for areas between the altitudes of 300 and 500 metres above mean sea level (amsl) is 10.0°C. The absolute maximum air temperatures are recorded in July and are in the range of 37 to 42°C for lower lying areas. The absolute minimum air temperatures are recorded in January and range from -20 to -36°C for mountainous areas. The majority of Serbia has a continental precipitation regime, with precipitation occurring fairly consistently throughout the year and the greatest rainfall typically occurring during May and June. (Republic Hydrometeorological Service of Serbia).

Regional climatic datasets have been summarized and values applicable to the Project and suitable for preliminary design have been selected. All data presented here are either from the Serbian Government, or from a study of climatic and hydrologic parameters produced by the University of Belgrade in 2016. This study presented climatic data collected at a weather station in Crni Vrh (20 km to the northwest of the Project), and two precipitation gauges at Metovnica and Brestovać Banja (6 km to the southwest of the Project at elevation 195 m amsl and 5 km to the northwest of the Project at elevation 350 m amsl, respectively).

The temperature data from the weather station at Crni Vrh, as provided in the University of Belgrade report, are for an elevation of 1,037 m amsl, and were adjusted to Project representative values at an elevation of 350 m amsl by using a typical lapse rate of 6.5°C per 1,000 m (see Table 5.1). The extreme daily temperatures recorded at the Crni Vrh station are 36.5°C on 24 July 2007 and -23.2°C on 24 January 2006, and temperature extremes in the Project area are expected to be similar.

Table 5.1: Estimated Mean Monthly Temperature for the Project Area

Parameter	J	F	M	A	M	J	J	A	S	O	N	D
Mean Monthly Temperature (°C)	0.9	1.1	5.0	10.6	15.8	19.0	21.4	21.4	16.8	11.5	6.1	2.2

Mean monthly lake evaporation values for the site were obtained from the Serbian government for the station at Kragujevic (elevation: 185 m amsl) and were calculated using the Penman-Monteith equation (see Table 5.2). These values are considered to be reasonably representative of evaporation conditions in the Project area and sum to a mean annual evaporation total of 786 mm.

Table 5.2: Estimated Mean Monthly Evaporation for the Project Area

Parameter	J	F	M	A	M	J	J	A	S	O	N	D	Annual
Mean Monthly Evaporation (mm)	15.5	23.1	47.7	72.9	101.3	117.0	130.1	114.0	77.3	46.6	24.7	15.3	785.5

Brestovać Banja is located at an elevation of 350 m amsl, which is essentially the same elevation as the Project facilities. The rainfall values for Brestovać Banja indicate a mean annual precipitation of approximately 685 mm (Table 5.3).

Table 5.3: Monthly Precipitation Data for Brestovać Banja 1960 to 2010

Parameter	J	F	M	A	M	J	J	A	S	O	N	D	Annual
Mean Monthly Precipitation (mm)	47.3	44.7	49.1	61.6	71.8	73.3	58.6	52.7	50.4	51.0	63.3	60.7	**684.5**
Maximum Monthly Precipitation (mm)	128.8	195.7	125.1	141.9	203.2	185.5	160.7	179.3	189.1	151.1	246.4	167.3	**1045.9**
Minimum Monthly Precipitation (mm)	4.1	0.9	4.0	9.7	11.2	11.6	1.7	0.8	1	0	3.7	5.8	**454.4**

The return period 24-hour precipitation events for Brestovać Banja were used to determine the anticipated return period precipitation events for the Project site. The University of Belgrade report provided values for return periods from 2 to 1,000 years, and statistics based on these values were used to estimate the probable maximum precipitation (PMP). The return period 24-hour precipitation events for the Project site are summarized in Table 5.4.

Table 5.4: Return Period 24-Hour Precipitation

Return Period (Years)	Precipitation (mm)
2	41
5	56
10	68
20	79
50	97
100	111
200	137
1,000	169
PMP	**477**

The anticipated snowpack depths for the Project were determined based on the most relevant regional snowpack data, which is from the Serbian government's weather station at the town of Nis (approximately 85 km southwest of the Project). Nis has a similar elevation (201 m amsl compared to 350 m amsl) and a similar annual precipitation to the site (580 mm compared to 685 mm). The maximum recorded snow depth at Nis over a 44-year period is 541 mm of snow water equivalent. The Nis record was used to calculate the one in 100-year snowpack depth of 660 mm for the Project site, as well as the one in two-year snowpack depth of 220 mm (similar to the expected annual average).

5.3 Local Resources

Bor is an active mining town, with a regional administrative centre possessing the facilities, services, and experienced work force required for advanced mineral exploration projects. Reliable power is available, with power lines (110 kV and 400 kV) passing through the Brestovać-Metovnica permit area (NTI, 2017). Rakita maintains an office in the town as a technical base for exploration activities on the Brestovać-Metovnica and other exploration permits in the Timok region. The project office is located close to the centre of drilling on site.

In January 2011, Outotec signed a contract with SNC Lavalin International to design, supply and install a new copper flash smelting furnace and related services for Rudarsko-topioničarski basen Bor (RTB Bor) in central Serbia.

The smelter was constructed by SNC Lavalin and commissioned in late 2015 to improve operational efficiency and reduce the environmental impact of the existing facility. The new flash smelter has a design capacity of 400 ktpa of concentrates at a design concentrate feed grade of 22% Cu. The flash smelter utilizes some of the existing infrastructure, with a newly constructed acid plant.

5.4 Physiography

The relief of the project area is marked by a gently rolling plateau with elevations ranging from 300 to 400 m amsl. The deposit itself is at an elevation of approximately 375 m amsl. The Crni Vrh hills to the west of the exploration permit rise to over 1,000 m amsl.

In the immediate project area, there is plenty of accessible flat or gently undulating land to accommodate surface processing facilities and waste storage as necessary. There are a few river valleys which are of sufficient depth to provide the necessary volume of tailings storage.

Vegetation in the area comprises mostly arable crops, some grassland and deciduous woodland.

The Timok River is the major drainage system in the project area, with multiple tributaries such as the Brestovać, Bor and Borska. It originates in the north of the Svrljig Mountains in the Carpathian-Balkan region in eastern Serbia running 203 km before discharging into the Danube River. Topographic elevation within the Timok catchment ranges from 142 m amsl at the Timok-Danube confluence, to 1,049 m amsl in the upper reaches of the catchment.

The Crni Vrh plateau is incised by the southeast-flowing drainage of the Brestovać River and its tributaries, and by the Bor River in the northeast of the Brestovać-Metovnica exploration permit area. The Brestovać River descends from about 280 m in the northwest corner of the property perimeter to about 160 m, where it flows across the south boundary of the exploration permit. The highest elevation is recorded as 464 m on the eastern margin of the property.

Anthropogenic features related to the mining activity, including waste dumps, dominate the physiography to the north of the exploration permit. The Bor open pit, approximately two kilometres north of the northern perimeter of the Brestovać-Metovnica exploration permit, is approximately 300 m deep and 1,000 m long.

5.5 Infrastructure

The municipality of Bor has well developed infrastructure due to its strong industrial activity. Consequently, there are many possible logistic options and solutions for the transport of materials from the proposed Timok project. The location of the Project is well-situated relative to existing rail/road/waterways networks. The logistics network is currently used by the RTB Bor mine complex and other industrial entities in the region (Elixir Prahovo, Aurubis Pirdop, etc.).

The Project is directly connected via state road 37 class 1B to the RTB Bor smelter (~9 km) and to the railroad cargo loading station at Bor Teretna (~7 km). The rail loading station would allow for transport via the state rail network to a smelter in Pirdop, Bulgaria and ocean ports in the region. There are also road and rail connections to the Danube river port of

Prahovo (~88 km), allowing for low cost material transport to the ocean port of Constanta, Romania.

Serbia's power infrastructure is well-developed and Bor is situated in a favorable location with access to a reliable power supply. Bor is one of the transfer points for the long-range 400-kV power line that transmits electrical power from the Đjerdap 1 hydroelectric plant, located on the Danube River, approximately 90 km northeast of Bor. The Bor 2 transmission substation is located five kilometres northwest of the project, and a 110-kV transmission line passes within one and a half kilometres of the Project.

6. History

6.1 Introduction

The history of exploration and mining in the Timok Bor district is described by Jankovic et al. (2002) as provided below.

The earliest known historic exploitation in the Region focused mainly on copper mining (and smelting) as early as 5500 BC (Vinca culture age). The next known historic exploitation of surface outcrops in the district was for gold in the massive sulphide mineralization at the Bor Coka Dulkan and Tilva Ros and Majdanpek gold (and iron) occurrences. This likely commenced during the Bronze Age and was continued by the Romans, who were active throughout the region and who also extracted alluvial gold from the Pek and Timok Rivers.

In the 19th century, Serbian investors (including Georg Weifert) financed prospecting and exploration in the Bor district from 1897 to 1902, which led to the discovery by Franjo Sistek of the copper and gold-rich Coka Dulkan and Tilva Ros deposits in 1902. Mine development began during 1903 and 1905 and mining commenced in 1907. The Serbian investors sold their interests to a French group (Society of the Bor Mines) who then controlled the mines until 1941. The mines and smelter were rehabilitated after the Second World War and were operated from then to the 1990s by the Yugoslav State, and then later by the state-owned Rudarsko-topioničarski basen Bor (RTB Bor).

During the Yugoslav State period, exploration focused on outcropping alteration and mineralization and drilling to maximum depth of approximately 700 m. During this time, the following porphyry deposits were discovered - Majdanpek, Bor River, Valja Strz, Veliki Krivelj, Cerovo/Cementation, Dimitri Potok and the high-sulphidation (HS) epithermal deposits - Lipa, Choka Marin, Choka Kuruga, Kraku Bugaresku. Most of these deposits were subsequently explored further and mined (Jankovic et al. 2002).

The earliest known exploitation in the Timok project area of Brestovać-Metovnica, was trial mining of copper and zinc mineralization south of Brestovać village, which was undertaken from an adit and blind shaft south of Brestovać by a French group in the 1930's; however, there are only incomplete records and no meaningful production was recorded.

During the Yugoslav State period, exploration along the Bor trend continued and there are records of approximately 41 RTB Bor drillholes at various locations in and near the project area from 1975 to 1988. Most drilling was relatively shallow, with depths less than 500 m, and took place in small clusters mainly targeting gravity and other geophysical anomalies. The records are not complete, and no drill core was retained. The Timok deposit mineralization was not intersected in any drillholes from this time. No other significant mineralization was found apart from a hole south of Brestovać village (near the old workings), which showed elevated gold grades in altered andesite, for which the sampling and analytical records are also incomplete. This hole was followed up by a Eurasian Minerals Inc. exploration program in 2006 (see below).

During the period 1990 to 2002 and the political uncertainty and conflict in the former Yugoslavia and Serbia, no mineral exploration of any significance was undertaken. The Serbian government issued exploration licences and concessions in 2002 and mineral exploration activities in the Timok area began in 2004 with the arrival of companies including Phelps Dodge, Eurasian, Euromax and Dundee.

6.2 Exploration 2004 – 2016

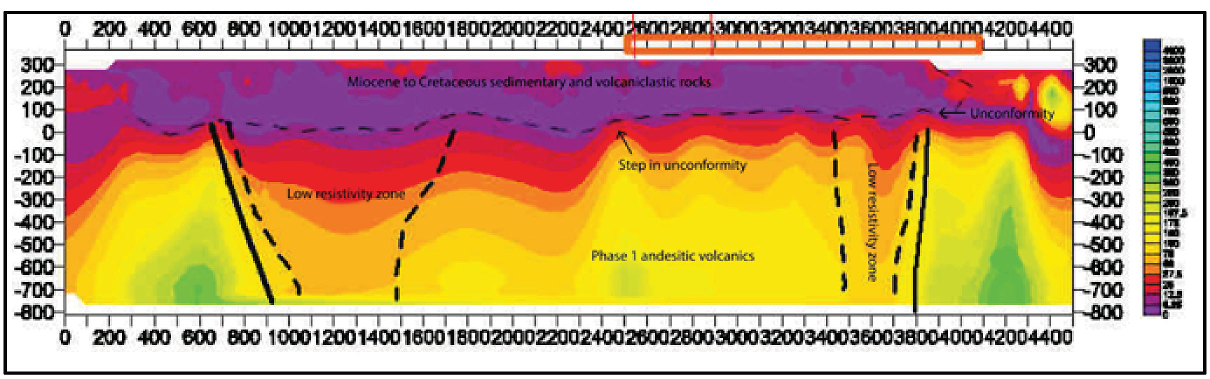
In 2004, the first Brestovać exploration licence was awarded to Southeast Europe Exploration d.o.o., a 100% owned subsidiary of Eurasian Minerals. Southeast Europe followed up on reports of the gold mineralization encountered in an historical drillhole from the 1970s, and in 2006, they confirmed this with shallow drilling. Drillhole BN-01 (terminated at 296.80 m) intersected gold and copper mineralization in the upper 60 m (including 22.4 m at 4.51 g/t gold) and zinc mineralization from 286 to 294 m. Follow-up ground magnetometry, induced polarization, and resistivity geophysical surveys defined a target area with high chargeability and conductivity characteristics in the “Corridor Zone” which extended through drill site BN-01.

In 2007, Southeast Europe became a 100% owned subsidiary of Reservoir Capital Corp and during 2007 to 2008, Reservoir undertook further geophysics and soil geochemical surveys in the Brestovać area and outlined a high-grade epithermal copper-gold system in the Corridor Zone along a strike length of 550 m, defined by 14 drillholes (total 1,937 m). Drillhole BN-19, at the eastern end of the Corridor Zone, close to the interpreted extension of Bor Fault, intercepted a massive sulphide zone with 24.8 m at 0.33% copper and 0.16 g/t gold, which supported the concept that the epithermal gold mineralization of the Corridor Zone grades into a copper-rich zone.

Phelps Dodge Exploration Corp obtained exploration licences at Timok in 2004, including the Metovnica exploration licence which was adjacent to the Brestovać exploration licence. In 2007, Phelps Dodge was acquired by Freeport McMoran and exploration continued. During 2006 to 2009, Phelps Dodge/Freeport undertook geological and large-scale induced polarization geophysical surveys on the Metovnica permit. Freeport also completed 14 drillholes (including three holes drilled during a joint venture with Euromax Resources) in the west and south of the Metovnica exploration licence. None of these holes intersected significant mineralization.

In 2010, Reservoir Capital and Freeport formed a joint venture (the Rakita Earn-in Agreement) with combining the Brestovać-Metovnica, Jasikovo, and Leskovo licences. Joint venture exploration continued with geophysical surveys and drilling over all licences. In 2011, drilling in the Brestovać-Metovnica licence discovered further intermediate sulphidation gold mineralization in the Ogasu Kucajna Prospect north of the Corridor Zone and then moved to exploration beneath the Miocene Basin area in the Slatina-Miocene project. In 2012, the third hole on this program, FMTC1210, intersected Upper Zone style HS mineralization with 266 m grading an average of 1.23% copper equivalent from 598 to 864 m and also porphyry style Lower Zone mineralization at depth. This became the Project Discovery hole.

From 2010 to 2016, joint venture exploration continued with exploration drilling on both the Upper Zone (59,333 m) and Lower Zone (42,380 m) deposits. Field work also included geological mapping, geochemical surveys and large-scale controlled source audio magneto-telluric (CSAMT) (Figure 6.1) and additional induced polarization surveys. These surveys covered certain target areas where Miocene sediments overlie the Upper Cretaceous volcanic rocks. Orientation surveys over known deposits in the Bor district were also carried out.



Source: Nikova, L., 2014

Figure 6.1: Line 60 CSAMT Geophysical Survey, Looking North-Northwest

The CSAMT survey highlighted the position of the base of the Miocene and the location of high/low resistivity zones and potential structural zones overlying the Upper Zone. The CSAMT data was used for exploration targeting and contributed significantly to the initial drill intersection of the Upper Zone, as documented by Reservoir in a press release on 16 July 2012. Drillhole to surface induced polarization resistivity measurements were conducted around drillhole FTMC1210, but the results were not sufficiently encouraging to justify further downhole induced polarization surveys.

After acquiring 55% equity interest under the Rakita agreement, Freeport gave notice to Reservoir in July 2012 that it had elected to sole fund expenditures on or for the benefit of the Project, until the completion and delivery of a feasibility study to bankable standards.

6.3 Historical Estimates

SRK and Reservoir Minerals produced a Mineral Resource estimate for Čukaru Peki Upper Zone with an effective date of 27 November 2013, reporting an inferred mineral resource, above a 1% copper equivalent cut-off grade, of 65.3 Mt grading 2.6% Cu, 1.5 g/t Au and 0.1% As (SRK (UK), 2014).

In 2016, SRK and Reservoir Minerals produced a PEA report, with an updated Mineral Resource estimate for Čukaru Peki, with effective date of 31 March 2016. The mineral resource included an indicated mineral resource of some 1.7 Mt above a cut-off grade of 0.75% copper, with average grades of 13.5% copper, 10.4 g/t gold and 0.23% arsenic and an inferred mineral resource of some 35.0 Mt above a cut-off grade of 0.75% copper, with average grades of 2.9% copper, 1.7 g/t gold and 0.17% arsenic (SRK (UK), 2016).

Both historical mineral resources are relevant and reliable. They use mineral resource categories as defined in CIM Definition Standards (and by extension NI 43-101). The historical mineral resources demonstrate that the current Mineral Resource estimates are robust. The historical estimates are superseded by the estimates presented in this PFS report.

6.4 Historical Production

Trial mining of copper and zinc mineralization was undertaken from an adit and blind shaft south of Brestovać in the past century; however, no meaningful production was recorded. There has been no significant production registered from the Brestovać-Metovnica exploration licence.

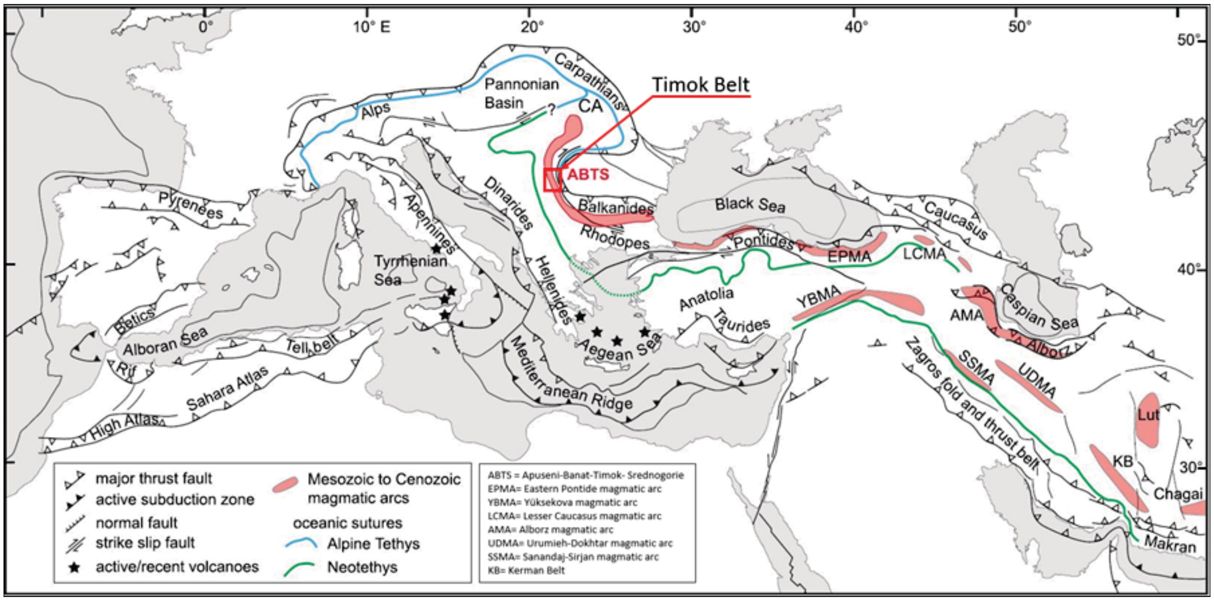
7. Geological Setting and Mineralization

7.1 Regional Geology

The Čukaru Peki HS copper-gold deposit is located within the central zone (or Bor District) of the TMC, which represents one of the most highly endowed copper and gold districts in the world. The TMC is located within the central segment of the Late Cretaceous Apuseni-Banat-Timok-Srednogorie (ABTS) magmatic belt in the Carpatho-Balkan region of southern-eastern Europe. The ABTS belt forms part of the western segment of the Tethyan Magmatic and Metallogenic Belt (TMMB) which lies along the southern Eurasian continental margin (Figure 7.1) and extends over 1,000 km from Hungary, through the Apuseni Mountains of Romania, to Serbia and Bulgaria to the Black Sea.

The TMMB, which is one of the longest magmatic arc systems in the world (Jankovic, 1997; Perelló et al., 2008; Richards et al., 2012; Richards, 2015), formed during Mesozoic to Tertiary evolution (and closure) of the Neotethys ocean and involved multiple subduction events and collision (orogenic) tectonism within the Eurasian segment. The TMC occurs within the Alpine-Balkan-Carpathian-Dinaride geo-tectonic province that also forms the westernmost part of the Tethyan Alpine-Himalayan orogenic system. TMMB magmatism and orogenic events occurred during closure of the Neotethys Ocean, which resulted in multiple subduction events and subsequent collision between the Indian, Arabian, and African plates with the Eurasian craton. Tectonic-magmatic events were initiated in the Middle Jurassic and continued through the Cretaceous to the present time.

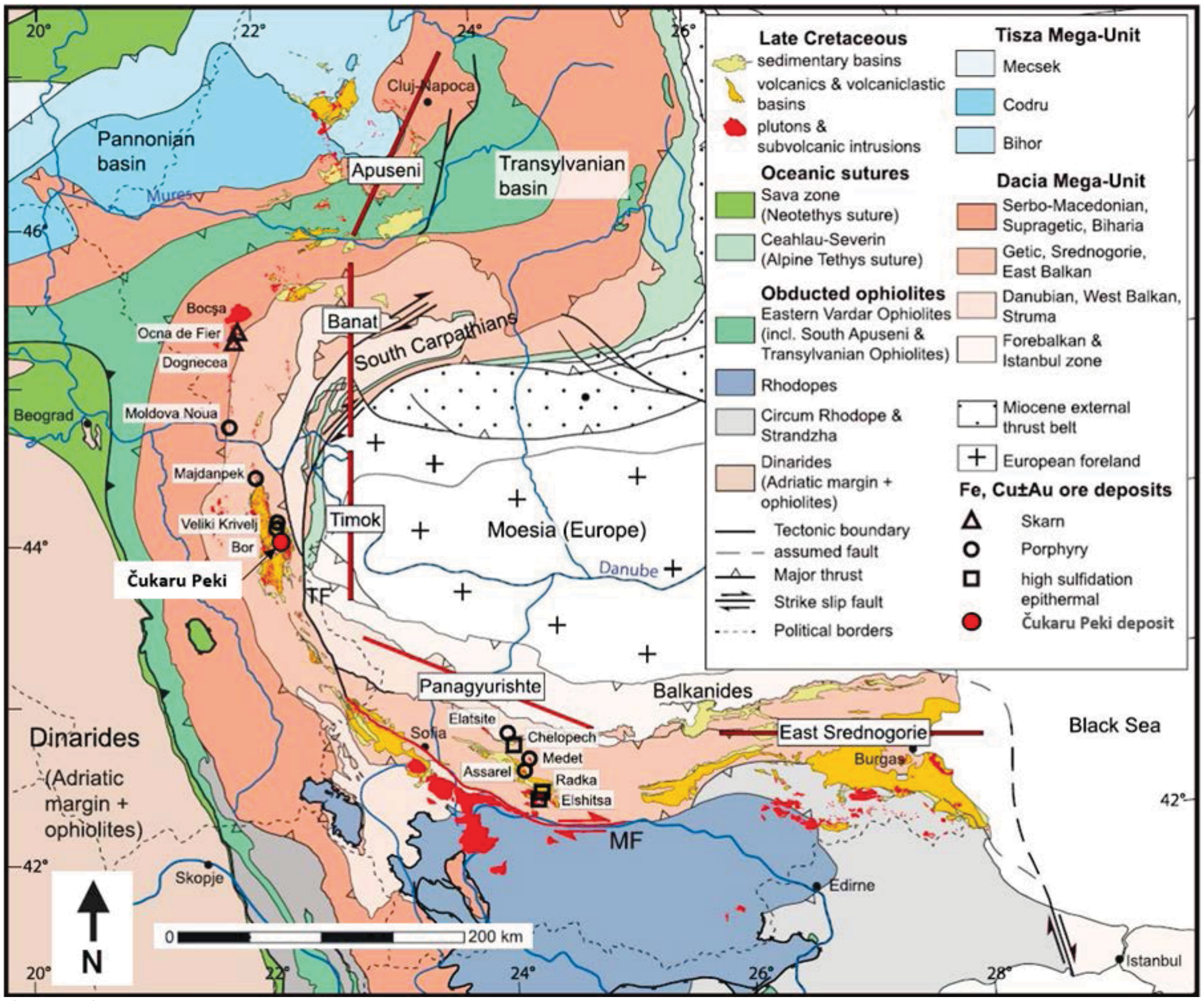
In the Carpatho-Balkan sector, magmatism terminated at the time of the collision and was subsequently overprinted by major collision-related deformation (Dewey et al., 1973; Schmid et al., 2008). The collisional overprinting makes arc magmatic reconstruction and interpretation of the associated geo-tectonic setting very difficult (Sosson et al., 2010; Bouihol et al., 2013).



Source: Modified from Morelli and Barrier 2004, in Gallhofer et al., 2015

Figure 7.1: Tectonic Map of the Western Eurasia Continental Margin

In southeast Europe (Carpathian-Balkans) the Late Cretaceous ABTS belt has been separated into five different segments showing distinctive magmatic and mineralization trends along the arc (Figure 7.2), defined by geographic regions and also by major crustal fault zones. The timing and the evolution of the magmatism and its associated deposits are relatively well studied in the central and eastern segments at Timok in central-east Serbia and Panagyurishte and Srednogorie in Bulgaria (Gallhofer et al., 2015).



Source: Schmid et al., 2008, 2001

Figure 7.2: Geological Map of the Carpathian-Balkan Orogen Showing the Five Segments of ABTS Belt and Timok Deposit

The ABTS magmatic arcs and associated metallogenic belts occur within the Balkan, Northern Rhodopes, South Carpathian and the Apuseni mountain ranges, which generally rise to an altitude less than 2,000 m amsl, and are partially covered by Miocene to recent sediments of the Pannonian Basin. The Late Cretaceous subduction-related intrusive rocks were intruded into pre-alpine basement of metamorphosed Palaeozoic and Proterozoic meta-sediments and granitoids, and Mesozoic clastics and limestone that had already undergone multiple Mesozoic orogenic episodes including accretion and major nappe formation (Figure 7.2).

The ABTS arc was subsequently intensively deformed during the Alpine Orogeny (Late Cretaceous to Neogene), bending around the Moesian platform (micro craton) to create the present “L” shape form of the arc. This deformation was not pervasive but was confined to brittle fault structures and ductile shear zones at the current level of exposures. The TMC underwent both compressional and extensional tectonics which partly predated and partly overprinted the Late Cretaceous magmatic arc, resulting in the distinctive segments of ABTS belt (Gallhofer et al., 2015).

The Timok Region occurs within the Getic sector of the Dacia terrane at the western margin of the Moesian craton (Figure 7.2). The Dacia terrane is a major nappe unit, which comprises slices of Proterozoic to Mesozoic aged continental, continental margin and ocean crust that were accreted during Cretaceous subduction and collision. Transition from compressional to extensional tectonics occurred during slab break-off which generated fertile but relatively short-lived Late Cretaceous regional multiphase magmatic and mineralizing events (over ~30 million years ago (Ma)) from ~90 Ma to 60 Ma.

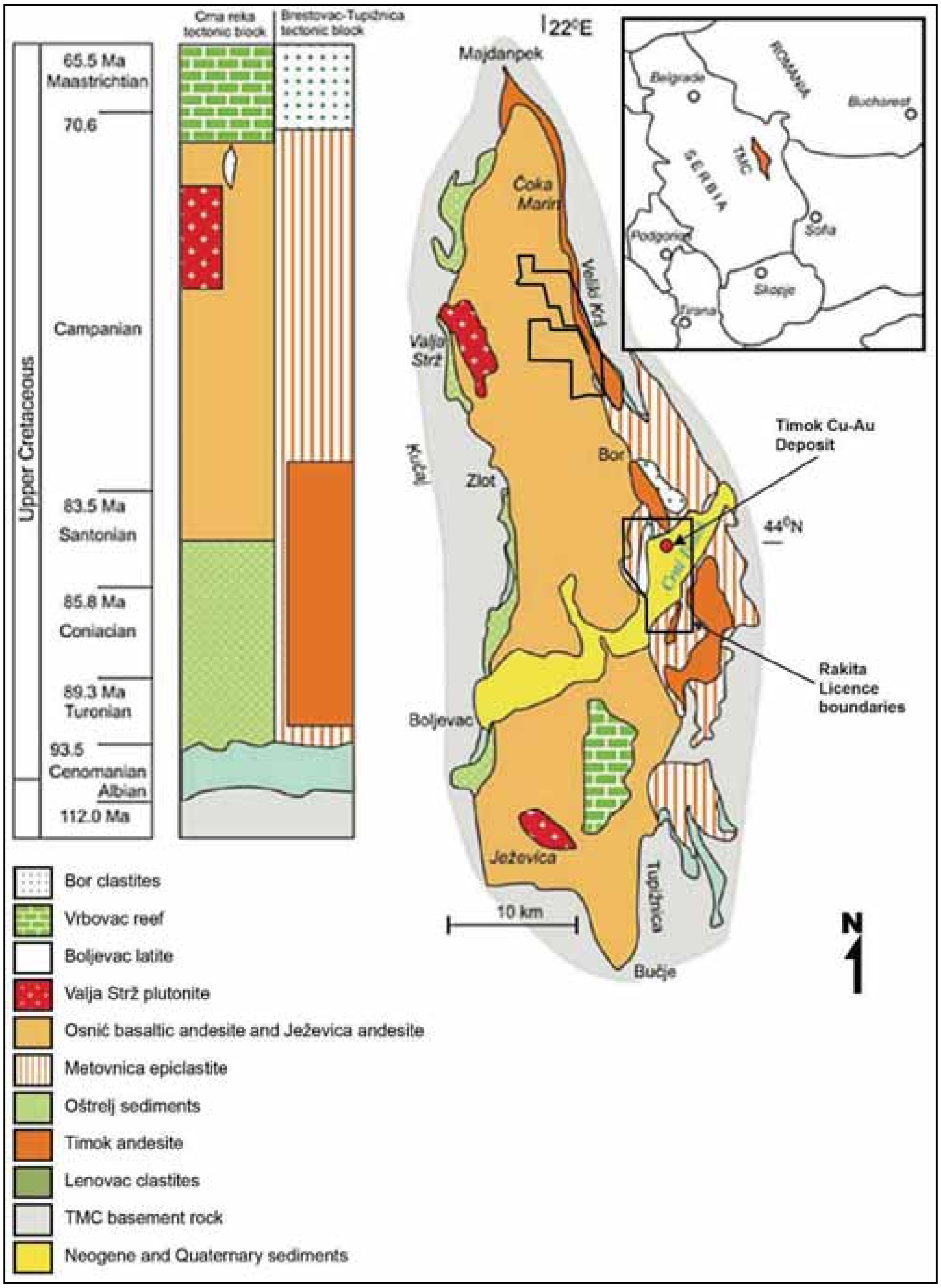
The Timok segment of the ABTS belt, has one of the highest concentrations of copper and gold metal in the Tethyan Belt, and its metallogenic endowment is a significant contributor to that of the entire Tethyan Eurasian Metallogenic Province. The metal endowment is contained mainly in porphyry copper-gold and associated massive sulphide HS epithermal copper-gold systems, as well as occurrences of low to intermediate sulphidation epithermal and skarn mineralization. In the TMC, the world-class Bor HS epithermal system and Majdanpek porphyry system have contributed to an estimated historical production of approximately 11.5 Mt of copper and 1.2 Moz of gold (Armstrong et al. 2005; Jelenkovic et al., 2016).

7.2 Property Geology

The Timok deposit is located within the central part of TMC, in the Bor District of northeastern Serbia (Figure 7.3). The project area is approximately five kilometres south of the mining municipality of Bor. The TMC is a north-south elongated lens-shaped graben feature with dimensions of approximately 85 km long and up to 25 km wide. The TMC comprises a series of andesitic to dacite-andesitic subvolcanic, volcanic and volcano-sedimentary sequences and plutonic intrusions (mainly monzonite to diorite and granodiorite compositions). The TMC is generally erosionally well-preserved when compared with both the Banat and Panagyurishte segment belts to the north in Romania and to the east in Bulgaria respectively. The largest porphyry and porphyry-epithermal deposits in the Timok segment are represented by the Majdanpek and Bor copper and gold deposits (Figure 7.2) which are hosted within in Upper Cretaceous andesitic volcanic units.

The exposed geology in the project area is dominated by Upper Cretaceous andesitic volcano-sedimentary sequences partially covered by a north-south to northwest elongated belt of poorly consolidated Tertiary clastic sedimentary rocks. Basement Mesozoic stratigraphy exposed around the TMC consists of Jurassic to Lower Cretaceous limestones and clastic sedimentary rocks (Figure 7.4).

Within the Jurassic succession, it is possible to recognize Lower and Middle Jurassic clastic units and sandy limestones up to 320 m thick. These units are overlain by Upper Jurassic reef limestone (Djordjević & Banješević, 1997) which are unconformably overlain by Lower cretaceous (Albian) sandstone (Djordjević & Banješević, 1997; Toljić, 2015). A transgressive Aptian-Albian carbonate sequence (up to 150 m thick) is conformably overlain by lower Later Cretaceous (Cenomanian) sandstone.



Source: Jelenkovic et al., 2016

Figure 7.3: Simplified Geological Map of the Timok Magmatic Complex, Showing the Position of the Timok Deposit and the Rakita Licenses

The Cenomanian sedimentary rocks are conformably overlain by volcanic, volcaniclastics and sedimentary units of the Upper Cretaceous TMC. The TMC complex is dominated by Late Cretaceous (Turonian to Campanian) andesitic lavas, lava domes and shallow intrusions, volcaniclastic and epiclastic units and basaltic andesites, volcaniclastics and clastic sedimentary rocks that formed in an extensional rift basin.

The TMC andesite volcanic rocks are typically calc-alkaline in composition. Kolb et al. (2013) describe a geochemical signature similar to adakites, which are commonly associated with porphyry and epithermal copper and copper-gold deposits elsewhere in the world. The western and eastern borders of the TMC complex are structurally controlled by major faults (Figure 7.4). In the centre and southeast of the TMC, Miocene clastic sedimentary rocks unconformably overlie the Late Cretaceous units (Figure 7.5).

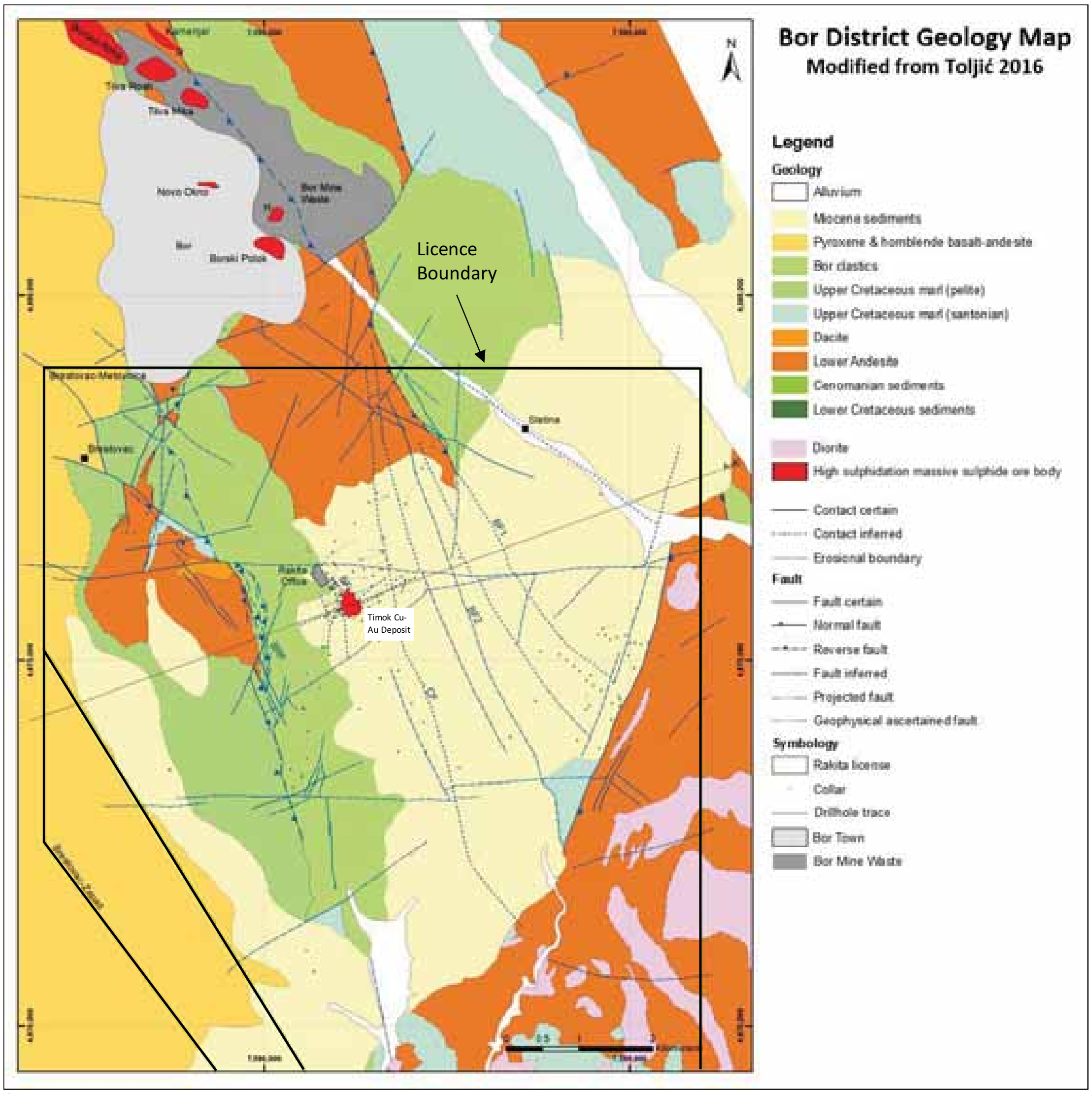
Three different phases of volcanic andesitic and minor dacitic sequences are recognized. The intrusive units are composite complexes from gabbro, diorite, monzonite and granodiorite. Volcanic activity was initially predominantly sub-terrestrial and subaerial which changed to submarine in later phases (Banješević, 2010).

Banješević (2010) describes in detail the volcanic stratigraphy, lithologies and age-dating in the central TMC and notes that the TMC is divided into an eastern 'Phase 1' Bor-Lenovac volcanic facies (or 'Timok andesite') and a western 'Phase 2' Crna Reka volcanic facies (or 'Osnić basaltic and Jezevica andesite'), possibly related to separate tectonic blocks (Figure 7.3). Phase 1 units comprise andesitic hornblende-andesite volcanics (dated ~ 89.0 to 84.3 Ma), subvolcanic intrusions with intercalated volcaniclastics, epiclastics, marls and fine-grained clastics that are restricted to the eastern Brestovać-Tupižnica tectonic block. Phase 2 (dated ~82.3 to 81.8 Ma) consists of basaltic (pyroxene-bearing) andesite volcanics and volcaniclastics which are restricted to the western Crna Reka tectonic block.

The boundary between the Phase 1 - eastern and Phase 2 - western volcanic sequences has not been observed to date. The contact between facies may be transitional and/or tectonic and may occur along a suture through the Brestovać river valley, which trends north-northwest to south-southeast through the western part the Brestovać-Metovnica exploration permit.

A third phase of younger more felsic intrusive rocks, though rare, does occur in the permit area and is generally the youngest phase in the TMC. In the eastern block, a quartz-bearing dacitic intrusive is mapped south of Brestovać village, where it is associated with an east-west fault and trachy-andesite dykes (81.5 Ma) outcropping at the Brestovać cross roads. In the western block, the Valja Strž monzonite to granodiorite suite (78.6 Ma) was intruded during the final phases of magmatism.

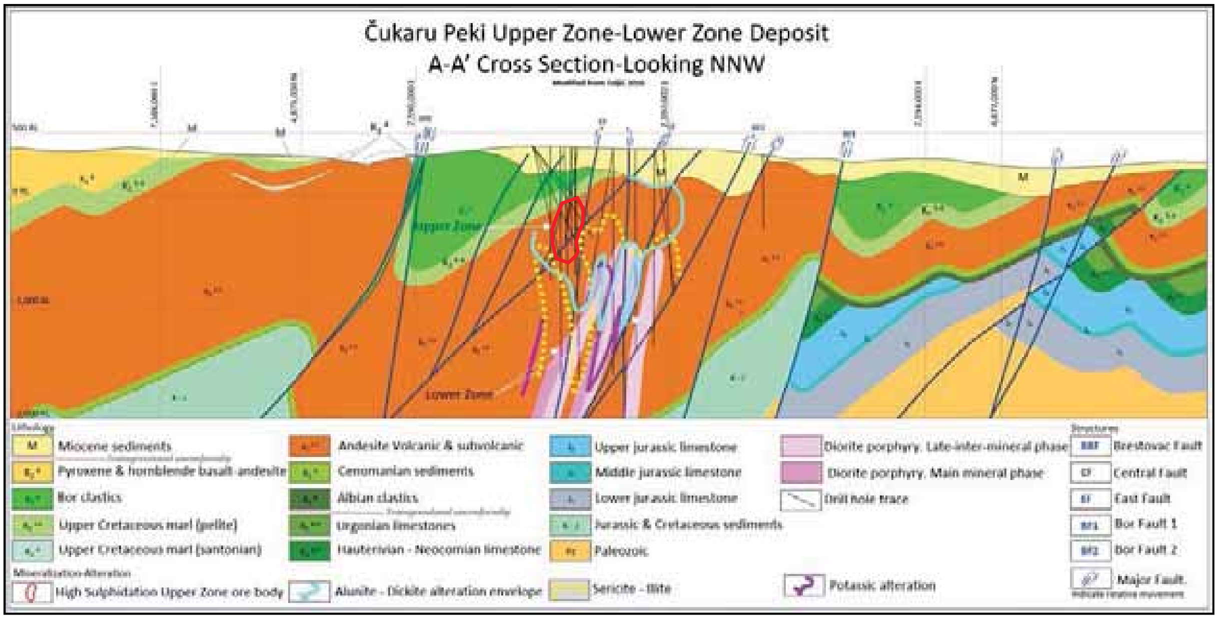
Following this final period of volcanism and magmatism, there was a period of sedimentation and uplift (including deposition of reef carbonates in the central TMC and formation of coarse clastics (Bor Conglomerate)) in the eastern TMC. After deformation (which included compression and nappe formation) and uplift (Alpine Orogeny) in the early Cenozoic, Miocene lacustrine clastic sediments (siltstones, sandstones and conglomerates up to 400 m thick in the TMC) were unconformably deposited on the underlying Late Cretaceous volcanic and volcaniclastic rocks.



Source: Rakita, 2017

Figure 7.4: Geology Map of Bor District Showing the Location of the Known HS Epithermal and Porphyry Deposits

In summary, the geology of the Brestovać-Metovnica licence area is dominated by the Phase 1 Upper Cretaceous Timok andesite volcanic unit, which comprises volcanic flows, locally subvolcanic intrusions, volcaniclastic rocks, and clastic sedimentary rocks typical of the eastern tectonic block. A small area of Phase 2 pyroxene-bearing basaltic andesite typical of the western Crna Reka Tectonic Block occur in the northwest part of the permit. Miocene clastic sedimentary rocks (locally with fault-bound basin margins) unconformably overly the Late Cretaceous in the centre of the permit area (Figure 7.4).



Source: Rakita, 2017 (modified from Toljić, 2016)

Figure 7.5: Cross-Section through the Čukaru Peki and Underlying Timok Lower Zone Deposit

7.3 Mineralized Zones

The Upper Zone (Čukaru Peki) and the Lower Zone are both part of the Timok copper-gold project.

7.3.1 Timok Upper Zone

Timok Upper Zone is a high-grade HS epithermal deposit typically associated with an advanced argillic alteration system with a discrete footprint.

The top-third of the HS epithermal mineralization of the UZ is characterized by a massive sulphide lens located on the top of a volcanic to volcaniclastic sequence (Figure 7.6). This lens has a variable but overall similar dip to the overlying stratigraphy. With increasing depth from the top of the UZ, the proportion of massive sulphide mineralization intruding or replacing the host rock reduces, as does the sulphide content and presence of fragmental volcanic units. With depth, the mineralization becomes more characterized by veins and stockworks hosted by more coherent andesite (Figure 7.7).

The massive sulphide comprises mainly pyrite and covellite and hosts the highest grades of copper and gold; multiple pyrite replacement phases are observed, which in some places comprise up to 95 wt% of the deposit. Locally, different pyrite phases can be recognized by cross-cutting relationships; however, in general they are difficult to distinguish. Covellite is interpreted to be later than pyrite and is observed transgressively cutting and brecciating massive pyrite; however, pyrite can also locally be observed cross-cutting covellite stringers or massive aggregates of covellite flakes intergrowing with alunite.

Pyrite with enargite is also present. Enargite is commonly observed rimmed and sometimes replaced by covellite and is therefore interpreted to represent an earlier phase of mineralization.

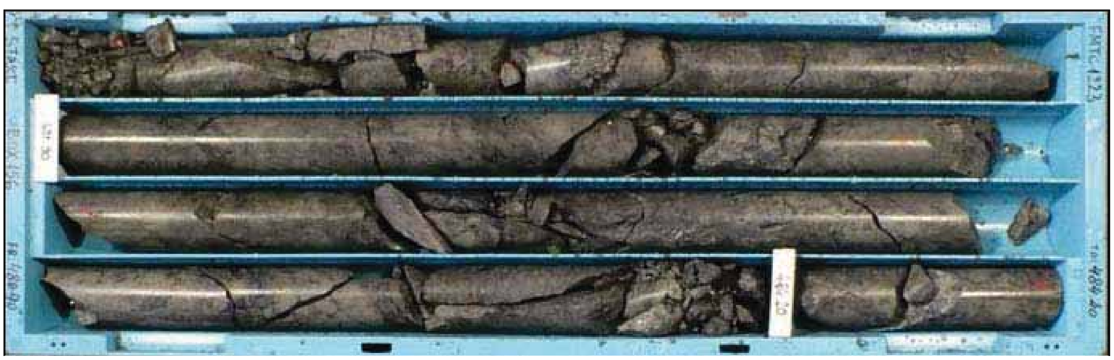


Figure 7.6: High Grade Covellite Breccia in Massive Pyrite - Hole FTMC1223 480.9 to 484.4 m

Figure 7.7: Massive Sulphide, Veins and Stockwork in More Coherent Andesitic Rock - Hole TC170150 at 549.5 to 5536.208 m

Mineralized hydrothermal breccias have also been locally observed in the UZ and at least two events have been recognized: an early syn-mineralization phase and an inter-mineral phase, hosting fragments of massive sulphide (Figure 7.8).

Figure 7.8: Probably Early Hydrothermal Breccia Matrix Filled by Covellite-Pyrite Mineralization in Advanced Argillic Altered Andesite. Hole FTMC1223 at 698 m

Gold mineralization is present in a number of forms, including tellurides such as calaverite (Au), sylvanite (Au-Ag) and kostovite (Au-Cu), altaite (Pb) and is mostly hosted in pyrite but also locally found encapsulated in bornite (Cornejo, 2017). Native gold is not common; however, where observed it is very fine, approximately 2 to 6 µm in diameter.

Low temperature galena and sphalerite as disseminations and in veins are noted in the peripheral zones of the UZ mineralization, mostly related to kaolinite-pyrite alteration fronts.

7.3.2 *Timok Lower Zone*

The Timok Lower Zone consists of lower grade porphyry copper-gold style mineralization comprising quartz-sulphide veins and disseminated sulphides and larger alteration footprint. The Lower Zone is situated approximately 200 m beneath the Upper Zone and extends from this point to the north of the Upper Zone, and most likely represents the source of fluids for the Upper Zone epithermal mineralization.

The Lower Zones constitutes a telescoped porphyry system related to multi-stage diorite intrusion, whereby the Lower Zone potassic zone with well-developed A-type vein stockwork is overprinted by quartz-sericite alteration (with classic D-type veinlets) and Upper Zone HS alteration and mineralization assemblages including alunite-dickite, covellite-pyrite and vuggy silica replacement.

Upper Zone HS epithermal mineralization at the Project occurs at depths from 450 to 850 m below surface. Lower Zone porphyry style mineralization is found from 700 to 2,200 m and is still open to the northwest at depth. To date, the deepest drill hole intercepting Lower Zone mineralization is hole TC170131A which terminated in porphyry style mineralization at 2,268.1 m.

7.4 Upper Zone Alteration

The footprint of Čukaru Peki mineralization is directly associated with the advanced argillic alteration and has a narrow alteration front or halo of kaolinite-pyrite, which typically varies from 1–3 to 10 m wide and is noted to have a distinctive gold, lead and zinc geochemical signature.

The advanced argillic alteration assemblages observed in the UZ are typical of HS epithermal systems. Spectrometer analysis and core logging confirm the presence of quartz-alunite, quartz-alunite-dickite and locally pyrophyllite and diaspore typically associated with massive and vein/stockwork sulphide mineralization. This extends downwards forming a transition zone overprinting the LZ porphyry alteration and mineralization.

The kaolinite-pyrite (to kaolinite-smectite) envelope represents the alteration front of the mineralized body and is characterized by high gold values without copper, particularly at the margins of the high-grade copper and gold zones. Average gold grades of 1 g/t in small drillhole intervals are common, and locally higher gold values up to 10 g/t can be found. Discontinuous but anomalous values of lead and zinc (occurring as fine dissemination or veinlets of sphalerite and galena) are observed with up to 1,000 ppm zinc and lead. Crystals of native sulphur (occurring in vein-fractures and disseminations) are also commonly observed within this zone.

A more distal argillic alteration halo (outside of the kaolinite-pyrite envelope), comprising mainly smectite and montmorillonite in fractures, is noted to extend laterally for approximately 100 m around the margins of the deposit. A weak, poorly developed argillic footprint is also observed above the deposit although this is terminated at the overlying unconformity. The main argillic alteration halo occurs at the margins and beneath the UZ mineralization and is observed as a more intense clay altered zone, which may also relate to faulting.

A summary of the main alteration minerals used to model each of the UZ alteration zones is provided in Table 7.1 and a schematic illustration of the distribution of the alteration in context of the main mineralized zones is illustrated in Figure 7.9.

Table 7.1: Indicator Minerals for Alteration Assemblages

Assemblage	Unique Indicator Minerals
Advanced Argillic	Alunite, dickite or pyrophyllite. Quartz is present in this and other assemblages.
Kaolinite	Kaolinite without alunite, dickite or pyrophyllite. Commonly occurs with pyrite.
Argillic	Montmorillonite, smectite or related complex clays.
Sericite-illite (Phyllic)	Sericite, illite or related complex clays.

At depth, circa -500 m amsl, the UZ alteration footprint is interpreted to be more structurally controlled comprising a mixture of advanced argillic alteration, kaolinite, sericite-illite to sericitic assemblages and covellite-pyrite mineralization overprinting early chalcopyrite-sericite and remnant potassic alteration. This is considered to represent the transition into the LZ porphyry style mineralization and alteration.

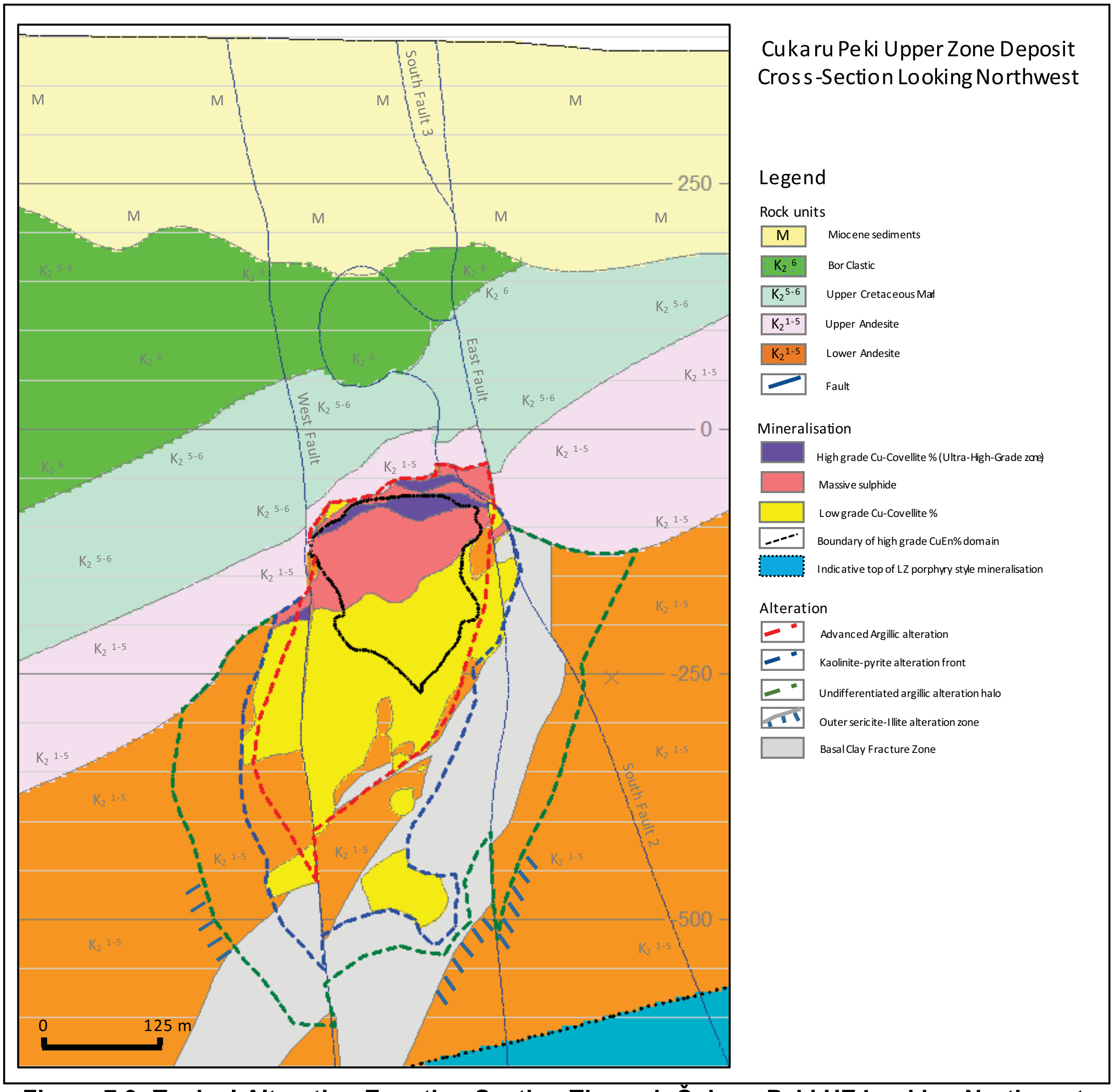


Figure 7.9: Typical Alteration Zonation Section Through Čukaru Peki UZ Looking Northwest, Showing the Geology, Alteration and Mineralized Units

7.5 Lithology

The Timok deposit is located within the Bor metallogenic zone of the TMC. The deposit is hosted within Upper Cretaceous (Phase 1) andesitic volcanic rocks. The volcanic rocks are overlain conformably by an Upper Cretaceous clastic sequence (up to 250 m thick) of Senonian marls and calcareous siltstone (Oštrelj Formation) and then overlain conformably by Maastrichtian conglomerate and sandstone of the Bor Clastic Formation (Banješević, 2010, 2015). A poorly consolidated sequence of Miocene clastic sedimentary rocks (sand and gravel to sandstone, conglomerate and mudstone), up to 400 to 500 m in thickness, unconformably overlies the Upper Cretaceous clastic sequence.

The andesitic units that host Čukaru Peki and Timok LZ can be further subdivided into the following facies:

- Lava flows (coherent and autoclastic).
- Shallow intrusions (lava domes, dykes and sills).
- Various volcaniclastic rocks.

The andesites comprise medium to fine porphyritic and aphanitic hornblende-plagioclase and hornblende-biotite-plagioclase volcanic as well as various andesitic volcaniclastic and epiclastic rocks, dacitic and dacitic-andesitic sub-volcanic intrusions as stocks, sills and dikes. In the deposit, the coherent volcanic or subvolcanic units are more abundant compared with the fragmental volcanic units.

U/Pb zircon and 40Ar/39Ar age dating on volcanic units (Jelenkovic et al., 2016) in the TMC suggest Late Cretaceous (Upper Turonian to Upper Santonian) ages spanning 89.0 ± 0.6 to 84.26 ± 0.67 Ma.

Recent laser ablation zircon dating by Rakita on samples of different volcanic and sub-volcanic rocks confirm the Late Cretaceous age and show that host volcanic unit ages range from 86.4 ± 1.5 Ma to 83.2 ± 1.4 Ma. Volcanic and volcaniclastic or fragmental andesitic rocks have reported a significant population of zircon ranging 88 to 90+ Ma, which could represent inherited zircons from the older basal part of the Andesite sequence (Valencia, 2017).

7.6 Structural Geology

7.6.1 Regional Scale

The geo-tectonic setting of the Timok region is the result of complex and multiphase subduction, collision/orogeny and large-scale oroclinal bending during post-collision tectonism throughout the Tertiary, which also involved major strike-slip faulting with overall dextral displacements in excess of 100 km. The Timok Region occurs within the Getic sector of the Dacia terrane at the western margin of the Moesian craton (Figure 7.2).

Principal structures in the Getic Nappe are northwest to north-south trending fold axes, thrusts and faults (Figure 7.10). The main structures in the Mesozoic sediments are north-northwest, northwest to north-south trending open fold axes (synclines) around the basement anticlinal dome with gentle dips to northwest, west-northwest and east-northeast. There is also a number of major north-northwest and north-south trending faults and thrusts running along the margin of the TMC. Thrusts are generally westward verging, and on the local scale, there are numerous northwest, northeast and east-west trending normal faults, including the (northwest-trending) Bor Fault which displaces the Bor deposits, as described in Section 7.6.2.

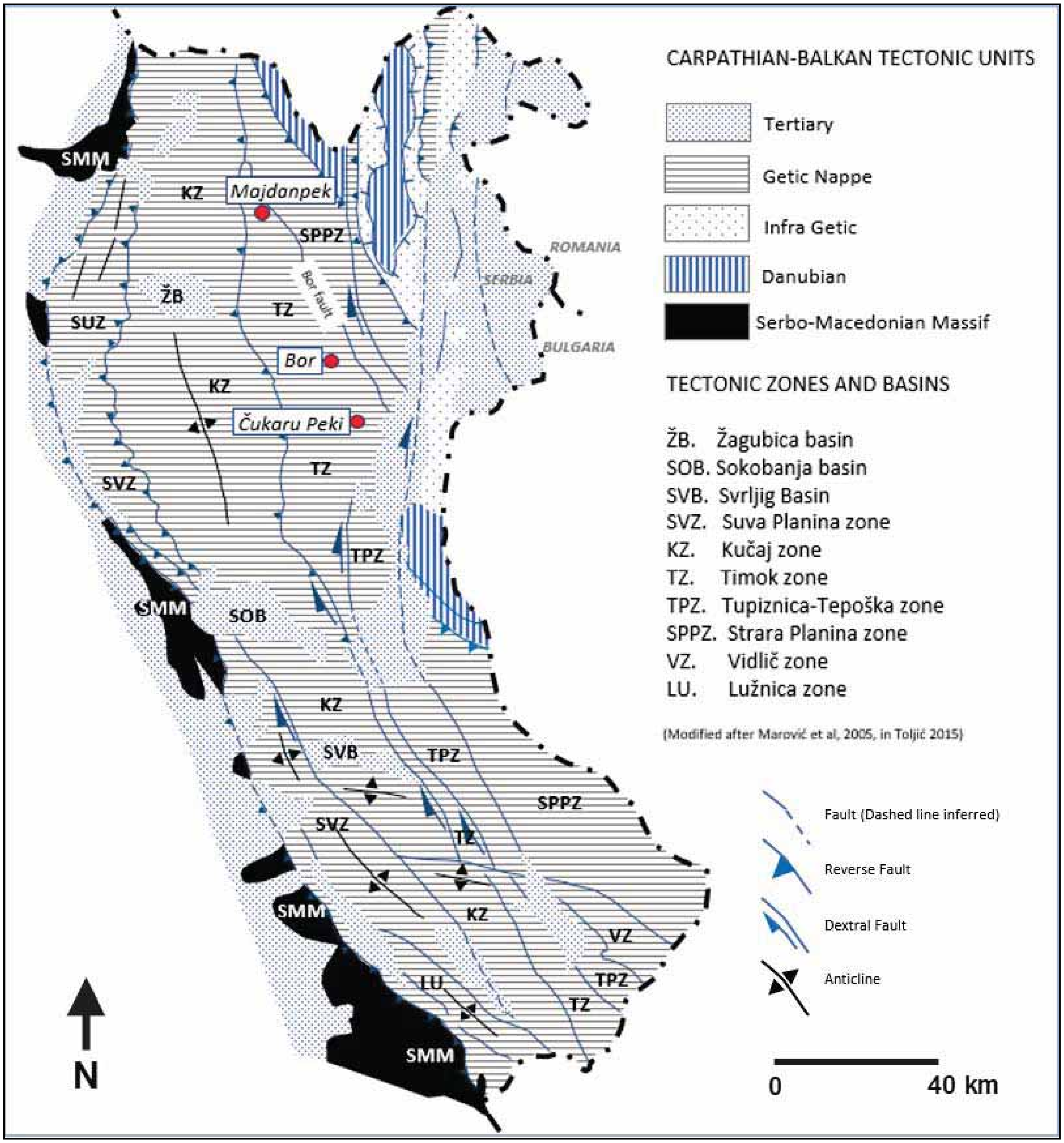


Figure 7.10: Principal Tectonic Units (and Tertiary Cover - Basins) and Structures of the Carpatho-Balkan Region of Eastern Serbia

The extrusive and intrusive units of the TMC formed within a north-northwest to south-southeast trending pull-apart graben (Drew, 2005) produced during transcurrent and strike slip faulting generated by Late Cretaceous orogenic transpression during oblique subduction (Popov, 1974; Drew, 2005). These extensional events were followed by the intrusion of the magmas and associated hydrothermal mineralizing systems which exploited major faults and pre-existing reactivated structures that have a pronounced north-northwest trend. The magmatic and associated mineralization (porphyry and epithermal deposits) follow a pronounced north-northwest to south-southeast to north-south regional structural trend.

7.6.2 Local Scale

The Timok area went through complex multiphase tectonic compressional and extensional episodes during the Alpine Orogeny (Late Cretaceous to Tertiary). Associated volcanism and intrusive magmatism may have led to deposit formation.

Three main structural events can be recognized (Canby et al., 2015):

- An early extensional phase in the Upper Cretaceous, which caused the subsidence and formation of the TMC volcanic basin (graben) along marginal normal faults, characterized by volcanism, sedimentation and (after volcanic activity ceased) deposition of calcareous siltstones.
- A phase of compression followed (roughly normal to the current north-northwest orientation of the TMC) and period of rapid uplift led to formation of Bor clastic sediments.
- A later phase of extension, normal faulting and subsidence during the Miocene, which led to formation of sedimentary basins/deposits unconformably overlying the Late Cretaceous sedimentary units.

Major faults and most geologic units within the Timok area strike parallel to the north-northwest elongation of the TMC. In the Bor and Timok area, well-documented major structures include the Bor and Krivel faults. The former is a west-dipping high-angle reverse fault which displaces the Bor deposits upward and eastward by at least several hundred metres into their current position adjacent to or above the younger Bor clastic unit.

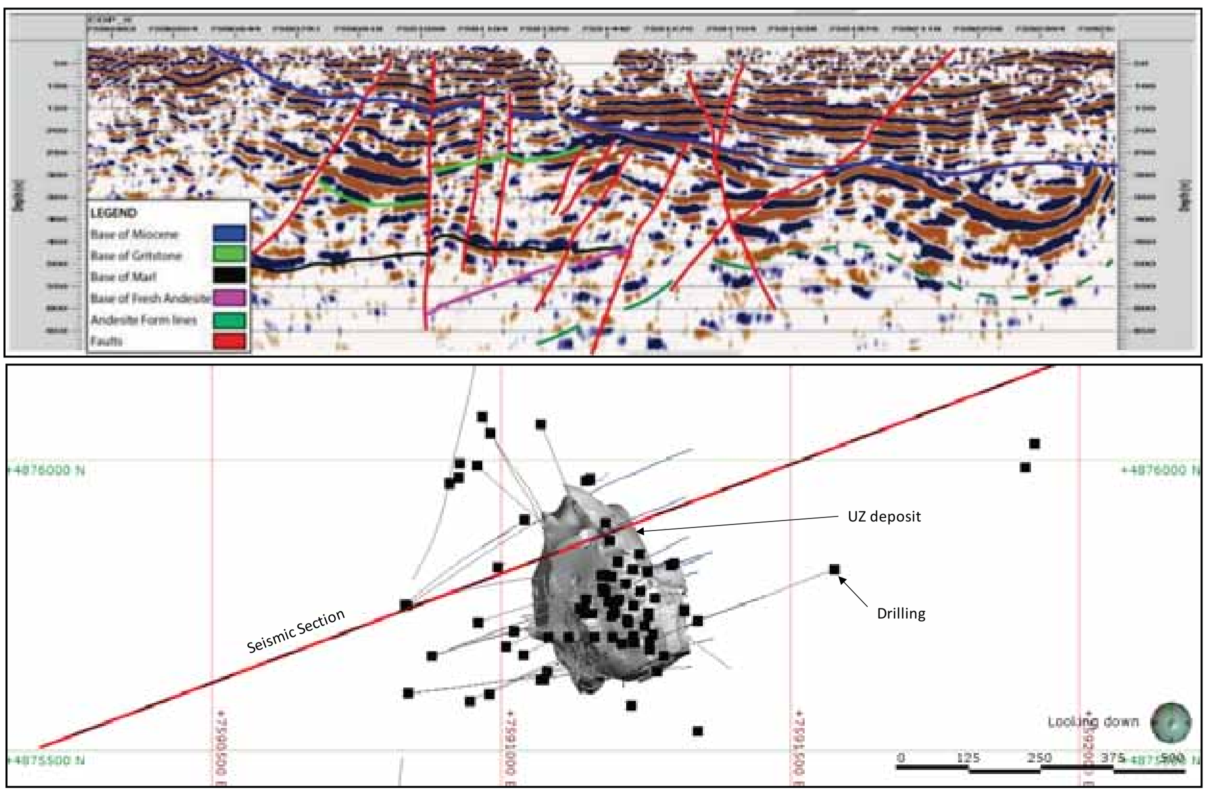
Deposit-scale drilling has possibly identified the ‘Bor 2’ fault beneath Miocene sediments, located to the east of the deposit, which has similar strike and dip to the Bor fault and labelled as ‘BF2’ alongside other permit-scale fault interpretations in Figure 7.4 and Figure 7.5. BF2 is considered by Rakita to represent the eastern (faulted) margin of the mineralized andesite within the Timok area.

Other prominent parallel north-northwest and east-west trending faults (lineament features) appear to be concealed beneath Miocene sediments as indicated by geophysical surveys (magnetic, gravity and CSAMT) as illustrated in Figure 7.11.

7.6.3 *Deposit Scale*

At the deposit scale, the Upper Zone (Čukaru Peki) HS deposit is located at an intersection between north-northwest and east-west structural corridors (Figure 7.12). The main mineralized zone is interpreted to be largely bound within the two major faults referred to as the East and West Faults (Figure 7.12). Mineralization is also observed as largely bound to the north of the 'South Fault' (Figure 7.12). However, limited drilling to date has been completed to the south of this structure and therefore the current interpretation may change with additional drilling campaigns.

Significant displacement along the West Fault has been interpreted largely based on drillhole TC150091, where the steep drilling orientation results in repetition downhole of very high grade massive sulphide mineralization and stratigraphy, explained (in context of logged fault rock and visual offset in copper grade and overlying stratigraphy in adjacent holes) by offset along a major structure.



Source: SRK, modified from Curtin university/ GEOING group d.o.o., 2014

Figure 7.11: Preliminary Seismic Interpretation (top) and Section Reference (bottom) Illustrating Numerous Faults, Looking North-Northwest

The faults have been interpreted based on 3D modelling of a combination of the visual sharp contact between the high-grade mineralization and very low-grade host rock, structural and geotechnical core logging, fault orientation data from acoustic borehole imaging (ABI) and also by the apparent displacement of the overlying Upper Cretaceous stratigraphic layers of marl and Bor clastic units.

In addition to the modelled faults shown in Figure 7.12, a relatively narrow, discontinuous zone of clay alteration and geotechnically weaker rock occurs at the upper margin of the lower andesite and UZ mineralized body. This is interpreted to represent the upper limits of the UZ argillic alteration front, rather than a discrete zone of faulting.

Whilst the predominantly vertical orientation of the drilling makes interpretation of the steep faulting at Čukaru Peki relatively difficult, in general the deposit-scale fault interpretations are considered to have a reasonable level of confidence. Internal zones of small-scale faulting, fracturing and zones of weaker clay altered rock within the mineralized zone have been modelled for geotechnical purposes using a combination of geotechnical logging, ABI and drill core structural logging. These do not significantly affect the geometry of the mineralization.

Beneath the UZ mineralized body (and partly along its eastern margin), a broad zone of clay and clay fractures has been modelled (as shown in Figure 7.9) based on drill core observations and geotechnical data, primarily to inform geotechnical assessment. It currently remains unclear whether this zone primarily relates to structure or alteration; however, the most recent interpretation suggests this coincides with the argillic alteration front.

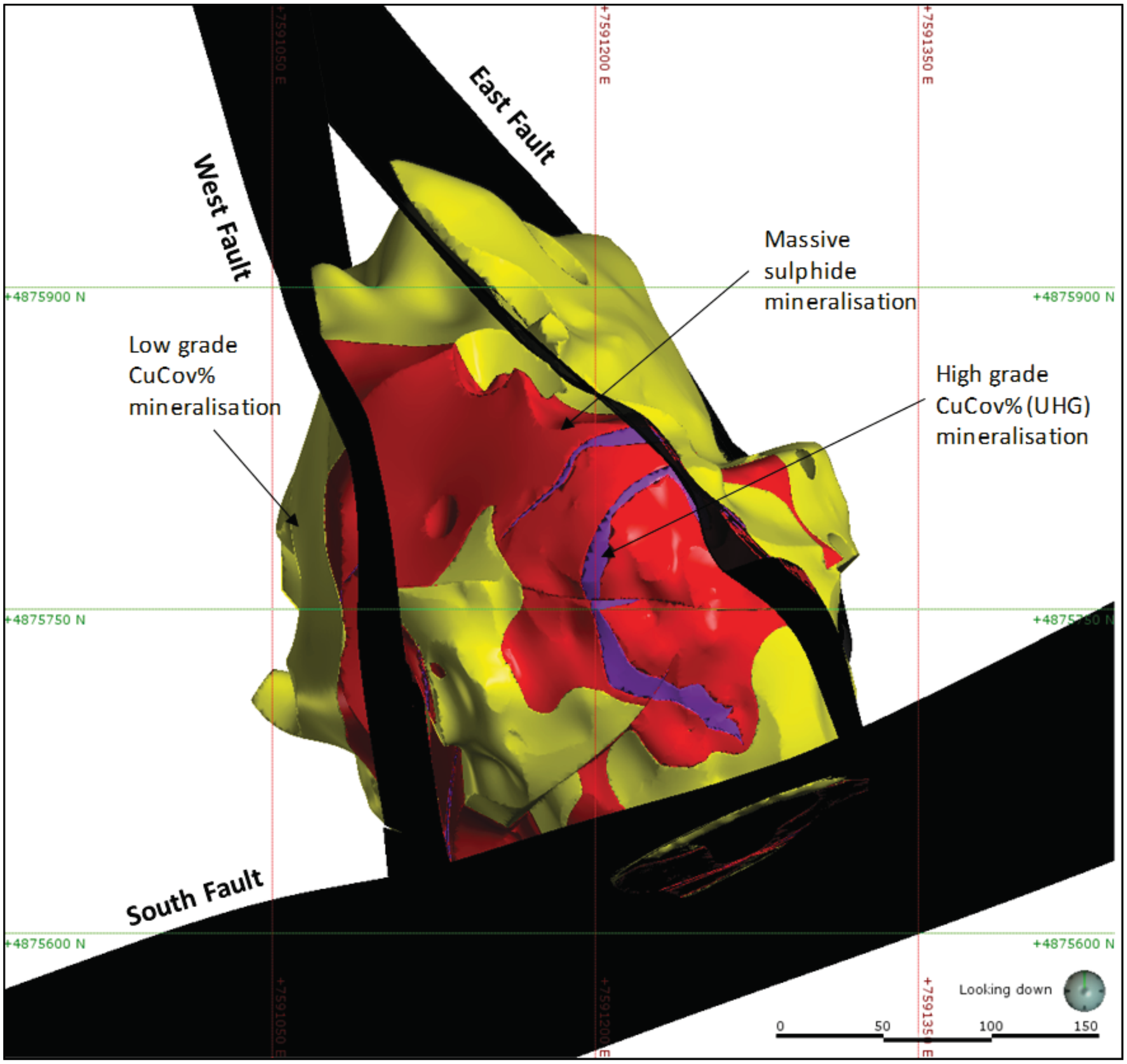


Figure 7.12: 3D (Plan) Image of Major Deposit-Scale Faults Interpreted at Čukaru Peki

8. Deposit Types

8.1 Mineralization in the Bor District

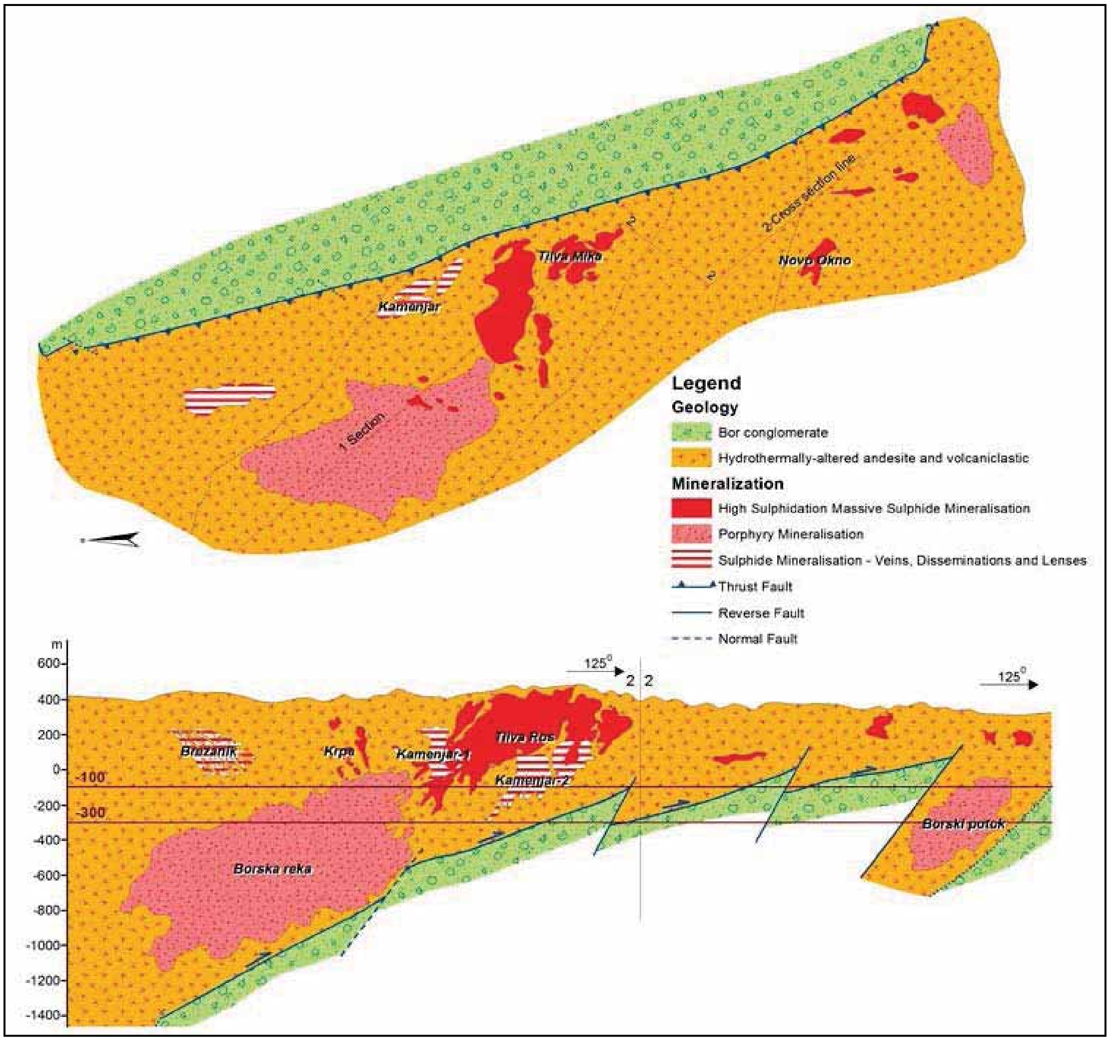
There are a number of different copper-gold deposits within the TMC, most of which are related to porphyry or epithermal style mineralization. Jelenković et al. (2016) have summarized the most significant styles of mineralization from the Bor district, including:

- Porphyry copper-gold quartz-sulphide vein/stockwork and disseminated sulphide mineralization.
- Epithermal, HS “massive sulphide” mineralization.
- Skarn and contact/carbonate replacement deposits.
- Vein/fault-controlled vein.
- Hydrothermal volcanogenic polymetallic/sulphide-bearing matrix in intrusive hydrothermal breccia mineralization.
- Mechanical transported sulphide fragments/clasts in sedimentary mineralization.
- Sediment hosted gold deposits.

The Bor copper-gold mining area is located approximately five kilometres north of the Timok project area. Bor contains two porphyry systems (Borska Reka, Borski Potok) as well as various spatially associated bodies of HS epithermal mineralization. The HS mineralization consists of covellite, bornite and locally chalcopyrite and enargite found in masses of fine-grained pyrite and occurs in several deposits including Coka Dulkan, Tilva Mika and Tilva Ros, which were very high grade and were originally mined out from the surface. Coka Dulkan, which was discovered in 1902, contained 5.45% to 19.4% of copper and an average of 1.5 g/t gold (Jankovic et al., 2002). Novo Okno and some of the other smaller “massive sulphide” deposits shown Figure 8.1, are interpreted to have been eroded and re-deposited from the original site of mineralization.

Figure 8.1 illustrates the close spatial relationships between the Bor porphyry and HS mineralization and the proximity to the Bor reverse fault. The principle styles of mineralization recorded at Bor have also been identified in the Timok area.

There are several other important porphyry related copper-gold deposits in the district to the northeast and north-northeast of Bor, including Veliki Krivelj, Mali Krivelj, Cementacija and Cerovo. Mineralization at Bor is invariably hosted by hornblende-andesites of the first volcanic phase (eastern block) and commonly related to north-northwest-striking reverse faults, including the important Bor Fault.



Source: modified from Jankovic et al., 2002.
Note: Horizontal and vertical scales are the same.

Figure 8.1: Plan and Cross-Section of the Mineralization in the Bor Mining District

8.2 Other Analogues

The HS epithermal and porphyry deposits of Timok bear many similarities to both the circum-Pacific southeast Asia copper-gold provinces and the neighbouring eastern extension of the Panagyurishte Belt in Bulgaria. Two examples of well-preserved porphyry copper systems with associated epithermal HS mineralization (outside of Timok Belt), are described below and include Panagyurishte, Bulgaria and Lepanto, Philippines.

8.2.1 Panagyurishte, Bulgaria

The Panagyurishte Belt, located east of the Timok Belt, represents a comparable well-preserved porphyry with associated epithermal mineralization, and clear evidence of telescoping (coalescence of the epithermal alteration assemblages of alunite).

The spatial association of the copper-gold epithermal deposits with porphyry copper deposits have been described in Bulgaria at the Panagyurishte Belt (Petrunov et al., 1991; Sillitoe, 1999). The Elshitsa HS epithermal deposits and nearby (1 km) past producing Vlaykov Vruh porphyry copper deposit constitute the best examples for the close spatial association of HS epithermal with porphyry Cu deposits (Kouzmanov, 2001) (Figure 8.2).

Cu–Au epithermal HS occurrences have also been described in the immediate proximity of the Assarel and Petelovo porphyry copper deposits (Petrunov et al., 1991; Sillitoe, 1999). The Chelopech HS massive sulphide deposit (1.28% Cu and 3.4 g/t Au) belongs to a deposit cluster in the northern Panagyurishte district, which also includes the vein-type Vozdol base metal occurrence, the Karlievo porphyry copper occurrence, and the major producing porphyry copper Elatsite deposit (1.13% Cu and 1.5 g/t Au in Popov et al., 2000).

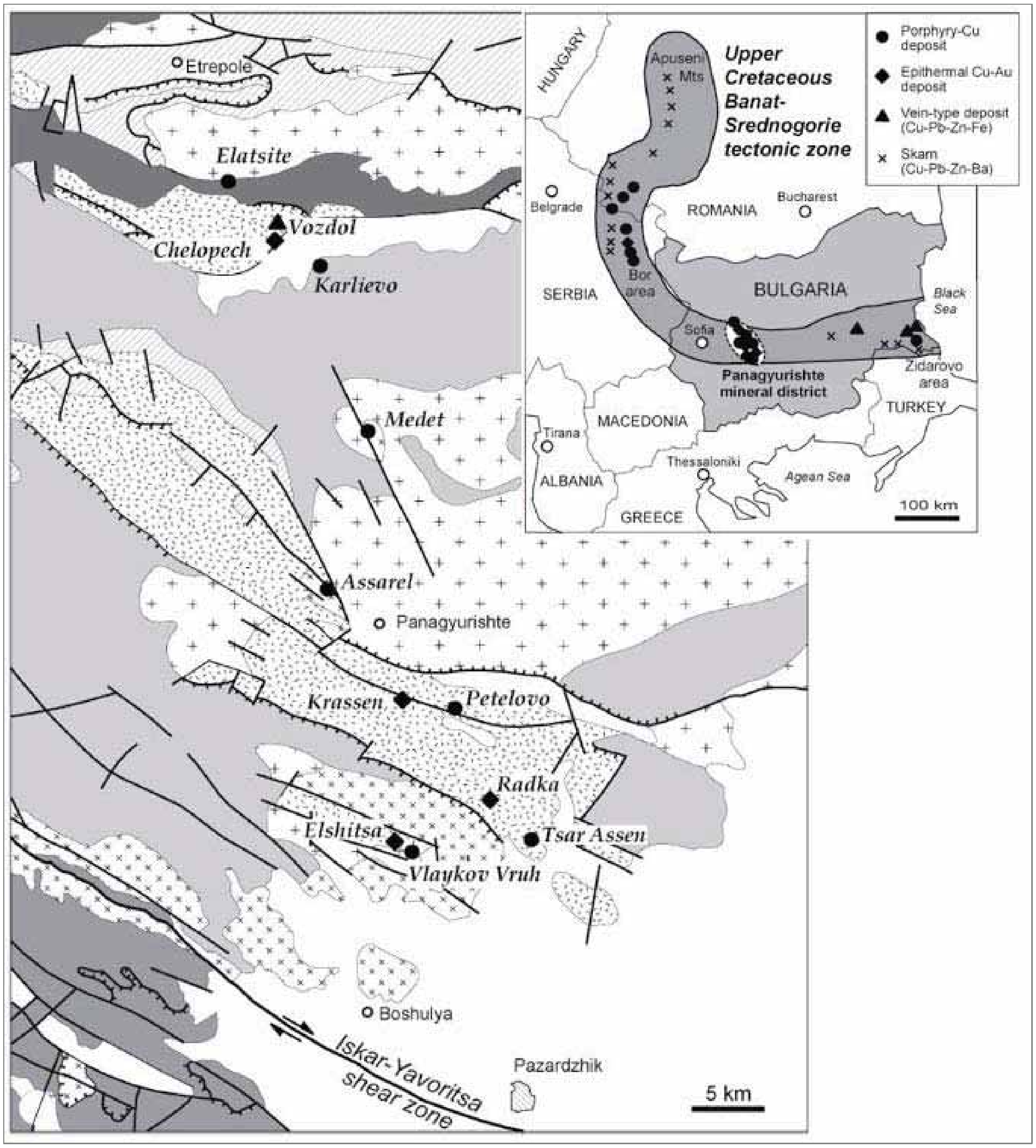
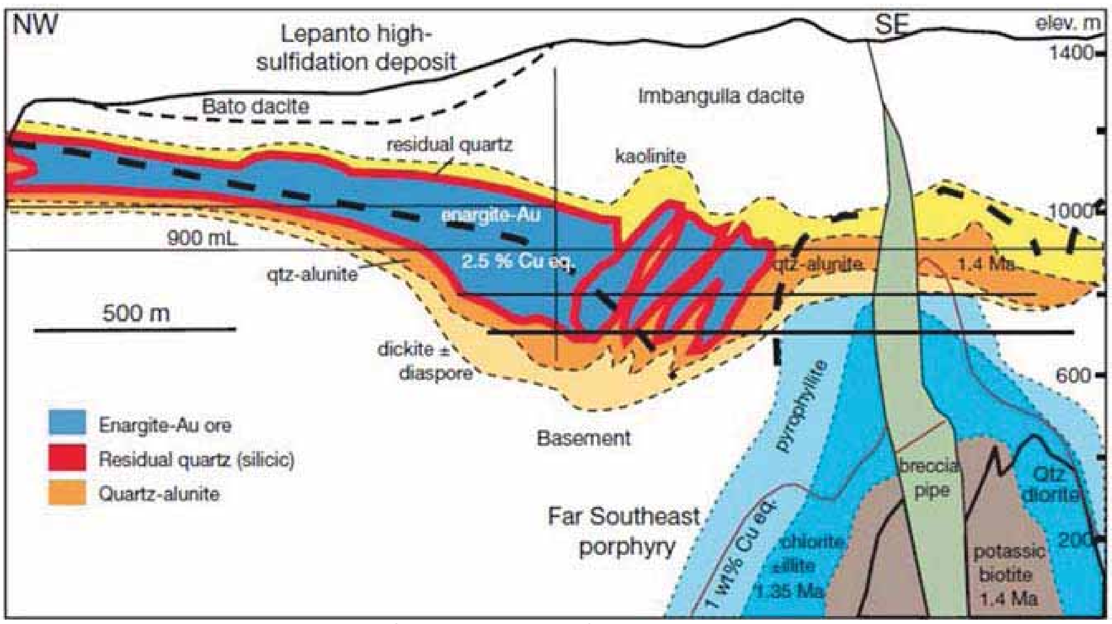


Figure 8.2: Panagyurishte Belt Located Eastern Timok Belt, Showing Clusters of Copper and Gold Porphyry and its Associated Massive Sulphide Epithermal Deposits

8.2.2 *Lepanto, Philippines*

At the Lepanto deposit, the orebody mineralogy, alteration assemblages and deposit architecture are broadly comparable to those observed and interpreted at the Timok deposit (Hedenquist et al., 1998; Hedenquist & Taran, 2013). The combination of steeply-dipping structural controls combined with shallow-dipping lithological controls is broadly similar to that at the Timok deposit; however, the shallow dipping mineralization at Lepanto is related to a laterally extensive pre-mineralization unconformity as opposed to the relatively restricted lateral extent of the volcaniclastic breccia units at the Timok deposit (Figure 8.3).



Source: Hedenquist & Taran, 2013

Figure 8.3: Schematic NW-SE Long Section through the Lepanto Enargite-Au Deposit

The smaller gold occurrences and prospects near Brestovać village (Corridor Zone and Ogasu Kucajna) are associated with base metals, in particular zinc, and display alteration and mineralogy similar to epithermal intermediate sulphidation mineralization (Sillitoe & Hedenquist, 2003). This style of mineralization is commonly developed in the distal parts of mineralizing systems and can form stand-alone deposits, such as Victoria at Lepanto (Hedenquist et al., 2000).

9. Exploration

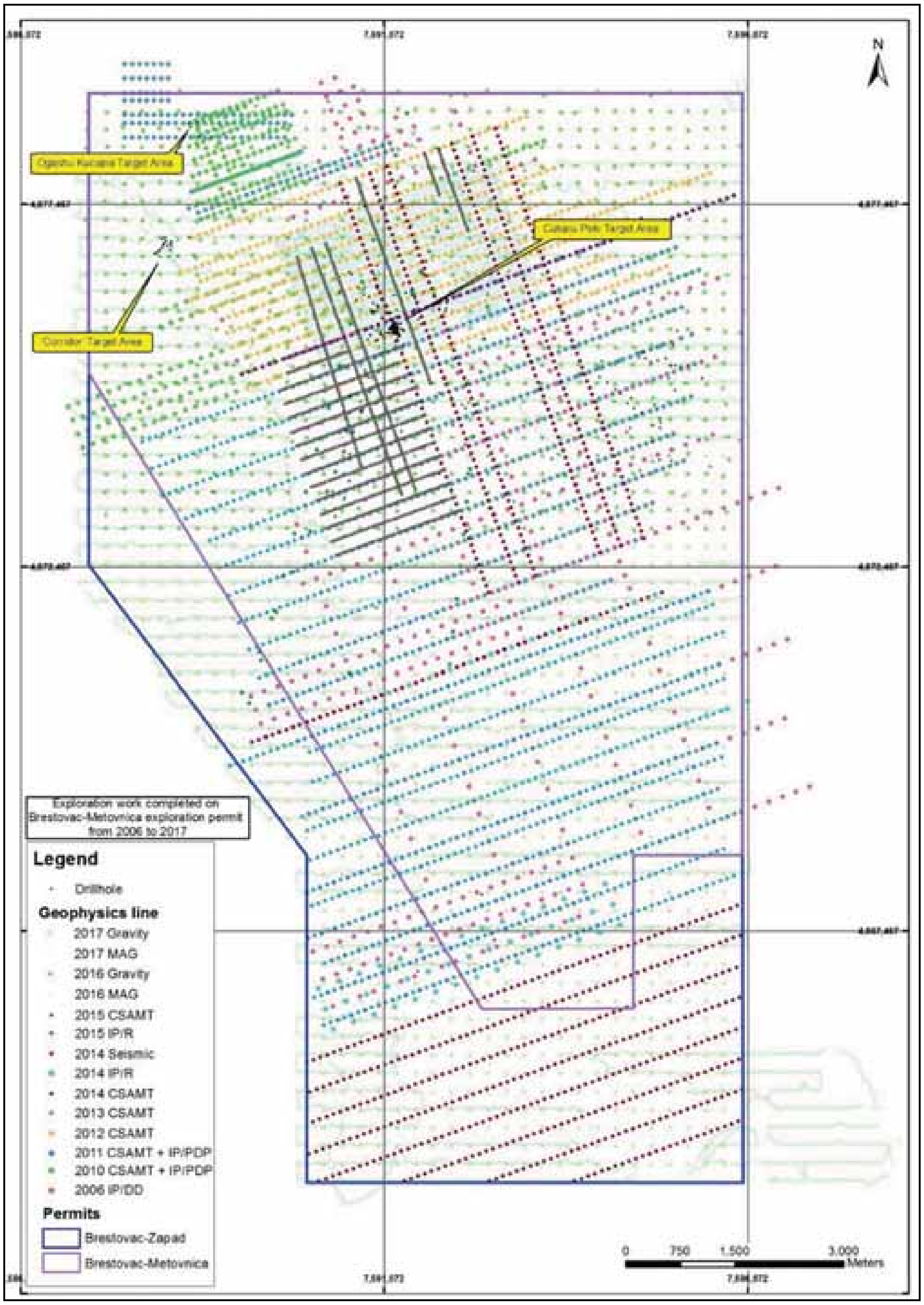
This section summarizes the work completed by Rakita and its partners within the Brestovać-Metovnica exploration permit since the start of the (2016) joint venture between Nevsun and Freeport-McMoRan Exploration Corporation. The history of mineral exploration in the Brestovać and Metovnica permits prior to 2016 is summarized in Section 6.

Since the start of the joint venture in 2016, several diamond drilling programs have been completed from surface mainly targeting the Timok UZ and LZ deposits, as described and illustrated in Section 10.

In addition, exploration drilling is currently in progress in the immediate surroundings of Čukaru Peki, focussing on targets generated from an updated interpretation of the geological, geochemical, geophysical, downhole electromagnetic, spectral and assay information. Geophysical work includes new lines completed in 2017 illustrated against previous geophysical lines, drilling completed to date and historical exploration target areas as shown in Figure 9.1.

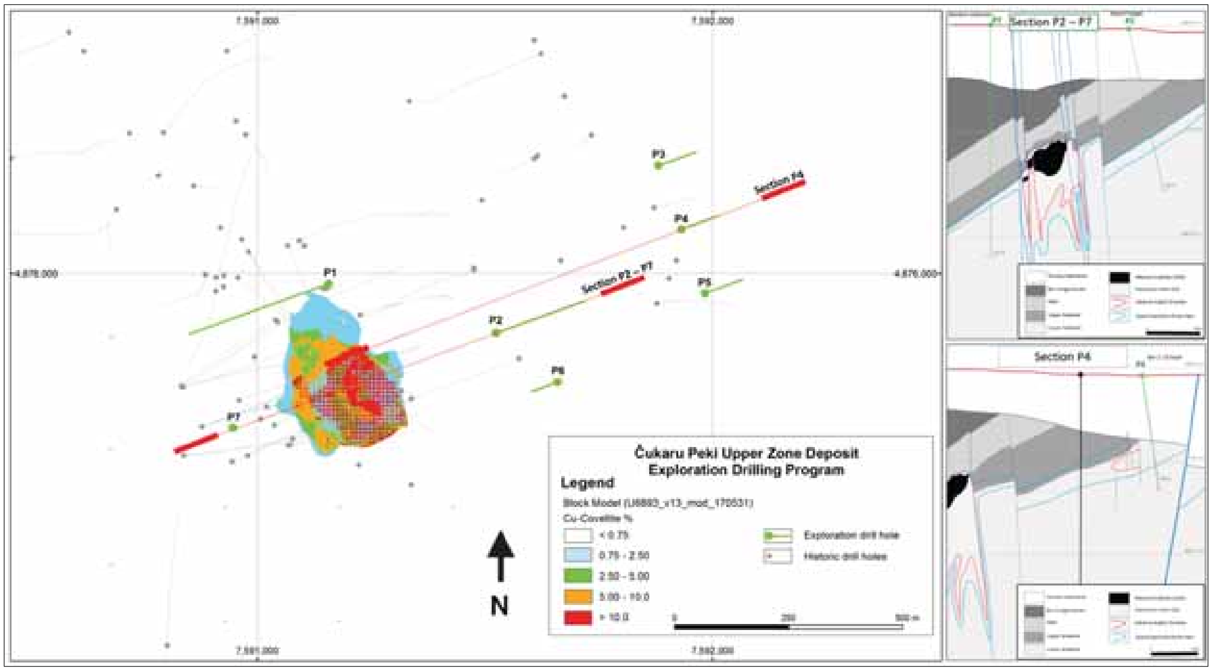
The currently ongoing exploration work on the licence includes proposed drilling that is targeting possible extensions to the east and west of Čukaru Peki, based on plans illustrated in Figures 9.2 and 9.3 and includes downhole electromagnetic geophysical surveys to help search for the presence of nearby off-hole conductive mineralized bodies, similar to the massive sulphide (pyrite-covellite) mineralization at Čukaru Peki.

Rakita has also started a condemnation drill program based on plans illustrated in Figure 9.3. The condemnation drilling campaign has 21 vertical holes planned with depths between 800 and 1000 m. These are partly on a grid spacing of 500 m in the vicinity surrounding the deposit, the process plant area and partly along the path of the proposed decline. To date, 13 condemnation drill holes have been completed, with results confirming absence of significant mineralised rock in these locations.



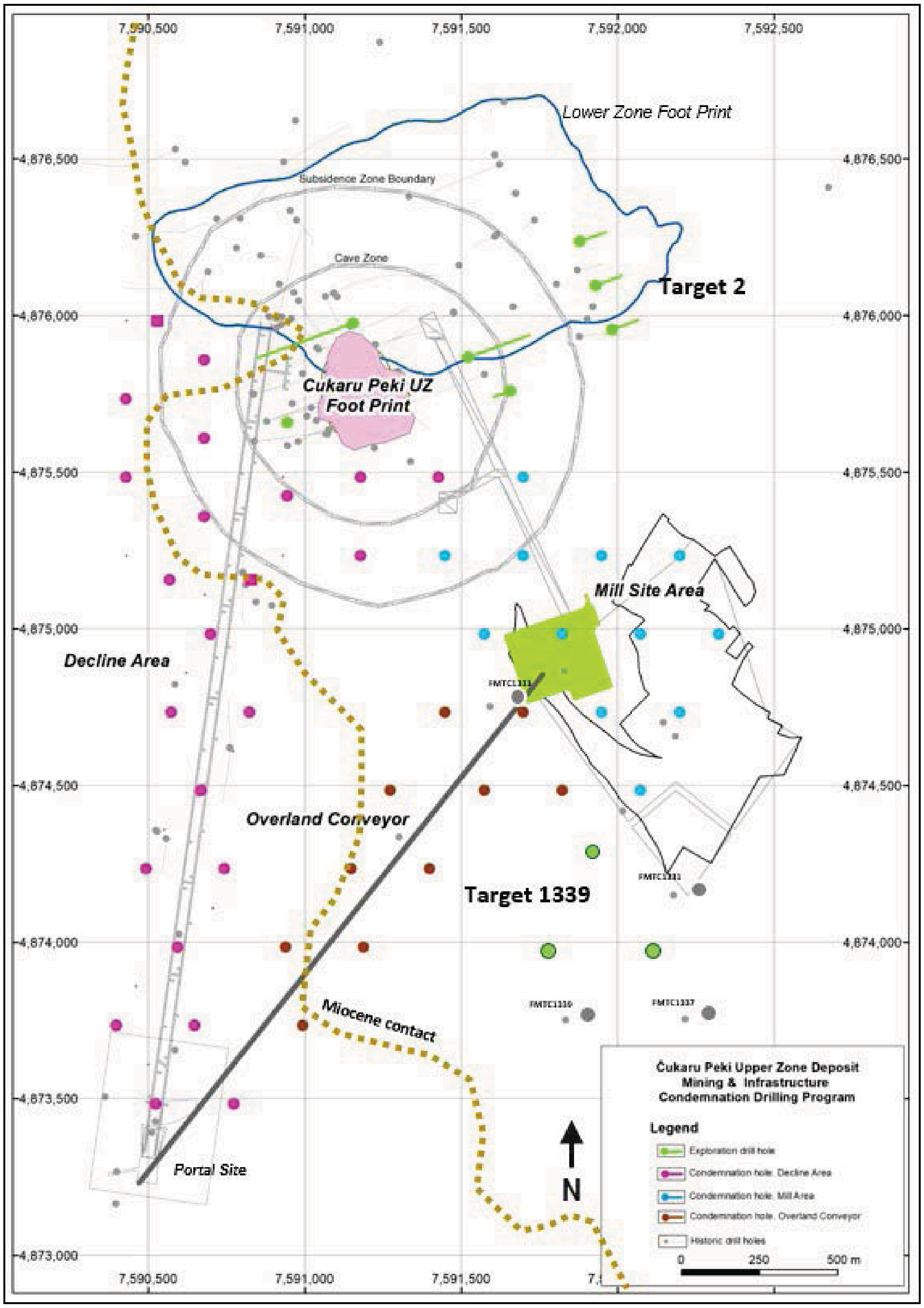
Source: Rakita, 2017

Figure 9.1: Exploration Work Completed on the Brestovać-Metovnica Exploration Permit from 2006 to 2017 (excludes LZ drillholes)



Source: Rakita, 2017

Figure 9.2: Rakita's Exploration Drilling at the Margins of Current Čukaru Peki UZ Mineral Resource



Source: Rakita, 2017

Figure 9.3: Rakita's Proposed Exploration and Condemnation Drilling Program

10. Drilling

10.1 Historical Drilling Programs

Historic drilling completed on both the Brestovać and Metovnica sectors of the Brestovać-Metovnica exploration permit is summarized in Section 6, SRK has relied on summary documents provided by Rakita in relation to historic drilling outside of the Timok deposit area and has not completed a detailed review of collar, assay, survey and geology as part of the current phase of work. SRK is not aware of any historic drilling programs completed within the Timok deposit area, prior to the exploration completed by Rakita.

10.2 Current Drilling Programs

This section summarises the Rakita drilling:

- completed up to the cut-off date of 24th of April 2017, most of which was used for the UZ Mineral Resource;
- the drilling available as of 14th of April 2018 for the Lower Zone Mineral Resource. This section does not cover a relatively small number of exploration drillholes completed elsewhere in the project licence area; and
- the condemnation holes completed before 20th of July 2018.

10.3 Drilling Databases

It is important to note that the UZ and LZ resource models were undertaken separately using drillhole databases that were exported from Rakita's master database at different times. It is also important to note that for the UZ database, owing to different ownership and confidentiality constraints existing at the time, some drillholes that intersected both UZ and LZ were artificially shortened to exclude LZ data and those drillholes which only intersected LZ were excluded from the UZ database. Similarly, any drillholes that intersected only the UZ were excluded from the LZ database.

The total of mother and wedged daughter drillholes completed at Cukaru Peki amounts to 188,929.9m in 268 drillholes. The two databases described below are not mutually exclusive their combined drillhole count and meterage exceed this total because 14 drillholes are present in both databases.

10.3.1 Upper Zone

10.3.1.1 Summary of Data Quantity

All drilling data available for the Upper Zone as of 24 April 2017 was made available to SRK (UK). A summary of the Upper Zone holes completed by the Rakita as at 24 April 2017 is provided in Table 10.1; the complete UZ drillhole database comprised 180 holes for a total of 100,337.9 m.

No additional drilling for UZ resource purposes has been completed since 24 April 2017. However, in the intervening period to 20 July 2018, one of the metallurgical holes which intersects the UZ resource model was assayed; the assay results agreed very well with the block model and in SRK (UK)'s opinion would have no material impact on the tonnage and grade in the block model if the model were to be updated to incorporate this new data. Also, in the intervening period to 20 July 2018, an additional 12 holes for a total of 9,072.9 m have been completed for condemnation purposes over planned infrastructure areas; these do not intersect the UZ resource model.

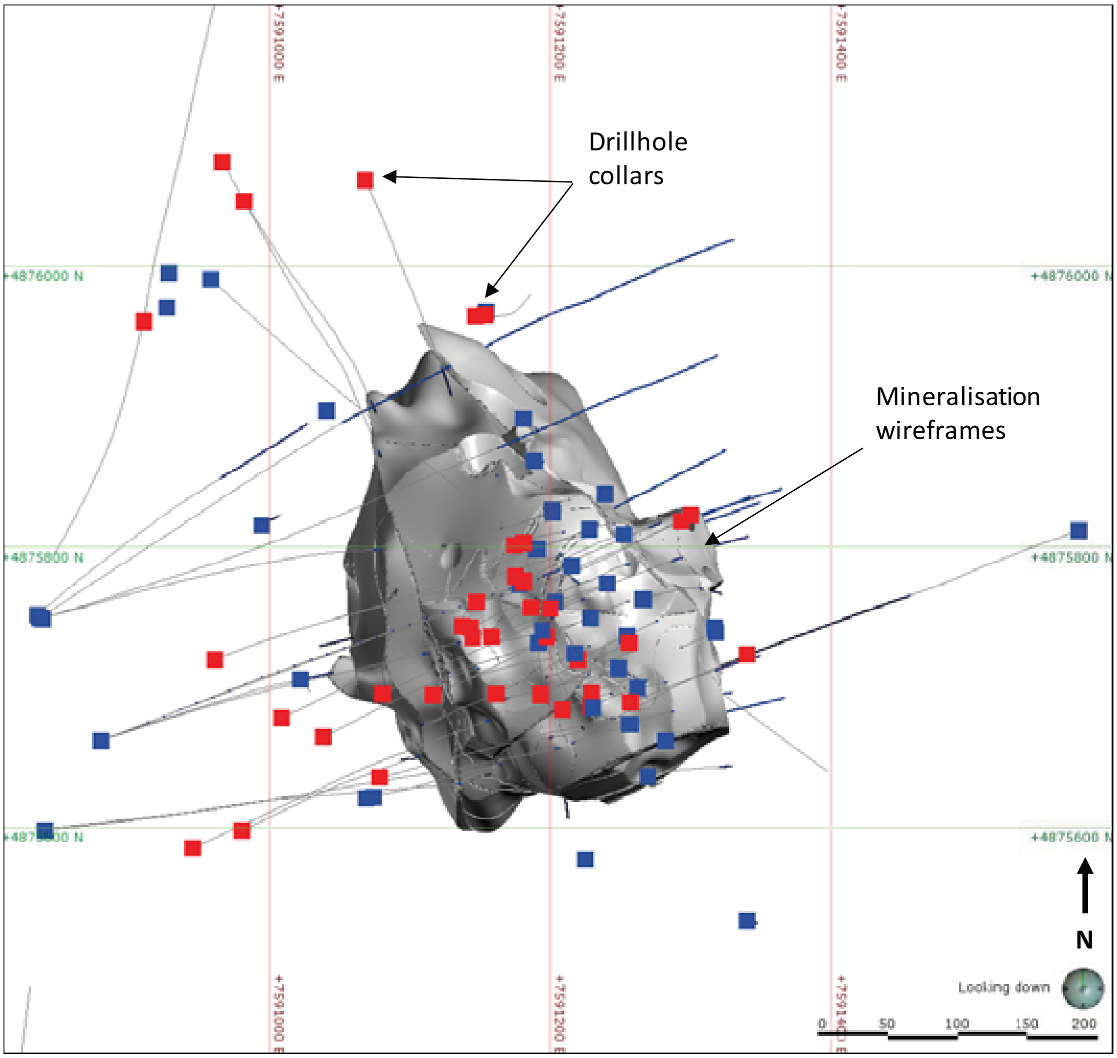
Table 10.1: Summary of Upper Zone drilling as at 24 April 2017*

Purpose	Drilling Type	Count	Total length (m)
Condemnation	Diamond Drilling	1	405.00
Exploration		65	53,245.40
Geotechnical		26	5,542.80
Metallurgical		3	2,492.30
Monitoring well		32	3,647.20
Resource		48	32,126.70
Vibrating Wire Piezometer		5	2,878.50
Total		180	100,337.90

* The UZ drilling database includes non-sampled hydrogeological, condemnation and geotechnical holes and excludes LZ and historic holes located away from the UZ deposit.
** One hole (FMTX1224) has a reverse circulation (RC) pre-collar and was finished with diamond drilling (DD)

10.3.1.2 Collar Surveys

Since hole FMTC1210, all Upper Zone collars were surveyed using high precision GPS based on total station measurements, which give a high degree of confidence in terms of the XY and Z location. Data for some of the older holes (outside of the Čukaru Peki area), was obtained by hand-held GPS. This data was provided to SRK (UK) in digital format using Gauss-Krüger MGI Balkans 7 coordinate system grid coordinates.



*In some cases, two or three drillholes have been completed from a single drill site.

Figure 10.1: Location of UZ database collars (red = completed since march 2016 NI 43-101)

10.3.1.3 Downhole Surveys

SRK was supplied with downhole survey information for the start and the end of each hole, with intermediate readings taken at intervals of up to 50 m. A range of tools was used, including a Reflex Gyro and Reflex EZ-Trac, Camteq Proshot Camera probe (CTPS200), north-seeking gyro and DeviTool Standard survey measurement. All azimuth readings are corrected to grid north in the database. In general, the data collected are considered to be of high precision and accuracy suitable for use in this resource estimation.

In order to compare the accuracy of the various systems, several holes, including TC150061, TC150067 and TC150064, were surveyed using both the DeviTool and Reflex Gyro or Camteq and north-seeking gyro tools. The results show that different tools in the same drillhole can provide pierce-point horizontal location errors at the top of the UZ deposit (some 450 m below surface) of 2 to 8 m.

10.3.1.4 Hole Orientation

All drilling undertaken on the Upper Zone has been completed from surface intersecting the mineralized zone from the northeast, southwest and above. Drillholes are typically plotted on sections oriented north 70° east providing intersections spaced 50 to 100 m apart.

The dips for inclined holes range from -50 to -85º, with hole lengths typically ranging from 300 to 1,000 m. In places, wedged daughter holes have been completed to maximise the information made available from a single drill site. It is SRK's view that the drilling orientations are reasonable to model most of the geology and mineralization based on the current geological interpretation. An example of drilling coverage and orientation is shown in Figure 10.2, which also shows mineralization wireframes.

10.3.1.5 Diamond Drilling Procedure

The drilling was performed by contractors and managed by Rakita's geological team and has been reviewed by SRK (UK) during several site visits. The drilling program to date has been completed by four drilling contractors: Drillex International, Geops Balkan Drilling Services Ltd, S&V Drilling Mine Services d.o.o and Geomag d.o.o.

All drilling was completed using diamond core, excluding one drillhole (FMTC1224), which had a reverse circulation pre-collar through the Miocene and Upper Cretaceous sedimentary rocks.

Diamond core drilling was performed with the use of a double tube with casing reducing from PQ to HQ and NQ rods at the appropriate depths.

Core was produced in three-metre core runs and then placed by hand into an open V-rail for measurement of recovered core length, before being transported to the drill site geologist who inspected the core and transferred it into numbered plastic core boxes. Cut plastic blocks were used to record core depths.

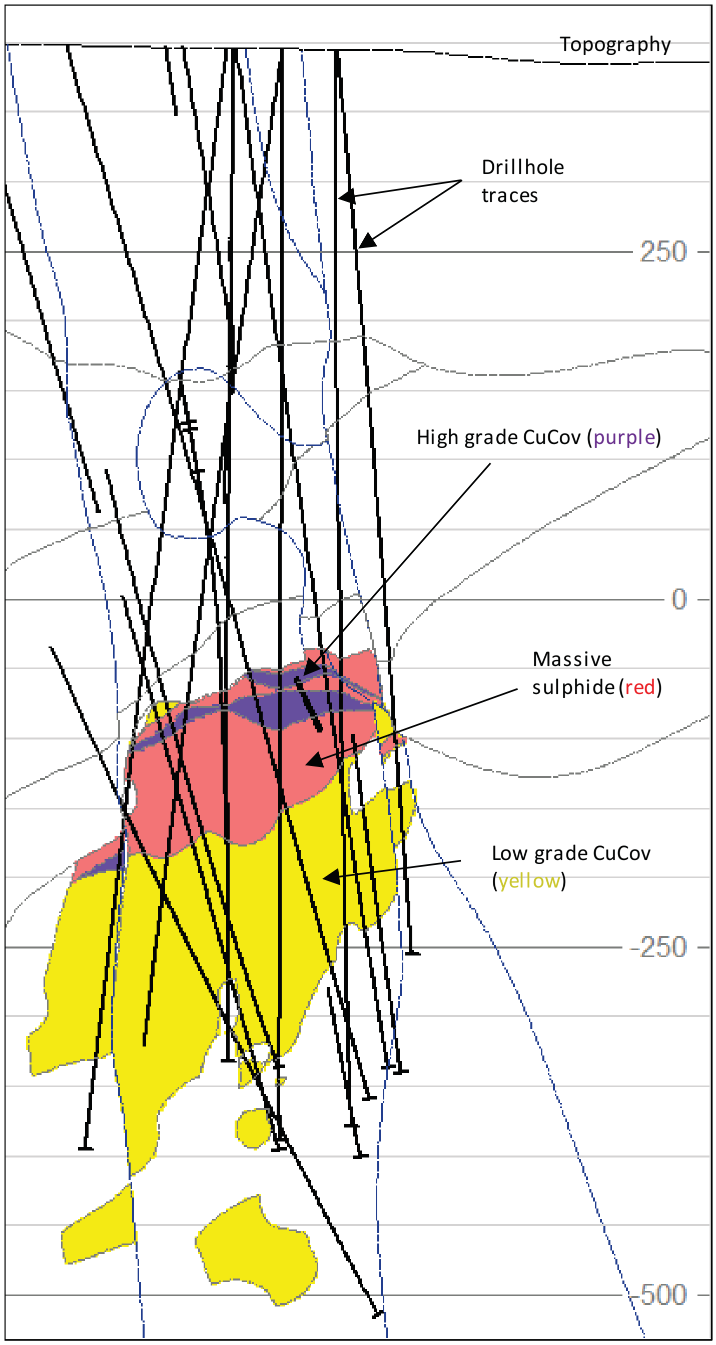


Figure 10.2: Example Cross-Section through the UZ Deposit (25 m Clipping Width)

10.3.1.6 Core Recovery

SRK (UK) reviewed the drill core recovery results and found that in general the recovery is good with an average recovery of 98.0% for the entire Upper Zone (Figure 10.3).

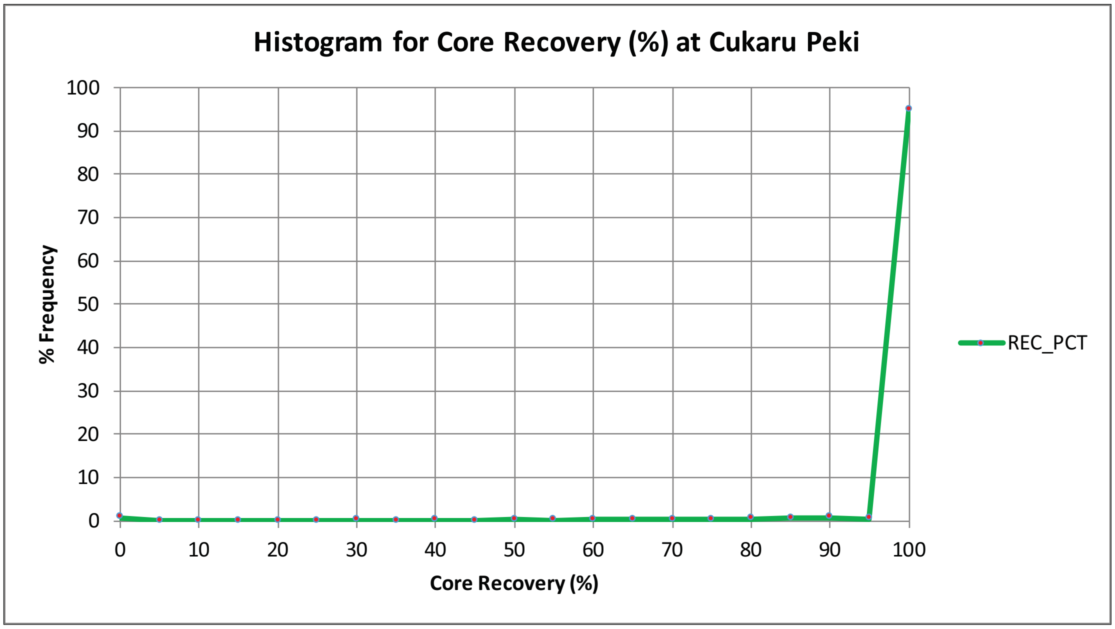


Figure 10.3: Core Recovery for the Upper Zone

10.3.2 Lower Zone

10.3.2.1 Summary of Data Quantity

All drilling for the Lower Zone as of 14th of April 2018 cut-off date was made available to SRK Canada. A summary of the Lower Zone holes completed by the Rakita is provided in Table 10.1. The complete drill hole database comprises 102 holes with a total of 108,611.4m among exploration, resources, monitoring wells and condemnation.

Table 10.2: Summary of Lower Zone drilling for the resource estimation purpose as at 14th of April 2018

Purpose	Drilling type	Count	Total length (m)
Condemnation	Diamond Drilling	1	849.6
Exploration	Diamond Drilling	68	68,195.1
	RC Drilling (*)	1	251.0
Monitoring well	Diamond Drilling	1	428.4
Resource		32	38,887.3
Total		102	108,611.4

10.3.2.2 Collar Surveys

Since the discovery hole FMTC1210, all Lower Zone collars were surveyed using high precision GPS based on total station measurements, which give a high degree of confidence in terms of the XY and Z location. Data for some of the older holes (outside of the Čukaru Peki area), was obtained by hand-held GPS. This data was provided to SRK (Canada) in digital format using Gauss-Krüger MGI Balkans 7 coordinate system.

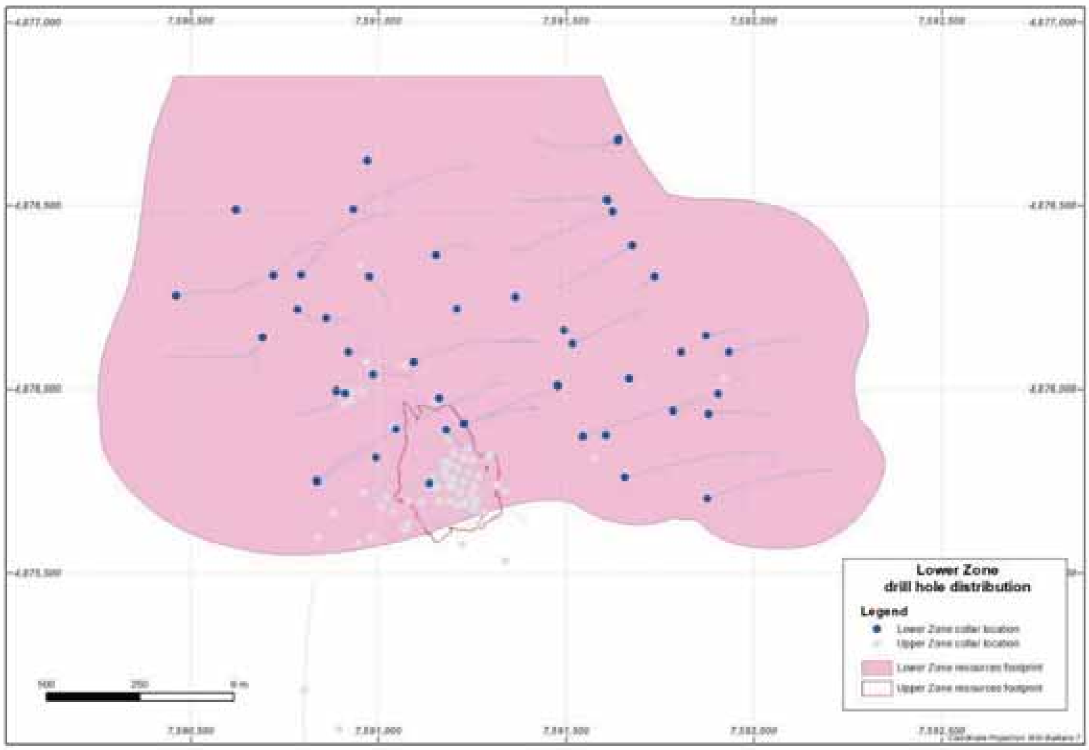


Figure 10.4: Location of collars completed up-to 14th of April 2018

10.3.2.3 *Downhole Surveys*

SRK was supplied with downhole survey information for the start and the end of each hole, with intermediate readings taken at intervals of up to 50 m. A range of tools was used, including a Reflex Gyro and Reflex EZ-Trac, Camteq Proshot Camera probe (CTPS200), north-seeking gyro and DeviTool Standard survey measurement. All azimuth readings are corrected to grid north in the database. In general, the data collected are considered to be of high precision and accuracy suitable for use in this resource estimation.

In order to compare the accuracy of the various systems, several holes, including TC150061, TC150067 and TC150064, were surveyed using both the DeviTool and Reflex Gyro or Camteq and north-seeking gyro tools. The results show that different tools in the same drill hole can provide pierce-point horizontal location errors at the top of the UZ deposit (some 450 m below surface) of 2 to 8 m.

10.3.2.4 Hole Orientation

As well as the Upper Zone, all drilling undertaken on the Lower Zone has been completed from surface plotted on sections oriented north 70° east providing intersections spaced 250 m, with a few exceptions at maximum of 500 m apart.

The dips for inclined holes range typically from -90º to -80º, with minor number of holes ranging from 75º to 60º. Hole lengths are ranging from 770 to 2,268 m. In some places, wedged or Navi-Drilling tolls were used to develop extension from mother into daughter holes have been completed to maximise the information made available from a single drill site.

It is SRK's view that the drilling orientations are showing a wide grid with mostly sub-vertical holes which in the future would require better coverage to improve the geology, alteration, and mineralization interpretation. An example of drilling coverage and orientation is shown in Figure 10.4, which also shows mineralization wireframes.

10.3.2.5 Diamond Drilling Procedure

The drilling was performed by contractors and managed by Rakita's geological team and has been reviewed by SRK (UK) during several site visits. The drilling program to date has been completed by four drilling contractors: Drillex International, Geops Balkan Drilling Services Ltd, S&V Drilling Mine Services d.o.o and Geomag d.o.o. and Geoing Group d.o.o.

All drilling was completed using diamond core, excluding one drill hole (FMTC1224), which had a reverse circulation pre-collar through the Miocene and Upper Cretaceous sedimentary rocks.

Diamond core drilling was performed with the use of a double tube with casing reducing from PQ to HQ and NQ rods at the appropriate depths.

Core was produced in three-metre core runs and then placed by hand into an open V-rail for measurement of recovered core length, before being transported to the drill site geologist who inspected the core and transferred it into numbered plastic core boxes. Cut plastic blocks were used to record core depths.

Figure 10.5: Cross-Section through the LZ Deposit, looking north-west (200 m clipping width)

10.3.2.6 Core Recovery

Drill recovery results were not reviewed by SRK Canada. Current core recovery for the Lower Zone are reported by Rakita at average 99.1%, as shown in Figure 10.6.

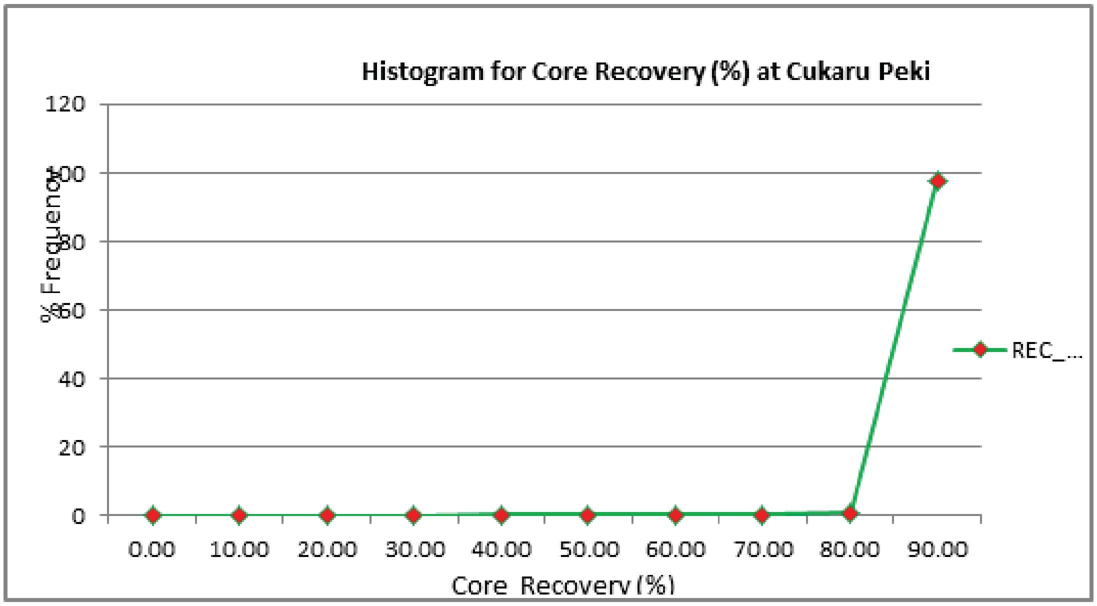


Figure 10.6: Core Recovery for the Lower Zone

10.4 Core Storage

The core shed is located adjacent to the deposit. SRK (UK) visited the storage facility several times and found the facility to be organized and clean.

SRK (UK) notes the potential for oxidation of the massive sulphide mineralization and recommends that Rakita consider vacuum packing or nitrogen purging the relevant core in plastic covers and/or storage in a freezing container. Preservation of core quality may be required for future detailed re-logging and physio-chemical metallurgical tests.

10.5 SRK Comments

In the opinion of SRK, the sampling procedures used by Rakita conform to industry best practices and the resultant drilling pattern is sufficiently dense to interpret the geometry, boundaries and different styles of the copper and gold mineralization in the Upper Zone with a relatively high level of confidence within well-drilled areas. Confidence in the geological interpretation decreases in areas of reduced sample coverage.

11. Sample Preparation, Analyses and Security

The following section relates to the methods and protocols used by Rakita for Upper and Lower Zones during its exploration campaigns to date.

11.1 Diamond Drilling Sample Preparation and Chain of Custody

All drilling completed prior to 2016 was transported by truck to the core storage facility for logging procedures, including photography and geological and geotechnical logging. Since then, prior to transport to the storage facility, all core has been logged for recovery and Rock Quality Designation at the drill site. A SWIR spectral analyzer, OreXpress TM spectrometer, was used to assist with mineral identification. Sampling intervals were marked using typically 1.5 m to rarely 1.0 m for the upper zone and 3.0 m and occasionally 2.0 m length for the lower zone and the core was subsequently split using a diamond core cutter.

Core is cut parallel to the core axis and the left half of core (when looking downhole) is selected for sampling, with the right half kept in storage for future reference. Sample numbers are subsequently assigned to sample intervals; this process is recorded using Microsoft Excel and then uploaded to Rakita's AcQuire database. Allocated sample numbers are marked on the core tray.

All samples are placed in a calico bag labelled with the sample number. A sample ticket is also placed inside the bag. Samples are stored in a dry, dust free room until they are shipped to the sample preparation laboratory. The typical weights of core samples arePQ3 = 7.0 kg, HQ3 = 3.8 kg and NQ3 = 2.05 kg.

11.2 Sample Preparation and Analysis

11.2.1 *2011 to 2013 Drill Program*

Prior to April 2012, sample preparation was carried out at the Balkan Exploration and Mining (BEM) laboratory in Belgrade. After this date, samples were sent for sample preparation to the Eurotest Control EAD Laboratory in Bulgaria (ETC Bulgaria). ETC was previously the laboratory of the Geological Survey of Bulgaria and then was privatised in 2000. Since September 2011, Rakita has new purpose-built premises that house in one building the entire laboratory and processing procedures as well as management and quality control. ETC Bulgaria has accreditation ISO 17025 for commercial analytical laboratories valid until 31 May 2016 and also ISO 9001 certification for their quality management system.

Sample preparation involved crushing to >95% passing -10 mesh (2 mm) using a jaw crusher prior to a 400-g split being taken and pulverized to better than 90% passing 140 mesh (140 µm) with an LM2 pulverising ring mill.

ETC Bulgaria analyzed the samples for gold by aqua regia digestion with atomic absorption spectrometry (AAS) finish until April 2013. After this date, samples were assayed for gold by fire assay with AAS finish (which showed improved accuracy), with high grade samples re-assayed using gravimetric finish. Copper, arsenic and multi-element analysis for 35 elements were assayed by aqua regia with inductively coupled plasma atomic emission spectroscopy (ICP-AES), with high grade copper (1 to 11%) re-assayed by AAS and very high-grade copper (>11%) re-assayed by ICP-AES using a 0.1 g aliquot.

11.2.2 2014 to 2017 Drill Program

For drilling completed subsequent to 2013, samples were sent for sample preparation to the ALS laboratory located at Bor (ALS Bor) in Serbia. Sample preparation involved crushing to >80% passing -10 mesh (2 mm) prior to a 400-g split being taken and pulverized to >80% passing -200 mesh (-75 µm).

Samples were sent for analysis to the ALS Laboratories in Loughrea, Ireland (ALS Loughrea) and (for gold only, based on a 100-g split) in Rosia Montana, Romania (ALS Rosia Montana). After March 2015, all samples were sent to ALS Loughrea.

Gold was assayed for by fire assay with AAS or ICP-AES; high grade samples >3 g/t gold were re-assayed using gravimetric finish. Copper, arsenic and multi-element analysis for 35 elements were assayed for using aqua regia with ICP-AES or inductively coupled plasma mass spectrometry (ICP-MS). Samples with copper >1% were re-assayed by ICP-AES using a 0.5 g aliquot. After September 2015, the assay digest methodology for copper, arsenic and multi-element analysis was changed from aqua regia to four-acid digest with ICP-MS.

During the 2015 to 2017 sampling programs, multi-element analysis also included total sulphur and sulphur in sulphate. Total sulphur was assayed using a Leco Analyzer, whilst sulphur in sulphate used potassium hydroxide leach. Drillhole samples completed prior to 2015 were sent as pulp rejects to ALS Loughrea to achieve sulphur analysis for these earlier holes.

In 2017 the analytical method for the Lower Zone was changed from ME-MS61 to ME-ICP61.

Both ALS Loughrea and ALS Rosia Montana are ISO 17025 accredited.

11.3 Bulk Density Data – Upper Zone

A density determination was made generally on every three-metre run of drill core. This was completed by weighing a piece of core in air and then determining the core volume by displacement of water.

Core samples selected for density analysis had an average length of 15 cm. The weight of the dry sample was initially determined using bench mounted electronic scales, before being submerged in water and to measure the submerged weight. The following equation was then applied by Rakita to determine the dry density:

Density = weight (in air)/[weight (in air) – weight (in water)]

With the exception of the density sampling completed prior to drillhole TC150071, which has been excluded from the database (as described in Section 12.2), a total of 19,022 measurements were supplied by Rakita. The density samples were coded with the modelled geological and mineralization wireframes; the descriptive statistics for the mineralized domains are provided in Table 11.1.

Table 11.1: Summary of Density per Mineralization Domain

CZONE	No. Samples	Type	Mean	Min	Max
101	292	High grade (UHG)	3.95	2.78	4.56
102	64	High grade (UHG)	3.63	2.40	4.49
103	1483	Massive sulphide	3.43	1.41	4.84
104	2585	Low grade CuCov	3.01	1.43	4.83
202	9	High grade	3.82	3.12	4.25
203	37	Massive sulphide	3.13	2.52	4.07
204	365	Low grade CuCov	2.83	2.01	3.44
303	36	Massive sulphide	3.17	2.36	4.20
304	126	Low grade CuCov	2.84	2.24	4.21

Figure 11.1 shows the relationship between total sulphur (sulphur %) and density for samples within each of the major modelled mineralization domains – ultra high grade (UHG), massive sulphide and low grade covellite. The graphs show that density and sulphur grade show a reasonable correlation. Based on the relationship between sulphur grade and density, the regression formula shown on each graph was used to calculate the dry density for each block in the block model based on its sulphur grades.

SRK notes that Rakita has not modified this approach for porous, weathered or vuggy samples.

In SRK's opinion, best practice for such samples would be to coat with wax or wrap with PVC film prior to immersion in water. While such samples are uncommon for the Project, SRK is satisfied that the method used is appropriate for the majority of samples.

11.4 Bulk Density Data – Lower Zone

A density determination was made generally on every three-metre run of drill core. This was completed by weighing a piece of core in air and then determining the core volume by displacement of water.

Core samples selected for density analysis had an average length of 15 cm. The weight of the dry sample was initially determined using bench mounted electronic scales, before being submerged in water and to measure the submerged weight. The following equation was then applied by Rakita to determine the dry density:

Density = weight (in air)/[weight (in air) – weight (in water)]

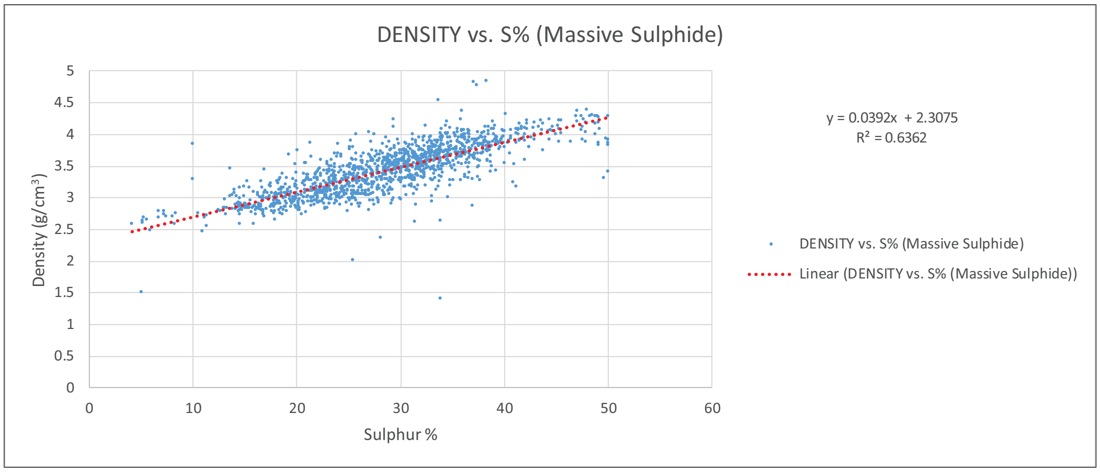
A total of 10525 measurements were supplied by Rakita. The density samples were coded with the modelled geological and mineralization wireframes; the descriptive statistics for the mineralized domains are provided in Table 11.1

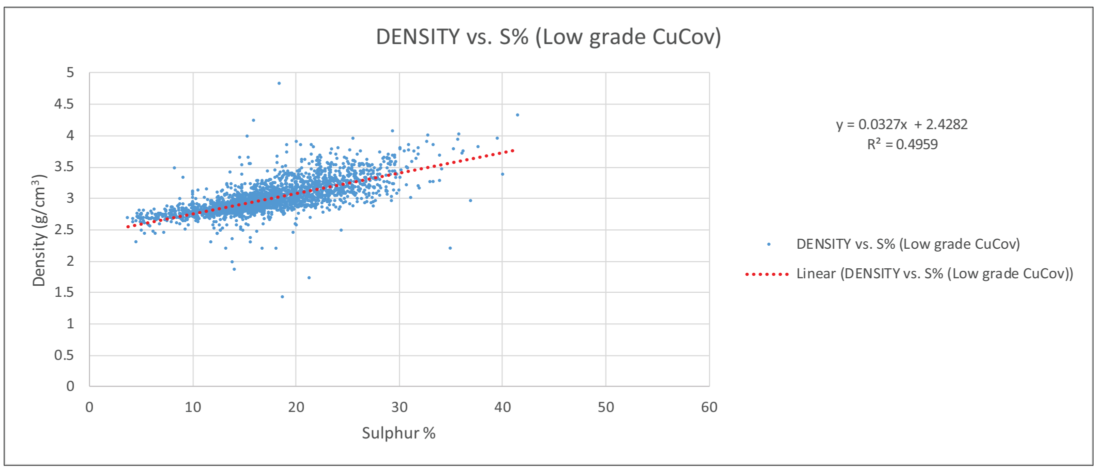
Table 11.2: Summary of Density per Mineralized Domains for the Lower Zone

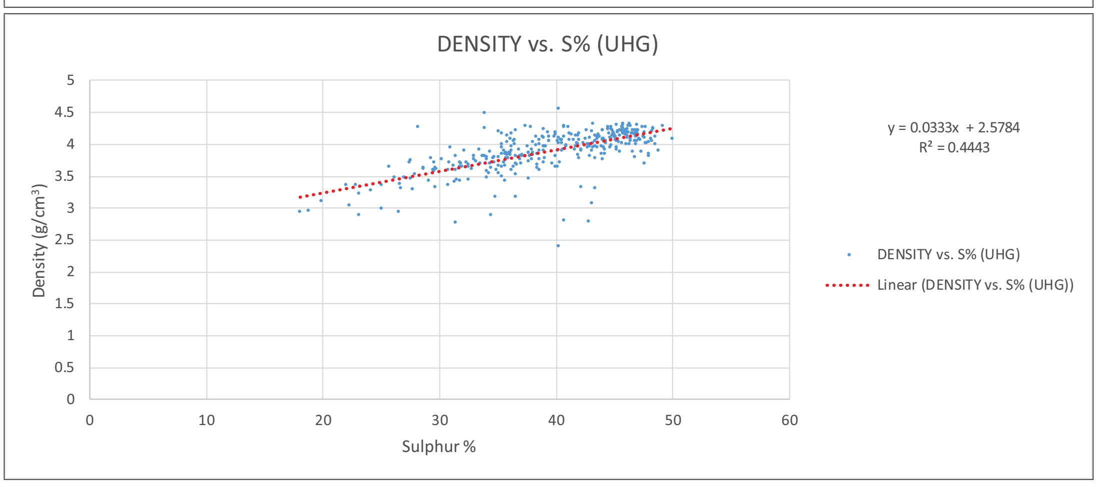
Zone	No. Samples	Type	Mean	Min	Max
201	1181	Low grade Epithermal	2.78	2.17	4.42
202	3459	Low grade Porphyry	2.80	1.43	4.41
501	189	High grade Epithermal	2.78	2.17	3.25
502	5696	High grade Porphyry	2.76	1.07	4.06

11.5 SRK Comments

In SRK’s opinion, the sampling preparation, security and analytical procedures used by Rakita are consistent with generally accepted industry standard practices and are therefore adequate for the purpose of resource estimation.







Source: SRK (UK), 2017

Figure 11.1: Density Regression Plots for UHG (UZ top), Massive Sulphide (UZ middle) and Low Grade CuCov (UZ bottom) Domains

12. Data Verification

12.1 Upper Zone

Rakita completed routine data verification as part of the on-going exploration program. Checks included validation for all tabulated data, including collar and down-hole survey, sampling information, assay and lithology interval data, with validation of sample results from the latest phase of drilling using standards, blanks and duplicate samples inserted routinely into each batch submitted to the laboratory and check assays completed at an umpire laboratory.

A routine quality assurance/quality control (QAQC) program was implemented by Rakita to monitor the ongoing quality of the analytical database. The QAQC system included the submission of blank, standard and duplicate samples in every batch of samples, QAQC samples account for approximately 15% of total laboratory submissions during 2011 to 2016 and 17% starting from 2017.

During the previous March 2015 model update, SRK (UK) noted a slight bias (on average 5%) towards higher grade in the gold certified reference material (CRM) data from ETC Bulgaria (2011 to 2013 drilling campaigns), representing some 38% of the (March 2015) sample database inside the mineralization wireframes. Despite this, the samples were used in the Mineral Resource estimate, with visual and statistical support provided by the sampling at ALS Loughrea, which was considered at the time to sufficiently smooth over this anomaly.

Since then, Rakita has re-assayed the gold data from ETC Bulgaria at ALS Loughrea, where CRM performance for gold shows a relatively good level of accuracy (as described below), therefore removing this slight bias from the assay results for gold.

12.1.1 Standards

Since the start of the drilling at Timok, Rakita has introduced 16 different standards into the analysis sample stream. These were sourced from CDN Resource Laboratories Ltd, Canada, Ore Research & Exploration Pty. Ltd., Australia, and Mineral Exploration Geochemistry, Nevada.

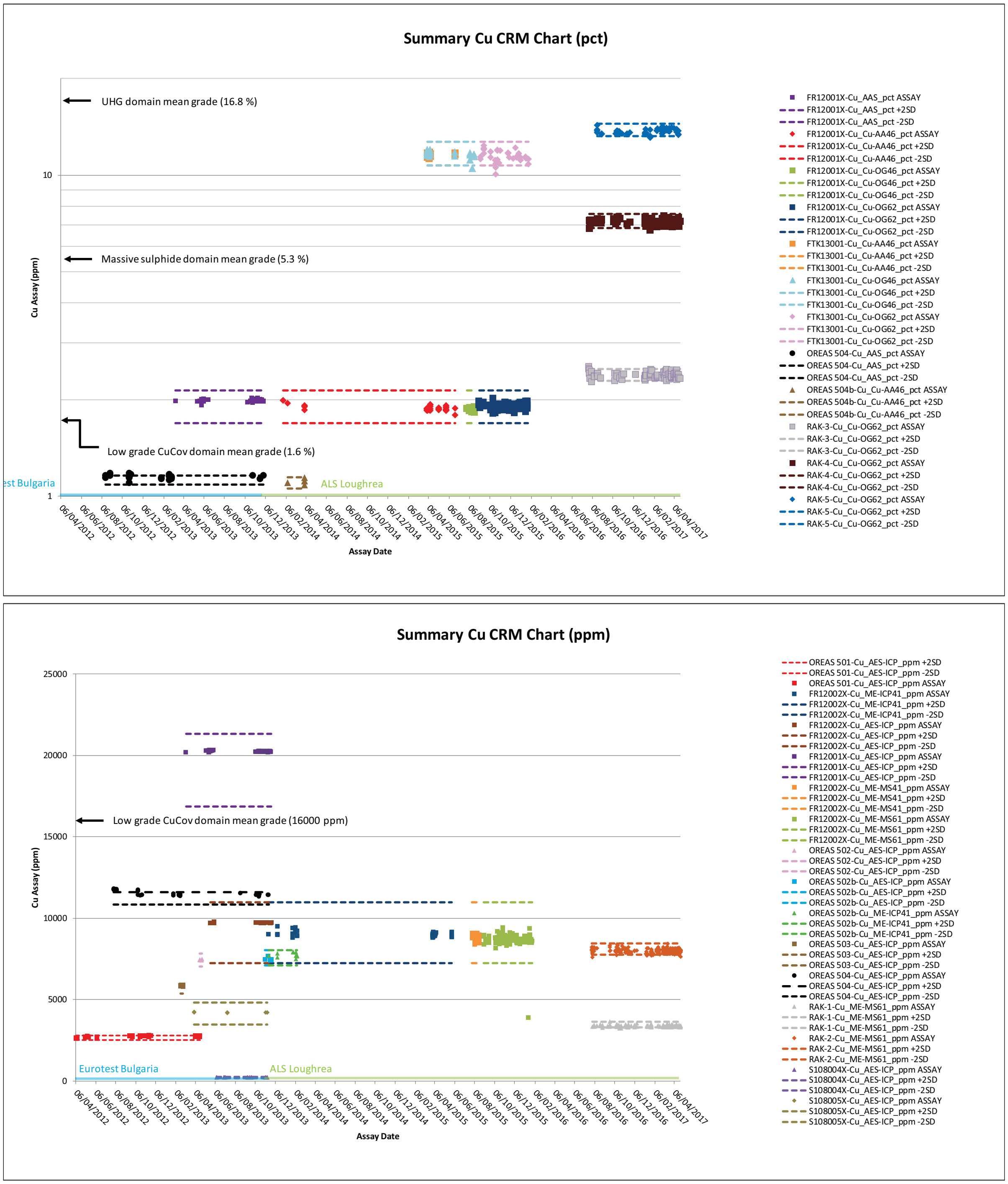
Eight of the standards have been developed based on material sourced from Čukaru Peki (namely the FR, and RAK series standards) to be used in the Upper Zone and the Lower Zone excluding those standards of high copper and gold grade (FTK13001 and RAK5). Certified limits were determined based on external round-robin analysis and historical performance at ETC and ALS when results are within reasonable tolerance (typically 2 to 7%) of the initial round-robin results (Jacks, 2015). The certified values per standard for copper, gold and arsenic are shown in Table 12.1.

SRK (UK) has reviewed the standard results for copper, gold and arsenic and is satisfied in general that (with the exception of a limited number of anomalies and the four-acid digest results for arsenic) they demonstrate a reasonable degree of accuracy at the assaying laboratory. The summary results of the CRM submissions used in the QAQC program to date are illustrated in Figure 12.1 to Figure 12.4. With regards to arsenic, the change in assay methodology from aqua regia to four-acid resulted in the slight under-reporting of higher grades. SRK (UK)'s analysis and rectifying of this issue for arsenic is described in Section 12.2.3.

Table 12.1: Summary of Certified Reference Material for Copper, Gold and Arsenic Submitted by Rakita in Sample Submissions

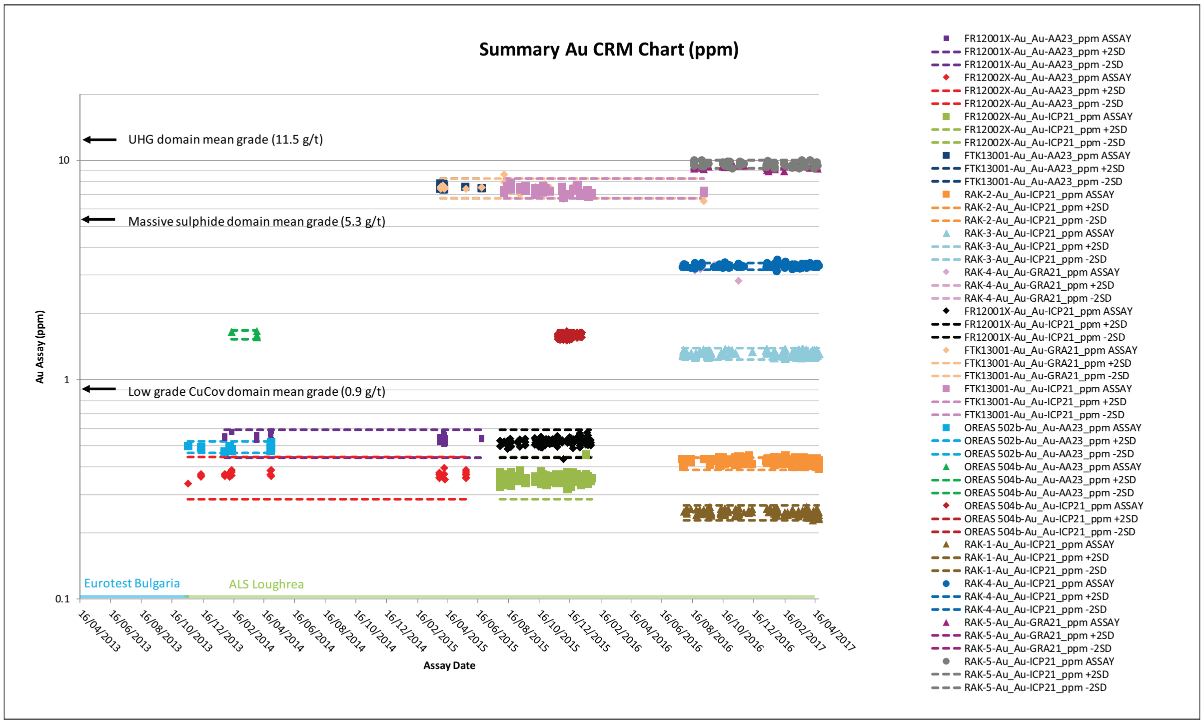
Standard Material	Copper Grade Results (ppm)			
	Certified Value	SD	Company	Certification Type
FTK13001	117,300.00	5,000.0	CDN Mineral Laboratories	Historical average from ALS and ETC Analysis (2013 to 2015), J. Jacks
RAK-1	3,480.00	90.0	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External certification
RAK-2	8,110.00	170.0	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External certification
RAK-3	23,900.00	500.0	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External certification
RAK-4	72,100.00	1,850.0	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External certification
RAK-5	138,400.00	3,100.0	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External certification
OREAS 501	2,670.00	70.0	ORE Research & Exploration Pty Ltd	External certification
OREAS 502	7,430.00	200.0	ORE Research & Exploration Pty Ltd	External certification
OREAS 502b	7,731.45	232.1	ORE Research & Exploration Pty Ltd	External certification
OREAS 503	5,630.00	130.0	ORE Research & Exploration Pty Ltd	External certification
OREAS 504	11,230.00	190.0	ORE Research & Exploration Pty Ltd	External certification
OREAS 504b	11,009.48	222.4	ORE Research & Exploration Pty Ltd	External certification
FR12001X	19,100.00	1,120.0	Shea Clark Smith/MEG Labs	Historical average from ALS and ETC Analysis (2013 to 2015), J. Jacks
FR12002X	9,105.00	932.0	Shea Clark Smith/MEG Labs	Historical average from ALS and ETC Analysis (2013 to 2015), J. Jacks
S108004X	215.00	19.0	Shea Clark Smith/MEG Labs	External certification
S108005X	4,139.00	340.0	Shea Clark Smith/MEG Labs	External certification
Standard Material	**Gold Grade Results (ppm)**			
	Certified Value	**SD**	**Company**	**Certification Type**
FTK13001	7.50	0.38	CDN Mineral Laboratories	Historical average from ALS and ETC Analysis (2013 to 2015), J. Jacks
RAK-1	0.25	0.01	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External Certification
RAK-2	0.42	0.01	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External Certification
RAK-3	1.32	0.04	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External Certification
RAK-4	3.29	0.06	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External Certification
RAK-5	9.60	0.21	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External Certification
OREAS 501	0.19	0.02	ORE Research & Exploration Pty Ltd	External certification
OREAS 502	0.46	0.03	ORE Research & Exploration Pty Ltd	External certification
OREAS 502b	0.49	0.02	ORE Research & Exploration Pty Ltd	External certification
OREAS 504b	1.61	0.04	ORE Research & Exploration Pty Ltd	External certification
FR12001X	0.52	0.04	Shea Clark Smith/MEG Labs	Historical average from ALS and ETC Analysis (2013 to 2015), J. Jacks
FR12002X	0.37	0.04	Shea Clark Smith/MEG Labs	Historical average from ALS and ETC Analysis (2013 to 2015), J. Jacks

Standard Material	Arsenic Grade Results (ppm)			
	Certified Value	SD	Company	Certification Type
RAK-1	175.00	6.50	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External Certification
RAK-2	253.00	8.50	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External Certification
RAK-3	2,209.00	97.00	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; External Certification
RAK-4	5,444.00	451.50	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; Provisional External Certification
RAK-5	2,440.00	199.50	CDN Mineral Laboratories	Smee & Associates Consulting Ltd; Provisional External Certification
FTK13001	2,065.00	140.00	CDN Mineral Laboratories	Historical average from ALS and ETC Analysis (2013 to 2015), J. Jacks
OREAS 501	17.00	1.00	ORE Research & Exploration Pty Ltd	Indicative
OREAS 502	19.50	-	ORE Research & Exploration Pty Ltd	Indicative
OREAS 502b	19.08	3.26	ORE Research & Exploration Pty Ltd	External certification
OREAS 503	18.00	1.00	ORE Research & Exploration Pty Ltd	Indicative
OREAS 504	5.50	1.25	ORE Research & Exploration Pty Ltd	Indicative
OREAS 504b	9.86	0.91	ORE Research & Exploration Pty Ltd	External certification
S108005X	352.00	4.50	Shea Clark Smith/MEG Labs	No indicative or certified value (average from sample data)
FR12001X	1,469.00	70.00	Shea Clark Smith/MEG Labs	Historical average from ALS and ETC Analysis (2013 to 2015), J. Jacks
FR12002X	991.00	58.00	Shea Clark Smith/MEG Labs	Historical average from ALS and ETC Analysis (2013 to 2015), J. Jacks



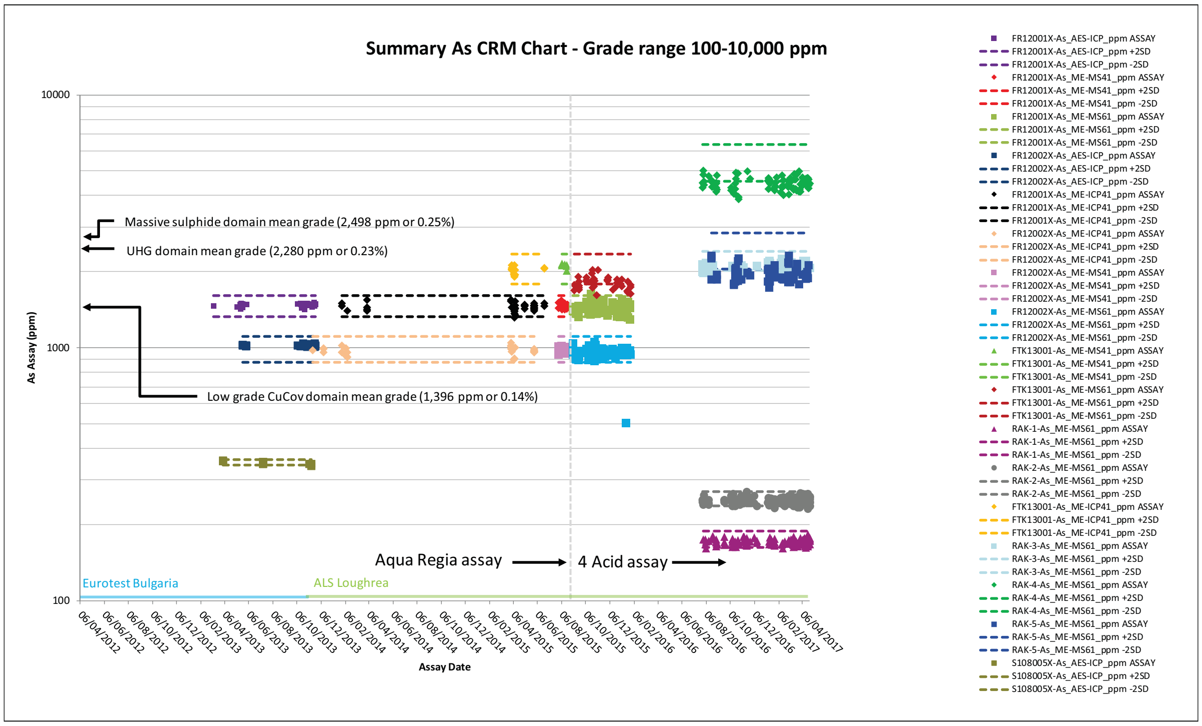
Source: SRK (UK), 2017

Figure 12.1: QAQC Standard Summary Charts for Copper from Submission of Čukaru Peki Samples



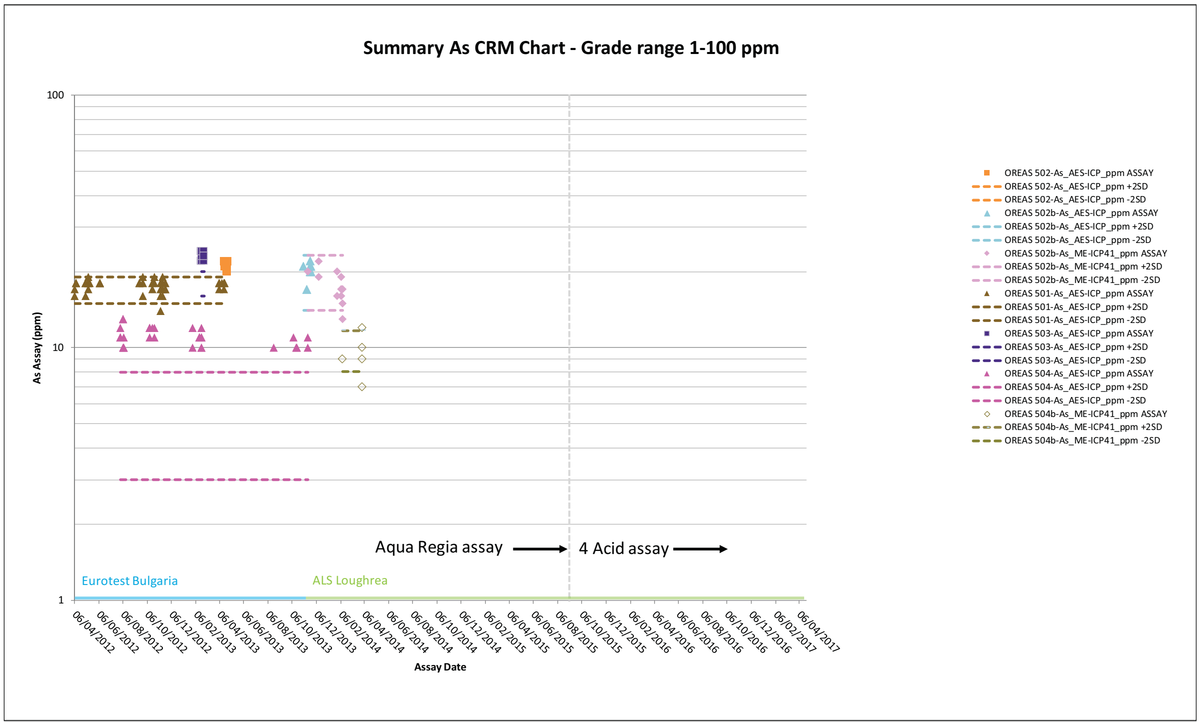
Source: SRK (UK), 2017

Figure 12.2: QAQC Standard Summary Charts for Gold from Submission of Čukaru Peki Samples



Source: SRK (UK), 2017

Figure 12.3: QAQC Standard Summary Charts for Arsenic from Submission of Čukaru Peki Samples; Grade Range 100 ppm to 10,000 ppm



Source: SRK (UK), 2017

Figure 12.4: QAQC Standard Summary Charts for Arsenic from Submission of Čukaru Peki Samples; Grade Range 1 ppm to 100 ppm*

*Standards OREAS504 and OREAS503 have mean values that are close to the lower analytical detection limit and are not certified for arsenic. Therefore, the observed consistent high-reporting of these standards is not considered a significant issue.

SRK (UK) notes that whilst the initial drilling programs (2011 to 2013) were largely supported by relatively low-grade standard material (with mostly provisional results for arsenic), Rakita introduced a number of higher-grade certified standards (FTK13001, RAK-4 and RAK-5) for the more recent infill drilling (2014 to 2017) which are closer to average deposit grades. Furthermore, QAQC completed during 2016 to 2017 is mostly limited to the RAK series standards, which represent the higher-grade range of the Rakita's suite of standard material (0.4-13.8 % Cu, 0.3-9.6 g/t Au and 0.02-0.54 % As).

The results for copper from ETC Bulgaria during early drilling campaigns (2011 to 2013) had a slight bias toward higher grade (on average +3% relative for copper). Results from standard sample submissions to ALS Loughrea have good analytical accuracy for copper and gold with average mean versus assigned grade per certified standard within 1% relative for copper and 0.5% for gold. SRK considers that the more recent samples assayed by ALS Loughrea, which comprise some 85% of the sample database inside mineralization wireframes, sufficiently moderate the 3% bias shown in the copper results from ETC Bulgaria.

For arsenic, with the exception of a small number of anomalous results in the near detection limit (very low grade) standard results from ETC Bulgaria, the results from the certified standards using aqua regia digest and corrected four-acid digest (as described in Section 12.2.3), in general are considered by SRK to have good accuracy.

12.1.2 Blanks

Coarse quartz material sourced from a local sandstone quarry (located some 20 km from Bor) was included as a blank in the sample stream prior to sample preparation. Blank samples were inserted at a rate of approximately 3%.

SRK has reviewed the results from the blank sample analysis and has determined that in general there is little evidence for sample contamination at the preparation facility.

12.1.3 Duplicates

Three types of sample duplicate were inserted into the routine sample stream, namely field duplicates (quarter core "sample duplicates"), coarse duplicates ("crush duplicates") and pulp duplicates. Duplicate samples were inserted at a rate of approximately 8%.

The coarse and pulp sample duplicate assay results for copper, gold and arsenic show a good correlation to the original assays, with a correlation coefficient typically in excess of 0.98, whilst field duplicates display a weaker correlation as typically expected, with a coefficient varying between 0.8 and 1.0. The elevated scatter in the mean grades for the field duplicates is considered to be a reflection of the geological variability and (resultant) inhomogeneous distribution of the mineralization in the drill core.

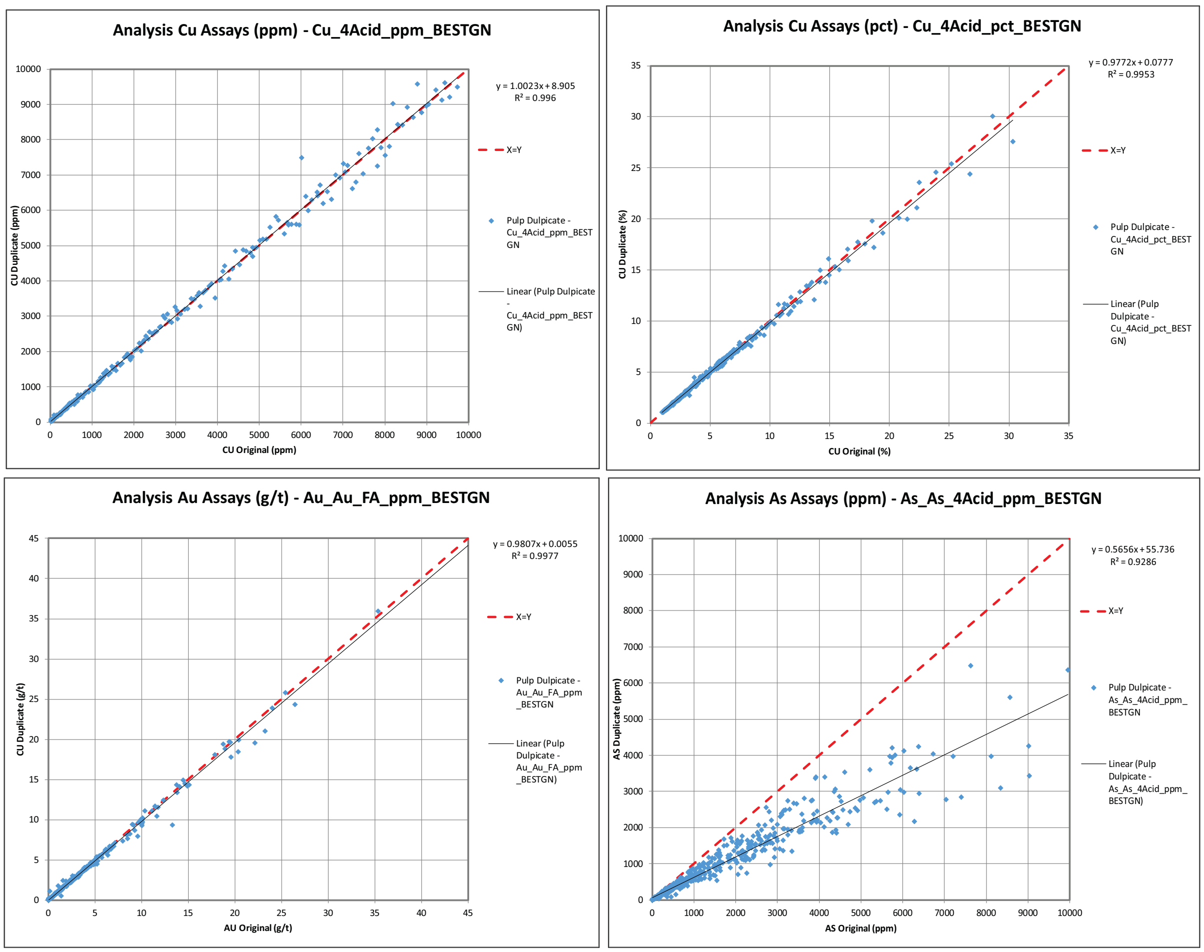
12.1.4 Verifications by Umpire Laboratory

During March 2017, Rakita sent 728 sample splits from original pulp samples from the 2015 to 2016 drilling programs to the Bureau Veritas Minerals (BVM) Laboratory in Vancouver, Canada for check analysis of the primary copper, gold and arsenic results from ALS Loughrea.

In general, the BVM Vancouver results for copper and gold were closely correlated to the ALS Loughrea assays, with correlation coefficients typically in excess of 0.98, with check assay charts for copper, gold and arsenic presented in Figure 12.5.

Analysis for arsenic however showed a relatively poor correlation, with a correlation coefficient of 0.57. There was a low bias at BVM Vancouver (20 to 40% relative) compared to Rakita's standards submissions. Whilst the four-acid digest methodology was used at both laboratories, based on discussions with BVM Vancouver, Rakita and its external geochemical consultant consider the poor correlation between sample results to be due to BVM Vancouver taking the digestion to total (rather than 'near') dryness which results in increased volatilisation of arsenic from the sample, hence introducing variability to the results.

In summary, the check assay work completed suggests that BVM Vancouver validates the analytical results for copper and gold from ALS Loughrea; however, limited only conclusions can be drawn from the arsenic check assays completed using four-acid (total dryness).



Note: Copper (top), gold (bottom left) and arsenic (bottom right); Original = ALS Loughrea, Duplicate = BVM Vancouver

Figure 12.5: Umpire Laboratory Results

12.1.5 Verifications by SRK – Upper Zone

In accordance with NI 43-101 guidelines, SRK has completed several visits to the Project, including:

- Martin Pittuck (QP for the Mineral Resource) - 11 and 12 September 2013.
- Paul Stenhouse (Structural Geologist) from SRK - 21 and 23 October 2013.
- Martin Pittuck, Paul Stenhouse, joint venture technical review - 12 and 14 May 2015.

- Paul Stenhouse, confirmatory logging - October 2015 and September 2016.
- Martin Pittuck, Robert Goddard Mineral Resource data technical review - September 2015, July 2016 and March 2017.

The site visits allowed SRK to review exploration procedures, define geological modelling procedures, examine drill core, inspect the site, interview project personnel and collect relevant information.

Overall, in SRK's opinion, the data supplied is adequate for the purposes described in this report.

12.1.5.1 Verification of Sample Database

SRK completed a phase of data validation on the digital sample database supplied by Rakita, which included the following:

- Search for sample overlaps, duplicate or absent samples, anomalous assay and survey results. No material issues were noted in the final sample database.
- Search for non-sampled drillhole intervals within the mineralised zones. SRK noted that some 4% of the sample database by length within the mineralised zones was not sampled. However, given that these non-sampled intervals relate mainly to superseded twin holes or metallurgical holes; they were then subsequently ignored during the calculation of composites that were eventually used for the grade estimation process. For the non-sampled metallurgical holes, the geology was used as a guide for geological modelling.

If possible, SRK (UK) recommends taking quarter core samples for resource modelling from metallurgical drillhole TC150101 given its central location within the UZ deposit.

12.1.5.2 Density Data Validation

During ongoing verification of the earlier drilling campaigns, Rakita noted an error in the recording of the sample weights used to calculate sample density which affected drillholes completed prior to TC150071, which account for some 30% of the density data inside the mineralization wireframes. The error resulted (on average) in a 6% under-reporting of density for the affected samples. Given the uncertainty in the quality of the density determinations prior to drillhole TC150071 and that the affected data is interspersed with correctly sampled density data, SRK (UK) has excluded the density data from drillholes completed prior to TC150071 from the estimation database.

12.1.5.3 Arsenic Grade Validation

In completing an updated assessment of QAQC CRM results (as presented in Section 12.1.1), SRK (UK) noted an issue with some of the arsenic analyses. Prior to

September 2015, when the arsenic assay method used aqua regia sample digestion, CRM results were acceptable but more recent results based on a four-acid digest method (intended to increase accuracy for copper) were under-reporting particularly for higher grades, as illustrated for the FR12 series and (more notably) the high-grade arsenic CRMs FTK13001, RAK3 and RAK5 in Figure 12.6 below; this affected some 70% of the data in the mineralised domains.

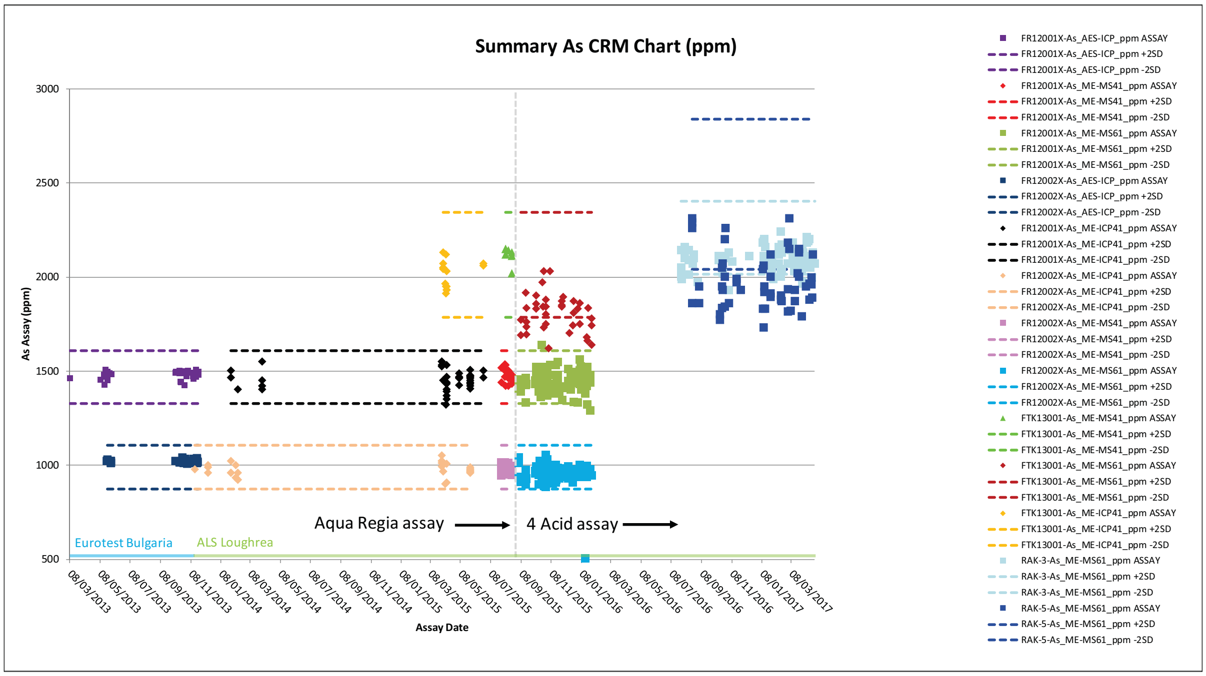


Figure 12.6: Arsenic CRM Results Illustrating the Change from Aqua Regia to Four-Acid Digest

Based on the CRM results, the difference appeared to range from 2% to 20% (relative). SRK (UK) and Rakita understand that this happens due to volatilisation of arsenic in the stronger acid digest, resulting in a systematic error.

To further review and fix the bias, Rakita undertook a dedicated re-assay program to get a regression formula to describe the relationship between four-acid and aqua regia digest assays. A total of 102 pulp samples with 12 CRM's that had been assayed using four-acid digest (and a small number of blanks and duplicate samples) were submitted to ALS Loughrea for check assay using aqua regia digest.

The results were plotted on a scatter chart (Figure 12.7) using averaged data in 1,000 ppm bins to remove scatter and better assess bias. A trend line was plotted on the chart to cover the range of arsenic grades representative of the sample composites in the UZ, namely

0 to 20,000 ppm. One sample that returned less than analytical detection was excluded from the analysis.

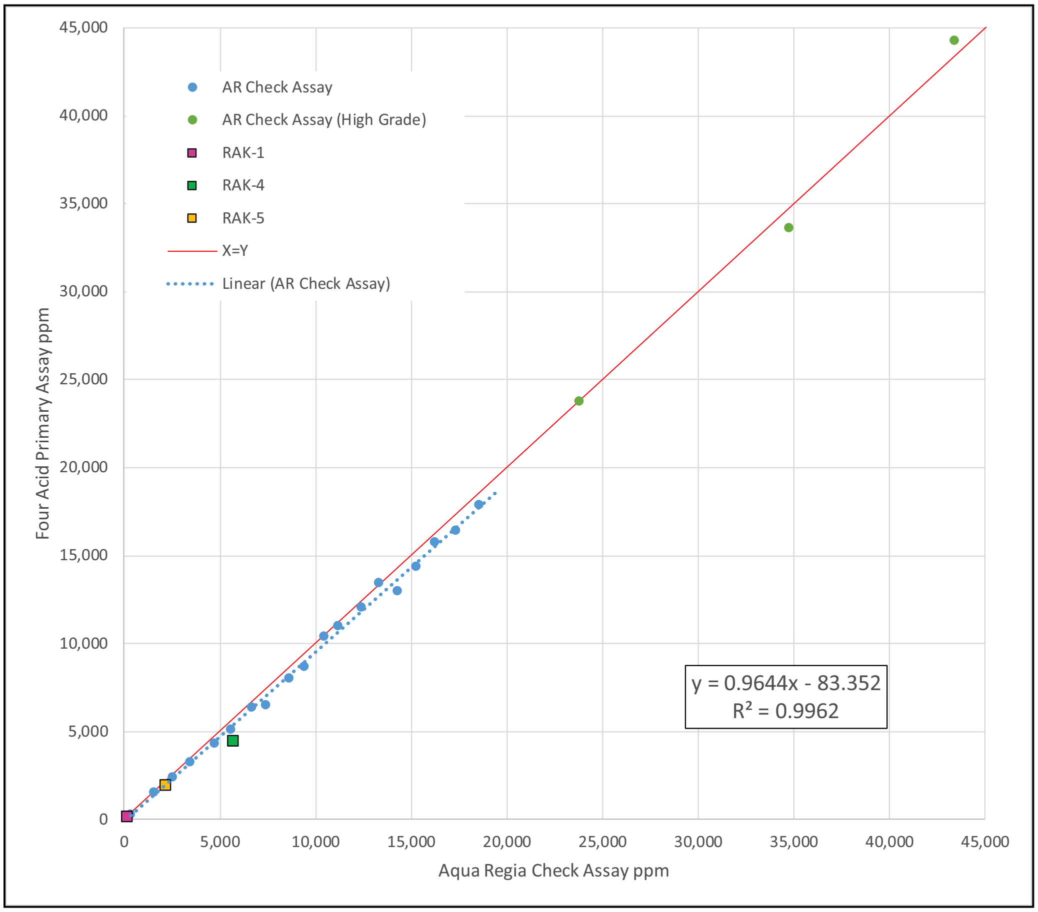


Figure 12.7: Scatter Plot for Arsenic ppm Samples Analyzed by Four-Acid and Aqua Regia

The results of the check assay suggest that whist the bias does not appear to be as significant as originally suggested by the CRM results, there is relatively 4% more arsenic when using aqua regia compared with four-acid assays. SRK (UK) therefore applied a regression formula to the four-acid sample data (as outlined below) in the assay database, based on the relationship from the graph shown in Figure 12.7.

As ppm Corrected = [As ppm from four-acid + 83.4]/0.96

12.1.5.4 SRK Comments

SRK has reviewed the data collection methodologies during several site visits and has undertaken an extensive review of the assay and geology database during the mineral resource estimation procedure.

With the exception of arsenic, for which there were issues associated with four-acid digest identified and rectified, assessment of the available QAQC data indicates the assay data for the drilling and sampling to date is appropriately accurate and precise.

With regards to arsenic analysis, given the tendency in the four-acid data for under-reporting particularly for higher grades and the sensitivity of the results to four-acid digest protocol (as illustrated by the check analysis at BVM Vancouver), SRK recommends using aqua regia for analytical digest during all future drilling campaigns. SRK also recommends further investigation into the significant (circa 20%) variance shown for arsenic in CRM RAK4 and RAK5, possibly by additional round-robin analysis and mineralogical study to determine whether arsenic in these Čukaru Peki-derived CRMs are more likely to undergo volatilization compared with more typical material from the deposit.

In addition, SRK recommends sourcing an additional certified standard whose copper and gold grade covers the top-end of the higher range (1 g/t Au to 15 g/t Au and 2% Cu to 18% Cu) to further add to the confidence in laboratory accuracy for samples in the high grade (UHG) domain of the UZ deposit.

The QAQC CRMs used between 2011 and 2014 were limited to relatively low-grade values. In the UHG domain, these represent 27% of the samples and they are well supported by and have a similar grade distribution to the 73% of the data in the domain for which QAQC included high grade CRMs (i.e. Cu >10% and As >2,000 ppm). Despite this, SRK has a high overall confidence in the block tonnage and grade estimates in the well-drilled parts of the UHG and other domains, sufficient for measured mineral resources; but for completeness, SRK recommends selecting 5% to 10% of the 2011 to 2014 sample pulps from the UHG domain and re-submitting these with the current QAQC standards to provide a retrospective validation of the arsenic and high-grade copper assays reported at that time.

12.2 Lower Zone

During May 2018 Rakita sent 837 pulp samples to Bureau Veritas Minerals (BVM) Laboratory in Vancouver, Canada for check analysis of 2017-2018 LZ drilling program. One UZ drill hole, TC150101, which was a met hole additionally logged and sampled in SGS, Lakefield, also included in the list. Approximately 10% of the drill samples from 2017-2018 drilling programs were analysed using comparable analytical methods in order to validate previous analyses. Drill samples from the 2017-2018 drilling programs were originally analysed at ALS Global, Ireland.

Gold was analysed using a 30-gram fire assay digestion with an ICP finish. Gold results exceeding 9 grams per tonne were reanalysed using a 30-gram fire assay digestion with a gravimetric finish. Samples with Copper <1% were analysed using a 0.1 gram four-acid digestion with an ICP-AES finish. Samples with over 1% Cu were re-assayed using a .0.5 g four-acid digestion with an ICP finish. The comparable analytical method codes are listed for each laboratory in Table 12.2.

The samples were prepared using splits from the original pulp samples (~100g each). These were homogenized, re-numbered and submitted in eight submittal batches to BVM in Vancouver during May, 2018. BVM released the results from the re-analysis in eight certificates.

The batches contained 66 standards, 22 blanks and 50 pulp duplicates embedded with the samples and were blinded to the analytical laboratory.

Table 12.2: Analytical methods used for the primary and check analysis program

Analyte	Analytical Method Description	ALS	BVM
Au < 9 gpt	30 g Fire Assay with AA Finish (ppm)	Au-ICP21	Au-FA330
Au > 9 gpt	30 g Fire Assay with Gravimetric Finish (ppm)	Au-GRA21	Au-FA530
Cu 1-10,000 ppm	Four acid digestion ICP Finish (ppm)	Cu-ME-MS61	Cu-MA200
Cu > 10 %	Four acid digestion ICP Finish (pct)	Cu-OG62 (1-40%)	Cu-MA370 For (Cu 1-10%); Cu-MA404 for (Cu>10%)

12.2.1 *Standards*

Rakita notes that all standards used in Lower Zone drilling programs from 2012 to 2018 show accuracy within acceptable limits. During these programs none of the high-grade copper and gold standards were used (FTK13001 and RAK5).

Only few of standard samples have values outside ± 3SD which not affect the overall summary statistics.

- RAK-1: Cu assays showed acceptable accuracy. Bias is -3.54% and relative standard deviation 2.11%
- RAK-1: One of 22 CRM's returned assay value for gold greater than Mean+3SD. Control samples before and after failure showed acceptable assays tolerance. Assay for this sample could be classified as random failure
- RAK-1: As assays showed 28.2% lower bias, which is, most probably, caused by different analytical protocols within labs
- RAK-3: Cu assays returned with acceptable accuracy. Bias is 1.29%, relative standard deviation is 1.61%

- RAK-3: Au assays returned with acceptable accuracy. Bias is -1.04% and relative standard deviation is 2.51%
- RAK-3: As assays showed 24.4% lower bias, most probably caused by different analytical protocol within labs
- RAK-4: Cu analyses showed acceptable accuracy. Bias is -0.31% and relative standard deviation is 2.27%
- RAK-4: Au assays returned three analytical failures lower than Mean-3SD. Those samples are not consecutive in the sequence and returned assays are within -10% of the Mean. Bias is -2.44%, relative standard deviation is 2.65%
- RAK-4: As assays showed 26.8% lower bias, most probably caused by different analytical protocol within labs

12.2.2 Blanks

Coarse quartz material sourced from a local sandstone quarry (located some 20 km from Bor) was included as a blank in the sample stream prior to sample preparation. Blank samples were inserted at a rate of approximately 3%.

Fourteen (of 22) blank samples returned assay values > 15 ppm Cu. Pulp blank samples were submitted for analysis, but all of the samples were re-pulverized after shipment, so the low level of Cu present could have been introduced during the re-homogenization.

Rakita has reviewed the results from the blank sample analysis and has determined that in general there is little evidence for sample contamination at the preparation facility.

Blank material is regularly analyzed prior every drilling campaign.

12.2.3 Duplicates

The check analysis program conducted at BVM was set up to compare the analytical methods used in ALS laboratory. The check analysis validates the original analytical program if the inter-laboratory bias is less than 10% and more than 80% of the check analyses are within 10% of the original analytical value.

BVM copper analyses validate earlier Cu analyses conducted at ALS. The BVM and ALS Cu analyses using four acid digestions are directly comparable. All of these copper analyses are acceptable for resource evaluation.

BVM lower-grade gold analyses (Au < 9 ppm) (do not) validate earlier ALS gold analyses using the fire-assay with ICP finish analytical method.

For higher-grade Au analyses (Au > 9 ppm) using fire-assay with a gravimetric finish, BVM assays validate earlier ALS gold analyses using the fire-assay with gravimetric finishes.

BVM arsenic analyses do not validate earlier As analyses conducted at ALS. The BVM and ALS analyses using a 4-acid digestion are not comparable as there is a significant bias between the analytical results obtained from the two laboratories. The As difference is caused by a difference in protocol between the two labs. ALS takes the four-acid digestions to near dryness, which means that there is still fluid left before they bring the sample back up with HCl and the As is retained in the fluid. BVM takes the sample to total dryness and then brings the sample back up with HCl. That results in the volatilization of As, which causes an almost 41% lower difference between As analyses by both laboratories.

12.2.4 Check analyses Au re-assay

From June to October 2017 a total of 4844 pulp samples including 12% of QAQC control material, from Lower Zone drilling program originally analyzed by Eurotest Laboratory Bulgaria between 2012 to 2014, were sent for gold reanalysis to ALS laboratory in Ireland

During the Au re-assay program in ALS there were no standard failures. Inter-lab bias was concluded to be 34%. All the results after Au re-assay in ALS imported as Priority 1 into the Database.

12.2.5 Verifications by SRK – Lower Zone

In accordance with NI 43-101 guidelines, SRK has completed a visit to the Project, for the preparation of the Lower Zone resource estimate:

- Gilles Arseneau (QP for the Mineral Resource) – 29 May to June 1, 2018.

The site visit allowed SRK to review exploration procedures, define geological modelling procedures, examine drill core, inspect the site, interview project personnel and collect relevant information.

Overall, in SRK’s opinion, the data supplied is adequate for the purposes described in this report.

12.2.5.1 Verification of Sample Database

SRK completed a review of the Lower Zone digital sample database supplied by Rakita, which included the following:

- Search for sample overlaps, duplicate or absent samples, anomalous assay and survey results. No material issues were noted in the final sample database.
- A review of the digital data against original assay certificates provided by ALS Global, the independent laboratory responsible for assaying drill core at Čukaru Peki. No significant errors were noted in the Lower Zone assay records provided by Rakita.
- Collected four samples of drill core representative of the Lower Zone mineralization to confirm the presence of copper and gold mineralization. The samples collected by SRK

returned similar values to the original assays reported by Rakita. Table 12.3 compares the SRK assay results with the original data.

Table 12.3: Comparison of SRK Check Samples With Rakita Original Assays

Sample Number	SRK Cu%	Rakita Cu%	SRK Au (g/t)	Rakita Au (g/t)
I951078	1.22	1.32	0.21	0.19
I951079	0.46	0.63	0.16	0.17
I951080	3.30	2.87	0.29	0.22
I951081	1.00	1.02	0.17	0.13

The results of the SRK samples agreed very well with the original assay data provided by Rakita. The sample sites were selected by SRK at random but to reflect typical Lower Zone style of mineralization. The sampling was only intended to confirm that copper and gold mineralization could be duplicated in the ranges that had been reported by Rakita and it was successful in doing that.

13. Mineral Processing and Metallurgical Testing

13.1 Metallurgical Test Work – Upper Zone

13.1.1 Summary

As part of the PEA published in 2016 (SRK UK, 2016), preliminary testing on samples from the Upper Zone, was conducted between November 2015 and March 2016 by SGS Canada Inc. (SGS, 2016). The scope of this first test program consisted of sample preparation, head sample characterization, mineralogical examination, grindability, flotation, and cyanidation testing. The main objectives were assessing the mineralogical characteristics and evaluating the flotation response of seven composite samples. The scoping level program was considered successful in obtaining reasonable copper recoveries and concentrate grades and in splitting the mineable resource into its four main components, covellite, enargite, pyrite and gangue.

The optimized flowsheet, developed during the 2016 PEA used conventional reagents, applicable to all process feed types and achieves good separation of the copper minerals from pyrite and gangue into a saleable bulk copper concentrate and a pyrite concentrate slurry stream. The pyrite concentrate was to be stored in a dedicated facility for possible later treatment, but for the purposes of that study it was considered a waste stream.

During the course of the PFS study (PFS, 2018), a second test program was completed on samples from the Upper Zone between September 2016 and September 2017 (SGS, 2017a). The emphasis of the test work was to optimize the flotation conditions established in the 2016 PEA and to provide samples to evaluate processing options (as opportunities in the study) for the high arsenic copper concentrate option, referred to as the complex copper concentrate, and the pyrite concentrate.

Later, in a third program, the scope was increased to include: mineralogy, comminution test work, process feed aging test work, bulk flotation test work, further flotation optimization and variability testing, solid-liquid separation testing, and environmental characterization.

A goal of testing for the 2017 study was to develop and optimize the flowsheet selected in the 2016 PEA, which produced two copper concentrates, a low arsenic and a complex concentrate. However, during the variability testing program of this study, it was realized that a proportion of the orebody was not likely to respond well to this two-concentrate production scenario. Testing, and analysis of a flowsheet producing a single bulk concentrate was then carried out, and in conjunction with Nevsun's marketing consultants, a decision was made to change to this simpler, more robust single concentrate approach.

Subsequent to the completion of the 2017 revised PEA it was also determined that, since there was no economically viable proven method for recovering the gold from the Timok pyrite concentrate, no pyrite concentrate would be produced or separately stored. Should a method be proven later then the whole tailings, including the contained pyrite, would be retreated to recover the pyrite, potentially using the copper flotation circuit which, by that time, would have ceased operation.

The optimized flowsheet, developed during the current study uses conventional reagents, applicable to all process feed types and achieves good separation of the copper minerals from pyrite and gangue into a bulk copper concentrate. The pyrite remains in the flotation tailings and is stored at site in the tailings facility for possible later treatment.

The process specifies a moderate primary grind P_{80} of 108 µm. The subsequent flotation process is as follows:

1. A bulk copper concentrate is produced using lime and a collector. Lime is added in the roughers to depress pyrite and allow a high recovery of copper minerals.
2. Following a regrind of the rougher copper concentrate to P_{80} of between 15 and 28 µm and further lime addition, one or two stages of cleaning are used.

During the 2017 PEA once the initial optimization work and the comminution test work were completed, the data were passed to Orway Mineral Consultants, in Mississauga, Ontario to complete a comminution plant sizing study. Their report and the SGS flotation and other results were passed to Ausenco Engineering in Toronto to produce design criteria, flowsheets, layouts and capital and operating cost estimates for a grinding and flotation plant to treat plant feed from the Upper Zone for the PEA design.

All test work results and study reports from the 2017 PEA were subsequently passed to Hatch for further optimization, cost estimation, execution planning and completion of the PFS for Upper Zone (PFS, 2018).

In addition four trade-off studies (ToS's, #1, 2, 3 and 4) were completed as part of the 2017 PEA to define options regarding:

1. Concentrate sale, bulk vs. separate high and low arsenic concentrates.
2. Process options for gold recovery from pyrite.
3. Process options for reduction of arsenic in the complex concentrate.
4. Definition of concentrate transportation considerations.

As part of ToS #2, samples of the pyrite concentrate were tested to determine the applicability of certain gold recovery processes, i.e. pyrite roasting (Outotec) and atmospheric oxidation following fine grinding (Albion).

The results of the tests from ToS #2 and preliminary reports from the respective process suppliers were reviewed by Ausenco Engineering of Brisbane who prepared preliminary scoping level, capital and operating cost estimates to assist in determining if the processes would be economic for future consideration. This work is outside the scope of this study.

As part of ToS #3, samples of the complex copper concentrate were tested for various arsenic removal processes, i.e. partial reductive roasting (Outotec), ferric oxidation (FLSmidth ROL®), and caustic leaching (Toowong). Each process supplier compiled a preliminary report summarizing its potential application.

13.1.2 Sample Description

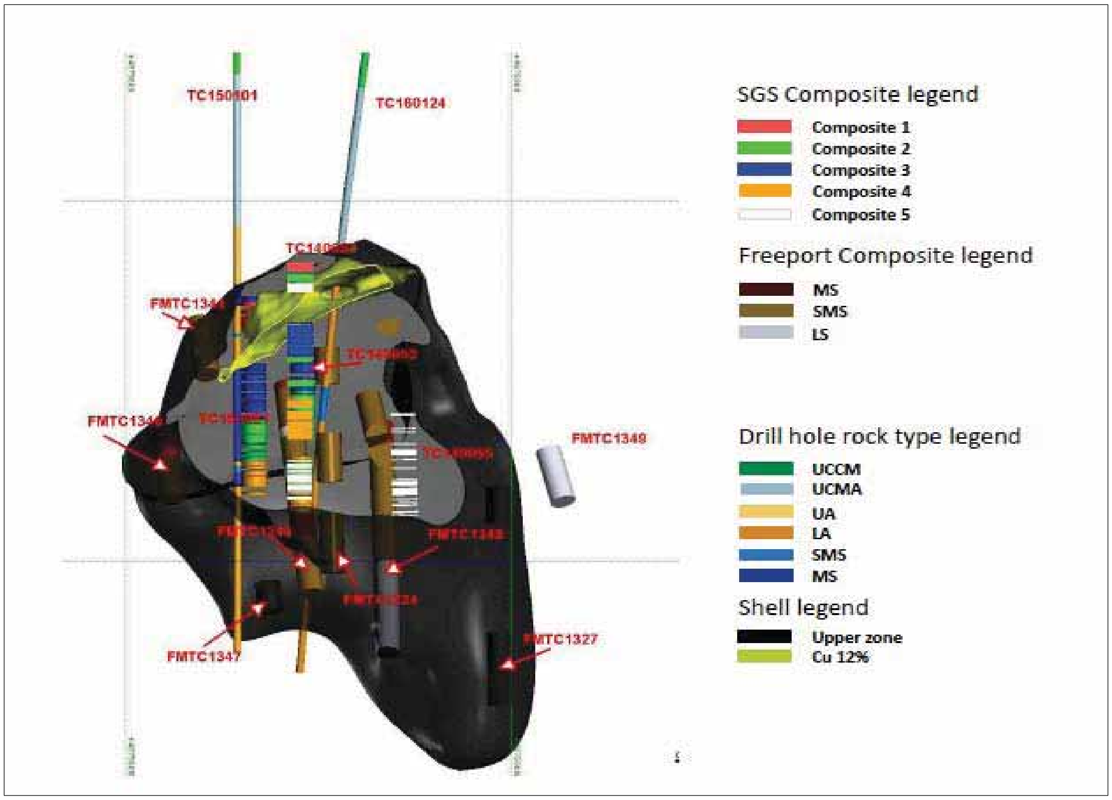
Samples for metallurgical test work were obtained from successive rounds of drilling as the project was developed. An overview of the samples is presented in Figure 13.1, which shows a section through the deposit and identifies the metallurgical sample locations.

13.1.2.1 2016 PEA Samples

For the 2016 PEA, Reservoir Minerals prepared seven composite samples to represent the major sulphide mineralogies and zones within the deposit, as understood at that time.

Five composite samples representing the Upper Zone massive and semi-massive sulphide copper mineralization were prepared from intercepts selected from five diamond drill holes at depths ranging from 446 to 640 m down the hole from surface. Two additional composite samples representing the Lower Zone porphyry mineralization were also prepared.

The samples and test results are described in the SGS report (SGS, 2016).



Source: SRK, 2017

Figure 13.1: Metallurgical Sample Locations

13.1.2.2 Flotation Optimization Composite Samples

To build on the 2016 PEA metallurgical program, a flowsheet optimization program was conducted at SGS to improve and characterize mineral responses under varying test conditions. Quantities of concentrates were also required for evaluating potential treatment methods for the copper concentrate for arsenic removal and for gold recovery from the pyrite concentrate.

For scheduling reasons, and to ensure sufficient quantity of sample was on hand to support the test programs, the relevant 2016 PEA composites were combined to produce two master composites. The first master composite sample (MC) was prepared by blending equal weights (60 kg each) of Composites 2, 3, 4 and 5 from the 2016 PEA program (SGS 2017a).

Subsequently, when additional material was required to produce copper concentrates for further work, a second composite MC1, was produced by blending the balance of the 2016 PEA composites with unused MC samples from the optimization program. Head analyses of these composites are presented in Table 13.1.

Table 13.1: Composites Used in Optimization Tests

Composite Sample	Cu, %	As, %	Au, g/t	S_{tot}, %	Cu/As
MC – Master Composite	4.15	0.30	2.88	22.6	13.8
MC1 – Master Composite 1	3.86	0.29	2.78	22.6	13.3

13.1.2.3 Comminution and Variability Samples

Two “metallurgical holes”, TC150101 and TC160124, were drilled to provide samples for comminution and flotation variability tests. The holes were located close to the initial holes that provided the 2016 PEA composites. Both holes were shipped to SGS.

Hole TC160124 was divided into test samples by depth through the entire mineralized zone resulting in 20 variability samples (SGS 2017b).

Hole TC150101 is held in storage at SGS for additional work during the next stage of the project.

13.1.2.4 Variability Samples - Sub-Level Caving Modelling

To model the latest mine plan, samples from drillhole TC150108 were used in metallurgical tests that model the ore to be produced from the SLC mining method. This mining method is characterized by:

- Dilution by waste derived from above the mining zone (25% dilution in Year 1, 15% in subsequent years).
- Mixing of mineable resource from upper zones into current mine production. For simplicity, this has been approximated by blending 50% “current” mineable resource with 25% mineable resource from each of the two previous production periods, prior to dilution with waste.
- Samples were prepared by combining crushed one-metre sample intervals over successive 20 m lengths through the mineralized zone, see Table 13.2 and Table 13.3.

Table 13.2: Samples Var 1 to Var 20

Sample ID	Hole TC160124A		Assays				Note
	From, m	To, m	Au,g/t	Cu,%	As,%	S,%	
VAR-1	264.5	312.5	<0.02	0.01	<0.001	0.1	marl
VAR-2	312.5	460.3	<0.02	<0.01	<0.001	0.08	andesite
VAR-3	460.3	470.5	0.68	1.13	0.027	6.44	top of orebody
VAR-4	470.5	476.5	0.69	1.42	0.021	14.4	
VAR-5	482.5	487	7.28	5.95	0.16	25.6	
VAR-6	487	499.8	8.13	17.3	0.17	42.8	
VAR-7	502.8	513.3	7.83	10.5	0.63	47.3	poor response -Group 3
VAR-8	513.3	525.3	4.73	6.61	0.66	41	poor response -Group 3
VAR-9	525.8	535.8	2.36	4.21	0.27	36.5	poor response -Group 3
VAR-10	535.8	544.8	3.07	4.96	0.47	43.7	poor response -Group 3
VAR-11	553.8	564.3	2.18	5.84	0.3	25.7	
VAR-12	564.3	576.3	1.92	5.42	0.14	25.8	
VAR-13	576.3	588.3	2.27	6.76	0.24	29.8	
VAR-14	588.3	597.3	3.47	9.82	0.41	34.5	
VAR-15	600.3	610.8	1.18	2.32	0.2	23	
VAR-16	615.3	627.3	1.6	2.53	0.37	23.6	
VAR-17	627.3	639.3	0.96	2.27	0.18	17.5	
VAR-18	651.3	664.8	0.48	2.08	0.12	18.4	
VAR-19	664.8	675.3	0.3	0.83	0.01	10.6	
VAR-20	679.8	690.3	0.27	0.25	0.01	7.16	exiting orebody

Table 13.3: Samples Var 21 to Var 35

Sample No	TC150108		Mine RL		Length of Intercept	Sample Wt, (kg)	Analysis of Test Samples			
	from	to	from	to			Cu, %	Au, g/t	As, %	$S^{=}$, %
Var 21	434	455	-39	-60	21	41	11.800	9.14	0.050	27.60
Var 22	455	475	-60	-80	20	37	12.500	5.79	0.160	27.00
Var 23	475	495	-80	-100	20	43	3.930	3.97	0.320	36.70
Var 24	495	515	-100	-120	20	39	3.100	2.92	0.280	30.10
Var 25	515	535	-120	-140	20	40	4.220	2.67	0.350	24.50
Var 26	535	555	-140	-160	20	41	4.220	3.07	0.210	22.60
Var 27	555	575	-160	-180	20	41	1.690	1.82	0.120	17.50
Var 28	575	595	-180	-200	20	39	1.840	1.40	0.120	17.90
Var 29	595	615	-200	-220	20	40	1.420	1.09	0.270	15.20
Var 30	615	635	-220	-240	20	39	2.090	1.16	0.250	17.40
Var 31	635	655	-240	-260	20	38	1.950	1.02	0.180	14.60
Var 32	655	675	-260	-280	20	38	1.760	0.87	0.110	12.50
Var 33	675	695	-280	-300	20	38	1.620	0.62	0.040	11.90
Var 34	374	404	21	-9	30	na	0.028	<0.02	<0.001	0.37
Var 35	404	434	-9	-39	30	na	0.016	<0.02	<0.001	0.64

Waste material extending for 60 m directly above the deposit was combined into two 30m lengths representing marl (Var 34) and upper andesite (Var 35). The variability samples were then blended, split, and combined into yearly blends representing annual mine production, as shown in Table 13.4.

Table 13.4: Blending Procedure for Yearly Blends

Yearly Blend Sample Preparation (kg)								
Sample	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Var 21	8.0	7.0	-	-	-	-	-	-
Var 22	8.0	5.0	6.0	-	-	-	-	-
Var 23	8.0	5.0	6.0	-	-	-	-	-
Var 24	-	5.0	4.0	6.0	-	-	-	-
Var 25	-	5.0	4.0	6.0	-	-	-	-
Var 26	-	-	4.0	5.0	-	-	-	-
Var 27	-	-	4.0	5.0	7.0	-	-	-
Var 28	-	-	-	5.0	7.0	7.0	-	-
Var 29	-	-	-	-	7.0	7.0	7.0	7.0
Var 30	-	-	-	-	6.0	7.0	7.0	7.0
Var 31	-	-	-	-	-	6.0	7.0	4.0
Var 32	-	-	-	-	-	-	6.0	4.0
Var 33	-	-	-	-	-	-	-	4.0
Var 34	4.0	2.5	2.5	2.5	2.5	2.5	2.5	2.5
Var 35	4.0	2.5	2.5	2.5	2.5	2.5	2.5	2.5
Total wt	32.0	32.0	33.0	32.0	32.0	32.0	32.0	31.0

Head analyses of the Year 1 blend through Year 8 blend are shown in Table 13.5.

Table 13.5: Yearly Blends

Sample	Analysis of Yearly Blend Samples			
	Au, g/t	Cu, %	As, %	$S^{=}$, %
Year 1	4.70	6.98	0.130	22.40
Year 2	4.33	6.27	0.190	23.80
Year 3	3.02	4.58	0.200	22.50
Year 4	1.97	2.54	0.180	18.40
Year 5	0.94	1.47	0.150	13.90
Year 6	0.96	1.49	0.170	13.10
Year 7	0.80	1.57	0.170	12.00
Year 8	0.78	1.46	0.150	11.70
Year 1 Waste	<0.02	<0.01	<0.001	0.39
Year 2-8 Waste	<0.02	0.01	<0.001	0.43

13.1.2.5 Other Samples

13.1.2.5.1 Freeport McMoRan

Additional metallurgical samples were also available from drilling conducted by Freeport-McMoRan when they were joint venture partners in the Timok project with the previous owners, Reservoir Minerals. The remaining Freeport samples, 2632 kg, were shipped to SGS Lakefield. However, the samples were not considered suitable for optimization test work as they had all been previously crushed to -10 mesh (-1.70 mm) and their storage history was unknown, leading to a risk of partial oxidation. It was considered that the samples could be suitable for generating the large weights of concentrates required for downstream testing.

The Freeport samples were composited into level composites, and a 150-kg sample of TM600 was used to produce samples of complex copper concentrate for pyrometallurgical testing by Outotec.

13.1.2.5.2 MMI Program

A second variability test program has been completed by the Institute for Mining and Metallurgy (MMI) in Bor, Serbia. This program involved a total of 50 samples, 10 from each of five diamond drillholes. The selected holes were chosen because they are outside the areas previously sampled for metallurgical work.

Samples were selected in continuous runs to cover the range of copper grades from approximately 1 to 10%. No attempt was made to adjust the levels of gold, arsenic or sulphur in the selected samples.

13.1.3 Mineralogy

The early composite samples tested by SGS, (SGS, 2016) ranged from very high grade massive sulfides in the upper levels of the deposit, through semi-massive sulfides in the mid zones, down to low grade vein and disseminated sulfide mineralization at depth.

Each composite was characterized by quantitative XRD and QemScan in Particle Mineral Analysis (PMA) mode. Covellite, CuS, was identified as the main copper mineral accounting for up to 80% of the copper in the Upper Zone samples; enargite, Cu_3AsS_4 is the second copper mineral and accounts for 5 -22% of the total copper. Bornite and chalcopyrite are also present in minor amounts.

Pyrite is the dominant sulfide mineral. The main non-sulfide gangue minerals were silica and alunite while clays/micas account for around 5% of the lowest grade samples at the base of the deposit.

At a grind size of 100% passing 150 microns the covellite was well liberated with 82 – 65% categorized as free or liberated (particles with >80% mineral surface exposure). Enargite was only poorly liberated at this size with only 56-79% of particles characterized as free and liberated. Enargite occurred primarily in binary grains with pyrite or with covellite.

The arsenic levels in these early composites averaged 0.31% whereas the deposit average appears closer to 0.17% As. A subsequent geometallurgical review (XPS, 2017) indicated that the lower grade stockwork zones were under-represented in the composites and there was a degree of clustering of samples from the mid and upper zones of the deposit. This was not considered to be a major red flag from a sampling or geo-metallurgical perspective and the Master Composites were considered reasonable for flowsheet development.

Additional mineralogical studies were completed on variability samples 3-20 as detailed in SGS reports (SGS, 2017a and 2017b):

- QEMSCAN analysis on the 18 variability samples (Var 3 to 20):
  - Samples underwent Rapid Mineral Scan to provide the modal mineral analysis for each sample.
  - Observations were similar to earlier QemScan results except that clays and micas were more prevalent, averaging 5% over the 18 samples and reaching high levels around 20-30% in end members of the sampling sequence (entering and leaving the mineralized zone). These levels can impact metallurgical response.
- QEMSCAN analysis on the three copper rougher tails from variability samples Var 7, Var 8 and Var 9 which performed poorly in the standard flotation test and are identified as Group 3 samples:
  - Around 60% of the covellite and enargite identified in the rougher tailings were free and liberated particles.
  - Pyrite was the dominant mineral in the rougher tailings and microprobe analysis indicated from 0.4% to 0.7% Cu in solid solution within the pyrite grains. Copper in this form could account for 20%,28%, and 48% of the Cu reporting to the Var 7,8,and 9 rougher tailings, respectively.
  - Other than the partial loss of Cu in pyrite solid solution, there were no obvious mineralogical or liberation issues that could explain the poorer performance of these Group 3 samples (a group of samples, spanning a length of 42 m, from 503 to 545 m, down hole TC160124A).

Limited mineralogical studies were also conducted to evaluate the nature of the gold occurrences in pyrite (AMTEL, 2013 and SGS, 2017c).

The AMTEL study had indicated that there are several different pyrite morphologies within the Timok mineralization and that the gold content varies widely between the different pyrite occurrences.

SGS used optical microscopy, QEMSCAN and D-SIMS to study a pyrite rougher concentrate, and a copper concentrate and tailing to determine the following:

- Whether the pyrite is significantly different in the high-grade sample.
- If the gold in pyrite reporting to the copper concentrate differs from the gold content of pyrite depressed into the copper rougher tails.

The study identified four different pyrite morphologies described as coarse, porous, fine, and disseminated. The gold content of the coarse pyrite was 4 ppm, while in the other species it was in the range 15 ppm to 20 ppm. All four types were identified in the pyrite concentrate, copper concentrate, and copper rougher tails and the gold contents were consistent. The average gold content of covellite was 9 ppm and 31 ppm for enargite, although the latter was based on only five particles identified in the copper concentrate.

Unfortunately, the analysis did not immediately point to a processing route for improved gold recovery and the work was not pursued.

The modal mineralogy of four of the Yearly Blend samples was investigated as part of the XPS test program (XPS, 2018). It was noted that while the covellite content decreased progressively down the series of samples, the enargite content remained relatively constant around 1% of the sample mass. As a result the Cu:As ratio decreases and flotation separation becomes progressively more challenging. Other findings were broadly the same as in the previous studies.

13.1.4 Comminution Testing

13.1.4.1 Comminution Test Results

For the 2017 PEA, two “metallurgical holes”, TC150101 and TC160124, were drilled to provide samples for comminution tests. Test work was conducted by SGS on samples from TC160124. Orway Mineral Consultants were selected to analyze the results, and recommend a comminution circuit suitable for the most recent mine plan, available at the time (OMC 2017). The Orway report is summarized below.

The following comminution test work was performed at SGS:

- Integrated JK drop-weight and SMC tests on five samples.
- SMC test on 15 samples.
- Bond low-energy impact test on five samples.
- Bond ball mill grindability test on 20 samples.
- Bond abrasion test on 20 samples.

The detailed results are contained in the SGS report (SGS, 2017b). The main points are summarized in Table 13.6.

13.1.4.1.1 JK Drop-Weight and SMC

The JK drop-weight test (DWT) was performed on five variability samples (samples Var 3, Var 6, Var 8, Var 12, Var 16) and the standard SMC test was performed on all 20 variability samples. All SMC testing was conducted on a single size fraction of -31.5/+26.5 mm.

Table 13.6: Comminution Results - Summary

Sample	Relative Density		JK Parameters				CWi	BWI	AI
Name	DWT/SMC	CWi	A x b1	A x b2	ta	SCSE	(kWh/t)	(kWh/t)	(g)
Var 1	2.63	-	-	38.9	0.38	9.9	-	13.6	0.079
Var 2	2.59	-	-	70.8	0.71	7.7	-	8.9	0.000
Var 3	2.59	2.74	78.8	72.4	0.76	7.4	6.8	10.4	0.037
Var 4	2.85	-	-	95.2	0.87	7.1	-	9.4	0.174
Var 5	2.87	-	-	111.0	1.01	6.7	-	11.5	0.249
Var 6	4.22	3.97	133	127.0	2.24	5.9	6.5	9.5	0.175
Var 7	-	-	-	-	-	-	-	10.1	0.139
Var 8	3.87	3.81	56.9	53.2	0.40	8.8	8.7	9.1	0.565
Var 9	3.72	-	-	58.0	0.40	8.9	-	8.9	0.486
Var 10	4.10	-	-	70.3	0.44	7.7	-	9.0	0.336
Var 11	3.30	-	-	47.6	0.37	10.0	-	10.9	0.773
Var 12	3.25	3.27	44.6	41.4	0.25	10.3	10.4	10.7	0.654
Var 13	3.43	-	-	49.0	0.37	9.9	-	12.0	0.675
Var 14	3.38	-	-	73.6	0.56	8.2	-	10.4	0.310
Var 15	3.12	-	-	54.1	0.45	9.3	-	10.7	0.673
Var 16	3.16	3.08	65.8	61.0	0.56	8.5	7.2	14.1	0.573
Var 17	2.91	-	-	77.4	0.69	7.7	-	12.6	0.261
Var 18	2.94	-	-	61.7	0.54	8.5	-	10.4	0.529
Var 19	2.81	-	-	67.3	0.62	8.0	-	12.3	0.220
Var 20	2.72	-	-	65.8	0.63	8.0	-	12.0	0.105

[1] A x b from DWT.

[2] A x b from SMC.

The SMC test results are preferably calibrated against reference samples submitted for the standard DWT in order to consider the natural ‘gradient of hardness’ by size, which can widely vary from one process feed type to another. For this project, the SMC results were calibrated against the respective DWT test results in the cases where both tests were performed. The rest of the SMC results were calibrated against the average of the five DWT-SMC results.

The A x b values of the samples submitted to both the DWT and SMC tests were similar, with the difference not exceeding 8% in all cases. The five samples were quite variable and were characterized as very soft to medium with respect to resistance to impact breakage (A x b) and very soft to hard with respect to resistance to abrasion breakage (t_a). The A x b of the other 14 samples also fell in the same range of competency with the exception of one sample (Var 1) which had an A x b value of 38.9 and was categorized as moderately hard. Sample Var 6 was the softest out of all the samples submitted for testing with an A x b value of 133.

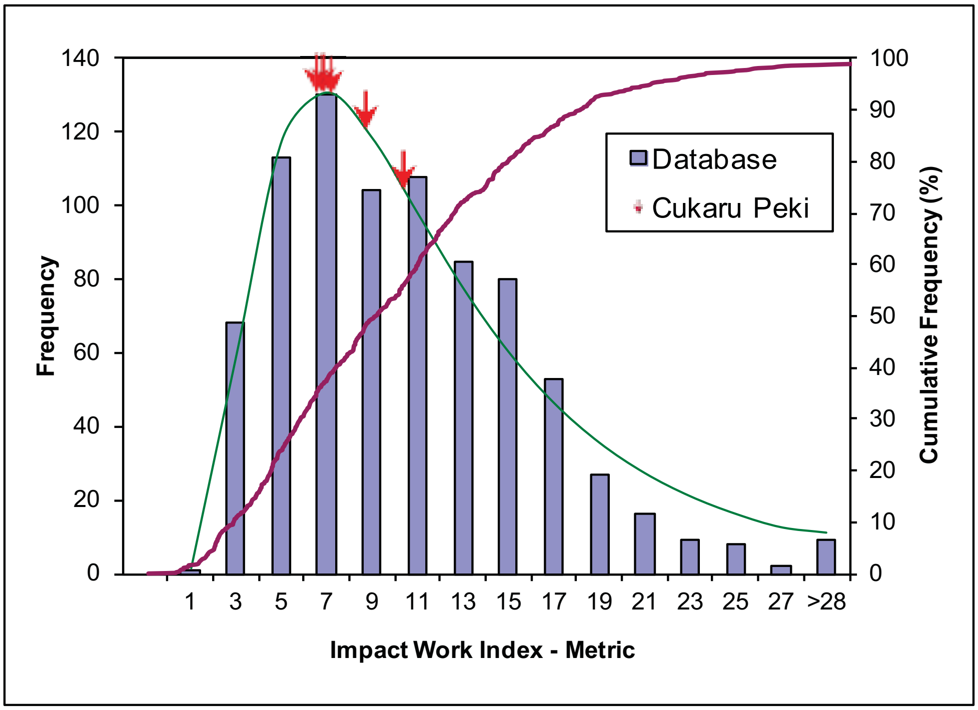
The measured rock relative densities were also quite variable and ranged from 2.59 to 4.22.

The DWT and SMC test results are detailed in the integrated DWT and SMC JKTech report, which is appended to the SGS report, along with the procedure, calibration and test details.. Cumulative distributions of A x b parameters from the 20 SMCs in the current program, as well as the JKTech database and various other test programs completed at SGS, are presented for comparison.

13.1.4.1.2 Bond Low-Energy Impact Testing

The Bond low-energy impact test determines the Bond Crusher Work Index (CWi), which can be used with Bond’s Third Theory of comminution to calculate power requirements for crusher sizing. For each of five variability samples tested, between 10 and 15 rocks in the range of 2 to 3 inches (”) were shipped to SGS Vancouver for the completion of the Bond low-energy impact test.

The results are presented in Table 13.6 and the CWi values are compared to the SGS database in Figure 13.2. Three samples (Var 3, Var 6, Var 16) fell in the moderately soft range of hardness of the SGS database, and two samples (Var 8, Var 12) were categorized as medium. The relative densities varied from 2.74 to 3.97 g/cm^3.



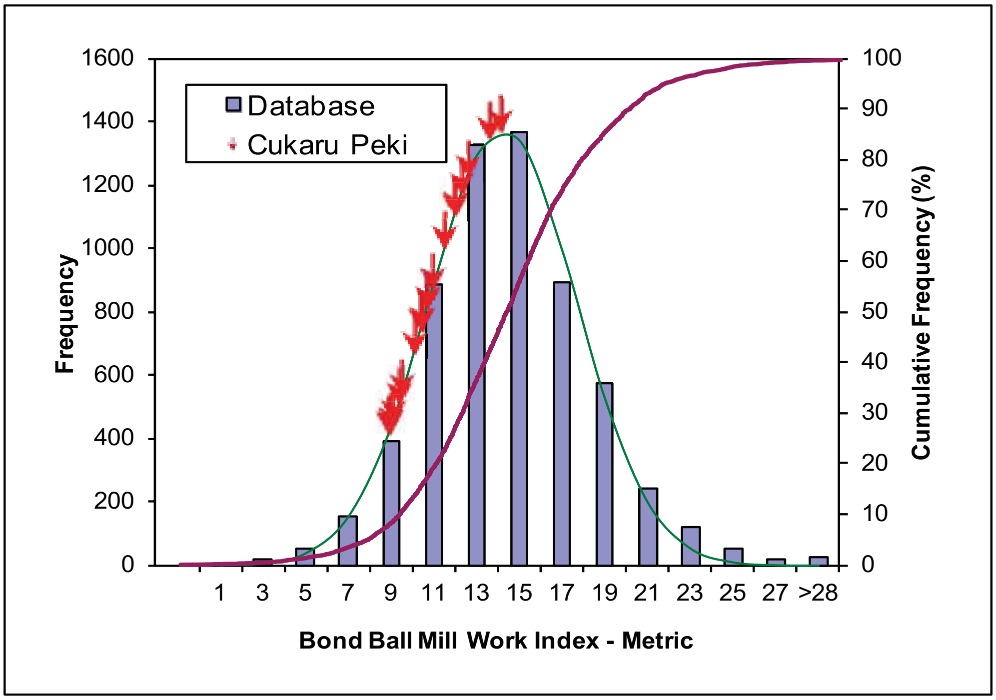
Source: SGS 2017b

Figure 13.2: CWi SGS Database Histogram Comparison

13.1.4.1.3 Bond Ball Mill Grindability Test Work

The 20 variability samples were submitted for the Bond ball mill grindability test at 150 mesh of grind (106 µm). Cumulative distributions of Bond Work Index (BWi) values from the current program as well as the SGS database and various other test programs completed at SGS are presented in Figure 13.3.

With BWi values ranging from 8.9 to 14.1 kWh/t, the samples fell in the very soft to medium range of hardness of the SGS database. The average BWi was 10.8 kWh/t.

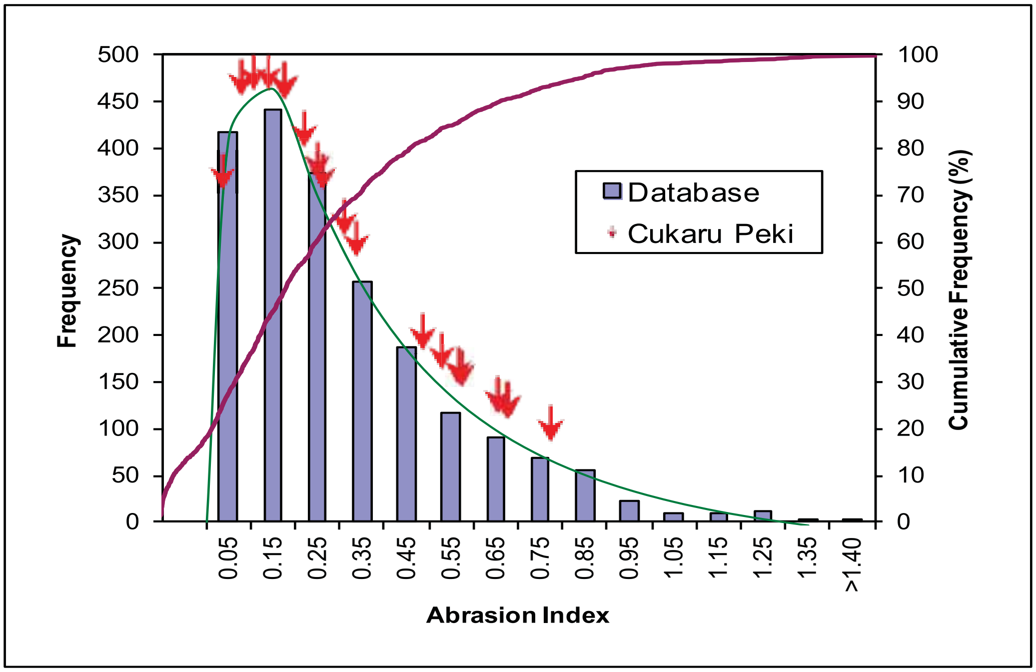


Source: SGS 2017b

Figure 13.3: BWi SGS Database Histogram Comparison

13.1.4.1.4 Bond Abrasion Test Work

The 20 variability samples were submitted for the Bond abrasion test (Table 13.6). Cumulative distributions of Bond Abrasion (Ai) values from the current program as well as the SGS database of various other test programs completed at SGS are presented in Figure 13.4.



Source: SGS 2017b

Figure 13.4: Ai SGS Database Histogram Comparison

The samples varied broadly in terms of their degree of abrasivity. With Ai values ranging from 0.0005 to 0.139 g, five samples fell in the very mild to moderately mild range of abrasiveness in the SGS database. With Ai values ranging from 0.174 to 0.336 g, seven samples were classified as medium abrasive. The remaining eight samples had Ai values varying from 0.486 to 0.773 g and covered the abrasive to very abrasive range in the SGS database.

13.1.4.2 Orway Comminution Circuit Sizing

The following is summarized from Orway's mill sizing report (OMC, 2017):

Two circuits were considered, SAB (semi-autogenous grinding (SAG) mill – ball mill) and SABC (SAG mill – ball mill – crusher), the former was selected. Initially, the SAB circuit mill sizing was based on the specific energy requirements calculated from the 85th percentile process feed characteristics of the 2016 PEA test work distribution. The mill sizing did not change following review with the additional 2017 comminution test work. The SAB circuit includes a SAG mill with an inside shell diameter of 7.92 m and an effective grinding length (EGL) of 3.20 m (26 x 10.5' (ft)), with a grate discharge and a 4,400-kW variable speed drive. The ball mill is a 5.50-m inside shell diameter by 8.69-m EGL (18 x 28.5') overflow mill, equipped with a 4,400-kW fixed speed drive. The SAG mill drive power is slightly oversized to provide a commonality of motors with the ball mill to reduce the project capital spares cost.

The comminution circuit power is suitable for the entire life of mine; however, this considers a coarsening of the mill product after Year 7. The Project is aware of this but the latest results show that coarsening to 108 µm from 75 µm will not impact copper recovery. This explains why the ball mill power was not increased following the review of the current PEA comminution test work results. This conclusion should be reviewed during the next phase of project development.

The mill sizing provides for operating power contingency of 24% for the SAG mill and 10% for the ball mill. In the case of the SAG mill, the extra contingency could be utilized to reduce the final grind size, but in view of possible coarser product in the latter half of the mine plan, the additional power could be used to further increase throughput. The analysis predicts a forecasted SAG mill throughput of 454 t/h (+14% compared to design) for Years 1 to 7. At this increased tonnage rate, the circuit is limited by the SAG mill therefore coarsening the grind from 75 µm to 108 µm would have no impact on mill throughput. If mill throughput is pushed beyond design in Years 1 to 7 then a review of the mine plan may be required.

13.1.5 Flotation Optimization

13.1.5.1 Composite Testing

As part of the 2017 PEA, a second testing program was completed on samples from the Upper Zone (SGS, 2017a). The emphasis of the test work was to optimize the flotation conditions established in the previous study (SRK (UK), 2016) and to provide samples to evaluate processing options for the high arsenic copper concentrate, referred to as the complex copper concentrate, and the pyrite concentrate.

To complete the 2017 PEA, a third program was initiated (SGS, 2017d). The overall scope of this program included sample preparation, mineralogy, comminution test work, process feed aging test work, bulk flotation test work, further flotation optimization and variability testing, solid-liquid separation testing, and environmental characterization.

For much of the 2017 PEA metallurgical program, it was intended to develop and optimize the flowsheet selected in the 2016 PEA, which produced two copper concentrates, a low arsenic and a complex concentrate. However, during variability testing, it was realized that a significant proportion of the samples did not respond well to this flowsheet. Testing and analysis of a flowsheet producing a single bulk concentrate was then carried out, and in conjunction with Nevsun's marketing consultants, a decision was made to change to this simpler, more robust flowsheet.

13.1.5.1.1 Copper Rougher Kinetics Flotation Tests

A series of 22 rougher kinetics tests, which examined the effects of primary grind size, pH, collector type, sodium metabisulphite addition, ammonium sulphate addition and extended time/increased collector dosage, was completed.

Reducing the primary grind K_{80} size from 140 microns to 89 microns improved copper recovery from 90.5% to 93.2% and Au recovery in the Cu rougher concentrate by 1% (to 32.9%). However, there was considerable noise in these initial test data and, given the high degree of covellite liberation noted previously, a "standard" grind of 108 microns was selected for the balance of the test program. The stage recovery of gold in the pyrite circuit was not affected by the changed primary grind.

When using Aerofloat 211, the pH in the copper rougher circuit did not impact gold metallurgy, and a higher pH resulted in improved copper metallurgy. When using Aerofloat 5100, the highest pH tested produced the best results for both gold and copper. Different types of collectors were explored: Aerofloat 211, Aerofloat 5100 and Aerofloat 7249. There was no significant difference in results among the three tests for either gold or copper metallurgy in the copper rougher circuit.

The use of sodium metabisulphite was explored at two pHs with Aerofloat 211 and with Aerofloat 5100. Ammonium sulphate was also tested with Aerofloat 211. Copper selectivity generally improved. The use of ammonium sulphate did not improve final results, although it did appear to improve copper selectivity in the earlier flotation increments.

There was no improvement in gold metallurgy in the copper rougher circuit at higher collector dosage and extended retention time. Copper metallurgy was improved in the earlier stages of flotation, but gradually converged to approximately the same results as the baseline test at the end of the copper rougher flotation.

13.1.5.1.2 Copper Circuit Batch Cleaner Flotation Tests

For the master composite sample, MC, 21 batch copper circuit cleaner flotation tests, including cleaner kinetics tests and two stage cleaner tests, were conducted. Variables studied in batch testing included the effect of regrind, the effect of depressants including sodium metabisulphite, SD200 and ammonium sulphate, cleaner pH level, and flowsheet configuration. Different optimization strategies were also explored in the cleaners, including solids loading and flotation times.

The optimum conditions for flotation, as per test F16, were determined as follows:

- Primary grind size: K_{80}~108 µm.
- Copper rougher: pH 10.0; 5 minutes aeration; 15.5 minutes flotation; 60 g/t Aero 211.
- Copper regrind size: K_{80}~28 µm.
- Copper first cleaner: pH 11.0; 8 minutes flotation; 20 g/t Aero 211.
- Copper first cleaner scavenger: pH 11.0; 6 minutes flotation; 30 g/t Aero 211.
- Copper second cleaner: pH 11.5; 4 minutes flotation; no collector.

Three tests examined the effect of regrind size. It appears that there was no benefit from regrinding finer than a K_{80} of 28 µm.

The effect of sodium metabisulphite addition was investigated. Copper and gold metallurgy were no better than in the baseline test F16 (without metabisulphite addition).

Four tests examined different techniques aimed at improving copper cleaner performance, though varying retention times and manipulation of pulp level, agitation and air flow rate

Different depressant schemes were tested in the copper cleaners, including the use of ammonium sulphate and the use of an increased pH throughout the cleaners. None of the tests performed better than the baseline test F16.

A single copper cleaner test was conducted at a coarse primary grind size (K_{80}) of ~200 µm. Performance was no better than the baseline test F16.

13.1.5.1.3 Copper Cleaner Kinetics Tests

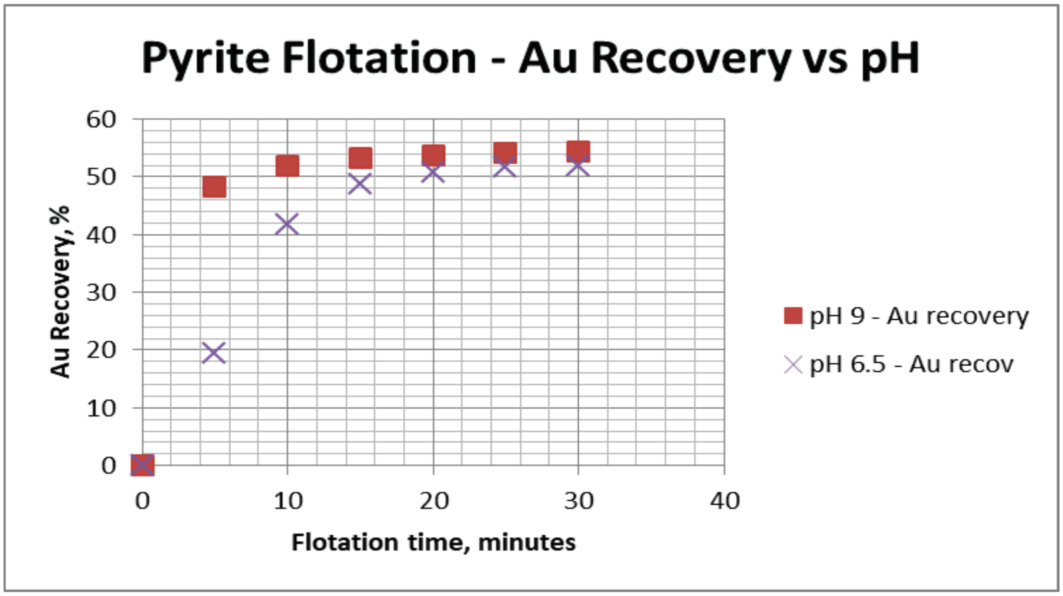
Two copper first cleaner kinetic tests, one copper second cleaner kinetic test, and five copper third cleaner kinetic tests were conducted. Copper recovery increased steadily with increasing retention time in all cases, suggesting that perhaps additional residence time may be warranted in order to reach the inflection point at which the curves plateau in a more pronounced fashion.

13.1.5.1.4 Pyrite Flotation

There was little difference in performance (in terms of the gold recovery versus mass pull relationships) among the pyrite rougher flotation tests carried out as part of the 2017 PEA, with all tests generally lying along a similar grade versus recovery trend.

Initial tests followed conventional pyrite flotation practice with sulphuric acid used to lower the pH to 6.5 and $CuSO_4$ added as an activator. Collector was potassium amyl xanthate (PAX) (200 g/t), and laboratory rougher flotation time was 20 minutes. However, the high lime requirements in copper flotation were matched by a high acid requirement in the pyrite circuit and it was found that both the acid and the copper sulphate could be omitted without significant effect on pyrite recovery. Surprisingly, pyrite flotation kinetics were significantly increased at higher pH, as shown in Figure 13.5.

Four tests with a pyrite first cleaner stage, gave a similar response in terms of upgrading. By rejecting non-sulphide gangue, the first cleaner stage was effective at reducing the concentrate mass at relatively low loss of gold recovery. Further reduction in concentrate mass was achieved by regrinding ahead of the cleaning stage.



Source: SGS 2017a

Figure 13.5: Pyrite Flotation Kinetics

Due to the intimate mineralogical association of gold with pyrite, regrinding to 50 µm did not significantly change the S/Au ratio and the mineralogical data suggests that very fine grinding would be required to achieve any significant liberation of gold. It therefore appears unlikely that additional flotation stages will increase the gold-to-pyrite ratio, which is the critical factor influencing the economics of oxidation and downstream processing. As a consequence, efforts to separate a viable gold bearing pyrite concentrate for separate storage have ceased at this time. Studies are continuing in this area with technology suppliers to look at alternative treatment technologies to recover the gold contained within the pyrite.

13.1.5.2 Variability Testing

At the time of preparation of the 2017 revised PEA, test work continued on variability samples from hole TC160124A. To date, the expected high copper recoveries, experienced with the earlier composites, have not been achieved, particularly on the Group 3 samples (spanning a length of 42 m from 503 to 545 m). They are characterized by a combination of high copper grades (4.0 to 10.5%), >60% pyrite content, and/or moderate to high arsenic levels (0.27 to 0.66%).

Group 3 also includes a fifth, deeper sample representing a 12-m interval from 615 to 627 m which has a lower grade, but a higher pyrite to covellite ratio (Var 16: 2.5% copper, 0.37% arsenic, 43% pyrite).

The metallurgy of the Group 3 samples is as follows:

- They yield less than 85% copper recovery in the rougher circuit compared to the 94% achieved with the composites.
- They have around 5% of the copper present in the pyrite crystal structure, which is therefore not available for recovery by differential flotation.
- They consistently return arsenic recoveries 15 to 20% lower than the copper recovery.
- Investigations are continuing into the causes of this Group 3 performance, concentrating on three areas:
  - Whether the high sulphide content causes redox/chemical conditions in which the covellite and enargite are less readily floatable: this would explain why the effect was not observed with master composite, MC, which contained only 38% pyrite.
  - Whether the low enargite recovery, as indicated by the arsenic results, is due to poor liberation from pyrite, especially in the high pyrite samples.
  - Whether poor performance is simply the result of adverse Cu:As and/or Cu:pyrite ratios, which would explain the absence of a “Group 3 effect” in earlier MC samples and in recent sample blends.

A more detailed sampling and variability test work program will be required during the feasibility phase of project development to define a range for acceptable process plant feed. The degree of blending occurring in the underground block cave is being quantified and additional blending, on surface, may be required. Methods of identifying the metallurgically difficult Group 3 zone in the underground mine, and blending it into the process plant feed, are also being analyzed.

13.1.5.3 Variability Testing at MMI

Data from the MMI program in Serbia are currently being evaluated. The results generally support the conclusions from the earlier test SGS programs concerning the benefits of a “flexible” flowsheet:

- High grade test samples generated low arsenic rougher concentrates of saleable quality (>20% Cu, 0.24% As, 9 g/t Au) at >93% Cu recovery.
- Intermediate feed grades required a single cleaner stage -34% of samples achieved a Cleaner 1 concentrate grading >35% Cu, 0.25% As, 7.9 g/t Au at 81% Cu recovery.
- The lower feed grades and samples with an less-favourable Cu:As ratio required a second cleaning stage to generate a final clean concentrate in which Cu grade was higher but recoveries were reduced (45% Cu, 0.35% As, 5 g/t Au at 74% Cu recovery).

Eleven of the test samples did not produce a “clean” concentrate containing <0.5% arsenic. The average Cu:As ratio in these samples was 9 compared with 55 for the samples that generated a clean concentrate. Further work will continue on these samples, and it is expected that blending to achieve a better Cu:As ratio will resolve the issue.

13.1.5.4 Testing of Yearly Blends

Tests have been conducted on yearly blends containing waste to emulate sublevel cave mixing of mineable resource over several levels, as modelled in the mine plan. Both the two-concentrate (clean plus complex) and the single bulk concentrate flowsheets were tested.

The two-concentrate flowsheet generated clean concentrates with low arsenic levels for Blend Samples 1 through 3. First cleaner concentrates in comparable tests averaged 36.7% Cu and 0.38% As at 81% Cu recovery. These blends all had Cu/As ratios greater than 20.

Blend Sample 4, with a Cu/As ratio of 14 had mixed success in achieving an As level below 0.5% with one successful test (0.33% As and one failure (0.68% As). Two cleaning stages were required. The average of two tests was 46.6% Cu and 0.51% As at 71% Cu recovery. Lowering the Cu recovery and/or grade would be expected to deliver the low arsenic target.

Blend Samples 5 through 8 were all similar with head grades 1.36 - 1.50% Cu, 0.15 – 0.17% As and a Cu/As ratio of only 9. None of these samples achieved low As level required for a clean concentrate (average of 5 tests was 42.2% Cu and 0.88% As at 63.5% Cu recovery). Extrapolation of several data points suggests that the clean copper recovery would need to be reduced to 35% in order to lower the arsenic grade to 0.4%.

As in previous test series the complex concentrates were not optimized for copper grade and recovery; this aspect of the program was deferred to allow development of the alternative bulk concentrate route.

The metallurgical response of most samples to the bulk concentrate flowsheet was very promising with respect to copper performance. Final concentrates were produced from roughing alone or after a single cleaning stage with grades ranging from 20 – 30% Cu at 88 – 95% Cu recovery. Arsenic grades in the bulk copper concentrate ranged from 0.51 – 2.50%. This can likely be optimized further by adjusting reagent additions, regrind size and by recirculating certain streams such as first cleaner tails and first cleaner scavenger concentrate process stream.

The bulk concentrate results, a selection of which are summarized in Table 13.7 and Table 13.8 have been used as a basis for the metallurgical predictions in Section 13.2.2.

Table 13.7: Yearly Blends Rougher Flotation

Test No.	Product	Wt %	Concentration Grade				Distribution			
			Au (g/t)	Cu (%)	As (%)	S (%)	Au (%)	Cu (%)	As (%)	S (%)
FB-1	Cu Ro Conc 1-5	22.7	6.92	29.6	0.51	32.5	34.8	95.1	85.2	31.8
FB-2	Cu Ro Conc 1-5	20.0	7.04	29.2	0.78	34.8	33.2	93.9	82.7	27.3

Table 13.8: Yearly Blends Cleaner Flotation

Test No.	Product	Wt %	Concentration Grade				Distribution			
			Au (g/t)	Cu (%)	As (%)	S (%)	Au (%)	Cu (%)	As (%)	S (%)
FB-3	Cu Ro Conc	16.8	5.69	25.1	1.06	39.2	35.3	93.8	93.8	27.5
FB-4	Cu 1st Clnr + Clnr Sc Conc1-2	11.6	4.39	19.7	1.45	40.5	26.9	92.0	93.6	23.8
FB-5	Cu 1st Clnr Conc1-4	6.1	4.57	21.3	2.29	36.5	25.7	90.5	93.4	14.6
FB-6	Cu 1st Clnr Conc1-4	6.1	4.05	22.1	2.50	34.3	24.4	90.7	90.6	13.9
FB-7	Cu 1st Clnr + Clnr Sc Conc	6.4	3.91	20.5	2.04	36.4	28.7	87.6	78.2	16.5
FB-8	Cu 1st Clnr Conc1-4	6.3		22.2	2.36	36.4		93.4	98.3	16.3

Notes: Ro = Rougher, Conc = Concentrate, Clnr = Cleaner, Sc = Scavenger

13.1.6 Ongoing Test Work at XPS | Expert Process Solutions

XPS of Sudbury, Ontario, were engaged in mid-2017 to provide a geo-metallurgical assessment of the Timok test work and, later, to confirm the relative capabilities of single concentrate versus dual (low and high arsenic) concentrate production.

In their 2017 review (XPS, 2017), conducted before the variability test results became available, XPS broadly agreed with the dual concentrate approach then being followed at SGS, but noted that the flowsheet had relatively low cleaning capacity for an overall Cu:As ratio of 10:1. They pointed out that this could limit the proportion of low arsenic concentrate to around 40% of total production, whereas up to 80% should be possible based on the mill head grades. XPS identified the opportunity to recover covellite from the high-As product and redirect it to the clean concentrate.

In February 2018 XPS released a draft report on their current test program (XPS, 2018) in which they confirmed that rougher flotation would be sufficient to produce a final concentrate in the early years of production. They also demonstrated that copper recover can be progressively improved as the concentrate arsenic limit is raised (Table 13.9).

Table 13.9: XPS Test Results

Blend	Test No	Concentrate with <0.5% As				Concentrate with 0.5 - 1% As				Concentrate with >1% As			
		Wt %	Cu	As	Cu recov.	Wt %	Cu	As	Cu recov.	Wt %	Cu	As	Cu recov.
1	T10-026	25.4	24.7	0.50	96.0								
2	T10-019	11.2	48.7	0.45	88.8	18.2	32.2	0.90	95.0				
3	T10-017	7.2	49.8	0.50	80.3	8.3	46.6	0.93	86.4	12.9	32.4	1.34	93.0
4	T10-014					5.5	33.00	0.95	76.5	6	32.2	1.13	81.4
5	T10-016					4.3	23.4	0.96	68.8	5.7	20.4	1.55	79.3

13.1.7 Oxidation Test Work

The detailed results of oxidation test work are contained in the SGS report (SGS, 2017d). The main points are summarized below.

The impact of process feed aging on metallurgical performance was evaluated through the artificial aging of four massive sulphide ores from the Upper Zone resource. Each of the composites from the 2016 PEA, that is Composites 2, 3, 4 and 5, were crushed to 100% passing 6 mesh, sealed in a plastic bag (10 kg per charge), and stored under dry ambient conditions for 10 months before starting the test program. Each composite was further crushed to 100% passing 10 mesh and split into five approximately 4-kg test charges. The first split 4-kg charge was used directly for process feed aging testing. Each of the remaining four 4-kg test charges was placed in a plastic pail, kept exposed to open-air, and was maintained damp with deionized water at different aged times (2, 4, 9 and 16 weeks) to simulate accelerated aging under humid conditions. For comparison purposes, each of the original four composites that had been crushed to 100% passing 10 mesh were also stored in a freezer for 10 months (2 kg per charge). These samples were also retrieved to perform identical metallurgical test work as the ambient stored samples.

The four frozen composites and four ambient-stored composites were submitted for copper speciation, sulphur and iron analysis. The soluble copper and iron increased for each composite, which indicated that the samples had been oxidized to some extent while under dry ambient storage for 10 months. The high sulphide ores seemed to be more easily oxidized.

The process feed aging test work consisted of batch rougher/cleaner flotation testing, ethylene diamine tetra acetic acid (EDTA) extraction, modified acid base accounting, and shake-flask extraction testing. The tests were conducted on samples that had been kept in freezer storage as well as those that had been stored under ambient conditions.

Sample aging under ambient conditions had a significant effect on batch cleaner flotation performance for Composites 2, 3 and 4. The copper grade of the final cleaner concentrate decreased with aging, while the arsenic grade increased compared to the frozen sample. This suggests separation of copper sulphides from enargite will become more difficult with aging. The metallurgical performance deteriorated drastically when aging was accelerated. Final cleaner copper grades were >55% for the Composites 2 and 3 samples stored in the freezer, where grades of less than 42% was the best that could be achieved for the same composites after only two weeks of accelerated aging. Composite 4 was even worse with a final copper concentrate grade of <25%. It was also almost impossible to achieve satisfactory separation between the copper and arsenic (enargite) sulphides, even after dry-aging the sample for 10 months. The sample aging effect on the flotation performance was less significant for the Composite 5 sample because of the relatively low head grades and generally poor recovery for both copper and sulphur.

Although the aged samples were tested on the two-concentrate flowsheet, and the separation problems will be less for the single bulk concentrate, this has still been identified as a risk, and will be evaluated further prior to and during the FS. It should be noted that the oxidation conditions experienced during the test are very aggressive and should be compared with actual conditions in the proposed mine.

13.1.8 Environmental Testing

Freeport had carried out a large number of tests geochemically characterizing the rock types within the deposit. Although a formal report was not received from Freeport, the data was collected and summarized by pHase Geochemistry for Knight Piésold in a report entitled ‘Review and Summary of Geochemical Data, Čukaru Peki Project’ dated 27 February 2017. The main points are summarized below.

- Elemental analyses (strong acid digest) determined that the pyrite tailing sample tested was comprised primarily of silicates (~29%) with moderate to minor amounts of aluminum (6.6%) and potassium (<2%). In comparison, the pyrite concentrate was predominantly comprised of iron (23%) with a lesser contribution from silica (15.5%) and minor aluminium (<1.5%). All other parameters reported at trace levels (<1%).
- Overall, the leachates from the toxicity characteristic leaching procedure typically reported significant concentrations (>1 mg/L) of calcium, copper, potassium and silica in solution.
- Analysis of the tailings EN 12457-2 extraction leachate reported a strongly alkaline final pH value (8.89), and concentrations of the typically controlled parameters (Hg, As, Cd, Cu, Fe, Ni, Pb, Zn) well within the guidelines designated by the World Bank. In contrast to the tailings, the concentrate EN 12457-2 extraction leachate reported an acidic pH value (4.57) and copper well in excess (two orders of magnitude) of the guideline designated by

the World Bank. Zinc was also observed at a concentration marginally greater than the specified guideline. All other parameters controlled by the World Bank were within the designated standards. As expected, elevated levels of most parameters were evident in the concentrate leachate in comparison to the tailings leachate.

- Analysis of the tailings process water reported near neutral pH (7.44), and all World Bank controlled parameters were well within the specified standards. While the concentrate process water also reported near neutral pH (7.28), concentrations of copper and iron, in excess of the World Bank guidelines, were observed. All other parameters reported within World Bank standards. Comparison of the total and dissolved metals indicated that the majority of analytes in the tailings process water were in the dissolved form, while the majority of analytes in the concentrate process water were suspended in the water column (total metals).
- Modified acid base accounting test results clearly identified both the tailings and the concentrate as potentially acid generating with major sulphide concentrations (≥5.54%) and very little neutralization potential.
- The tailings humidity cell leachates have maintained circum-neutral pH values (≥6.72) throughout the initial 10 weeks of the humidity cell test. Decreasing levels of alkalinity and sulphate are currently evident in the weekly leachates. The concentrate humidity cell leachates reported increasingly acidic pH values, well below the lower limit dictated by the World Bank (6.0), throughout the initial five weeks of weathering. Increasing concentrations of free acidity and sulphate are evident in the weekly leachates.
- Results of the particle size distribution analyses indicated that both samples (tailings and concentrate) were comprised primarily of fines with ≥70% of the samples passing the 75-µm sieve and ≤30% of the samples reporting as sand size particles. While the majority of these fines were comprised of silt size particles (67% to 75%), relatively significant clay size fractions (3% to 4%) were also reported.
- Results of the settling tests indicated that both samples (tailings and concentrate) will settle very quickly in a tailings pond setting. Both the settling test samples (standard and drained) generally settled out of solution within 15 to 30 minutes and terminal density was achieved shortly thereafter. The addition of drainage to the settling tests resulted in little difference in the final settled density of the tailings and concentrate samples (68.7% and 73.3% solids for the standard settling tests versus 69.7% and 74.8% solids for the drained settling tests, respectively).

13.1.9 Concentrate Characterization

13.1.9.1 Concentrate Description

This section describes testing on concentrates produced using the two-concentrate approach. The single bulk concentrate, which is the basis of this study, will have somewhat different characteristics and therefore the results presented should be considered as preliminary estimates. The “Copper Clean Concentrate”, “Copper Complex Concentrate”, and the “Pyrite Rougher Concentrate” were generated from a total of 12 batch copper cleaner flotation tests (BF23-BF34) which were conducted on the “TM Comp 600” sample by repeating the flowsheet and conditions described under report 15242-002 (SGS, 2017c).

Samples of the three concentrates were submitted for bulk density, specific gravity measurement, size fractional analysis, detailed concentrate analysis and whole rock analysis. The results are presented in Table 13.10 through to Table 13.13.

13.1.9.2 Concentrate Characterization Results

Table 13.10: Summary of the Bulk Density and the Specific Gravity

Test No.	Sample ID	Bulk Density kg/m^3		SG
		Uncompacted	Compacted	
BF24-34	Cu Clean Conc	1940	2336	4.58
--	Cu Complex Conc	1510	2125	4.65
Estimate	Bulk copper concentrate	1725	2230	4.60

Table 13.11: Size Fractional Analysis Summaries

BF24-34 Cu Clean Conc, P80 = 20 µm										
Fractions	Weight		Cu, As, S, Fe%				Distribution %			
	g	%	Cu	As	S	Fe	Cu	As	S	Fe
-38/+28 µm	4.0	3.8	61.9	0.098	33.2	3.72	4.1	0.5	3.8	2.6
-28/+21 µm	12.2	11.6	61.0	0.130	33.7	3.84	12.5	2.2	11.8	8.3
-21/+15 µm	20.5	19.5	60.2	0.220	34.1	4.39	20.7	6.2	20.0	15.9
-15/+10 µm	15.0	14.3	60.8	0.380	34.0	4.75	15.3	7.8	14.6	12.6
-10/+7 µm	13.2	12.5	58.2	0.600	34.3	5.31	12.9	10.8	13.0	12.4
-7 µm	40.3	38.3	51.3	1.320	32.0	6.73	34.6	72.6	36.9	48.1
Total (Calc.)	105.0	100.0	56.8	0.700	33.2	5.36	100.0	100.0	100.0	100.0
Total (Direct.)			56.3	0.640	34.5	4.98				
Cu Complex Conc, P80 = 18 µm										
Fractions	Weight		Cu, As, S, Fe%				Distribution %			
	g	%	Cu	As	S	Fe	Cu	As	S	Fe
-38/+28 µm	4.5	3.5	41.7	1.67	38.4	16.5	5.4	2.6	3.3	2.3
-28/+21 µm	12.6	9.7	31.8	1.88	41.9	22.9	11.5	8.1	10.0	8.8
-21/+15 µm	23.5	18.1	26.7	2.02	43.9	27.1	18.0	16.3	19.6	19.4
-15/+10 µm	20.8	16.0	23.8	2.10	44.2	28.2	14.2	15.0	17.4	17.9
-10/+7 µm	19.0	14.6	23.6	2.17	43.5	28.2	12.8	14.2	15.7	16.3
-7 µm	49.5	38.1	26.9	2.57	36.2	23.4	38.1	43.7	34.0	35.3
Total (Calc.)	130.0	100.0	26.9	2.24	40.6	25.3	100.0	100.0	100.0	100.0
Total (Direct.)			27.3	2.21	41.7					

Table 13.12: Detailed Concentrate Analysis

Element	Unit	Product ID		
		Cu Clean Conc	Cu Complex Con	Bulk Conc (est)
Au	g/t	5.37	5.06	See mine plan
Ag	g/t	11.0	11.0	See mine plan
Cu	%	56.3	27.3	See mine plan
As	%	0.64	2.21	See mine plan
S	%	34.5	41.7	See mine plan
Te	g/t	<60	<60	<60
Hg	g/t	2.2	1.9	2.0
F	%	<0.005	0.01	0.01
Cl	g/t	<50	<50	<50
Si	%	0.53	1.12	0.80
SO_4	%	0.8	1.0	0.9
CT	%	0.05	0.08	0.07
CCO_3	%	<0.05	<0.05	<0.05
Al	%	0.31	0.18	0.25
Ba	g/t	57.8	75.0	65.0
Be	g/t	<0.07	<0.03	<0.03
Bi	g/t	<80	<100	<100
Ca	%	0.13	0.28	0.20
Cd	g/t	<10	<20	<20
Co	g/t	<8	24.0	16.0
Cr	g/t	264	788	500
Fe	%	4.98	27.6	27.6
K	%	0.068	0.042	0.05
Li	g/t	<5	<20	<20
Mg	g/t	40.0	64.0	52.0
Mn	g/t	35.6	182	110
Mo	g/t	<20	<9	<20
Na	%	0.029	0.022	0.026
Ni	g/t	76.0	33.0	54.0
P	g/t	<200	211	211
Pb	g/t	196	290	250
Sb	g/t	103	290	290
Se	g/t	<100	<50	<50
Sn	g/t	<90	220	100
Sr	g/t	66.3	42	50
Ti	%	0.0085	0.017	0.017
Tl	g/t	<30	<30	<30
U	g/t	<20	<40	<40
V	g/t	186	126	150
Y	g/t	<0.3	<0.3	<0.3
Zn	g/t	938	168	550
Th	g/t	<0.5	1.0	<0.5

Table 13.13: Whole Rock Analysis

Oxide	Unit	Product ID		
		Cu Clean Conc	Cu Complex Conc	Bulk Conc (estimate)
SiO_2	%	1.13	2.40	1.75
Al_2O_3	%	0.59	0.35	0.47
Fe_2O_3	%	7.12	39.5	23
MgO	%	0.007	0.01	0.01
CaO	%	0.18	0.39	0.30
Na_2O	%	0.04	0.03	0.04
K_2O	%	0.08	0.05	0.07
TiO_2	%	0.01	0.03	0.02
P_2O_5	%	<0.05	0.05	0.05
MnO	%	0.005	0.02	0.02
Cr_2O_3	%	0.04	0.12	0.08
V_2O_5	%	0.03	0.02	0.03

The three concentrates have low uranium (less than detection limits) and thorium grade (<1.2 g/t) and thus they have low radiation. The arsenic grade of the Copper Clean Concentrate (0.64% As) is higher than the typical requirements of smelters. All other penalty elements are below the typical requirements.

As part of the ongoing program of test work at XPS it is planned that larger batches of bulk concentrate will be produced for further characterization of the concentrate quality and penalty elements.

13.1.9.3 Self-Heating Test work

A sample of each of the three concentrates was shipped to NesseTech Consulting Services Inc. for self-heating tests. The results are summarized in Table 13.14.

Table 13.14: Summary of the Self-Heating Test Work

Test No.	Sample ID	As Received Weight (kg)	SHC		
			% Moisture	Stage A	Stage B
2017-35	Cu Complex Conc	1.0	0.4	18.6	0.9
2017-36	Cu Clean Conc	1.2	4.1	19.6	2.1

The NesseTech draft report, the information from which were incorporated into the (SGS, 2017b) states that the Copper Complex Concentrate has a high Stage A reactivity but low Stage B reactivity, which seems somewhat unusual. The results suggest that the Copper Complex Concentrate may heat up to below 100°C with low potential to generate sulphur dioxide. The Copper Clean Concentrate has a similar and reactive Stage A (to the other copper concentrate) but higher Stage B values suggest that this is the product with some risk of self-heating if conditions in the field are conducive. This work will be repeated prior to and during the FS to determine the implications on concentrate transportation.

13.1.10 Tailings Characteristics

The 2017 PEA considered two tailings samples, a pyrite rougher concentrate (at that time to be stored in a pyritic TSF) and a pyrite rougher tails (to be stored in the main bulk TSF), from the bulk flotation tests which were subjected to solid-liquid separation and rheology tests to assist in design of waste management facilities.

13.1.10.1 Sample Characterization

The pyrite rougher concentrate and pyrite rougher tails samples were produced from flotation tests “BF5 to BF18”. The samples were received in the form of a pulp. In addition, a 20-L pail of process water was provided for additional test dilution. The pH was adjusted to pH 9.0 using lime as required. The results of the characterizations are summarized in Table 13.15.

Table 13.15: Sample Characterization

Sample I.D.	d80, µm	<1 µm, % vol	Dry Specific Gravity	Testing pH
Pyrite Rougher Tails	74	2.3	2.74	7.8
Pyrite Rougher Conc	118	2.8	3.60	9.0

13.1.10.2 Dynamic Thickening

Flocculant selection and preliminary static settling test results indicated that both samples responded well to BASF Magnafloc 333 flocculant, which is a very high molecular weight non-ionic polyacrylamide flocculant. The optimized results of the subsequent dynamic thickening tests are summarized in Table 13.16.

A thirty-minute period of extended underflow thickening, without feed and raking, resulted in an increase in underflow density. The underflow density of the pyrite concentrate increased to 74.6% w/w solids with a corresponding yield stress of 76 Pa. The underflow density of the pyrite tailings increased to 70.6% w/w solids with a corresponding yield stress of 59 Pa (versus the pre-extended thickening yield stress of 13 Pa).

Table 13.16: Dynamic Thickening Results Summary

Sample I.D.	Pyrite Rougher Tails	Pyrite Rougher Conc
Dosage flocculant g/t	10	20
Undiluted Feed %wt.	35.0	40.0
Diluted Feed %wt.	15.0	25.0
Thickener Underflow %wt.	68.0	64.8
TUFUA1 $m^2/(t/d)$	0.14	0.06
THUA2 $m^2/(m^3/d)$	0.04	0.02
Net Rise Rate $m^3/m^2/d$	39.8	46.7
Solids Loading $t/m^2/h$	0.298	0.694
Net Hydraulic loading $m^3/m^2/h$	1.66	1.94
Res.Time, Solid vs. U/F, h	1.38	0.78
Overflow Visual	Clear	Clear
TSS3 mg/L	19	51

Notes:

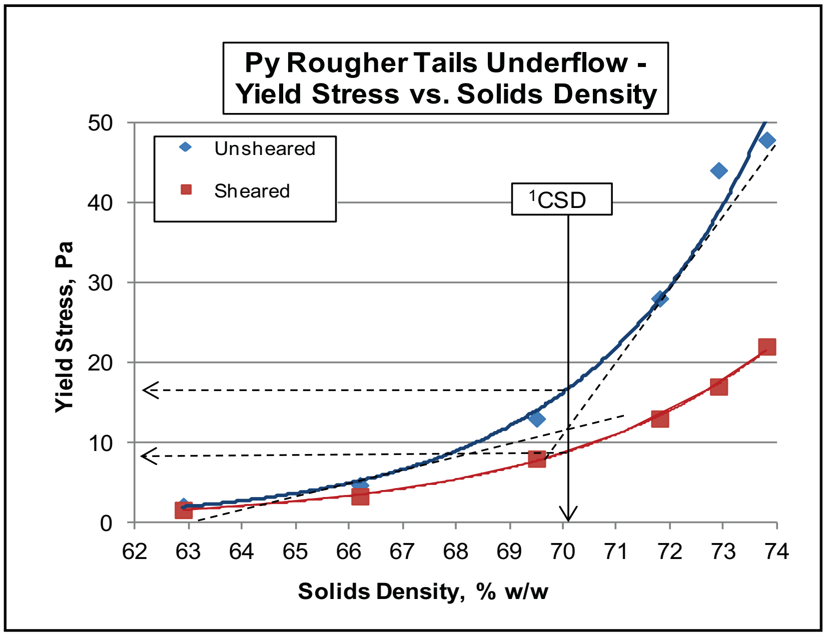
1. TUFUA – thickener underflow unit area.
2. THUA – thickener hydraulic unit area.
3. TSS – total suspended solids.
4. The base case for this PEA does not include a separate pyrite concentrate.

13.1.10.3 Underflow Rheology

A very important aspect relating to sample characterization within the context of a rheological study is the relationship between the solids specific gravity (density) and slurry solids density (content). Both samples tested displayed insignificant inter-particle interactions as suggested by “α” values near one, meaning that the dry solids specific gravity was comparable to their density in the slurry phase.

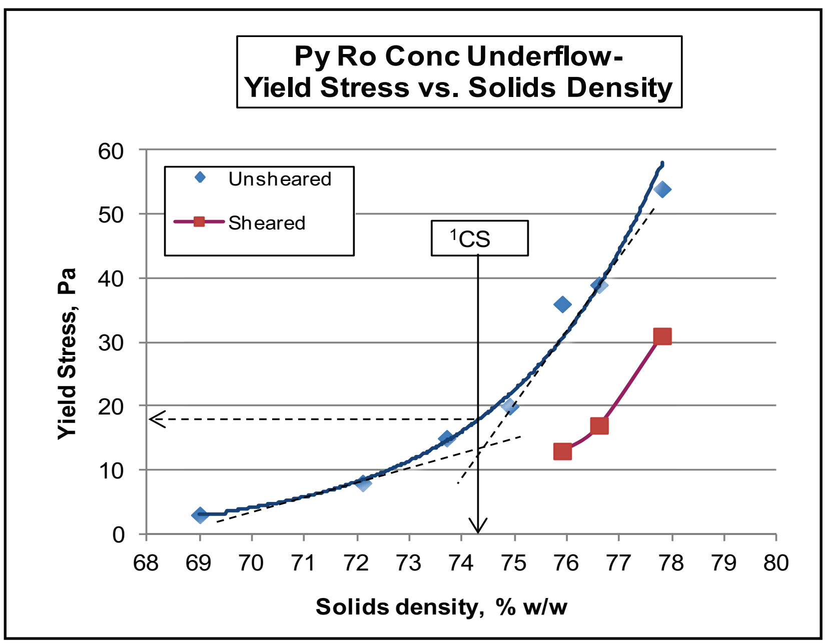
The rheology test measurement data allowed for Bingham modelling and subsequent interpretation, particularly with respect to the solids density rheological profile. The critical solids density of the pyrite rougher concentrate underflow sample was ~74% w/w solids, which corresponded to a yield stress of 18 Pa under unsheared flow condition. Due to the settling nature of the sample at or below 74.9% w/w solids, yield stress under sheared conditions could not be determined for these measurements.

The critical solids density of the pyrite rougher tails underflow sample was ~70% w/w solids, which corresponded to a yield stress of 17 Pa under unsheared flow condition and 8 Pa under sheared condition (i.e. post constant shearing). The test results are summarized in Figure 13.6 and Figure 13.7.



Source: SGS 2017a

Figure 13.6: Solids Density Rheological Profile – Pyrite Rougher Tails Underflow



Source: SGS 2017a

Figure 13.7: Solids Density Rheological Profile – Pyrite Rougher Concentrate Underflow

The combined bulk tailings that will result from the current flowsheet, which does not include pyrite flotation, will be placed in a single fully lined TSF facility. Examination of the above results indicates that a single bulk tailings stream would have a Critical Settled Density in the range of 70% to 74% solids, further testing is required to determine this value.

13.1.10.4 Concentrate and Tailings Filtration

Pressure filtration testing on copper concentrate was conducted using a filter feed at 74.0% w/w solids and at 5.5 bar and 6.9 bar pressure levels.. The cake thickness ranged from 15 mm to 30 mm. The resulting solids output (i.e. dry solids capacity) ranged from 3240 kg/m^2 h to 4052 kg/m^2 h. The discharge cake residual moisture content ranged from 8.8% to 11.1%w/w.

Vacuum filtration testing was conducted on the rougher tailings using a filter feed at 70.0% w/w solids based on the results of the settling/thickening and underflow rheology test results. The cake thickness ranged from 15 mm to 35 mm. The resulting solids output (i.e. dry solids capacity) ranged from 601 to 5629 kg/m^2 h. The discharge cake residual moisture content ranged from 12.5% to 20.2% w/w. The vacuum filtration was conducted at an average of 25 inches mercury vacuum level.

13.1.11 Processing Trade-off Study Summaries

As part of the 2017 PEA, it was decided to carry out four trade-off studies (ToS's #1, 2, 3 and 4) to define processing options for the project. A summary of each is included below.

13.1.11.1 ToS #1 Compare the Economics of a Single Copper Concentrate vs. Two Concentrates

The metallurgical process for Timok could be designed to produce either high and low arsenic copper concentrates, referred to as "clean" and "complex" respectively, or a single bulk concentrate.

Prior to completion of the 2017 PEA Nevsun's marketing consultants indicated that, based on the concentrate assays produced in the 2016 PEA all three concentrates would be marketable, assuming that a buyer can be found to purchase and treat the high arsenic copper concentrate. At treatment charges prevailing in early 2017 ToS #1 concluded that the combined NSR for the split concentrates and the NSR for the bulk concentrates were similar, suggesting there was no economic benefit from producing two concentrates. However, the marketing situation is changing as increasing amounts of complex concentrates enter the market.

Selecting a metallurgical process that produces split concentrates would concentrate the arsenic into a lower tonnage concentrate for treatment, should a suitable process route be identified. This would significantly reduce capital and operating costs of such an option.

13.1.11.2 ToS #2 Pyrite Concentrate Treatment Options

ToS #2 concluded that a technically feasible gold recovery process from a pyrite concentrate is possible, by either pyrite roast with acid production or the Albion process. Both processes have significant capital cost implications if installed as part of the initial project build and in the case of pyrite roast, would be hampered by a lack of market for the very significant amounts of acid the processes will produce.

The study recommended to go forward for further testing, engineering and estimating of pyrite roasting and the Albion Process. A second round of Albion test work focusing on low sulphide oxidation levels of 15% is ongoing.

13.1.11.3 ToS #3 Treatment Options for High Arsenic Copper Concentrates

ToS #3 concluded that economic reduction of the arsenic content of the complex concentrate is likely possible by either the Outotec reductive roasting or the Toowong caustic leach process.

Both processes underwent a first round of testing and both successfully reduced arsenic levels from around 4% to 0.1-0.2% in the treated concentrate. Subsequently it was decided not to pursue the Toowong process as it has not yet been demonstrated on an industrial scale.

The Outotec reductive or partial roasting process is now proven and reported to be working well at Ministro Hales for Chuquicamata high arsenic concentrates.

13.1.11.4 ToS #4 Review of Transportation Regulations for Arsenical Concentrates

A report on concentrate marketing, including general comments on transportation regulations, was delivered by Cliveden Trading AG. Additional opinions were sought from Hugh Hamilton, previously Manager, Raw Materials at Teck’s Trail Smelter Operation, and Laurie Reemeyer, previously Manager Metallurgy at Century Zinc Mine and Director, Process Strategy at Amec Foster Wheeler.

The views expressed in these three external reports have been summarized by the Nevsun-Rakita metallurgical team into several risk categories, where risk is defined as the product of exposure frequency and consequence or impact of an event.

Four scenarios or cases were developed to address options for Rakita to manage the identified risks:

Scenario A – Currently Acceptable Practice

Scenario B – Best Practice Based on Industry Trends

Scenario C – Potential Worst Case Due to Regulatory Changes

Scenario D – On-site Removal and Storage of Arsenic By-product

Although it appears that Rakita concentrates can be successfully managed under Scenarios A and B, the large quantity of arsenic involved greatly increases the complexity of the Rakita marketing challenge, particularly regarding the number of smelters that will be required to “share” the arsenic load. This issue could become acute if regulatory changes result in a reduced number of smelters able or willing to accept Rakita’s high arsenic concentrate (Scenario C).

The inevitable conclusion is that the best way to mitigate both the environmental and commercial risks of high arsenic production is to pre-treat the concentrate to remove the arsenic prior to concentrate sales and to store the arsenic in stabilized chemical form in a permanent storage facility (Scenario D). This assumes, of course, that economically viable pre-treatment technologies are available, thus the evaluations in ToS #3 and subsequent engineering studies.

13.1.12 Gold Recovery from Pyrite Concentrate

Significant test work and engineering efforts were undertaken to establish an economic route for gold recovery from pyrite. However the low gold content of the pyrite in all but the initial years of production, the up-front capital required to produce a pyrite concentrate and store it separately for later reclaim, and the revenue deferral from this element of the Timok project rendered it uneconomic.

It is not proposed to include pyrite recovery and separate storage in the flowsheet for Rakita. Pyrite in the copper tailings will report to the tailings storage facility.

13.1.13 Future Test Work

The following work is recommended during the FS stage:

- Run mini flotation pilot plant at XPS | Expert Process Solutions.
- Resolve the lower copper recovery, using the two-concentrate flowsheet, in the Group 3 variability samples and quantify its significance for the project.
- Further comminution and flotation variability tests, including those in progress at MMI.
- Conduct further concentrate characterization.
- Investigate arsenic removal from the complex concentrate by partial roasting, including stabilization for final storage.
- Finalise the investigation into gold recovery from pyrite under conditions of low-sulfide oxidation using the Albion process.
- Quantify implications of resource oxidation in the broken ore in the underground mine.

- Quantify implications of concentrate heating on concentrate transportation.

13.1.14 *Conclusions and Recommendations*

The 2017 metallurgical program covered flotation optimization, variability testing, testing of simulated annual blends, comminution testing, concentrate and tailings characterization, oxidation and environmental testing and mineralogy. Four trade-off studies were completed to provide guidance on concentrate marketing, gold recovery from pyrite, arsenic reduction in the complex copper concentrate and concentrate transportation.

For much of the recent metallurgical program, it was planned to develop and optimize the flowsheet selected in the 2016 PEA, which produced two copper concentrates, a low arsenic and a complex concentrate. However, during variability testing, it was realized that a significant proportion of the orebody did not respond well to this flowsheet. Testing, and analysis of a flowsheet producing a single bulk concentrate was then carried out, and in conjunction with the project's marketing consultants, a decision was made to base the PFS (PFS, 2018) on a simplified, more robust flowsheet that yields a single bulk concentrate.

13.2 Mineral Processing – Upper Zone

13.2.1 *Summary*

The copper mineralogy of the ore consists primarily of covellite with lesser enargite. The flowsheet was initially designed with the ability to generate two concentrates: a low (<0.5% As) arsenic content and elevated (>0.5% As) arsenic content. This same general flowsheet, utilized in a more simplified manner, has the capacity to produce a single bulk concentrate. Single bulk concentrate production is the basis of this study.

13.2.2 *Predicted Metallurgical Results*

The two-concentrate metallurgical flowsheet produced good recoveries and grades from almost all composites and from a good proportion of the variability samples. However, in conjunction with the study's marketing consultants, it was decided to base this PEA on the simpler, more robust approach, which produces a single bulk copper concentrate. It should be noted that the simplified single concentrate flowsheet developed for this PEAS is, for the most part, the same as the two-concentrate flowsheet and if future studies suggest reverting back to the two-product concept, it will be a relatively simple adjustment in terms of additional unit processes required at the back end of the flotation circuit.

Although the ore mineralogy and therefore copper and arsenic grades are important, geometallurgical relationships between metal recovery, concentrate grades and process feed type have not been identified so far. It was decided, based on an analysis of the flotation test results to identify five "ore grade bins" and assign average recoveries and concentrate grades achieved during testing of the yearly blends. The five feed grade bins and the representative yearly blend samples shown in Table 13.17 are expected.

Table 13.17: Feed Grade Bins

Process Feed Type	Yearly Blend Samples
Very High grade (>7% Cu)	FB-1
High grade (5 to 7% Cu)	FB 2
Medium high grade (4 to 5% Cu)	FB-3
Medium low grade (2 to 4% Cu)	FB-4
Low grade (<2% Cu)	FB-5, 6, 7 and 8

After consultation with the study's marketing consultant and modelling the impact of arsenic penalties charged by smelters, it was agreed to target a concentrate which maximized copper recovery without strongly constraining the arsenic content of the product.

In order to derive concentrate production data from the test work results it was necessary to manually adjust the concentrate copper grades to normalize the grade of the mill feed to the average grade bin feed copper value; to adjust the mass pull to match the copper recovery in the tests average for that grade bin; and to adjust the arsenic and gold grades to match their respective recoveries in the tests for the representative bins. This was done for each production quarter on the mine production schedule.

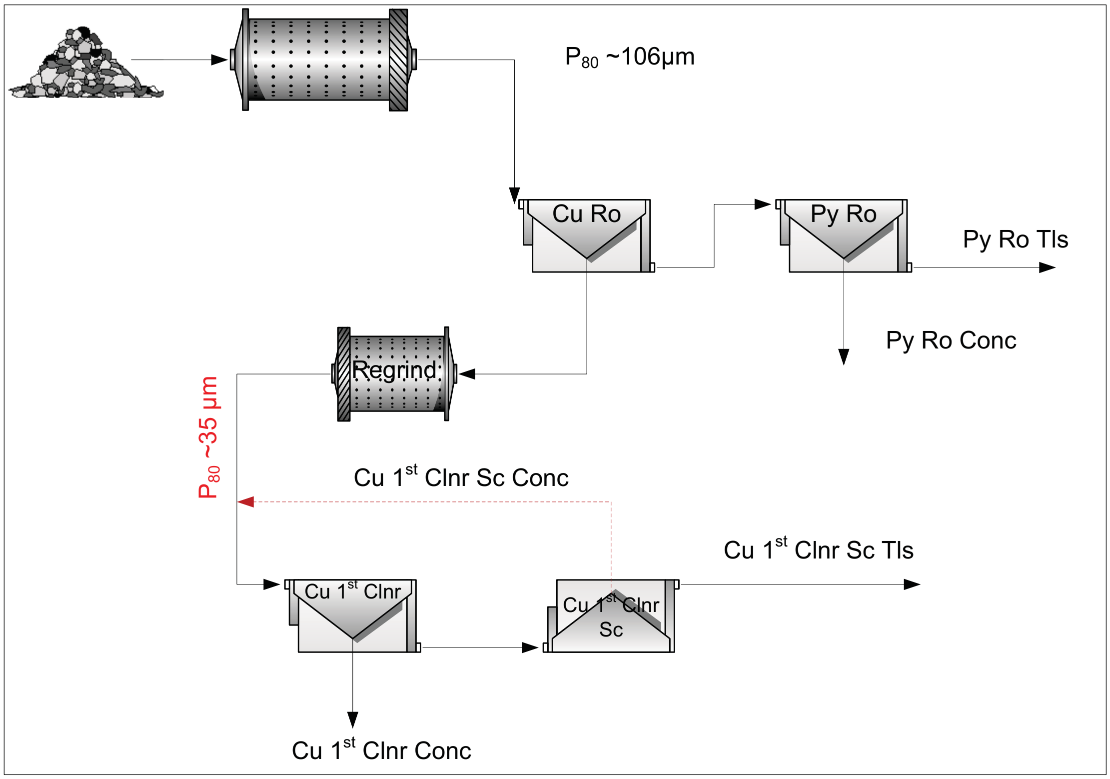
This manual, iterative approach was necessary because insufficient test results were available to derive mathematical relationships between all of the variables for copper, arsenic and gold.

The predicted metallurgical results averaged by grade bin, using the single concentrate flowsheet and maximising copper recovery are shown in Table 13.18.

Table 13.18: Predicted Metallurgical Results

Ore type	Flowsheet and Products	Concentrate Grade % Cu	Recovery Cu %	Recovery As %	Recovery Au %
High grade (>7% Cu)	Single rougher concentrate	19	96	92	50
High grade (5 to 7% Cu)	Single rougher concentrate	34	94	90	29
Medium high grade (4 to 5% Cu)	Single first cleaner concentrate	32	93	91	24
Medium low grade (2 to 4% Cu)	Single first cleaner concentrate	22	928	95	256
Low grade (<2% Cu)	Single second cleaner concentrate	21	917	94	22

A schematic representation of the single bulk concentrate flowsheet is shown in Figure 13.8 note that in the design for the PFS (PFS, 2018) the pyrite rougher is removed and all of the pyrite goes to the bulk tailings.



Source: SGS 2017

Figure 13.8: Bulk Concentrate Process

13.2.3 Flowsheet Selection

The process plant design is based on a combination of metallurgical test work, mine production plan and in-house information. Where necessary, benchmarking has been used to support the design.

The Timok process plant includes the following unit processes and associated facilities:

- Primary crushing located underground.
- Overland conveying of crushed process feed.
- Coarse feed storage bins and reclaim.
- SAB grinding circuit.

- Copper flotation comprising rougher flotation, concentrate regrind and two stage cleaning.
- Copper concentrate thickening and filtration.
- Copper concentrate load out and storage for each concentrate.
- Tailings storage and water reclaim.
- Effluent treatment.
- Reagents storage and distribution (including lime slaking, flotation reagents, water treatment and flocculant).
- Grinding media storage and addition.
- Water services (including fresh water, fire water, gland water, cooling water and process water).
- Potable water treatment and distribution.
- Air services (including high pressure air and low-pressure process air).
- Plant control rooms.
- Possible future equipment for flotation of a separate high arsenic complex copper concentrate.

The Processing Plant is described further in Section 17 – Recovery Methods.

13.3 Metallurgical Test Work – Lower Zone

A limited amount of metallurgical test work has been performed on the Lower Zone mineralization. In 2016, Aminpro performed tests on three types of mineralization found in the Lower Zone: DC1 Overprint, DC2 Mixed Zone and DC3 Primary. Primary, or porphyry copper mineralization is by far the most abundant and most important. Their results are summarized below.

- Three types of mineralization were treated from the Lower Zone (LZ). The received samples showed that the copper minerals are predominantly chalcopyrite and bornite with increasing chalcopyrite with depth. The following are the main assays and estimated minerals of the samples received.

Table 13.19: Lower Zone Metallurgical Assays

Sample	Assays					Estimated Minerals		
	Au g/t Au	As ppm	Cu %	Fe %	S %	Cu-S %	FeS2 %	Gangue %
DC1	0.11	86.0	0.69	6.00	7.02	1.19	10.20	88.6
DC2	0.12	61.0	0.65	6.65	7.70	1.44	9.90	88.7
DC3	0.17	11.2	0.67	4.16	4.79	1.81	5.80	92.4

The DC1 represent the Overprint Zone, below is the DC2 or Mixed Zone and the DC3 is the lowest zone or Primary porphyry zone.

The metallurgical test work done at Aminpro-Chile laboratory in Santiago was conceptual and focused on:

- Rougher Tests
    - Characterization of mineralization
    - Rougher Reagent test work
    - Mineralization response to variables (%Solids, pH and regrind)
- Cleaner Tests
    - Reagent tests
    - Mineralization response to variables (%Solids, pH and regrind)
- Locked cycle tests
- Tailings sedimentation and rheology tests

The work above was interpreted, modeled and put into a front-end-engineering package that would provide the owner's team information for a conceptual level study, namely potential flowsheet, optimum grind and design criteria with mass balances for a "nominal" and "design" case.

As general statement on the sample's test response to flotation, the following may be said:

- The locked cycle tests gave very encouraging results and with all mineralization types, maximum copper recoveries reached into the upper eighties. Gold recoveries were into the forties in the DC1 and DC2 and reached into the upper sixties for the primary porphyry mineralization. Copper concentrate grades were low because of the pyrite but did reach into the twenties with all samples.
- There were no deleterious elements that would incur penalty charges in the sale of concentrates except for DC1 ore if mined by itself.

- The mineralized samples responded well to floatation showing fast kinetics for copper in the roughers. The copper kinetics were slower in the cleaners and the maximum recoveries were in the low nineties because of the conditions set to depress pyrite, a mineral that appears highly activated.
- The test work did show that the pyrite content of the mineralized samples is high and a better selectivity in reagents will be required; perhaps even a mixture of reagents for the roughers and cleaners. The locked cycle test indicated that recoveries of copper will reach into the upper eighties and gold recoveries will be above 40 in the overprint and mixed material and reach the upper sixties in the primary porphyry mineralization.

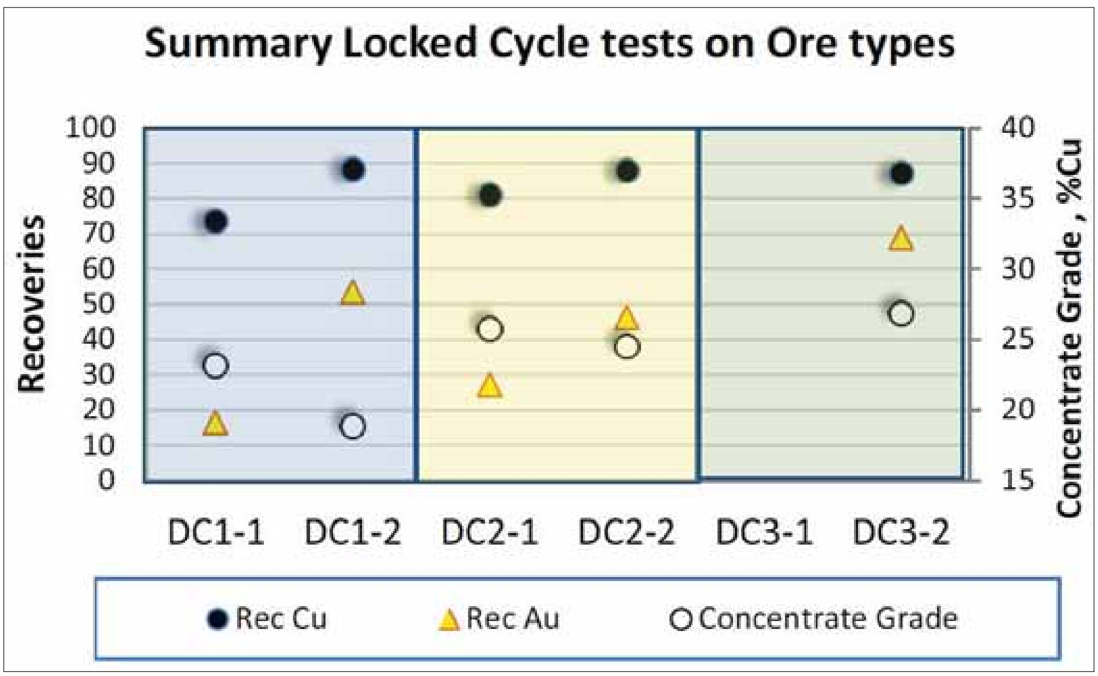


Figure 13.9: Lower Zone – Locked Cycle Test Summary

14. Mineral Resource Estimates

14.1 Introduction – Upper Zone

The Čukaru Peki Upper Zone (UZ) Mineral Resource statement presented herein has been reported in accordance with NI 43-101.

The Čukaru Peki UZ Mineral Resource model prepared by SRK (UK) is based on the drillhole database described in Section 10.3.1.1. The Mineral Resource estimate was supervised by Mr. Martin Pittuck, C.Eng., FGS, MIMMM an "independent Qualified Person" as defined in NI 43-101. This Mineral Resource estimate was prepared on 24 April 2017.

To the best of SRK's knowledge, there are no environmental, permitting, legal, title, tax, socio-economic, market, political or other relevant factors that would affect the Mineral Resource presented in this report.

14.2 Introduction – Lower Zone

The Čukaru Peki Lower Zone mineral resource statement presented herein represents first mineral resource evaluation reported in accordance with the NI 43-101. The Lower Zone mineral resource is in addition to the UZ mineral resource reported by SRK (UK) on April 24, 2017.

The Lower Zone resource model prepared by SRK (Canada) utilizes some 137,882 m of drilling for a total of 102 exploration, resource and metallurgical drillholes. The Mineral Resource estimate was prepared by Dr. Gilles Arseneau, P. Geo., an "independent qualified person" as defined in NI 43-101. The effective date of the mineral resource statement for Lower Zone is June 19 2018.

To the best of SRK's knowledge, there are no environmental, permitting, legal, title, tax, socio-economic, market, political or other relevant factors that would affect the mineral resource for the Lower Zone presented in this report.

14.3 Resource Estimation Procedures – Upper Zone

The resource estimation methodology involved the following procedures:

- Database compilation and verification.
- Construction of wireframe geological models and definition of resource domains.
- Data conditioning (compositing and capping review) for statistical analysis, geostatistical analysis.
- Variography, block modelling and grade interpolation.
- Resource classification and validation.

- Assessment of "reasonable prospects for economic extraction" and selection of appropriate reporting cut-off grades.
- Preparation of the mineral resource statement.

14.4 Resource Estimation Procedures – Lower Zone

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- Database compilation and verification.
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- Variography, block modelling and grade interpolation.
- Resource classification and validation.
- Assessment of "reasonable prospects for economic extraction" and selection of appropriate reporting cut-off grades.

Preparation of the mineral resource statement.

14.5 Resource Database – Upper Zone

SRK was supplied with the Čukaru Peki drilling data in a Microsoft Excel database on April 24, 2017. The database was reviewed by SRK and imported into Leapfrog Geo and Datamine. SRK is satisfied with the quality of the database for use in the construction of the geological block model and associated Mineral Resource estimate.

14.6 Resource Database – Lower Zone

SRK was supplied with the Lower Zone drilling data in comma separated values (CVS) format files on April 30, 2018. The database was reviewed by SRK and imported into Leapfrog Geo and Geovia's Gems 6.8.1 to complete the Mineral Resource estimate. SRK is satisfied with the quality of the database for use in the construction of the geological block model and associated Mineral Resource estimate.

14.7 Statistical Analysis – Upper Zone Raw Data

Metallurgical and mineralogical test work highlight that copper mineralogy primarily consists of covellite with lesser enargite and trace colusite, bornite and chalcopyrite. To allow for statistical assessment and modelling of the separate distributions of the primary copper minerals, SRK and Rakita derived two additional fields in the raw assay database using the copper and arsenic assays, based on the following formulae:

Copper in enargite (CuEn) % = As % * 2.55

(i.e. the copper to arsenic ratio in enargite, Cu_3AsS_4)

Copper in covellite (CuCov) % = Cu% - CuEn%
(i.e. the remaining copper is assumed to be associated with covellite, CuS)

An initial global statistical analysis was undertaken using the raw drill data. Summary statistics, incremental and log histograms were prepared. The skewed log normal distributions for CuCov, CuEn, gold and arsenic are shown in Figure 14.1, with the separate populations noted in the assays relating to host rock, low and high-grade zones.

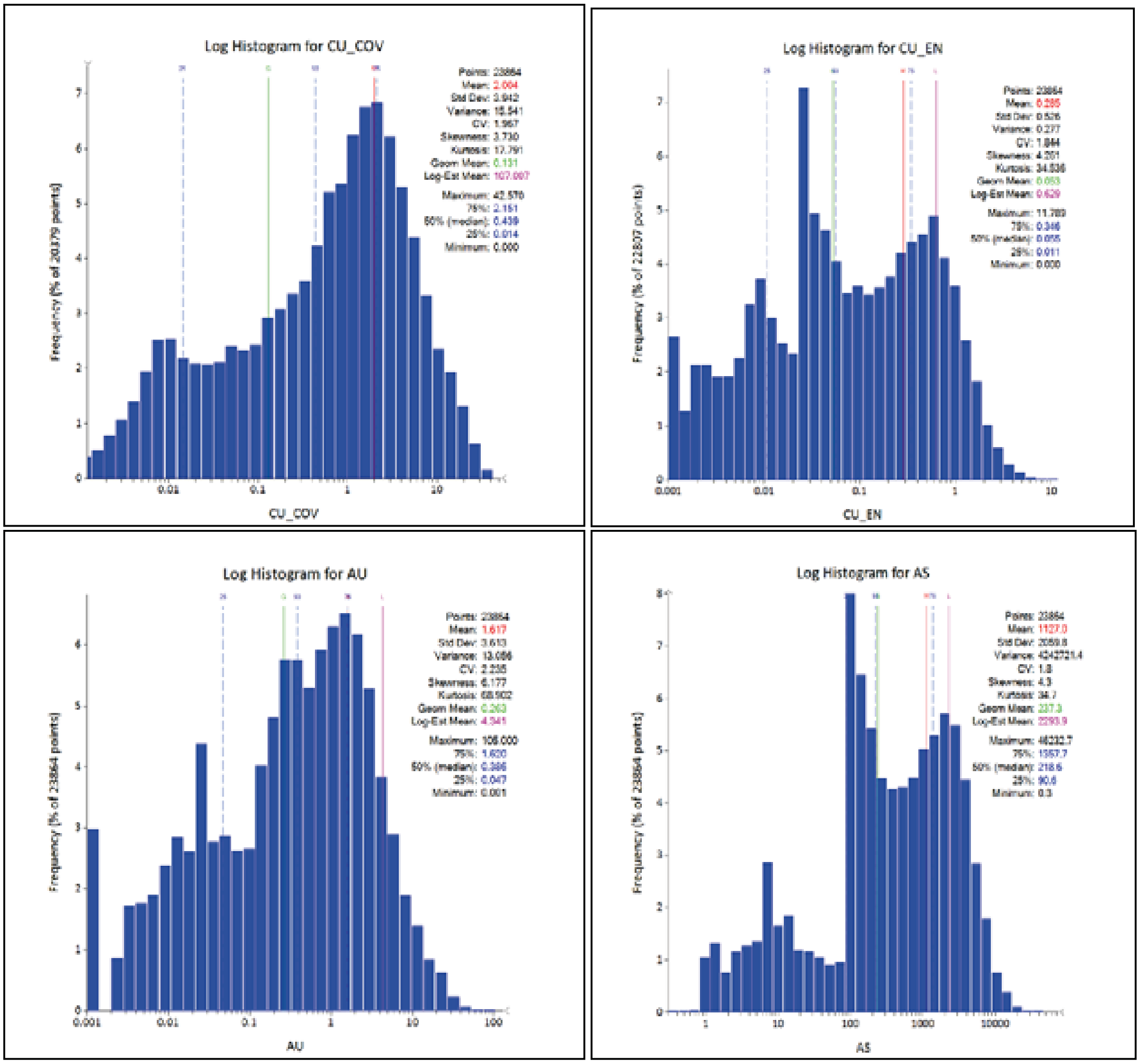


Figure 14.1: Incremental and Log Histogram of Length Weighted Project CuCov%, CuEn%, Gold and Arsenic Assays

14.8 Statistical Analysis – Lower Zone Raw Data

Initial review of the Lower Zone assay data seem to indicated that copper could be grouped into two distinct populations with values less than 0.5% copper being grouped into a lower grade population and assays greater than 0.5% copper grouped in a higher grade population (Figure 14.2).

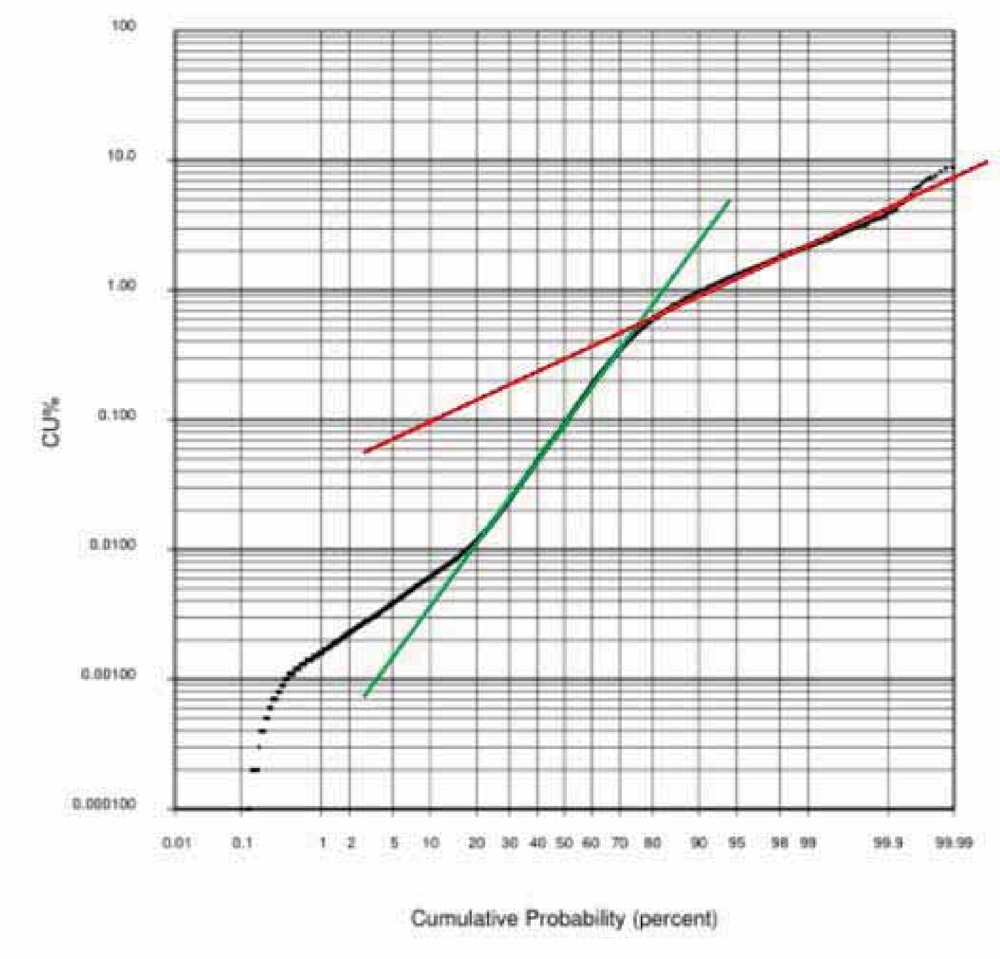
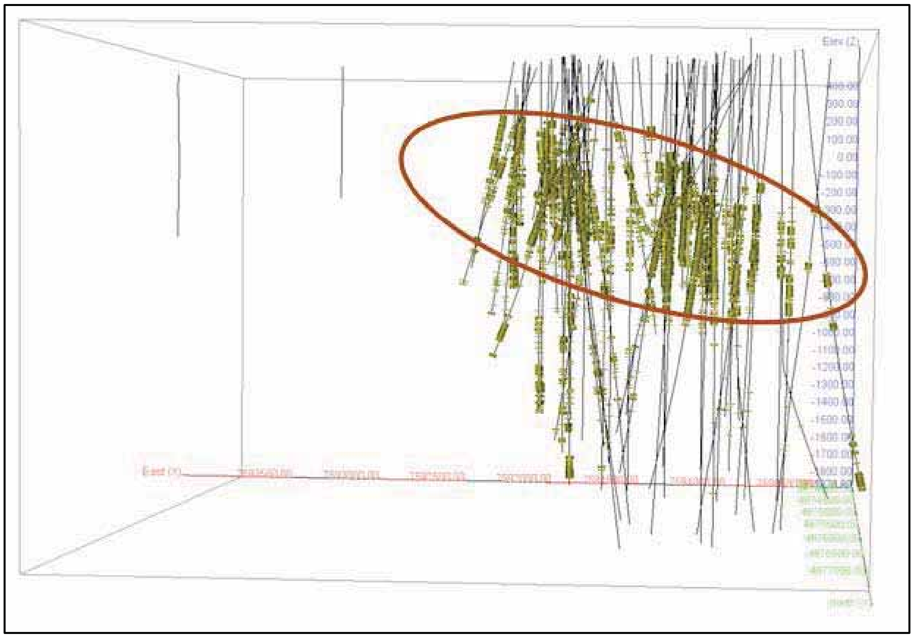


Figure 14.2: Probability Plot showing High Grade and Low Grade Copper Populations

A further analysis of the raw assay data showed that most of the base metal values, lead, zinc and bismuth are concentrated in the uppermost part of the Lower Zone Initial review of the Lower Zone (Figure 14.3). This is contrast with molybdenum which appear to be concentrated in the lower parts of the Lower Zone (Figure 14.4). SRK concluded that the apparent separation of base metal value from molybdenum values could be used to separate the porphyry associated copper mineralization from the copper associated with the epithermal style of mineralization associated with the Upper Zone deposit. A surface was generated from the assay data to separate the epithermal style mineralization from the more porphyry dominated style of mineralization.



Note: Markers are 100 m apart on the vertical axis and 500 m apart on the horizontal axes

Figure 14.3: Perspective View Looking South of the Lower Zone Drill Holes Showing values of (Pb + Zn + Bi) greater than 100 ppm Highlighted in Yellow

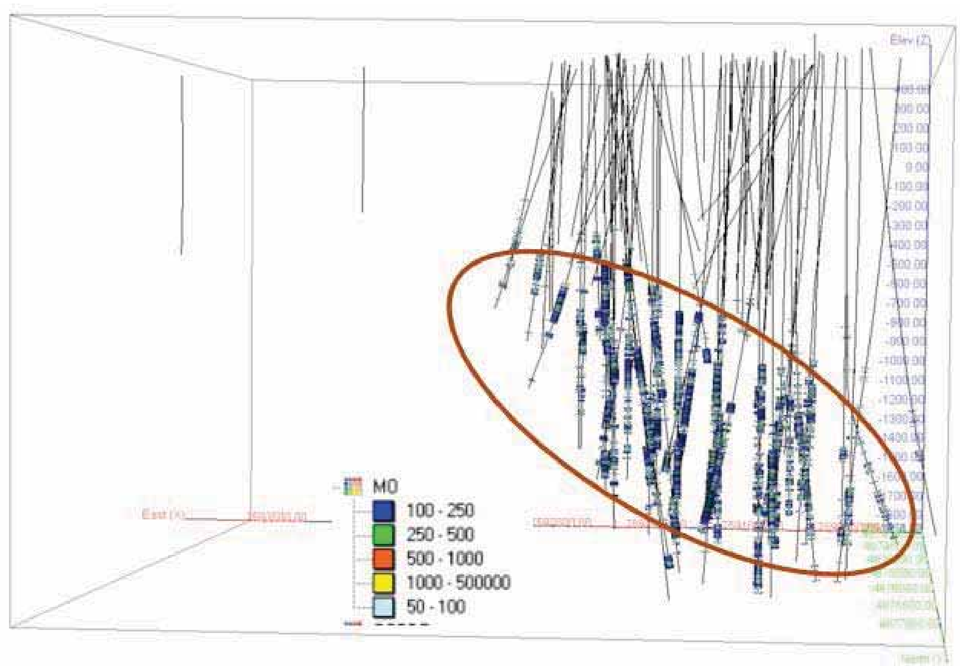


Figure 14.4: Perspective View Looking South of the Lower Zone Drill Holes Showing Molybdenum values Greater than 50 ppm

Note: Markers are 100 m apart on the vertical axis and 500 m apart on the horizontal axes

14.9 3D Modeling – Upper Zone

The 2017 Mineral Resource estimate update is based on drilling, site visit, core photo review and modelling of the following geological features for the deposit:

- Fault network.
- Lithology model.
- Alteration shell framework.
- Mineralization zones based on:
    - High-grade copper in covellite (UHG).
    - Massive sulphide.
    - Low grade copper in covellite.
    - High grade copper in enargite.
    - Low grade copper in enargite.

14.9.1 Geological Wireframes

14.9.1.1 Fault Network

SRK (UK) constructed a fault network model using a combination of core photo re-logging, ABI information on the orientation of interpreted primary slip zones, Rakita's major structure drillhole logs geotechnical logs and visual assessment of offsets in the geology and copper assay grades.

Leapfrog Geo software was used to model the fault network and associated structural domains, with four faults identified as significant with respect to the geometry of the deposit. Zones of geotechnically weaker rock associated with faulting, fracturing and increased clay content were also modelled in 3D during this process, most significantly including the broad (basal) zone of clay alteration and fracturing under the UZ mineralization (referred to as the 'Basal Clay Fracture Zone').

14.9.1.2 Lithology Model

SRK has modelled the un-mineralized stratigraphic cover sequence as a series of surfaces above the lower andesite (LA) which hosts the mineralization, based on geological logging of: upper andesite sill (UA), marl (UCMA), conglomerate (UCCM) and Miocene clastic sediments (MCS). The LA is interpreted to have zones of higher and lower porosity, with the more coherent, less porous rock being at the margins of the UZ deposit (referred to as 'LPA' and 'LAB' in logging codes) considered to coincide with the contact between mineralization and waste. SRK has modelled the LA using a combination of surfaces and implicit shells, based on geological logging codes, to reflect a relatively uniform contact with the overlying UA sill and more geometrically variable contact at depth between mineralized and un-mineralized parts of the LA.

Prior to constructing the lithology model, the fault network described above was used to generate a series of fault bounded domains, within which lithological surfaces were constructed independently to reflect faulted offsets.

14.9.1.3 Alteration Shell Framework

SRK has grouped together drillhole intervals based on similar alteration assemblages (which were interpreted using a combination of Terraspec data and core observations) and generated broadly concentric alteration wireframes using Leapfrog Geo. SRK has modelled the following alteration zones within each of the fault bounded domains:

- Advanced argillic halo.
- Kaolinite halo.
- Argillic halo.

The advanced argillic halo has been used as a guide for modelling the main mineralization domains.

14.9.2 Mineralization Wireframes

Mineralization domains were modelled for the HS epithermal style mineralization in the Čukaru Peki (UZ) deposit. Mineralization wireframes have been defined based on a combination of the following criteria:

- Within the advanced argillic alteration domain.
- Within the LA unit (rather than overlying UA and sediments).
- Differentiated by mineralization style (i.e. Massive sulphide vs. Andesite breccia).
- Differentiated by copper mineralogy (i.e. covellite vs. enargite) and copper grade.

14.9.2.1 Ultra High-Grade Copper in Covellite

The UHG domain has been modelled based on the visually evident top contact with waste rock and the step changes in the grade at around 12% CuCov at the lower contact. SRK created a 3D solid wireframe from selected sample intervals using the vein modelling tool in Leapfrog.

14.9.2.2 Massive Sulphide

The top third of the deposit comprises massive sulphide mineralization, including the UHG domain at the top, below which can be found medium grade copper in covellite mineralization with mainly stratiform grade distribution. Within the massive sulphide, (below the UHG), CuCov grade typically ranges between 5% and 10%.

14.9.2.3 Low Grade Copper in Covellite

The lowermost two-thirds of the deposit comprise the low-grade copper in covellite mineralization where the host rock is more recognizably andesitic with sulphide veinlets, CuCov typically ranges from 0.5 to 5.0% CuCov, but the overall copper grade distribution is more influenced by enargite which has relatively steep dipping continuity. The geometry of the low-grade mineralization at depth is based on relatively few drillholes and is interpreted to have an irregular contact with the un-mineralized part of the lower andesite. Consequently, this part of the model is restricted to lower confidence inferred mineral resources.

SRK created 3D solid wireframes from selected sample intervals using a combination of the vein modelling tool and implicit techniques in Leapfrog Geo.

14.9.2.4 Copper in Enargite Domains

Copper in covellite domains which are used to control the estimation of CuCov and Au overprint copper in enargite domains which are used to control the estimation of CuEn and

As. A high-grade copper in enargite domain has been modelled based on a visually evident step change in the CuEn grade at around 0.5% CuEn; this feature is located relatively centrally within the UZ deposit and shows relatively steeply dipping continuity. The advanced argillic altered lower andesite outside of this domain represents the low-grade copper in enargite, where the grade distribution is considered to be relatively isotropic.

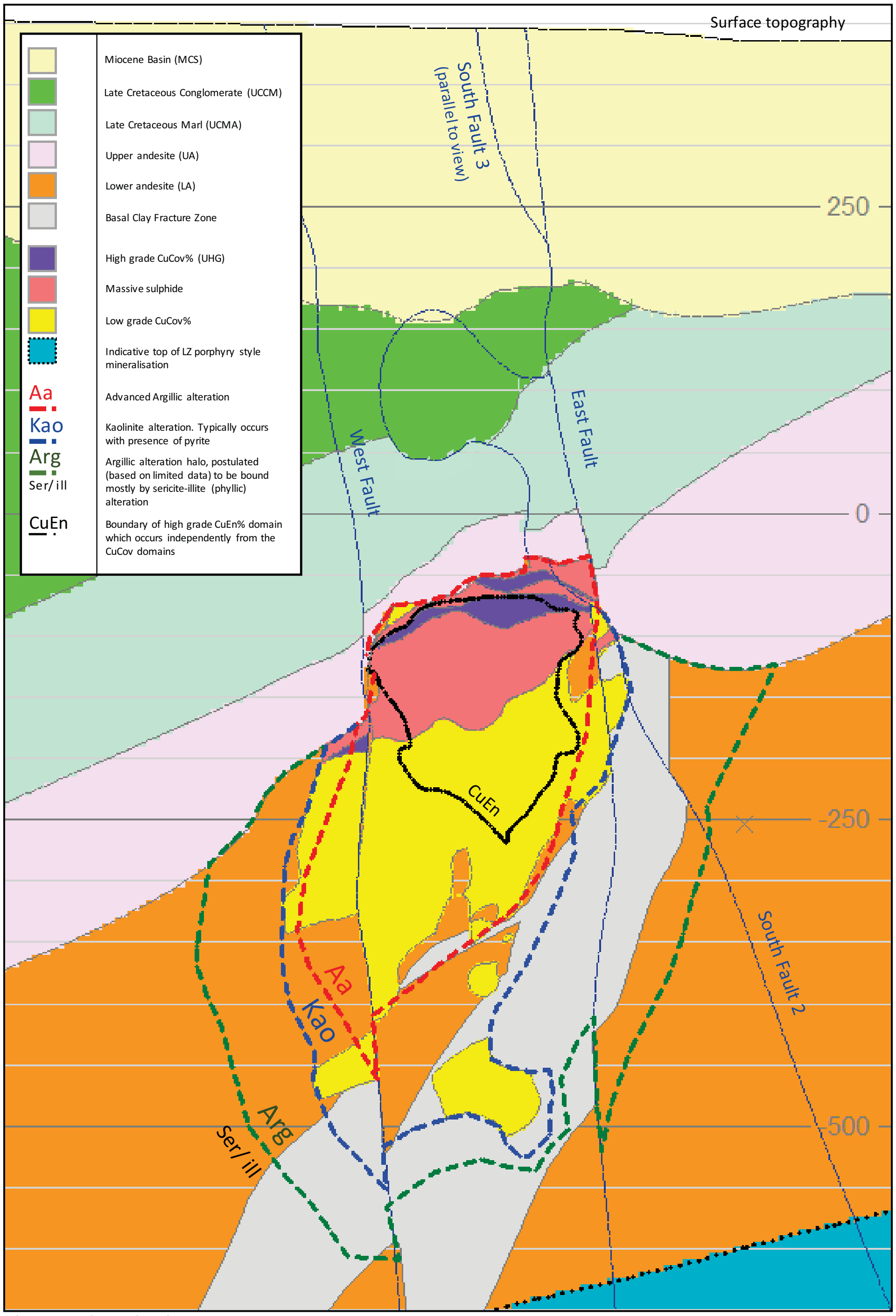
An example of a cross-section showing the mineralization domains in context of the modelled lithology, alteration and fault network is provided in Figure 14.5.

14.9.2.5 Statistical Analysis

Modelled domains were checked to ensure they formed appropriate sample populations for grade estimation, with the presence of any bimodal populations or high-grade histogram tails noted to ensure appropriate representation during block grade estimation.

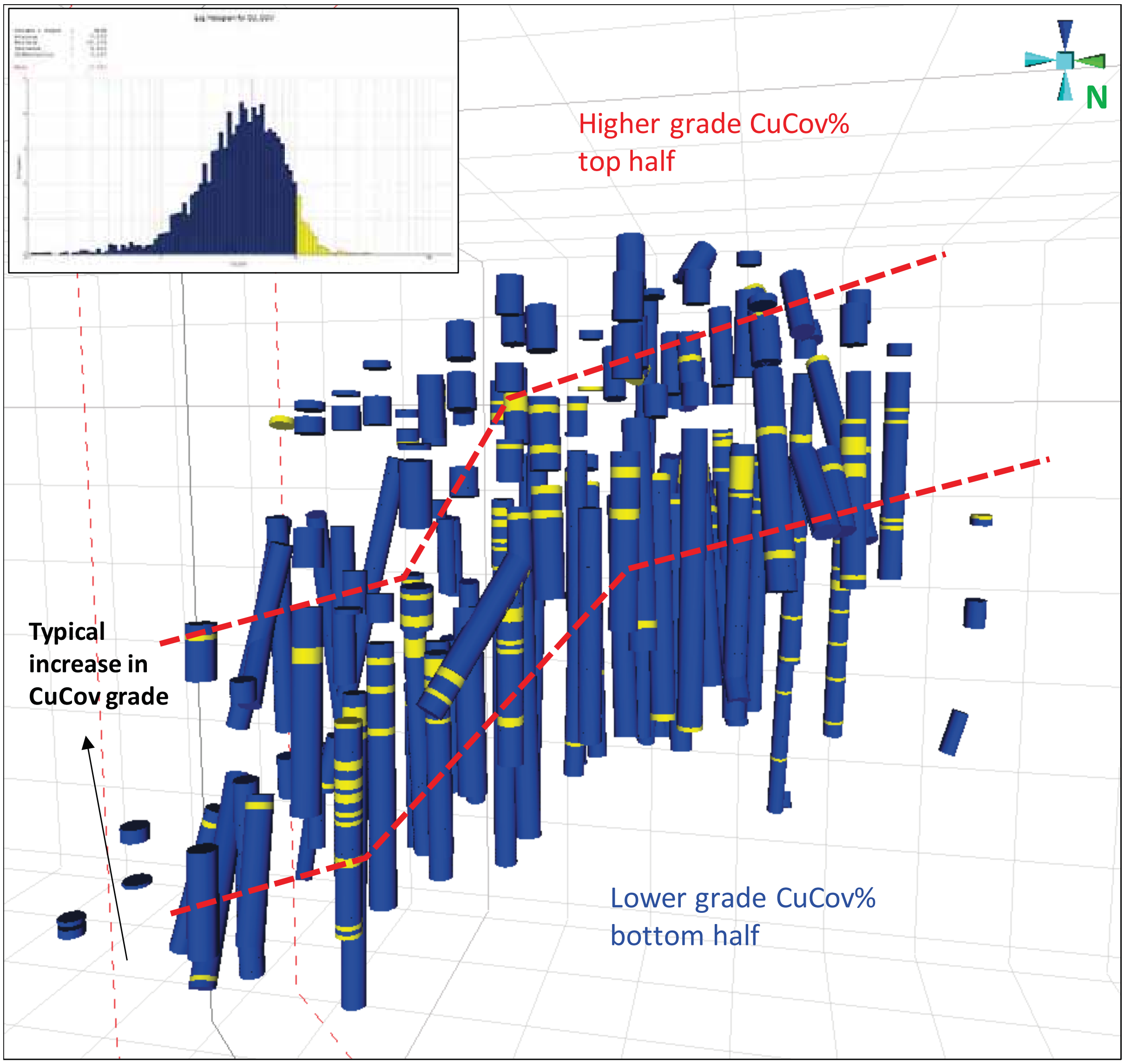
An example of the raw sample grade distribution for CuCov and gold for the massive sulphide domain is illustrated in Figure 14.6 and Figure 14.7. Within this domain, CuCov grades typically reflect a mostly higher-grade middle third (associated with increased presence of massive sulphide) and lower grade top and bottom third, whilst gold grades tend to gradually increase from the west (at depth) towards the upper eastern margin, which is a reflection of the typical gold grade distribution throughout the Upper Zone.

Whilst SRK noted a degree of sample grade zonation in the CuCov, CuEn, gold and arsenic grade data within each of the modelled mineralization domains (as illustrated for CuCov and gold in the figures below), based on visual and statistical assessment SRK considers this to be largely gradational and therefore no further internal statistical grade domaining was deemed necessary.



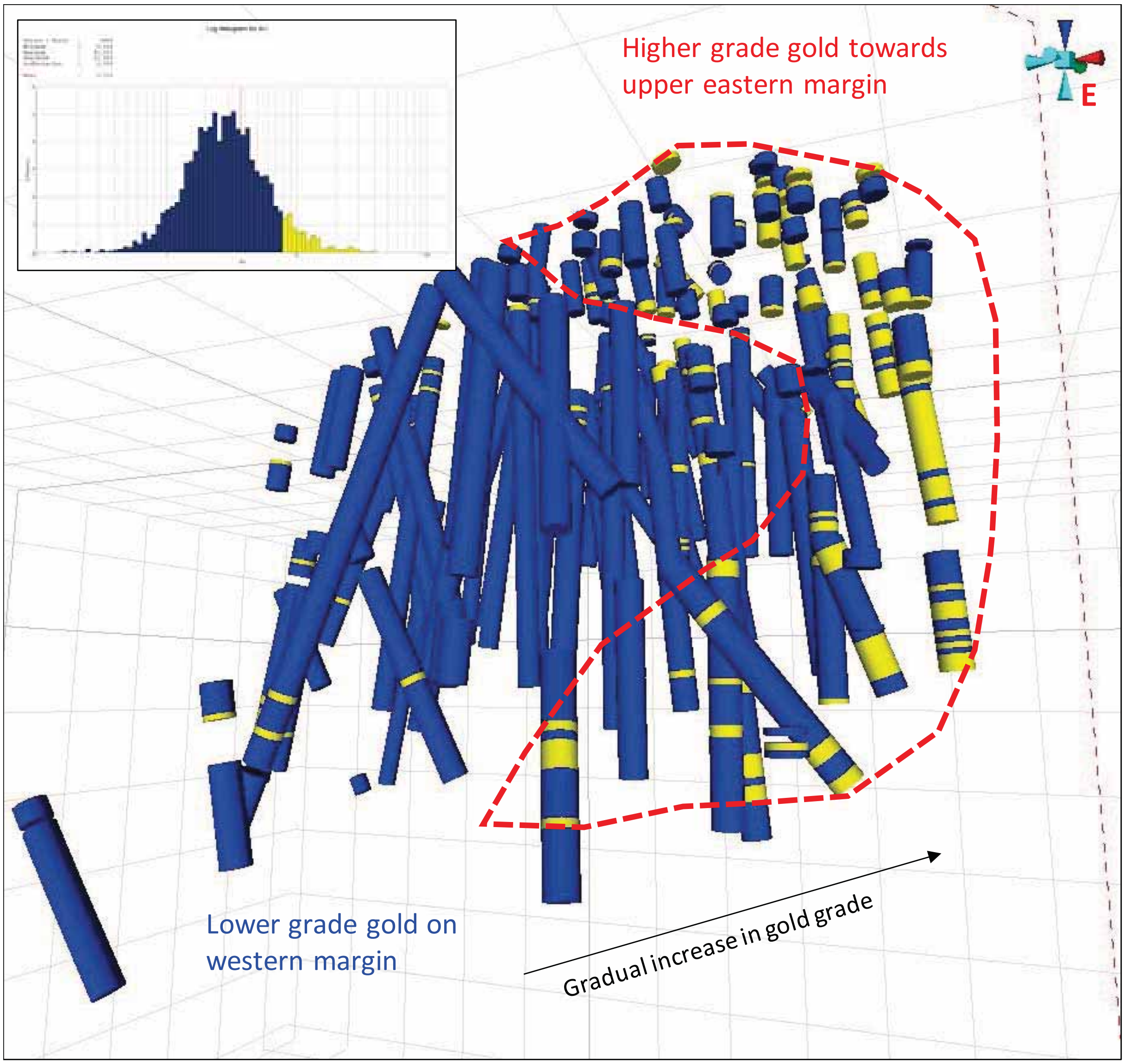
Source: SRK (UK), 2017

Figure 14.5: Schematic Section of the UZ Deposit Looking Northwest (Azimuth 343º)



Note: 25 m grid looking east

Figure 14.6: 3D Visual Review and Log Histogram Plot for CuCov for the Massive Sulphide Domain Samples



Note: 25-m grid looking northwest

Figure 14.7: 3D Visual Review and Log Histogram Plot for Gold for the Massive Sulphide Domain Samples

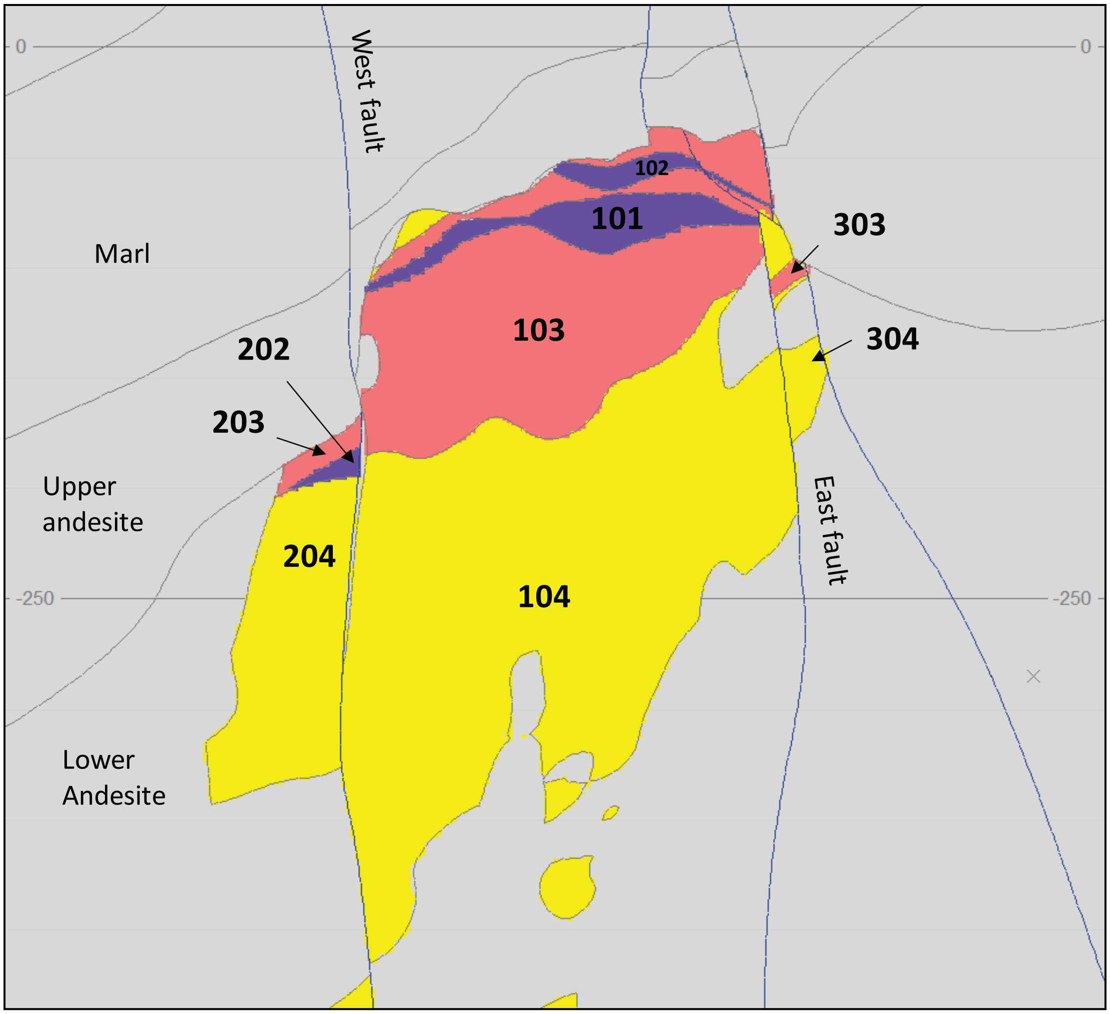
14.9.2.6 Mineralization Model Coding

A summary of the mineralization domains, the estimation domain codes and wireframe names for Čukaru Peki is provided in Table 14.1 and Figure 14.8 for the CuCov (CZONE) domains and Table 14.2 and Figure 14.9 for the CuEn (EZONE) domains.

Mineralization modelled for July 2017 shows a horizontal thickness of up to 250 m, with a vertical extent that ranges from 150 m to greater than 300 m.

Table 14.1: Resource Model CZONE (Copper% in Covellite Mineralization, CuCov) Codes

CZONE Domain Code and wireframe name	Mineralization Envelope Criteria/ Guide	Description
101 - fb3_6_uhg1	Advanced argillic alteration logging, within the lower andesite host unit	High grade copper in covellite ('UHG') domains
102 - fb3_6_uhg2		
103 - fb3_6_ms		Medium grade copper in covellite (massive sulphide) domain
104 - fb3_6_lg_aa		Low grade copper in covellite domain (main fault block)
202 - fb2_uhg		UHG domain located outside of the west fault
203 - fb2_ms		Massive sulphide domain located outside of the west fault
204 - fb2_lg_aa		Low grade covellite domain located outside of the west fault
303 - fb7_ms		Massive sulphide domain located outside of the east fault
304 - fb7_4_lg_aa		Low grade covellite domain located outside of the east fault
404 - fb1_aa		Low grade covellite domain located outside of the south fault
999	Material outside of the advanced argillic alteration domain	Un-mineralized host rock

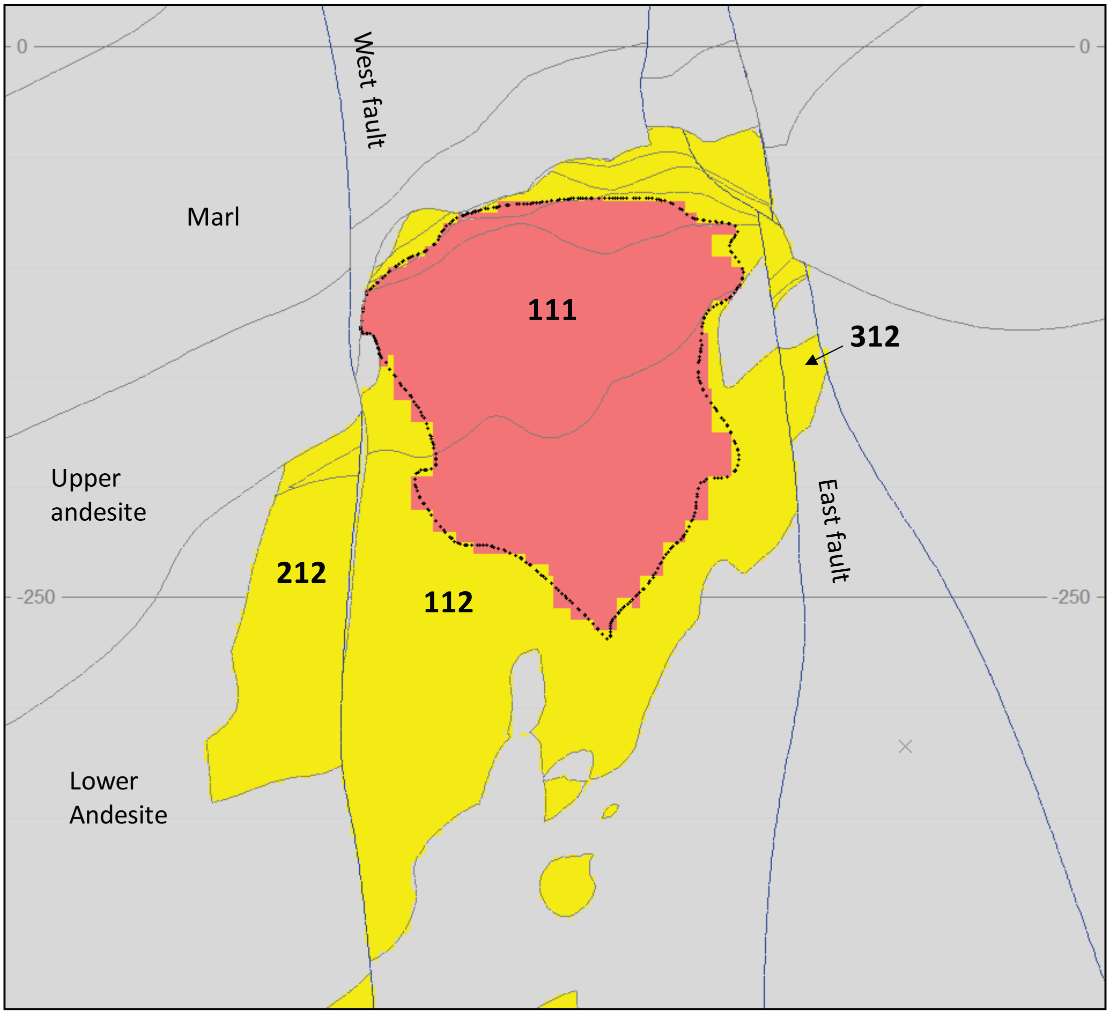


Note: 250 m grid looking northwest (azimuth 343^{o})

Figure 14.8: Resource Model CZONE Codes vs the Mineralization and Geology Domains

Table 14.2: Resource Model EZONE (Copper% in Enargite Mineralization, CuEn) Codes

EZONE Domain Code	Mineralization Envelope Criteria/ Guide	Description
111 - fb3_6_hg_enargite	Advanced argillic alteration logging, within the lower andesite host unit	High grade copper% in enargite (CuEn) domain
112 – (outside 111)		Low grade CuEn domain (main fault block)
212 – (outside 111)		Low grade CuEn domain located outside of the west fault
312 – (outside 111)		Low grade CuEn domain located outside of the east fault
412 – (outside 111)		Low grade CuEn domain located outside of the south fault
999	Material outside of the advanced argillic alteration domain	Un-mineralized host rock. This matches the CZONE 999 domain code.



Note: 250 m grid looking northwest (azimuth 343°)

Figure 14.9: Resource Model EZONE Codes vs the Mineralization and Geology Domains

14.10 3D Modeling – Lower Zone

The 2018 Mineral Resource estimate is based on drilling, site visit, core photo review and modelling of the following geological features for the deposit:

- Fault network.
- Lithology model.
- Mineralization zones based on:
    - Higher-grade copper domain (>= 0.5% Cu).

- Lower grade copper domain (>= 0.2% Cu and < 0.5% Cu).
- Epithermal dominated mineralization.
- Porphyry dominated mineralization.

14.10.1 Geological Wireframes

14.10.1.1 Fault Network

Leapfrog Geo software was used to model the fault network and associated structural domains with respect to the geometry of the Lower Zone.

Because of the sparse drilling, little is known of the location of faults affecting the Lower Zone mineralization. Some of the major faults modelled from the Upper Zone work were extended downwards and used to limit the extent of the Lower Zone mineralization. The Lower Zone mineralization was terminated against the Bor 2 fault and limited to the Lower Andesite Unit.

14.10.1.2 Lithology and Alteration Models

The geological and alteration models for the Lower Zone are not well defined and Rakita is in the process of re-logging all of the Lower Zone drill core in order to develop a better understanding of the geology and alteration associated with the Lower Zone mineralization.

The mineralization of the Lower Zone seems associated with quartz-veined porphyry bodies within andesitic host rock. At least four porphyry phases have been identified (Sillitoe, 2017).

Two principal alteration types have been observed within the Lower Zone thus far, a sericite dominated alteration associated with the high-sulphidation-epithermal mineralization and a potassic alteration assemblage associated with the porphyry style of mineralization.

The sericite alteration is generally pervasive and associated with pyrite and covellite with bornite, chalcopyrite and possibly enargite. This alteration type generally assays anomalous in lead, zinc, bismuth and arsenic.

The potassic alteration is generally best developed along veins and veinlets and is dominated by quartz and anhydrite, chalcopyrite and locally magnetite. This alteration type is generally anomalous in molybdenum and low in arsenic.

14.10.2 Mineralization Wireframes

Two primary wireframes were used to model the Lower Zone mineralization, a 0.5% copper grade shell was generated in Leapfrog Geo to model the higher grade copper population and a 0.2% copper wireframe was used to limit the extent of the block model interpolation. The mineralization wireframes were restricted (clipped) to the Lower Andesite domain and above the Bor 2 fault. While mineralization does appear to extend below the Bor 2 fault, too few drill holes cross the fault and SRK believes that until additional information can be collected across the fault that there isn't enough information to support extending the mineralized wireframes across the fault.

14.10.2.1 *Statistical Analysis*

Modelled grade domains were further subdivided to separate the porphyry style mineralization from the mainly high sulphidation-epithermal mineralization. Mineralized domains were examined to determined if the domain boundaries should be treated as hard or soft boundary and to determine the average grades of each of the domains. checked to ensure they formed appropriate sample populations for grade estimation, with the presence of any bimodal populations or high-grade histogram tails noted to ensure appropriate representation during block grade estimation. Figure 14.10 shows a contact plot for copper across the 0.5 and 0.2% copper grade shells. As can be seen there is a marked difference across the wireframe boundary indicating a hard boundary.

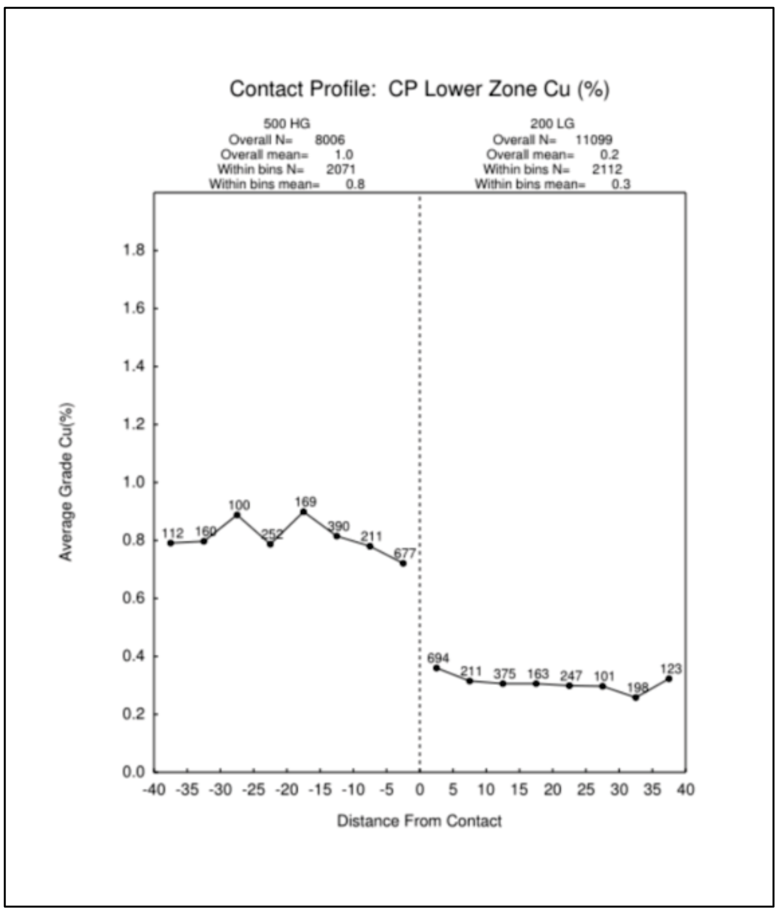


Figure 14.10: Contact Plot across High and Low Grade Copper Domains

Figure 14.11 shows a box plot of the arsenic values for the high sulphidation mineralization (codes 201 and 501) compared with the arsenic content compared with the porphyry mineralization (codes 202 and 502). The high sulphidation mineralization contains significantly more arsenic than the porphyry mineralization, almost by a factor of 10.

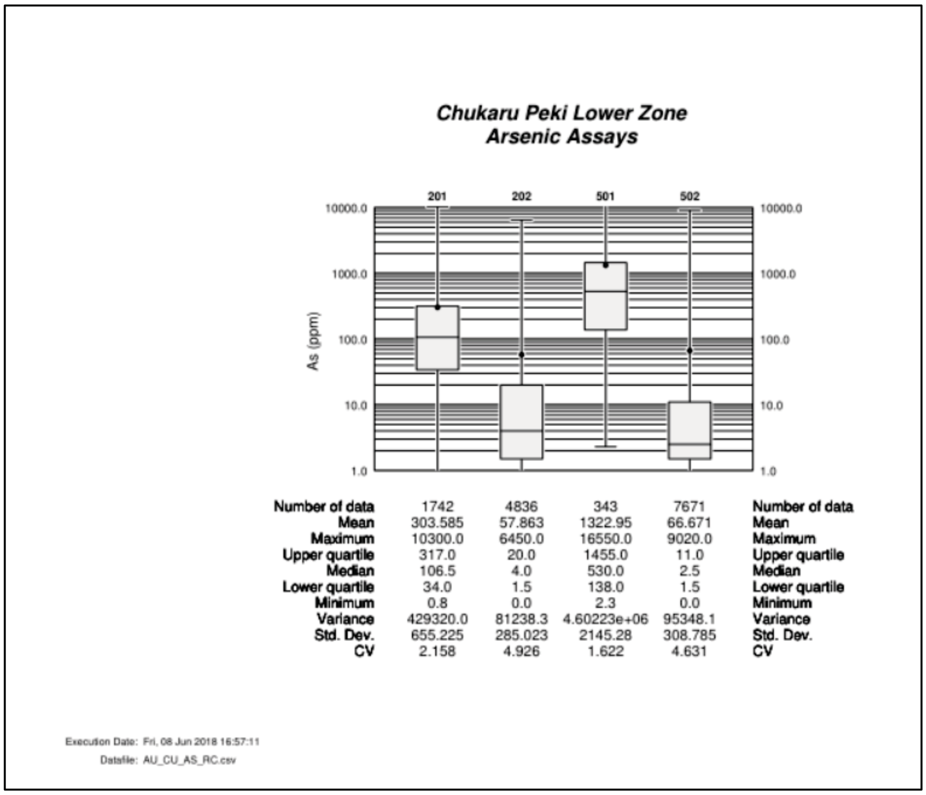


Figure 14.11: Box Plot of Arsenic Values for Lower Zone Mineralization

14.10.2.2 Mineralization Model Coding

A summary of the mineralization domains, the estimation domain codes and wireframe names for Čukaru Peki Lower Zone is provided in Table 14.3 and on Figure 14.12.

Mineralization modelled for July 2017 shows a horizontal thickness of up to 250 m, with a vertical extent that ranges from 150 m to greater than 300 m.

Table 14.3: Lower Zone Resource Model codes

Domain Code	Description
201	Low grade >=0.2% Cu and < 0.5% Cu High Sulphidation Domain
202	Low grade >=0.2% Cu and < 0.5% Cu Porphyry Domain
501	High grade >= 0.5% Cu High Sulphidation Domain
502	High grade >= 0.5% Cu High Porphyry Domain
1000	Upper Zone Mineralization (not used)
99	Un-mineralized host rock

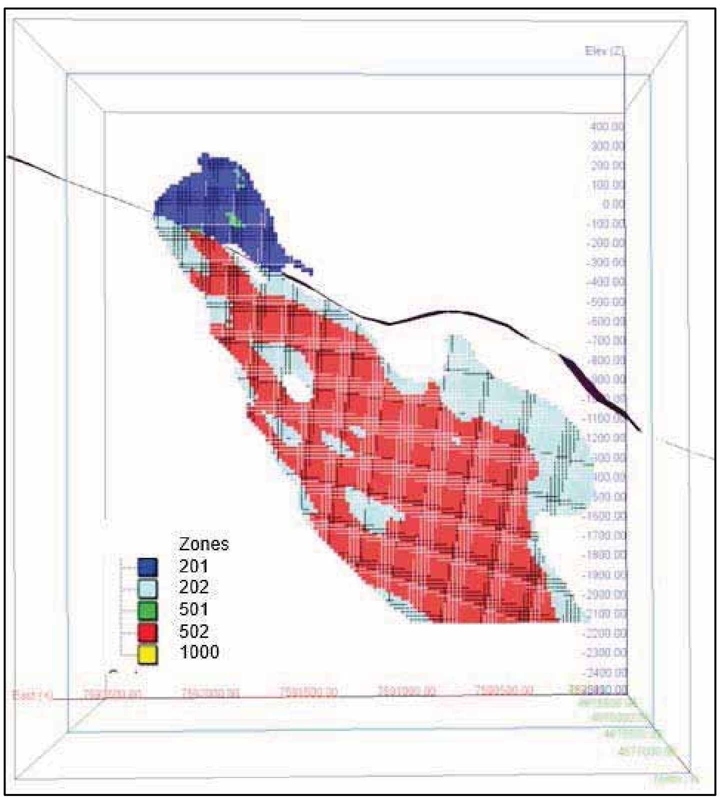


Figure 14.12: Vertical Section of Lower Zone Mineralization Showing Block Model Coding

Note: Section is Looking South, Markers are 500m apart on the horizontal axes and 100 m apart on the vertical axis

14.11 Compositing – Upper Zone

The mean length of the sample data is approximately to 1.2 m. For the UHG domains (CZONE 101, 102 and 202), given the visual observation of grade layering for CuCov and gold, SRK elected to create two-metre composites to ensure that the layering could be reflected during block grade estimation.

In the underlying, lower grade, CuCov domains (CZONE 103 to 104 and 203 to 404), given the broader-scale nature of grade zonation, SRK selected a 10-m composite, which provided a reasonable representation of grade trends, whilst retaining an appropriate number of samples for high confidence local grade estimation.

With regards to CuEn and arsenic, SRK created 10 m composites throughout modelled zones to reflect the grade variability at a visually representative scale.

14.12 Compositing – Lower Zone

The mean length of the sample data is approximately to 2.1 m. over 90 percent of the assays are less than 3.0 m in length, for this reason, SRK decided to composite all the assay data to a fixed 3.0 m length for resource estimation.

14.13 Evaluation of Outliers – Upper Zone

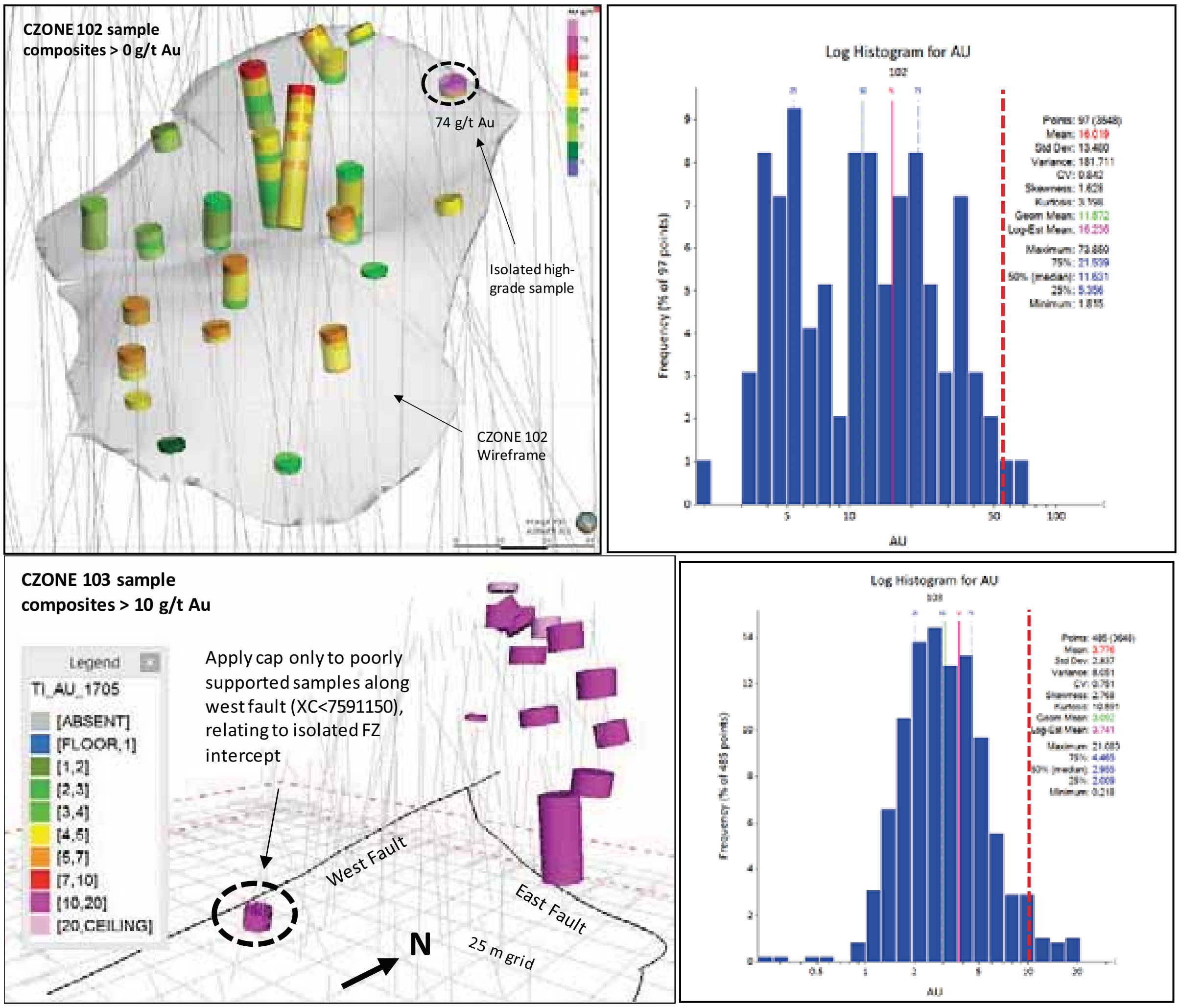
SRK has completed an analysis based on log probability plots, raw and log histograms to identify any very high-grade samples which might have disproportionate impacts on the local grade estimation.

Based on a review of histogram plots for each mineralization domain and visual assessment of sample support, high-grade capping was applied for gold in CZONEs 102, 103 and 203. SRK notes the following with regard to the applied capping:

- A high-grade cap for gold in domain CZONE 102 was applied at 55 g/t Au to prevent a single isolated high-grade sample composite (74 g/t Au) from overly influencing the domain volume.
- A high-grade cap for gold in domain CZONE 103 was applied at 10 g/t Au only to the isolated high-grade samples located west of X = 7591150; this high-grade cap does not apply to the well supported high-grade gold samples situated along the eastern deposit margin.
- A high-grade cap for gold in domain CZONE 203 was applied at 10 g/t Au to prevent a high grade drillhole intercept (within this poorly drilled fault-block domain) from overly influencing the domain volume.

Log histograms for gold showing selected high-grade capping limits and associated visual reviews are illustrated for domains 102 and 103 in Figure 14.13.

No high-grade capping was applied for CuCov, CuEn or arsenic.



Note: Capping limits in red on histogram plots for CZONE's 102 (top) and 203 (bottom)

Figure 14.13: High Grade Outlier Review for Gold Showing Selected Capping Limits

14.14 Evaluation of Outliers – Lower Zone

SRK completed an analysis of the assay data based on log probability plots, raw and log histograms to identify any very high-grade samples which might have disproportionate impacts on the local grade estimation.

Based on a review of histogram plots for each mineralization domain and visual assessment of sample support, high-grade capping was applied to gold, copper and arsenic as outlined in Table 14.4.

Table 14.4: Capping Levels for Lower Zone Estimation

Metal	Model Codes	Capping Levels	Number of Assays capped
Cu (%)	201	4.0	4
Cu (%)	501	3.5	16
Cu (%)	202	2.5	5
Cu (%)	502	4.0	10
Au (g/t)	201	No Capping	0
Au (g/t)	501	3.0	10
Au (g/t)	202	No Capping	0
Au (g/t)	502	No Capping	0
As (ppm)	201	2,000	32
As (ppm)	501	5,000	21
As (ppm)	202	1,500	20
As (ppm)	502	3,500	11

14.15 Statistical Analysis – Estimation Composites – Upper Zone

Estimation composites for grade interpolation comprise the 2.0-m capped-composite samples for the UHG domains (CZONE101, 102 and 202) and 10.0-m capped-composite samples for all other estimation domains. A log histogram and log-probability plot is illustrated for CuCov composites in Figure 14.14.

Table 14.5 and Table 14.6 present sample summary statistics for each of the domains and grade variables. The selected capping limit and a comparison of the mean grades within each estimation domain based on the grade capping (where applied) is also presented.

With the exception of the poorly drilled CZONE 203 domain where few high gold grade samples skew the mean of the raw composite samples towards higher grade, the results show that the global reduction in the gold grade in capped zones is in the order of 1%, which SRK deems to be within acceptable margins.

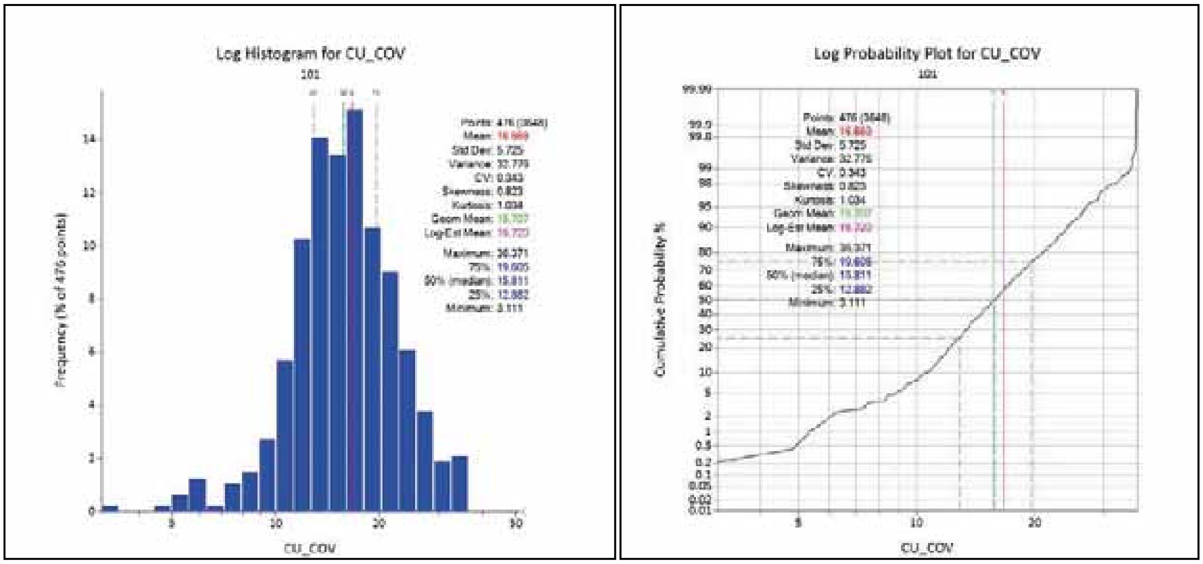


Figure 14.14: Log Histogram and Log Probability Plot for CuCov for the UHG CZONE 101 Domain at Čukaru Peki

Table 14.5: Comparison of Mean Composite Grades (Raw Composite versus Capped) for CuCov% and Gold g/t*

CZONE	Field	No. samples	Min	Max	Mean	Cap	Var	StdDev	COV	%DIFF
101	AU	476	0.85	37.63	11.18	-	43.19	6.57	0.59	-
	CU_COV	476	3.11	36.37	16.67		32.71	5.72	0.34	
102	AU	97	1.82	73.85	16.02	55.00	179.84	13.41	0.84	-1.3%
	AUCAP	97	1.82	55.00	15.82		160.38	12.66	0.80	
	CU_COV	97	3.77	31.73	15.23	-	35.49	5.96	0.39	-
103	AU	485	0.22	21.08	3.78	10.00*	8.03	2.83	0.75	-0.3%
	AUCAP	485	0.22	21.08	3.76		7.82	2.80	0.74	
	CU_COV	485	0.29	12.95	4.68	-	5.02	2.24	0.48	-
104	AU	1019	0.01	12.90	1.21	-	1.75	1.32	1.09	-
	CU_COV	1019	0.00	6.91	1.41		1.09	1.05	0.74	
202	AU	12	11.05	51.23	22.13	-	104.07	10.20	0.46	-
	CU_COV	12	14.07	27.96	20.41		17.04	4.13	0.20	
203	AU	11	0.85	15.46	3.93	10.00	17.24	4.15	1.06	-12.6%
	AUCAP	11	0.85	10.00	3.44		8.27	2.88	0.84	
	CU_COV	11	1.74	8.58	5.26	-	3.70	1.92	0.37	-
204	AU	101	0.03	1.59	0.41	-	0.09	0.29	0.71	-
	CU_COV	101	0.00	6.27	1.00		0.82	0.91	0.90	
303	AU	15	0.33	10.11	5.20	-	10.35	3.22	0.62	-
	CU_COV	15	0.53	13.87	4.34		13.07	3.62	0.83	
304	AU	65	0.18	12.07	1.73	-	6.71	2.59	1.50	-
	CU_COV	65	0.00	3.19	0.66		0.51	0.71	1.08	
404	AU	1	3.93	3.93	3.93	-	-	-	-	-
	CU_COV	1	2.85	2.85	2.85		-	-	-	

*The high-grade cap for gold in domain 103 is applied only to the isolated high-grade samples located west of X= 7591150. This high-grade cap does not apply to the well supported high-grade gold samples situated along the eastern deposit margin.

Table 14.6: Comparison of Mean Composite Grades (Raw Composite versus Capped) for CuEn% and Arsenic%

CZONE	Field	No. samples	Min	Max	Mean	Cap	Var	StdDev	COV	%DIFF
111	AS	635	0.02	1.43	0.35	-	336	0.18	0.13	-
	CU_EN	635	0.04	3.64	0.90	-	0.2	0.47	0.13	-
112	AS	971	0.00	0.48	0.08	-	51.1	0.07	0.15	-
	CU_EN	971	0.00	1.23	0.22	-	0.03	0.18	0.15	-
212	AS	114	0.01	0.36	0.05	-	38.8	0.06	0.17	-
	CU_EN	114	0.01	0.92	0.13	-	0.03	0.16	0.17	-
312	AS	78	0.00	0.22	0.03	-	13.2	0.04	0.16	-
	CU_EN	78	0.01	0.56	0.08	-	0.01	0.09	0.16	-
412	AS	1	0.00	0.001	0.001	-	-	-	-	-
	CU_EN	1	0.00	0.002	0.002	-	-	-	-	-

14.16 Statistical Analysis – Estimation Composites – Lower Zone

Estimation composites for grade interpolation comprise the 3.0-m capped-composite samples for all Lower Zone domains. Table 14.7 present summary statistics for each of the domains and grade variables for the capped composited assay data.

Table 14.7: Basic Statistical Data for capped 3 m Composites for the Lower Zone

ZONE	Metal	No. samples	Min	Max	Mean	Var	StdDev	COV
201	Cu (%)	1350	0	4.0	0.22	0.14	0.38	1.73
201	Au (g/t)	1350	0	3.70	0.14	0.04	0.20	1.43
201	As (ppm)	1350	0	3,730	196	114,357	338	1.72
501	Cu (%)	214	0	3.34	0.85	0.60	0.77	0.91
501	Au (g/t)	214	0	3.0	0.50	0.46	0.68	1.36
501	As (ppm)	214	0	5,000	978	1,572,148	1254	1.28
202	Cu (%)	4264	0	2.31	0.24	0.04	0.20	0.83
202	Au (g/t)	4264	0	1.76	0.06	0.01	0.09	1.50
202	As (ppm)	4264	0	1,500	34	14,793	122	3.59
502	Cu (%)	6541	0	4.0	0.84	0.30	0.55	0.65
502	Au (g/t)	6541	0	1.68	0.17	0.02	0.14	0.82
502	As (ppm)	6541	0	3,500	55	61,901	249	4.53

14.17 Geostatistical Analysis – Upper Zone

Variography is the study of the spatial variability of an attribute, in this case CuCov, CuEn gold and arsenic grade. Snowden’s Supervisor software was used for geostatistical analysis.

In completing the analysis for the mineralization domains, experimental semi-variograms were calculated in the along-strike, down-dip and across-strike orientations, with a short-lag variogram calculated to characterize the nugget effect.

With the exception of EZONE 112 for CuEn and arsenic, directional variograms were modelled for all mineralization domains. All variances were re-scaled for each mineralized zone to match the total variance (Var) for that zone.

As an example, the variogram model and parameters for the mineralization domain CZONE 103 for CuCov are shown in Table 14.8 and Figure 14.15.

Table 14.8: Summary of Modelled Semi-Variogram Parameters for the Čukaru Peki UZ Mineralization Domain CZONE 103

Variogram Parameter	CZONE 103-CU_COV
Co	0.83
C1	2.97
A1 – Along Strike (m)	37
A1 – Down Dip (m)	32
A1 – Across Strike (m)	23
C2	1.22
A2 – Along Strike (m)	98
A2 – Down Dip (m)	100
A2 – Across Strike (m)	40
C3	0.00
A3 – Along Strike (m)	0
A3 – Down Dip (m)	0
A3 – Across Strike (m)	0
Nugget Effect (%)	17%

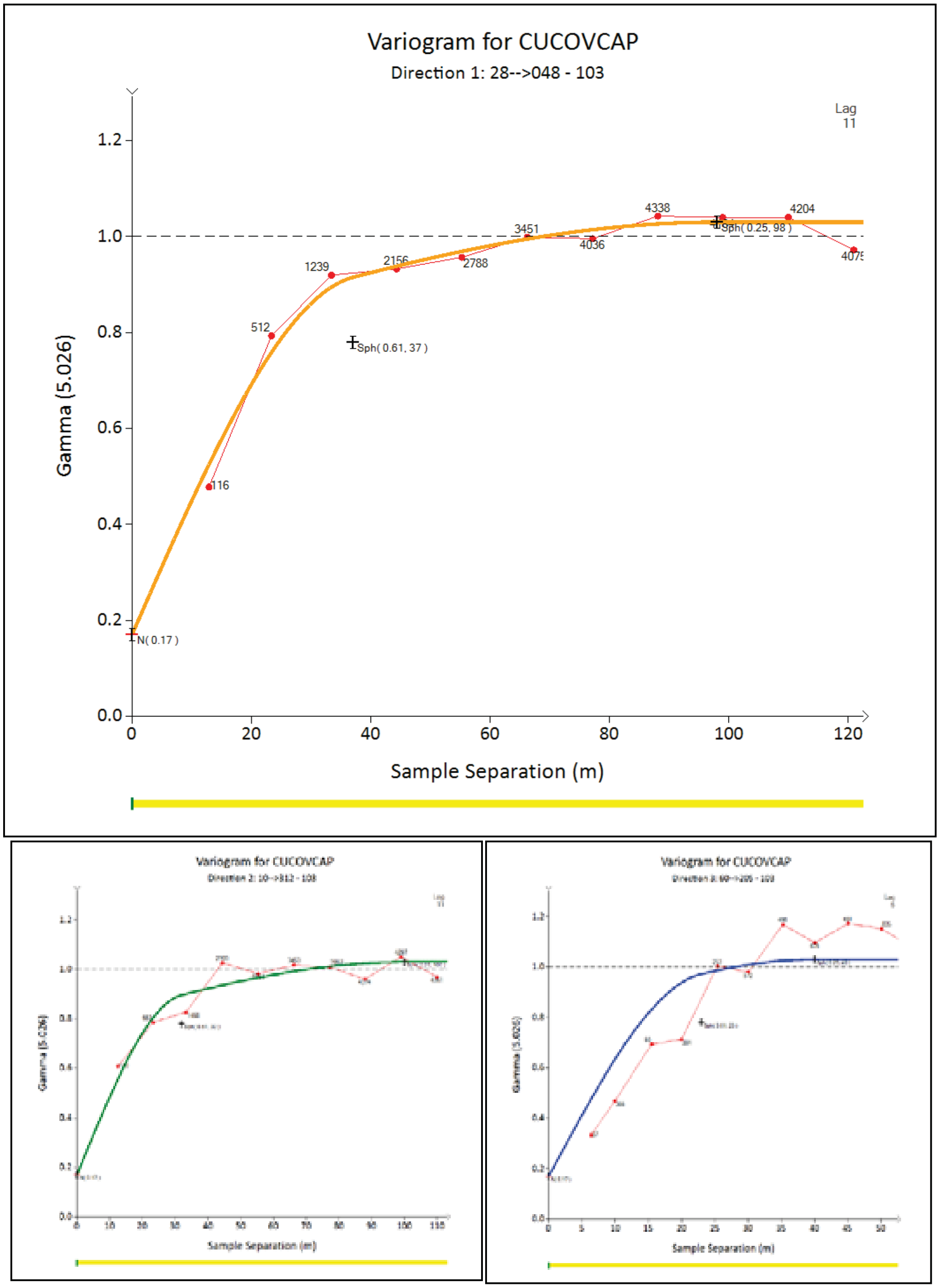


Figure 14.15: Variogram Models for CuCov for Domain CZONE 103 Showing Along Strike (top), Down Dip (bottom left) and Across Strike (bottom right)

14.18 Geostatistical Analysis – Lower Zone

Variography is the study of the spatial variability of an attribute, in this case copper, gold and arsenic grade. Sage2001 software was used for geostatistical analysis.

In completing the analysis for the mineralization domains, experimental correlograms were calculated in the along-strike, down-dip and across-strike orientations, with a short-lag variogram calculated to characterize the nugget effect.

Because of the prominent vertical component of the drilling in the Lower Zone domains, the range of continuity in any direction but vertical was difficult to discern. Indicator correlograms were used to guide the primary orientations where regular pairwise normalized structures were difficult to generate. The modelled correlograms parameters for the mineralization domains for the Lower Zone are shown in Table 14.9.

Table 14.9: Summary of Modelled Correlograms Parameters for the Lower Zone Mineralization

Domain	Metal	Model Type	Nugget (C0)	C1 & C2	Rotation			Range		
					(Z)	(Y)	(Z)	Rot X	Rot Y	Rot Z
High sulphidation (201, 501)	Au	Exponential	0.165	0.835	-2	-82	11	75	319	9
				ND	ND	ND	ND	ND	ND	ND
	Cu	Exponential	0.238	0.463	-84	28	85	57	138	8
				0.298	-84	28	85	60	350	263
	As	Exponential	0.23	0.769	1	-82	1	130	406	8
				ND	ND	ND	ND	ND	ND	ND
Porphyry (202, 502)	Au	Exponential	0.2	0.346	29	-2	-30	422	442	79
				0.453	29	-2	-30	11	909	954
	Cu	Exponential	0.439	0.56	2	75	82	362	187	50
				ND	ND	ND	ND	ND	ND	ND
	As	Exponential	0.15	0.442	44	43	-76	29	14	12
				0.407	44	43	-76	38	362	263

Note: ND = not determined

14.19 Block Model and Grade Estimation – Upper Zone

A block model prototype was created for Čukaru Peki based on the Gauss-Krüger coordinate system. Block model parameters were chosen per domain to reflect the average drillhole spacing (along strike and on section) and to appropriately reflect the grade variability both horizontally and vertically.

To improve the geometric representation of the geological model, sub-blocking was allowed along the boundaries to a minimum of 2 x 2 x 1 m (x, y, and z). A summary of the block model parameters for the UHG domains and underlying, lower grade CuCov domains are given in Table 14.10.

Table 14.10: Details of Block Model Dimensions for UZ Grade Estimation*

Model Domain	Dimension	Origin (UTM)	Block Size	Number of Blocks	Min Sub-blocking (m)
CZONE 101-102	X	7590900	10	57	2
	Y	4875500	10	54	2
	Z	-595	5	120	1
CZONE 103-304	X	7590900	25	23	2
	Y	4875500	25	22	2
	Z	-595	20	30	1

After grade estimation, all model domains were re-blocked to 10 x 10 x 5 m using the block model framework shown for CZONE 101-102. Given the relatively broad scale of the CuEn wireframes, no sub-blocking was used (beyond the 10 x 10 x 5 m block size) to represent the internal boundary between the high and low grade (EZONE) domains.

14.20 Block Model and Grade Estimation – Lower Zone

A block model was generated for the Čukaru Peki Lower Zone based on the Gauss-Krüger coordinate system. Block model parameters were chosen to reflect the average drillhole spacing (along strike and on section) and to appropriately reflect the grade variability both horizontally and vertically.

A summary of the block model parameters are given in Table 14.11.

Table 14.11: Details of Lower Zone Block Model Dimensions

Dimension	Minimum	Maximum	Extent	Block Size	Number of Blocks
X	7,590,000	7,592,700	2,700	18	150
Y	4,875,000	4,877,502	2,502	18	139
Z	-2,400	498	2,988	18	166

14.21 Final Estimation Parameters – Upper Zone

Ordinary kriging was used for the grade interpolation of CuCov, CuEn, gold and arsenic, with the sum of the CuCov and CuEn block grades used to derive total copper grade (Cu) %.

Search ellipses were orientated to follow the trend of each domain with Datamine's Dynamic Anisotropy used to control search ellipse orientation in the UHG and massive sulphide domain (CZONE 101 to 103). Domain boundaries have been treated as hard boundaries during the estimation process.

Inverse distance weighting was used for the interpolation of grade for the poorly drilled domains located outside of the main fault block (i.e. CZONE 202 to 304 and EZONE 212 to 412, given too few samples to define a variogram of sufficient clarity), the interpolation of total sulphur values (to derive density via regression) and for verification of the ordinary kriging estimates for CuCov, CuEn, gold and arsenic. The selected estimation parameters have been verified based on the results of a quantitative kriging neighbourhood analysis, and are presented in Table 14.12 and Table 14.13.

Table 14.12: Summary of Final Estimation Parameters for Čukaru Peki UZ CZONE Domains

Estimation Parameters								Description
KZONE	101, 102	103	104	104	202	203,303,204	304*,404	Kriging zone for estimation
FIELD	CU_COV, AUCAP		CU_COV	AUCAP	CU_COV, AUCAP			Field for interpolation
SREFNUM	1,2	3,8	4	5	6	7	9,10	Search reference number
SMETHOD	2	2	2	2	2	2	2	Search volume shape (2 = ellipse)
SDIST1	30	55	65	45	45	45	45	Search distance 1 (dip)
SDIST2	30	55	40	45	45	45	45	Search distance 2 (strike)
SDIST3	10	20	40	45	45	45	45	Search distance 3 (across strike)
SANGLE1	0	0	-20	0	0	0	0	Search angle 1 (dip direction)
SANGLE2	0	0	50	0	0	0	0	Search angle 2 (dip)
SANGLE3	0	0	90	0	0	0	0	Search angle 3 (plunge)
SAXIS1	3	3	3	3	3	3	3	Search axis 1 (z)
SAXIS2	1	1	1	1	1	1	1	Search axis 2 (x)
SAXIS3	3	3	3	3	3	3	3	Search axis 3 (z)
MINNUM1	9	12	12	12	10	4	4	Minimum sample number (SVOL1)
MAXNUM1	30	40	36	36	30	16	16	Maximum sample number (SVOL1)
SVOLFAC2	2	2	2	2	2	2	2	Search distance expansion (SVOL2)
MINNUM2	9	12	12	12	10	4	4	Minimum sample number (SVOL2)
MAXNUM2	30	40	36	36	30	16	16	Maximum sample number (SVOL2)
SVOLFAC3	3	3	3	3	3	3	3	Search distance expansion (SVOL3)
MINNUM3	3	4	2	2	2	2	1	Minimum sample number (SVOL3)
MAXNUM3	30	40	36	36	30	16	16	Maximum sample number (SVOL3)
MAXKEY	3	4	0	0	0	0	0	Maximum no. of samples per drillhole
SANGL1_F	TRDIPDIR	TRDIPDIR	0	0	0	0	0	Dynamic Anisotropy ("0" = not used)
SANGL2_F	TRDIP	TRDIP	0	0	0	0	0	

Table 14.13: Summary of Final Estimation Parameters for Čukaru Peki UZ EZONE Domains

Estimation Parameters						Description
KZONE	111	112	212	312	412	Kriging zone for estimation
FIELD	CU_EN, AS					Field for interpolation
SREFNUM	1	2	3	5	4	Search reference number
SMETHOD	2	2	2	2	2	Search volume shape (2 = ellipse)
SDIST1	65	45	45	45	45	Search distance 1 (dip)
SDIST2	40	45	45	45	45	Search distance 2 (strike)
SDIST3	40	45	45	45	45	Search distance 3 (across strike)
SANGLE1	-155	0	0	0	0	Search angle 1 (dip direction)
SANGLE2	120	0	0	0	0	Search angle 2 (dip)
SANGLE3	100	0	0	0	0	Search angle 3 (plunge)
SAXIS1	3	3	3	3	3	Search axis 1 (z)
SAXIS2	1	1	1	1	1	Search axis 2 (x)
SAXIS3	3	3	3	3	3	Search axis 3 (z)
MINNUM1	12	12	4	4	4	Minimum sample number (SVOL1)
MAXNUM1	36	36	16	16	16	Maximum sample number (SVOL1)
SVOLFAC2	2	2	2	2	2	Search distance expansion (SVOL2)
MINNUM2	12	12	4	4	4	Minimum sample number (SVOL2)
MAXNUM2	36	36	16	16	16	Maximum sample number (SVOL2)
SVOLFAC3	3	3	3	4	3	Search distance expansion (SVOL3)
MINNUM3	2	2	2	2	1	Minimum sample number (SVOL3)
MAXNUM3	36	36	16	16	16	Maximum sample number (SVOL3)
MAXKEY	0	0	0	0	0	Maximum number of samples per drillhole
SANGL1_F	TRDIPDIR	0	0	0	0	Dynamic Anisotropy ("0" = not used)
SANGL2_F	TRDIP	0	0	0	0	

14.22 Estimation Parameters – Lower Zone

Ordinary kriging was used for the grade interpolation of copper, gold and arsenic grades into the Lower Zone block model. Search ellipses were orientated to follow the main trend of the large 0.2% Copper wireframe. Domain boundaries were all treated as hard boundaries during the estimation process.

Grade interpolation was carried out in four passes with the first three passes having increasing search radii and requiring at least two drill holes to interpolate a block grade. Grade were only interpolated if they had not been estimated by the previous pass. A final fourth pass was run to interpolate any interpolated blocks that were within 50 m of drill holes contained within the mineralized wireframes. Table 14.14 summarises the grade interpolation parameters for all passes.

Bulk density was interpolated into the block model using inverse distance weighting in a single pass using a minimum of eight samples. Any blocks that had not been estimated were assigned a density equal to the average density value for the mineralized domain as outlined in Table 14.15.

Table 14.14: Lower Zone Grade Interpolation Parameters

Pass	Minimum Number of Composites	Maximum Number of Composites	Maximum Number per DDH	Range (X)	Range (Y)	Range (Z)
1	8	24	6	75	75	30
2	8	24	6	150	150	60
3	8	24	6	250	250	120
4	6	24	6	50	50	50

Table 14.15: Average Density Values Assigned to Un-estimated Blocks by Domains

Domain	Average Density
201	2.78
202	2.8
501	2.78
502	2.76

14.23 Model Validation and Sensitivity – Upper Zone

14.23.1 Sensitivity Analysis

Grade estimation was verified through a quantitative kriging neighbourhood analysis exercise which was based on varying kriging parameters for CuCov (i.e. number of samples and search ellipse size) to test a number of different scenarios. This focused on the UHG (CZONE 101) and massive sulphide (CZONE 103) domains given their significant contributions to copper metal (20% and 35% respectively) in the resource model.

In general, these domains are relatively insensitive to changes in the estimation parameters. SRK noted, however, that block grades (visually) better reflected the sample variability by restricting the search ellipse dimension and maximum number of composites per drillhole to within reasonable limits. The final parameters were selected to ensure that the CuCov grade layering and zonation interpreted to exist in the deposit were appropriately reflected in block grade estimates.

14.23.2 Block Model Validation

SRK has validated the block model using the following techniques:

- Visual inspection of block grades in comparison with drillhole data.

- Sectional validation of the mean samples grades in comparison to the mean model grades.
- Comparison of block model statistics using ordinary kriging and inverse distance weighting grade estimates.

14.23.2.1 *Visual Validation*

Visual validation provides a comparison of the interpolated block model on a local scale. A thorough visual inspection has been undertaken in section and 3D, comparing the sample grades with the block grades, which demonstrates in general good comparison between local block estimates and nearby samples, without excessive smoothing in the block model. Figure 14.16 to Figure 14.19 show examples of the visual validation checks and highlight the overall block grades corresponding with composite sample grades for CuCov, CuEn, total copper, gold and arsenic.

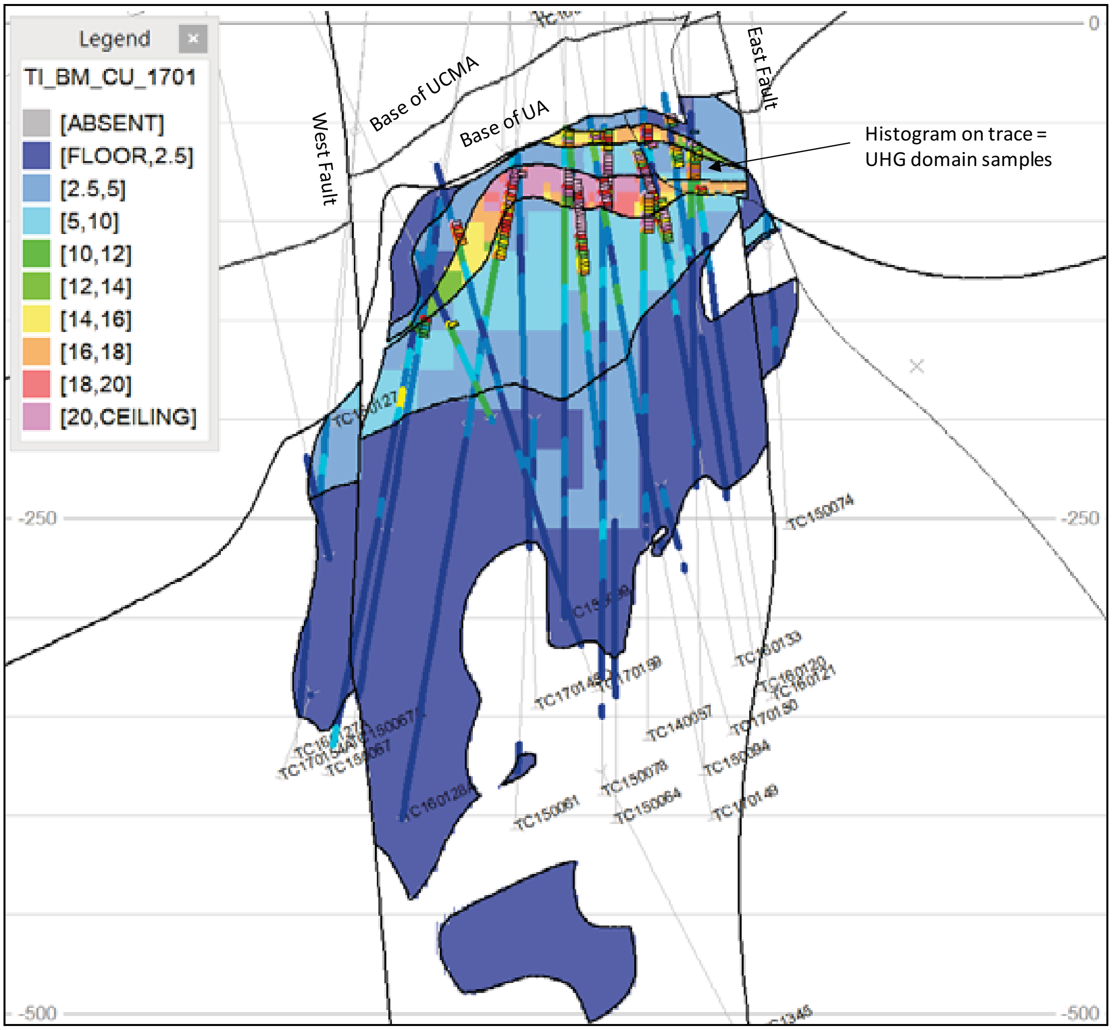


Figure 14.16: Čukaru Peki Block Model CuCov (%) Grade Distribution Looking Northwest

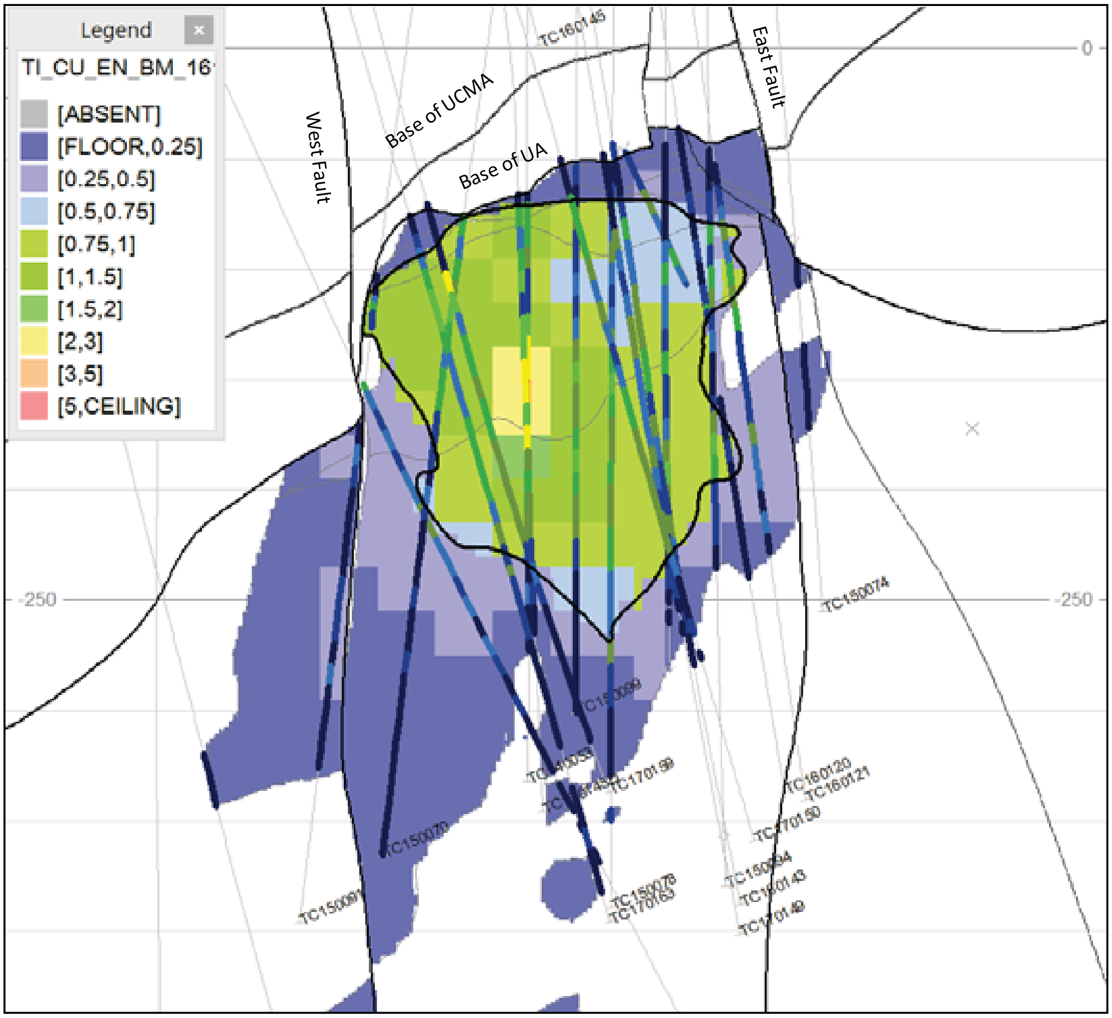


Figure 14.17: Čukaru Peki Block Model CuEn (%) Grade Distribution Looking Northwest

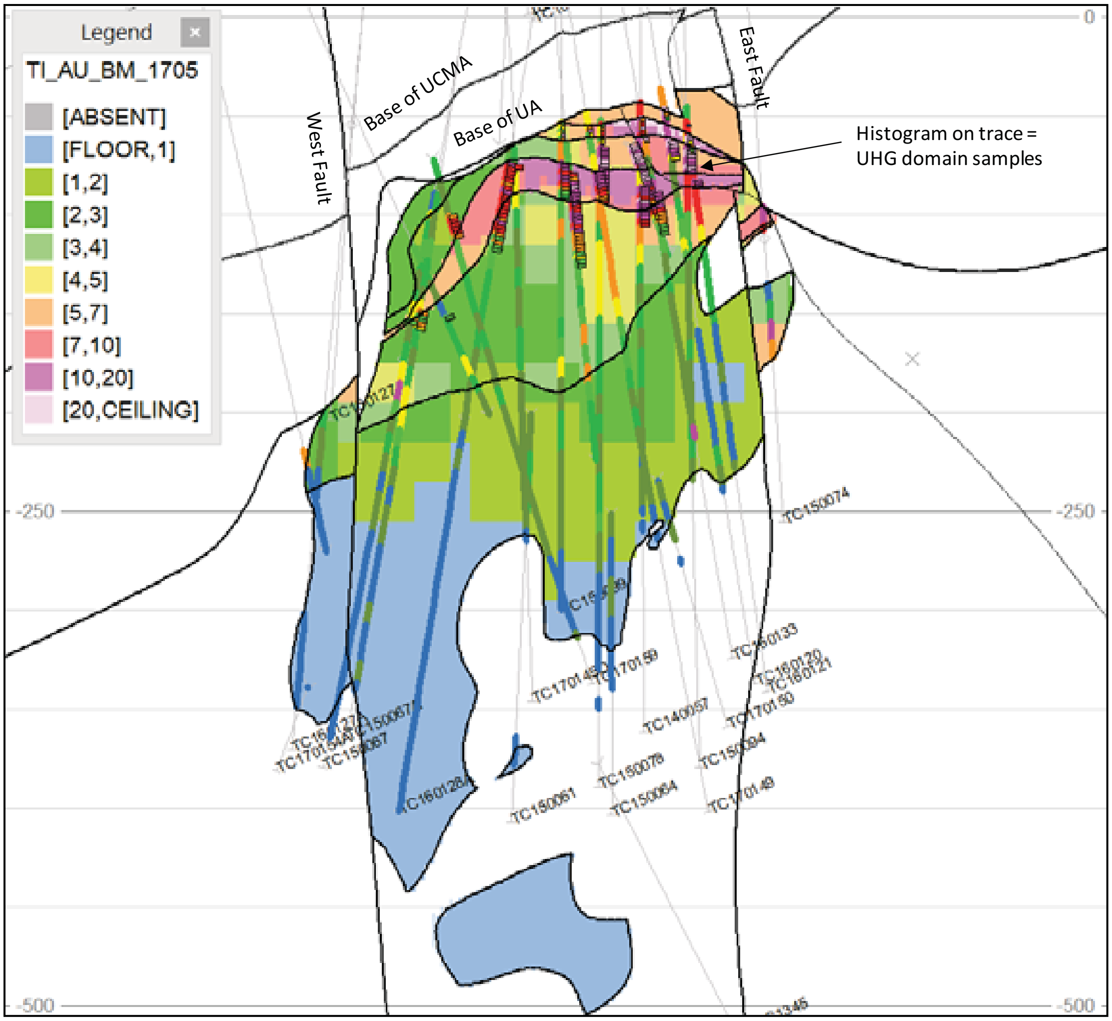


Figure 14.18: Čukaru Peki Block Model Gold (g/t) Grade Distribution Looking Northwest

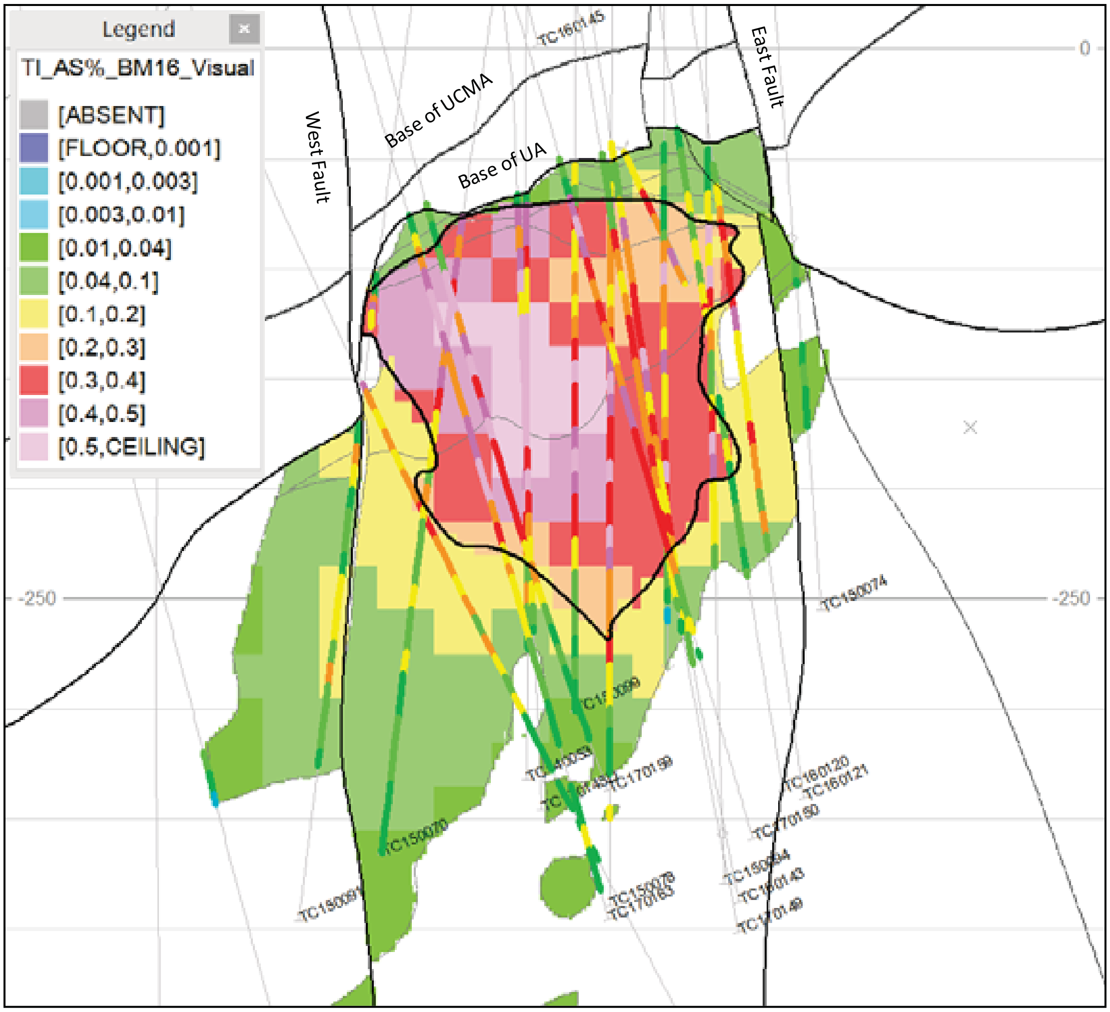
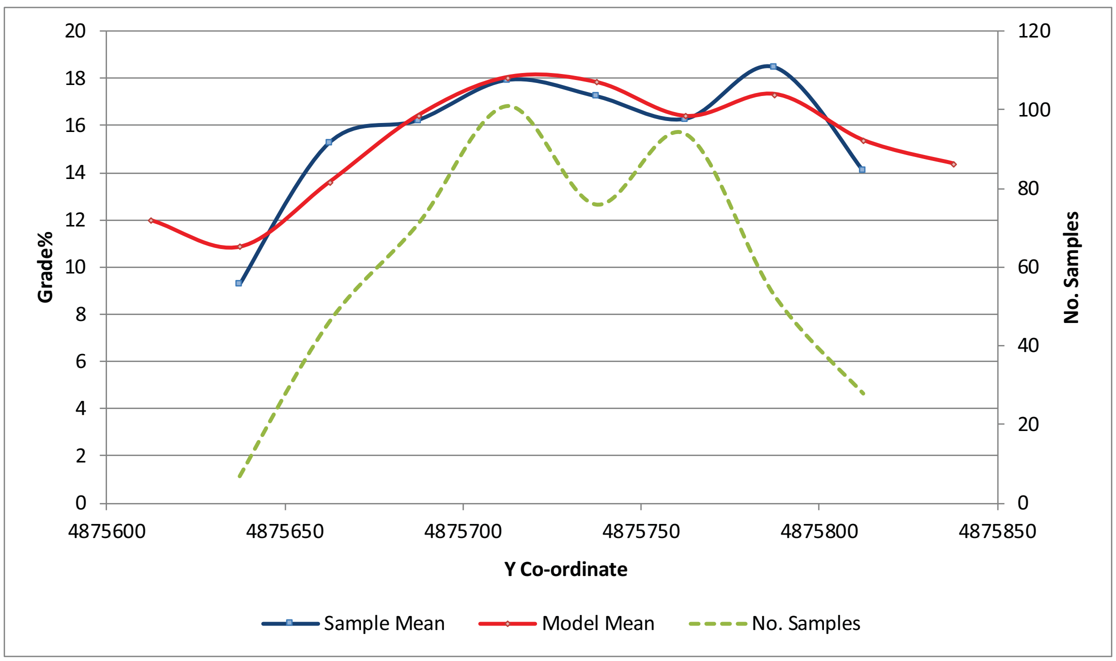


Figure 14.19: Čukaru Peki Block Model Arsenic (ppm) Grade Distribution Looking Northwest

14.23.2.2 Sectional Validation

As part of the validation process, the input composite samples are compared to the block model grades within a series of coordinates (based on the principle directions). The results of which are then displayed on charts to check for visual discrepancies between grades. Figure 14.20 shows the results for the copper grades for the UHG domain CZONE 101 based on section lines cut along y-coordinates.

The resultant plots show a reasonable correlation between the block model grades and the composite grades, with the block model showing a typically smoothed profile of the composite grades as expected. SRK notes that in less densely sampled areas, minor grade discrepancies do exist on a local scale. Overall, however, SRK is confident that the interpolated grades reflect the available input sample data and the estimate shows no sign of material bias.



Source: SRK (UK), 2017

Figure 14.20: Validation Plot (Northing) Showing Block Model Estimates versus Sample Mean (25 m Intervals) for UHG Domain CZONE 101 for CuCov

14.23.2.3 *Statistical Validation*

The block estimates have been compared to the mean of the composite samples (Table 14.16 and Table 14.17) which indicate the overall percentage difference in the mean grades typically vary between 1% and 10%, which SRK deems to be within acceptable levels.

SRK notes larger percentage differences in the means for domains CZONE 104, 204 and 303 to 304 and EZONE 312, which are less well drilled and have irregular sample coverage. As a result, the sample mean is skewed by relatively few high/low grade samples.

Based on the visual, sectional and statistical validation results, SRK considers the grades in the block model to be well estimated overall, with variable confidence in some areas.

Table 14.16: Summary Block Statistics for Ordinary Kriging (OK) and Inverse Distance Weighting (IDW) Estimation Methods for CuCov% and Gold g/t

CZONE	Field	Estimation Method	Block Estimate Mean	Composite Mean	% Difference	Absolute Difference
101	AU	OK	10.75	11.18	-4%	-0.43
	AU	IDW	10.77	11.18	-4%	-0.40
	CUCOV	OK	16.46	16.67	-1%	-0.21
	CUCOV	IDW	16.43	16.67	-1%	-0.24
102	AU	OK	15.06	15.82	-5%	-0.75
	AU	IDW	15.22	15.82	-4%	-0.59
	CUCOV	OK	15.02	15.23	-1%	-0.21
	CUCOV	IDW	15.12	15.23	-1%	-0.11
103	AU	OK	3.54	3.76	-6%	-0.22
	AU	IDW	3.50	3.76	-7%	-0.26
	CUCOV	OK	4.68	4.68	0%	0.01
	CUCOV	IDW	4.63	4.68	-1%	-0.04
104	AU	OK	0.90	1.21	-25%	-0.31
	AU	IDW	0.91	1.21	-25%	-0.30
	CUCOV	OK	1.23	1.41	-12%	-0.17
	CUCOV	IDW	1.25	1.41	-11%	-0.15
202	AU	IDW	22.20	22.13	0%	0.08
	CUCOV	IDW	20.44	20.41	0%	0.03
203	AU	IDW	3.23	3.44	-6%	-0.21
	CUCOV	IDW	5.27	5.26	0%	0.01
204	AU	IDW	0.36	0.41	-12%	-0.05
	CUCOV	IDW	0.84	1.00	-16%	-0.16
303	AU	IDW	5.91	5.20	14%	0.71
	CUCOV	IDW	4.76	4.34	9%	0.41
304	AU	IDW	0.77	1.73	-55%	-0.96
	CUCOV	IDW	0.52	0.66	-22%	-0.14
404	AU	IDW	3.93	3.93	0%	0.00
	CUCOV	IDW	2.85	2.85	0%	0.00

Table 14.17: Summary Block Statistics for Ordinary Kriging and Inverse Distance Weighting Estimation Methods for CuEn% and Arsenic%

EZONE	Field	Estimation Method	Block Estimate Mean	Composite Mean	% Difference	Absolute Difference
111	AS	OK	0.36	0.35	3%	0.01
	AS	IDW	0.39	0.35	10%	0.04
	CUEN	OK	0.92	0.90	2%	0.02
	CUEN	IDW	0.99	0.90	10%	0.09
112	AS	OK	0.08	0.08	-1%	0.00
	AS	IDW	0.08	0.08	-8%	-0.01
	CUEN	OK	0.21	0.22	-1%	0.00
	CUEN	IDW	0.20	0.22	-8%	-0.02
212	AS	OK	0.05	0.05	-8%	0.00
	CUEN	OK	0.12	0.13	-8%	-0.01
312	AS	OK	0.02	0.03	-38%	-0.01
	CUEN	OK	0.05	0.08	-38%	-0.03
412	AS	IDW	0.00	0.00	0%	0.00
	CUEN	IDW	0.002	0.002	0%	0.00

14.24 Model Validation and Sensitivity – Lower Zone

14.24.1 Block Model Validation

SRK has validated the block model using the following techniques:

- Visual inspection of block grades in comparison with drillhole data.
- Sectional validation of the mean samples grades in comparison to the mean model grades (swath Pots).
- Comparison of block model statistics with drill hole composited grades.

14.24.1.1 Visual Validation

Visual validation provides a comparison of the interpolated block model on a local scale. A thorough visual inspection has been undertaken in section and 3D, comparing the sample grades with the block grades, which demonstrates in general good comparison between local block estimates and nearby samples, without excessive smoothing in the block model. Figure 14.21 to Figure 14.23 show examples of the visual validation checks and highlight the overall block grades corresponding with composite sample grades for copper, gold and arsenic.

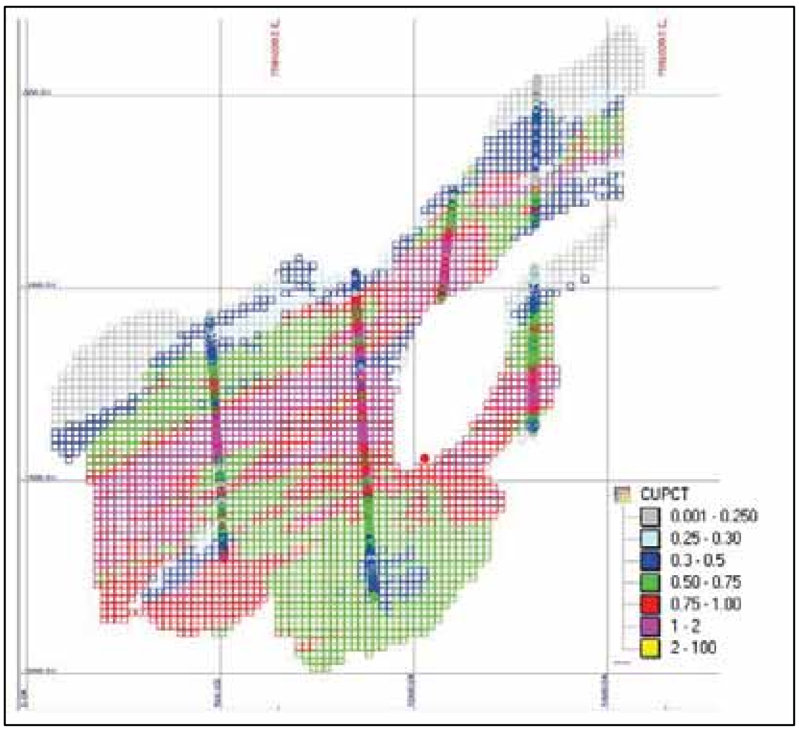


Figure 14.21: Lower Zone Block Model Copper Grades Compared with Dril Hole Composites

Note: Section 4876400N Looking North, Grid Lines are 500 m apart.

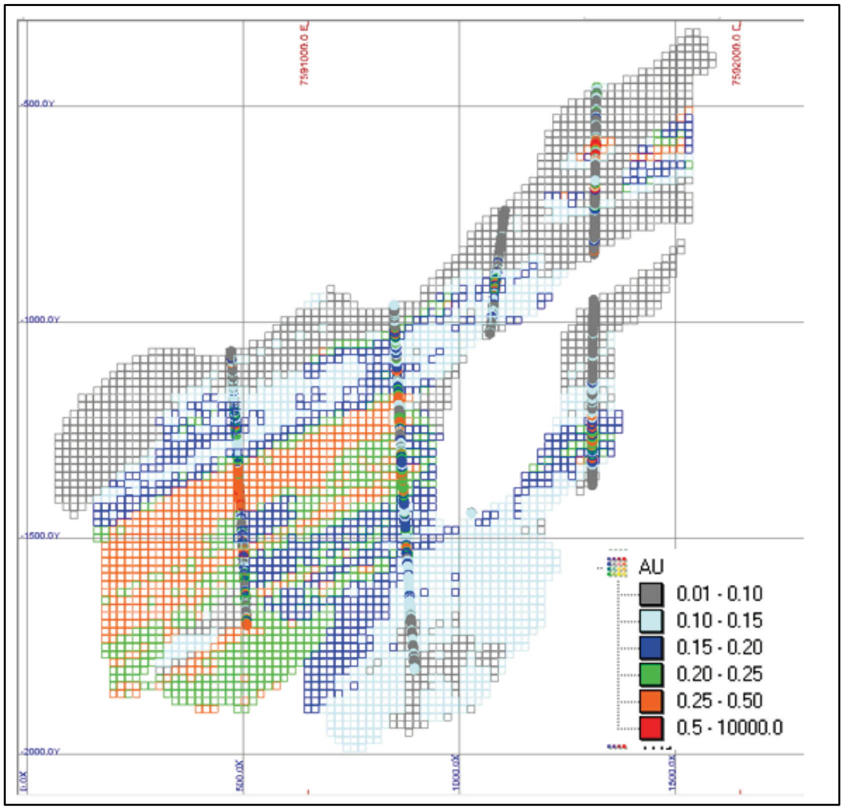


Figure 14.22: Lower Zone Block Model Gold Grades Compared with Drill Hole Composites

Note: Section 4876400N Looking North, Grid Lines are 500 m apart.

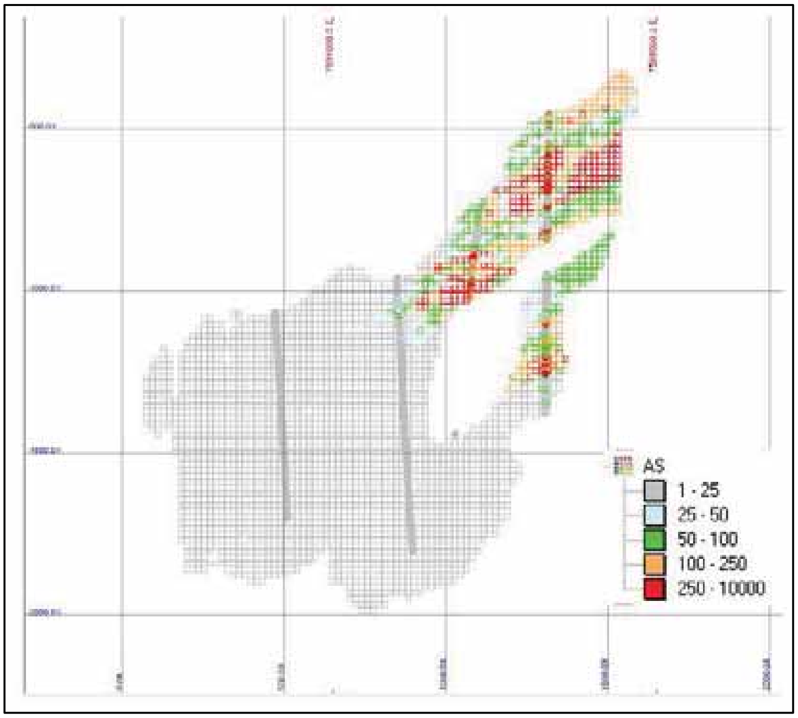


Figure 14.23: Lower Zone Block Model Arsenic Grades Compared with Drill Hole Composites

Note: Section 4876400N Looking North, Grid Lines are 500 m apart.

14.24.1.2 Sectional Validation – Swath Plots

As part of the validation process, the input composite samples are compared to the block model grades within a series of coordinates (based on the principle directions). The results of which are then displayed on charts to check for visual discrepancies between grades. Figure 14.24 shows the results for the copper grades swath for the porphyry mineralization in the Lower Zone block model.

The resultant plots show a reasonable correlation between the block model grades and the composite grades, with the block model showing a typically smoothed profile of the composite grades as expected. SRK notes that in less densely sampled areas, grade discrepancies do exist on a local scale. Overall, however, SRK is confident that the interpolated grades reflect the available input sample data and the estimate shows no sign of material bias.

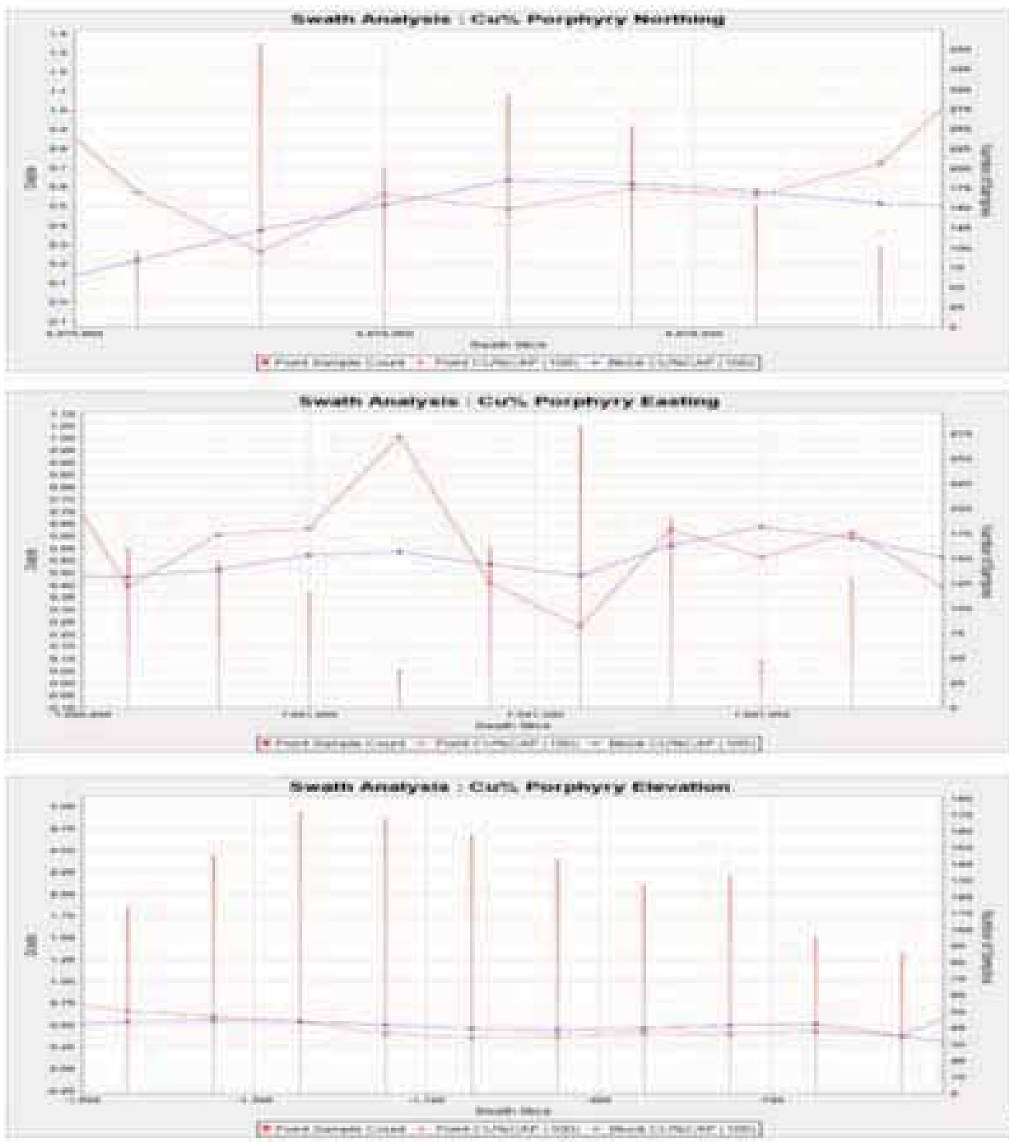


Figure 14.24: Swath Plot for Lower Zone Mineralization

14.24.1.3 *Statistical Validation*

The block estimates have been compared to the mean of the composite samples (Table 14.18) which indicate the overall percentage difference in the mean grades typically vary between 1% and 10% for gold inside the larger domains and within 15% for arsenic, which SRK deems to be within acceptable levels.

SRK notes larger percentage differences in the means for the 501 domain which is a relatively small domain with fewer composites and estimated blocks contributing to the wide differences between estimate and input data. As a result, the sample mean is skewed by relatively few high/low grade samples.

Based on the visual, sectional and statistical validation results, SRK considers the grades in the block model to be well estimated overall, considering the widely spaced drilling in some areas of the mineralized domains.

Table 14.18: Comparison of block Statistics and d Composited data for the Lower Zone Block Model

Domain	Metal	Block Estimate Mean	Composite Mean	% Difference	Number of Blocks
201	Cu	0.22	0.22	0.0	14,257
	Au	0.15	0.14	7.1	14,257
	As	224	196	14.3	14,257
501	Cu	0.67	0.85	-21	541
	Au	0.25	0.50	-50	541
	As	539	977	-44	541
202	Cu	0.27	0.24	12	62,758
	Au	0.06	0.06	0	62,758
	As	31	34	-8.8	62,758
502	Cu	0.87	0.84	3.5	101,947
	Au	0.18	0.17	5.8	101,947
	As	48	55	-12.7	101,947

14.25 Mineral Resource Classification – Upper Zone

Block model tonnage and grade estimates for the Čukaru Peki (UZ) deposit were classified according to the CIM Definition Standards.

Mineral resource classification is typically a subjective concept, industry best practice requires that resource classification should consider both the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the confidence in the tonnage and grade estimates. Classification should integrate both concepts to delineate regular areas of similar confidence.

Data quality, geological confidence, sample spacing and the interpreted continuity of grades controlled by the deposit have allowed SRK to classify the block model in the measured, indicated and inferred mineral resource categories. The following guidelines apply to SRK's classification:

14.25.1 Measured

Measured Mineral Resources are where block grades are estimated from multiple drillhole intercepts on an approximate 25-m spacing and where there is good continuity shown by both assay grades and geological wireframes. For each of these zones SRK has 'measured' confidence in the average grade, tonnes and grade distribution in volumes that are relevant for detailed mine planning.

14.25.2 Indicated

Indicated Mineral Resources are where block grades are estimated from multiple drillhole intercepts spaced typically at less than 50 m and where there is reasonable continuity shown by both assay grades and geological wireframes. In these volumes SRK has reasonable to good confidence in the suitability for long-term mine planning.

14.25.3 Inferred

Inferred Mineral Resources are where there is reasonable to low confidence in geometry, geological continuity and block grade estimates due to blocks being typically within 100 m of sample data. These areas require infill drilling to improve the quality of the geological interpretation and local block grade estimation before they can be used for long-term mine planning. SRK considers there to be a reasonable expectation that infill drilling in the inferred mineral resource areas will result in indicated mineral resources.

An example of SRK's Mineral Resource classification for the Čukaru Peki (UZ) deposit is shown in Figure 14.25.

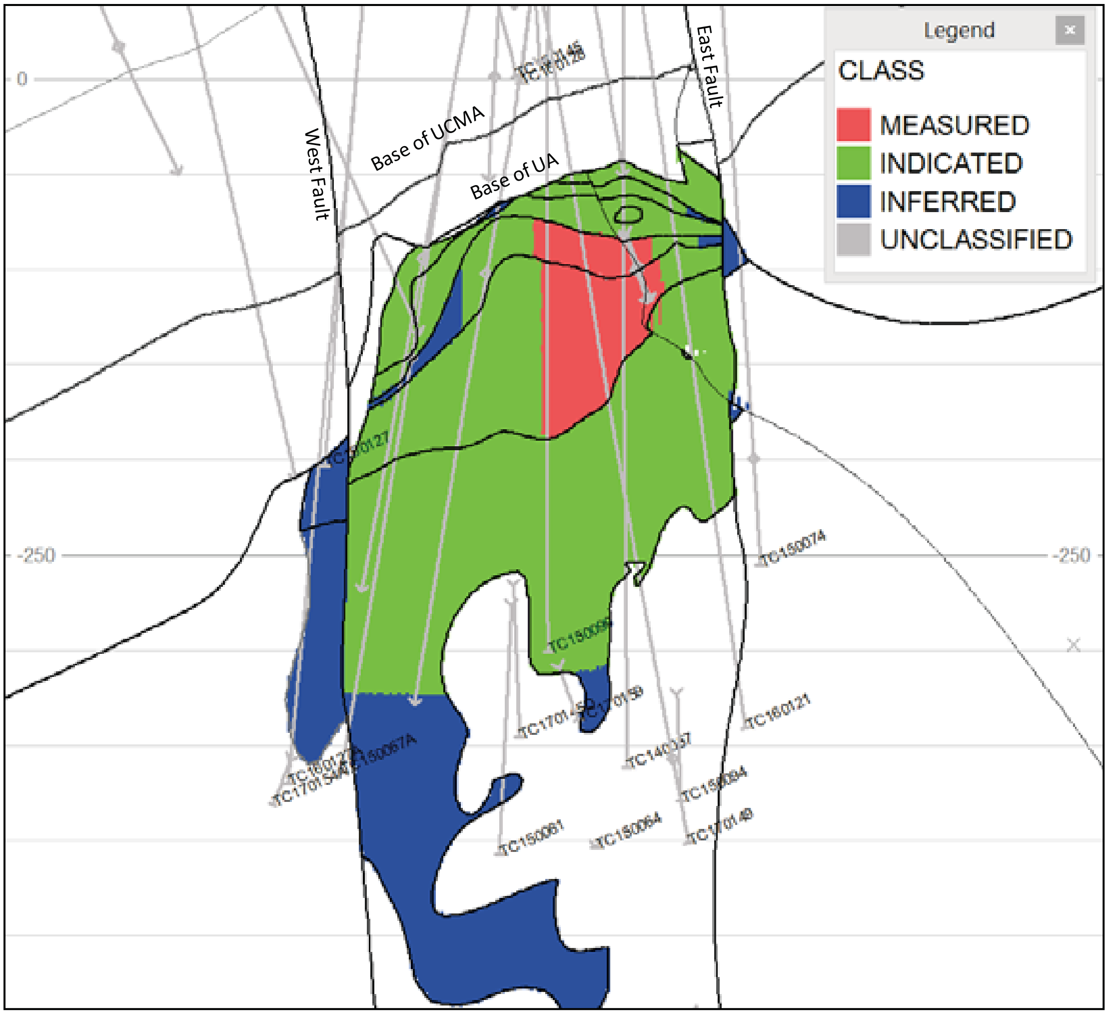


Figure 14.25: Cross-Section Showing SRK's Wireframe-Defined Mineral Resource Classification for the Timok Deposit, View North-Northwest

14.26 Mineral Resource Classification – Lower Zone

Block model tonnage and grade estimates for the Čukaru Peki (UZ) deposit were classified according to the CIM (2014) Definition Standards.

Mineral Resource classification is typically a subjective concept, industry best practice requires that resource classification should consider both the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the confidence in the tonnage and grade estimates. Classification should integrate both concepts to delineate regular areas of similar confidence.

Data quality, geological confidence, sample spacing and the interpreted continuity of grades have only allowed SRK to classify the block model as inferred mineral resource category. The following guidelines apply to SRK's classification:

14.26.1 Inferred

Inferred Mineral Resources are where there is reasonable to low confidence in geometry, geological continuity and block grade estimates due to blocks being typically within 250 m of at least two drill holes. These areas require infill drilling to improve the quality of the geological interpretation and local block grade estimation before they can be used for long-term mine planning. SRK considers there to be a reasonable expectation that infill drilling in the inferred mineral resource areas will result in indicated mineral resources.

14.27 Mineral Resource Statement – Upper Zone

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a mineral resource as a:

"concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling".

The "reasonable prospects for eventual economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade taking into account likely extraction scenarios and processing recoveries.

SRK (UK) has applied basic economic considerations developed for the PEA to restrict the mineral resource to material that has reasonable prospects for economic extraction by underground mining methods.

The Mineral Resource has been reported using a resource net smelter return (RscNSR) cut off value based on copper, gold and arsenic, using a copper price of $3.49/lb and gold price of $1,565/oz, derived from long-term consensus forecasts with a 20% uplift as appropriate for assessing eventual economic potential of mineral resources. Assumed technical and economic parameters selected were based on the results of the PEA study.

SRK UK considered that the blocks with a RscNSR value greater than an operating cost of $35/t had "reasonable prospects for eventual economic extraction" and could be reported as a Mineral Resource. Based on a review of average block values in 5-m horizontal slices, SRK

UK determined a level in the block model, (-445 m amsl), below which the average RscNSR fell short of covering this cost. The 2017 reported Mineral Resource therefore comprised all blocks inside the mineralization model above this elevation including a small number of individual blocks with RscNSR values lower than 35; this approach also excluded isolated blocks with >$35/t RscNSR below -445 m amsl.

The Mineral Resource statement for the Čukaru Peki (UZ) deposit is shown in Table 14.19; subtotals are given based on geological domains.

Table 14.19: SRK Mineral Resource Statement as at April 24, 2017 for the Upper Zone of the Čukaru Peki Deposit

Category	Resource Domain	Quantity Mt	Grade			Metal	
			% Cu	g/t Au	% As	Cu Mt	Au Moz
Measured	UHG	0.44	18.7	11.70	0.29	0.082	0.17
	Massive Sulphide	1.70	6	4.10	0.29	0.1	0.23
Indicated	UHG	0.95	17.1	11.80	0.24	0.16	0.36
	Massive Sulphide	6.70	5.2	3.40	0.25	0.35	0.73
	Low grade covellite	19.00	1.9	1.10	0.17	0.36	0.70
Measured and Indicated	UHG	1.40	17.6	11.80	0.26	0.24	0.52
	Massive Sulphide	8.40	5.4	3.60	0.26	0.45	0.96
	Low grade covellite	19.00	1.9	1.10	0.17	0.36	0.70
Inferred	UHG	0.45	15	10.80	0.16	0.07	0.16
	Massive Sulphide	0.80	4.9	3.40	0.11	0.04	0.09
	Low grade covellite	12.70	1	0.44	0.05	0.12	0.18
Total-Measured		2.20	8.6	5.70	0.29	0.190	0.40
Total-Indicated		26.60	3.3	2.10	0.20	0.870	1.80
Total-Measured and Indicated		28.70	3.7	2.40	0.20	1.050	2.20
Total-Inferred		13.90	1.6	0.90	0.06	0.230	0.42

1. The RscNSR value used to report the estimate is $35/t.
2. All figures are rounded to reflect the relative accuracy of the estimate.
3. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
4. The Mineral Resource is reported on 100% basis, attributable to Rakita Exploration d.o.o.

14.28 Mineral Resource Statement – Lower Zone

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a mineral resource as a:

“concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a mineral resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling”.

The “reasonable prospects for eventual economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade taking into account likely extraction scenarios and processing recoveries.

SRK (Canada) considers that the Lower Zone mineralization is amenable to underground mining by block caving mining methods and has applied basic economic considerations to identify blocks that have a reasonable prospects for economic extraction by underground mining methods.

The mineral resource has been reported using a US dollar equivalent cut off value based on copper price of $3.00/lb, gold price of $1,400/oz, a 69% recovery for copper and a 69% recovery for gold for assessing eventual economic potential of mineral resources. SRK further determined that any high-sulphidation epithermal mineralization did not currently satisfy the reasonable prospect of eventual extraction until further exploration drilling was carried out to better outline this style of mineralization in the Lower Zone.

Based on large block cave operations, SRK (Canada) considers that the blocks with a combined value of $25/t have “reasonable prospects for eventual economic extraction” and can be reported as a mineral resource. The mineral resource statement for the Čukaru Peki (LZ) deposit is shown in Table 14.20; subtotals are given based on geological domains.

Table 14.20: SRK Mineral Resource Statement as at June 19, 2018 for the Lower Zone of the Čukaru Peki Deposit

Category	Resource Domain	Quantity Mt	Grade			Metal	
			% Cu	g/t Au	% As	Cu Mt	Au Moz
Inferred	Lower Zone Porphyry	1,659	0.86	0.18	0.01	14.3	9.6

1. The Dollar equivalent cut-off value used to report the estimate is $25/t.
2. All figures are rounded to reflect the relative accuracy of the estimate.
3. Mineral resources are not mineral reserves and do not have demonstrated economic viability.
4. The Mineral Resource is reported on 100% basis, attributable to Rakita Exploration d.o.o.

14.29 RscNSR Cut-off Sensitivity Analysis – Upper Zone

The results of RscNSR cut-off sensitivity analysis completed for Čukaru Peki are shown in Table 14.21 and Table 14.22.

This is to show the continuity of the grade estimates at various cut-off increments and the sensitivity of the mineral resource to changes in RscNSR cut-off. The tonnages and grades in these tables, however, should not be interpreted as mineral resources.

Table 14.21: Gradations for Measured and Indicated Material at Čukaru Peki Upper Zone at Various RscNSR Cut-Off Grades

Cut-off RscNSR ($/t)	Reporting Elevation (m RL)	Quantity Mt	Grade			Metal	
			% Cu	g/t Au	% As	Cu Mt	Au Moz
105.00	-235.00	18.3	4.9	3.3	0.23	0.89	1.96
95.00	-255.00	20.4	4.6	3.1	0.22	0.93	2.03
85.00	-275.00	22.3	4.3	2.9	0.22	0.96	2.08
75.00	-295.00	23.9	4.1	2.7	0.22	0.99	2.11
65.00	-315.00	25.3	4.0	2.6	0.21	1.01	2.13
55.00	-415.00	28.7	3.7	2.4	0.20	1.05	2.18
45.00	-435.00	28.7	3.7	2.4	0.20	1.05	2.18
35.00	-455.00	28.7	3.7	2.4	0.20	1.05	2.18
25.00	-475.00	28.7	3.7	2.4	0.20	1.05	2.18
15.00	-520.00	28.7	3.7	2.4	0.20	1.05	2.18

Table 14.22: Gradations for Inferred Material at Čukaru Peki Upper Zone at Various RscNSR Cut-Off Grades

Cut-off RscNSR ($/t)	Reporting Elevation (m RL)	Quantity Mt	Grade			Contained Metal	
			% Cu	g/t Au	% As	Cu Mt	Au Moz
105.00	-235.00	3.4	3.9	2.8	0.09	0.13	0.31
95.00	-255.00	4.3	3.3	2.3	0.08	0.14	0.32
85.00	-275.00	5.2	3.0	2.0	0.08	0.15	0.34
75.00	-295.00	6.1	2.7	1.8	0.08	0.16	0.35
65.00	-315.00	6.9	2.5	1.6	0.07	0.17	0.36
55.00	-415.00	11.5	1.8	1.1	0.06	0.21	0.41
45.00	-435.00	12.7	1.7	1.0	0.06	0.22	0.42
35.00	-455.00	13.9	1.6	0.9	0.06	0.23	0.42
25.00	-475.00	15.2	1.5	0.9	0.06	0.24	0.43
15.00	-520.00	16.1	1.5	0.8	0.06	0.24	0.43

14.30 Cut-off Sensitivity Analysis – Lower Zone

The results of cut-off sensitivity analysis completed for Čukaru Peki Lower Zone are shown in Table 14.23.

This is to show the continuity of the grade estimates at various cut-off increments and the sensitivity of the mineral resource to changes in cut-off values only. The tonnages and grades in these tables, however, should not be interpreted as mineral resources.

Table 14.23: Gradations for Inferred Material at Čukaru Peki Lower Zone at Various Cut-off

Cut-off ($/t)	Quantity Mt	Grade			Contained Metal	
		% Cu	g/t Au	As ppm	Cu Mt	Au Moz
55.00	706	1.12	0.24	49	7.9	5.3
50.00	906	1.06	0.22	45	9.6	6.5
45.00	1,126	0.99	0.21	44	11.1	7.5
40.00	1,330	0.94	0.20	45	12.5	8.4
35.00	1,483	0.91	0.19	49	13.5	9.0
30.00	1,572	0.89	0.18	49	14.0	9.3
25.00	**1,659**	**0.86**	**0.18**	**50**	**14.3**	**9.6**
20.00	1,945	0.78	0.16	47	15.2	10.3
15.00	2,286	0.71	0.15	44	16.2	11.0

14.31 Comparison to Previous Mineral Resource Estimates – Upper Zone

The updated Mineral Resource estimate represents a significant increase in metal content within the Indicated category for copper from 0.2 to 0.9 Mt and gold from 0.6 to 1.8 Moz which is primarily due to additional geological confidence provided by infill drilling which has allowed a significant portion of the Inferred resource to be upgraded to Indicated. In addition, SRK has upgraded 2.2 Mt at a grade of 8.6% copper and 5.7 g/t gold to the Measured category.

Within the Inferred category, in comparison to the previous (March 2016) mineral resource, which was reported at a cut-off grade of 0.75% copper, the updated Inferred Mineral Resource estimate (reported above an RscNSR cut-off of $35/t) represents a decrease in metal content, from 1.0 to 0.2 Mt for copper and from 1.9 to 0.4 Moz for gold. The change in contained metal within the Inferred category is the result of 60% reduction in tonnage and approximately 45% (relative) decrease in copper and gold grade mainly due to the material upgraded to Measured and Indicated and partly due to the change in deposit geometry at the margins and at depth.

SRK considers that the key changes in the mineral resource result from a combination of the following factors:

- Metal converted to measured and indicated resources, primarily due to new infill drilling confirming the continuity of the geology and mineralization, typical grade distribution and average grades within better drilled areas of the deposit.
- Reduction in geological continuity outside of interpreted fault boundaries; this impacts on the margins of the highest-grade mineralization.
- Refinement from infill drilling to the distribution of medium to high grade layering within parts of the massive sulphide domain.
- Change in the cut-off approach from using copper grade to RscNSR value (and elevation limit) which has added low grade material at depth.

In addition, the Lower Volcano-sedimentary Breccia domain postulated in the previous model has been re-interpreted based on new drilling information. Instead, low grade CuCov mineralization continues to depth, constrained by more competent, un-mineralized andesite.

14.32 Exploration Potential

The full extents of Čukaru Peki UZ mineralization have now been relatively well defined; however, there is good potential for discovery of additional zones of HS (and associated porphyry style) mineralization proximal and along trend from the current deposit. This potential is highlighted by the similarities with the Bor deposit camp (some 10 km to the north), where there are numerous clusters of discrete HS and porphyry-style mineralized bodies clustered within an approximately three-kilometre long area.

The favourable Bor geological and metallogenic trend continues into the Brestovać-Metovnica permit area (86 km^2) shown in Figure 7.4 which is dominated by the same prospective Upper Cretaceous Phase 1 ('Lower') andesite volcanic unit (exposed and continuing below the Miocene cover) and controlling structures (north-northwest and east-west intersecting cross structures) associated with Timok and the wider Bor district deposits, which have yet to be fully explored. The permit is therefore considered to have good potential for discovery of additional HS bodies and porphyry mineralization.

Although the permit area has been covered by geophysical surveys (CSAMT, IP/resistivity and locally one line of seismic), geological mapping and geochemical sampling, there has been limited drilling completed outside of the immediate Čukaru Peki UZ & LZ deposit area. Further evidence for permit potential is highlighted by the discovery of epithermal gold and base metal intermediate sulphidation mineralization systems found in historic Rakita drilling at Brestovać proximal to the Bor Fault, some two kilometres west of Čukaru Peki.

More recently (2013), Rakita has also observed intersections of HS advanced argillic alteration (quartz-alunite-pyrite) with copper (and gold) mineralization, quartz-illite pyrite alteration and locally lead-zinc mineralization in three holes approximately 2.5 km south of Čukaru Peki which may indicate proximity to another hydrothermal system or centre.

15. Mineral Reserve Estimates – Upper Zone

Mineral Reserve statements are based on material class as economically recoverable Measured and Indicated Mineral Resources with dilution and mining/processing recovery factors applied. Depletion has been included in these estimates. No Proven Mineral Reserves have been declared.

Factors which may affect the Mineral Reserve estimates include commodity prices and valuation assumptions; changes to the proposed sublevel cave design, geotechnical, mining, and processing plant recovery assumptions; appropriate dilution control; changes to capital and operating cost estimates.

The Mineral Reserve statement for the Čukaru Peki deposit is presented in Table 15.1.

Table 15.1: Mineral Reserve Statement, Čukaru Peki Deposit, Republic of Serbia, March 8, 2018

Description	Quantity (kt)	Grade			Contained Metal		
		(% Cu)	(g/t Au)	(% As)	(klbs Cu)	(kOz Au)	(kt As)
Proven	0	0.00	0.00	0.00	0	0	0
Probable	27,121	3.25	2.06	0.17	1,944,074	1,792	47
Total	**27,121**	**3.25**	**2.06**	**0.17**	**1,944,074**	**1,792**	**47**

Notes:

1. The Mineral Reserves and Resources in this news release were estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions and Guidelines prepared by the CIM Standing Committee on Reserve Definitions and adopted by CIM Council.
2. Metal prices used include US$3.00/lb Cu and US$1,300/oz Au.
3. A Reserve NSR cut-off of US$35/tonne was used to optimize the SLC Ring layout.
4. Contained metal figures and totals may differ due to rounding of figures.

16. Mining Methods – Upper Zone

16.1 Geotechnical Engineering

From 2016 to 2017 SRK has conducted a number of geotechnical studies on Čukaru Peki deposit, as well as providing guidance on geotechnical drilling program. Following is a summary of geotechnical assessment results and ground support recommendations used for the PFS mine design.

16.1.1 Geotechnical Assessments

SRK has conducted the following geotechnical assessments for Čukaru Peki:

- Rock mass assessment.
- Intact rock strength assessment.
- Structural assessment.
- Caveability assessment.
- Ground support.

A rock mass assessment has been carried out for the deposit represented by the advanced argillic domain to understand the variability in rock mass quality for all the mining levels from -80 m level to -420 m level. Q-logging has been evaluated for the argillic footwall domain to define ground support recommendations for development and major infrastructure excavations. The data shows the following rock mass quality distribution: Good (13%), Fair (16%), Poor (57%) and Very Poor (14%).

The intact rock strength assessment used field-based empirical intact rock strength estimates (hammer tests), point load testing and uniaxial compressive testing.

The structural assessment focussed on defining the small-scale structure for the deposit, hanging wall and footwall. The hanging wall is defined by the structure in the UA; the deposit by the advanced argillic; and the footwall by the argillic.

A caveability assessment was carried out to define how many drawpoints will need to be advanced and retreated before sustaining caving of the hanging wall is achieved. The lower and upper quartile rock mass rating, RMR_{90} values, for the UA have been used to define the lower and upper bound RMR values and have been multiplied by adjustment percentages to define the mining rock mass rating (“MRMR”), which reflects the mining activities.

For ground support, based on the rock mass assessment, the proportions of "Poor", "Average" and "Good" rock were estimated for each mining level and a design RMR value was chosen to represent the rock mass quality. The design RMR_{90} was adjusted to reflect the mining activities to calculate an MRMR value and Laubscher stability chart was used to estimate hydraulic radius for continuous caving of the hanging wall.

16.1.2 Geotechnical Assessment Outcomes

Geotechnical domains are based on geological units and alteration zones. The overall rock mass quality of the mineralized zone is fair to good (RMR_{L90} of 40 to 60). However, this zone is transected by several fractured rock and clay zones of poor to very poor-quality rock mass (RMR_{L90} of <40). The mineralized zone is overlain by Miocene sedimentary units of variable rock mass quality (RMR_{L90} of 20 to 60) but with some units highly susceptible to weathering. The laboratory results indicate that the mineralization hosted in the advanced argillic domains is classified as very strong with average intact rock strength (IRS) results of 109 MPa. The hangingwall defined by the UA is classified as Strong with average IRS results of 97 MPa and the footwall defined by the Argillic domain is classified as Strong with an average IRS of 75 MPa.

The in-situ stress regime is currently unknown and in-situ stress measurements are planned for the next stage of investigation. Although in-situ stress is currently unknown, a stress adjustment factor of 90% was applied to RMR values, which resulted in a hydraulic radius of 14 to ensure continuous caving.

The selected base case mining method is sub-level caving. Ground support and development rates have been adjusted to consider the presence of fracture and clay zones. SRK considers the geotechnical data to be suitable for a PEA. The main risks are: i) uncertainty in the extent and location of the in-situ fractured zone in relation to main infrastructure, ii) the unknown stress regime and behaviour of MCS in terms of dilution and mudrush potential, and iii) fragmentation and fines generation in Miocene sediments potentially affecting dilution.

16.2 Mining Method and Access Selection

16.2.1 Deposit Context

The choice of a mining method is primarily aimed at achieving the lowest cost to finished metal with manageable risk, while maintaining a safe mining environment and achieving optimum production rates and productivities.

The mining method selection for the Čukaru Peki portion of the Timok project was guided by the following:

- Geology: The exposed geology in the Timok project area is dominated by Upper Cretaceous andesitic volcano-sedimentary sequences partially covered by a north-south to north-west elongated belt of poorly consolidated tertiary clastic sedimentary rocks.

Basement Mesozoic stratigraphy exposed around the TMC consists of Jurassic to Lower Cretaceous limestones and clastic sedimentary rocks. Overburden thickness ranges from 0.7 to 16.0 m, and consists predominantly of medium to high plasticity clay.

- Geometry: The Čukaru Peki (UZ) deposit has a large extent both horizontally (200 m x 250 m in plan view) and vertically (about 450 m in section), where a value cut-off of $35/t NSR was used to define the mineable resource.
- Grade Distribution: The grade distribution follows general trends with high grade on the top of the deposit and lower grades at depth.
- Rock Mass: The rock mass at Čukaru Peki is variable within the mineralization. The faults and fracture zone domains contain poorer quality rock than the alteration and geology domains, that are mostly of good quality.
- Geological Structure: Resent structural geology interpretation identifies that the deposit is contained within structural faults with different timings of the faulting: East Fault, West Fault and South Fault.
- Alteration: The footprint of the UZ mineralization is directly associated with the advanced argillic alteration and has a narrow alteration front or halo of kaolinite-pyrite, which envelope represents the alteration front of the mineralized body and is characterized by high gold values without copper, particularly at the margins of the high-grade copper and gold zones.

16.2.2 Selected Strategy – Sub-level Cave

The recommended strategy for mining the orebody is Sublevel Caving (SLC) method. A sensitivity analysis showed that the selection of this strategy was robust across a wide range of assumptions.

In SLC, mining starts at the top of the orebody and progresses downwards. Mineable resource is extracted from sublevels spaced at regular vertical intervals throughout the deposit. A series of ring patterns are drilled and blasted from the drawpoints on each sublevel; broken mineable resource is mucked from the drawpoints after each ring blast.

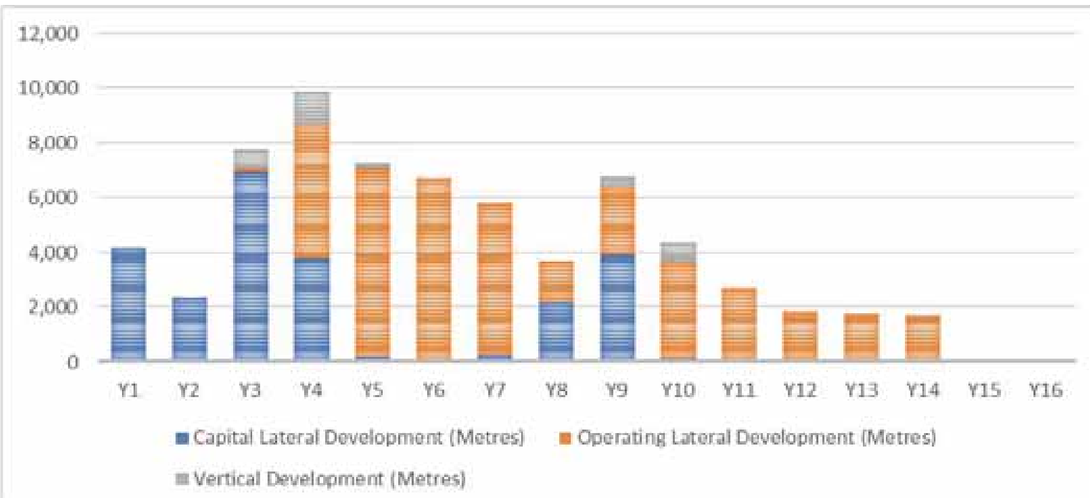
SLC is applicable through a wide range of geotechnical conditions, but as with most mining methods it is most efficiently applied in strong rock conditions, making it a relatively easy method to mechanize. This method is normally used in massive, steeply-dipping orebodies with considerable strike length. SLC typically has dilution ranging from 15 to 30% and mining recovery ranging from 80 to 90%, and is dependent on effective management of the SLC operations.

16.3 Sub-level Cave Design

16.3.1 Mine Development

Total mine development consists of 24 km of lateral development, inclusive of declines, 3 km of vertical development and 40 km of operating development. Capital and operating development are outlined in the life-of-mine (LOM) schedule (Figure 16.1 and Table 16.4).

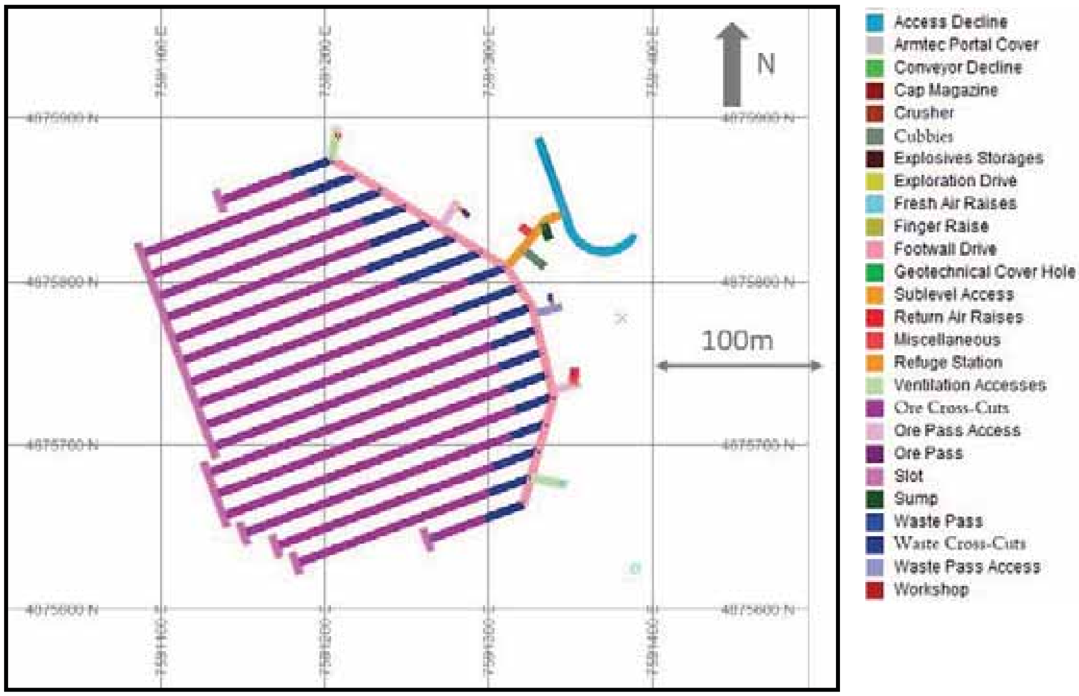
Capital and operating development are outlined in the life-of-mine schedule (Figure 16.1). Note that the quantities presented in Figure 16.1 do not represent the quantities in the economic model due to the truncation of unprofitable periods at the end of the mine plan.



Source: SRK, 2018

Figure 16.1: LOM Capital and Operating Development Physicals

Due to the orebody geotechnical variability, the levels are spaced at 20 m vertical intervals. There is opportunity to increase the level spacing should the geotechnical conditions be better than anticipated. A level access drive is developed from the decline at each level and is typically 50 m in length. The level access is the access to the footwall drive. The footwall drive is offset approximately 25 m from the deposit. The standoff distance will be based on actual conditions at time of construction. The footwall drive will follow the geometry of the orebody. Cross-cuts are spaced 14 m apart along the footwall drive. Slot drives are developed at the end of each cross-cut and may connect adjacent cross-cuts when they are in alignment. The footwall drive also provides access to the ore and waste passes and to the ventilation system. This layout is shown in Figure 16.2.

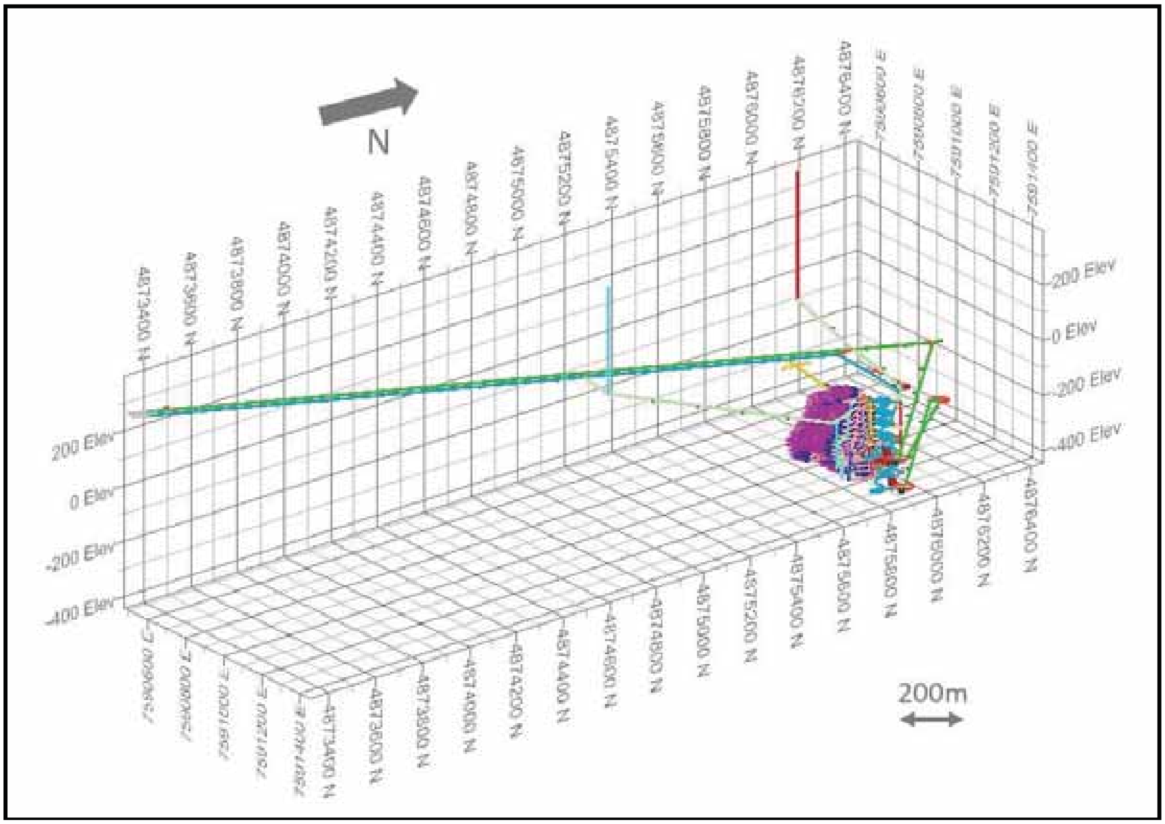


Source: SRK, 2018

Figure 16.2: Plan View of a Typical Sublevel Layout

16.3.2 Overall Development

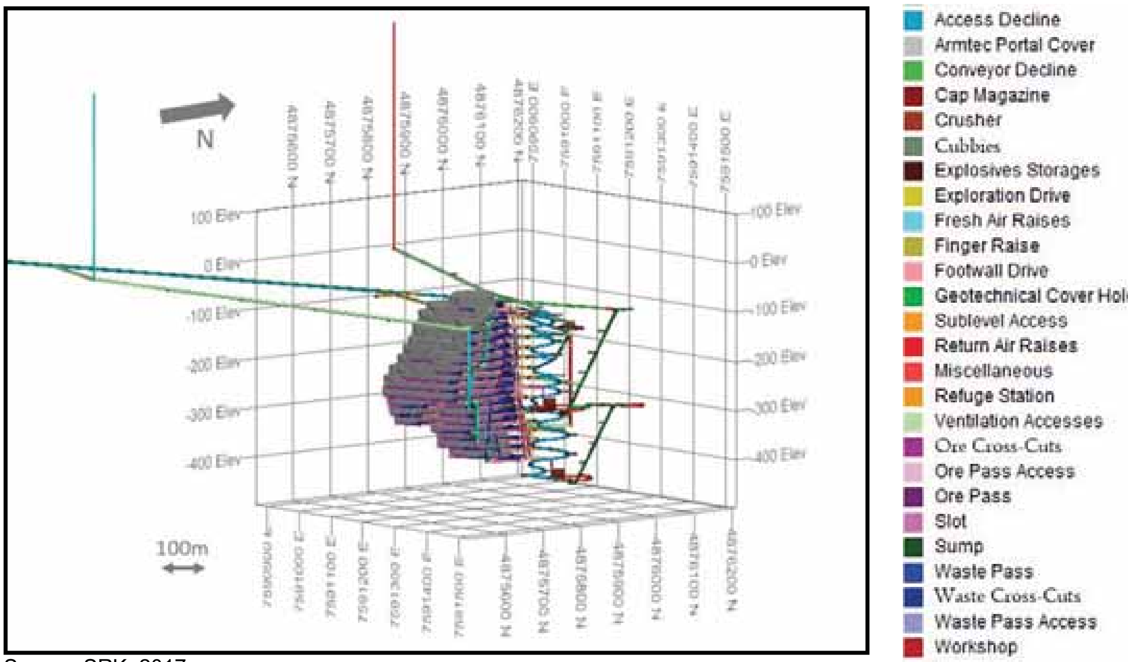
The total underground development layout plan for Timok is shown in Figure 16.3.



Source: SRK, 2018

Figure 16.3: 3D View Of Timok Development Layout - Looking Northwest

Figure 16.4 shows the key mine design features of the mine, including the production rings, looking northwest.



Source: SRK, 2017

Figure 16.4: Magnified 3D View of Timok Overall Mine Layout - Looking Northwest

16.3.3 Cave Design

Levels have been designed at 20 m vertical spacing and cross-cuts at 14 m horizontal spacing, along the footwall drive. The main driver for this selection is the varied ground conditions and designing to the poorest expected rock mass conditions. This is a conservative design parameter that will be revisited base actual ground conditions encountered during initial development. The following ring design criteria have been used:

- 31.3 m height from floor to apex.
- 14 m width.
- 60° apex pillar angle.
- 2.5 m burden.
- 80° dump angle.
- For design simplification and expediency, the following was used in PCSLC:
- A ring burden of 8 m; equal to 3.2 rings at a typical burden of 2.5 m.

- A dump angle of 0°.

16.3.4 Production Cycle

A V-30 blind bore is used to create the slot raises. Each slot activity has 23 m of V-30 drilling and 390 m of additional longhole drilling. Each slot activity takes three days to complete, including the longhole drilling, charging and blasting.

A drill factor of 0.08 m/t has been considered for all the 8-m long shapes, with a longhole drilling productivity of 235 m/d, per drill. Emulsion will be used as the primary explosive.

In general, only three production levels are mined in parallel. In some occasions, when the orebody widens and allows for a second front to open without affecting the first one regarding dilution, up to four production levels are mined in parallel to keep reaching the target production rate of 3.25 Mt. At any given time, a maximum of six mucking LHDs are working in parallel, with a maximum of two LHDs assigned per level, each mucking at an average of 1,900 t/d.

Blasted material is mucked to the ore passes (two per level) that lead to the crushers, at either the -275 m amsl or the -435 m amsl (levels numbered per their elevation above mean sea level).

16.4 Mine Scheduling

16.4.1 Development Schedule

Specific jumbo advance rates have been estimated for each geotechnical domain.

Development in the Timok Upper Zone will encounter 19 geotechnical domains as described in Table 3.2. These 19 domains have been grouped into three ground quality categories, as shown in Table 16.1.

Table 16.1: Rock Types and Qualities Within Timok Upper Zone Development

Geotechnical Domain	Ground Quality
Blue: RMR>50	Good
Green: 50>RMR>45	Fair
Orange: 44>RMR>40	Fair
Red: RMR<40	Poor

The ground support required for each heading will depend on the ground quality around that heading. Three different ground support sets will be used at Timok, corresponding to each of the three ground quality categories. The ground support sets should be revisited as new data becomes available during decline development.

Table 16.2 outlines the assumptions for the lateral development advance rates per jumbo in a single heading, according to the ground quality category. When multiple development headings become available, the advance rate per jumbo will increase, but the advance rate per heading will decrease. Development of multiple headings will increase equipment utilisation, but reduce advance rate per heading.

Table 16.2: Lateral Development Advance Rate

Ground Quality	Rate (m/month)
Good	170
Fair	150
Poor	95

Table 16.3 details the vertical development advance rates. The vertical development advance rate is independent of the geotechnical domains.

Table 16.3: Vertical Development Advance Rate

Type - Diameter	Applied to	Pilot/Ream	Vertical Advance Rate (m/d)	Vertical Advance Rate (m/month)
Raisebore – 5.0 m	MRAR, MFAR	Pilot	15	450
		Ream	4	120
Raisebore – 3.0 m	RAR, FAR, OPA, WPA	Pilot	20	600
		Ream	6	180
Drop Raise – 2.4 m	FIR, SRAR, SFAR	NA	6	180

16.4.2 Production Schedule

The production rate is a function of the deposit geometry and continuity, the prevailing ground conditions, the number of available stoping areas, and the expected productivity for each stope. The production resources were scheduled to achieve a practical production output.

The expected drawpoint productivity has been limited to 260 tonnes per drawpoint per day, based on benchmarking comparable mines. This productivity rate results in annual targets of 3.00 Mtpa for the SLC operations and an additional 0.25 Mtpa for development, providing 3.25 Mtpa of feed to the processing plant.

The Timok LOM mine production schedule, which includes stockpiling material prior to the processing plant being commissioned, is presented in Table 16.4.

The resulting LOM ore production schedule for the Timok mine is presented in Table 16.4.

After the initial pre-production period of four years (ore development starting in the first quarter of Year 4), the Upper Zone will be mined at an average rate of 3.2 Mtpa over 12 years. The 12-year schedule does not consider any truncation resulting from the consideration of sub-economic years at the tail-end of mine life that may result from reduced production rates.

In the final years of production, the annual mining rate decreases to account for the reduced extraction rings. As well, due to the nature of SLC, towards the end of mine life the dilution in the mined material increases to the point where the material extracted has insufficient grade to sustain the operation. This has resulted in the 12-year mine life being truncated in the economic model to a 10-year mine life.

Table 16.4: Mine Production Schedule

Reporting Period	Units	Total	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15	Y16
Development Ore	kt	1,926	0	0	6	262	358	340	291	51	107	166	115	75	75	81	0	0
Run-of-mine Copper Grade	%	3.44%	0.00%	0.00%	5.86%	8.35%	4.77%	3.30%	2.10%	1.66%	1.67%	1.55%	1.33%	1.16%	1.27%	1.26%	0.00%	0.00%
Run-of-mine Gold Grade	gpt	1.97	0.00	0.00	3.15	5.31	2.54	2.16	1.27	0.79	0.77	0.58	0.52	0.46	0.40	0.41	0.00	0.00
Run-of-mine Arsenic Grade	%	1.98%	0.00%	0.00%	1.57%	2.68%	2.76%	2.20%	1.53%	2.39%	0.99%	1.80%	1.28%	1.19%	1.00%	0.95%	0.00%	0.00%
Run-of-mine Iron Grade	%	13.47%	0.00%	0.00%	21.88%	21.82%	16.29%	13.98%	11.15%	12.25%	8.69%	9.95%	8.70%	8.61%	8.65%	9.53%	0.00%	0.00%
Production Ore	kt	27,695	0	0	0	378	2,117	2,939	2,772	3,063	3,158	2,811	3,024	2,788	2,158	1,424	950	112
Run-of-mine Copper Grade	%	3.07%	0.00%	0.00%	0.00%	9.73%	6.77%	5.14%	4.46%	3.29%	2.57%	2.01%	1.83%	1.48%	1.37%	1.24%	1.23%	1.22%
Run-of-mine Gold Grade	gpt	1.93	0.00	0.00	0.00	6.93	4.37	2.92	3.05	2.37	1.77	1.27	1.15	0.76	0.61	0.48	0.45	0.42
Run-of-mine Arsenic Grade	%	1.70%	0.00%	0.00%	0.00%	1.90%	2.17%	2.17%	1.99%	1.87%	1.72%	1.39%	1.44%	1.32%	1.44%	1.48%	1.42%	1.24%
Run-of-mine Iron Grade	%	11.81%	0.00%	0.00%	0.00%	21.51%	19.18%	16.12%	14.08%	12.15%	10.77%	9.99%	9.49%	8.61%	8.26%	8.66%	9.23%	8.50%
Total Ore	kt	29,621	0	0	6	640	2,475	3,279	3,063	3,114	3,265	2,976	3,139	2,863	2,233	1,505	950	112
Run-of-mine Copper Grade	%	3.08%	0.00%	0.00%	5.86%	9.15%	6.45%	4.92%	4.20%	3.26%	2.53%	1.98%	1.81%	1.47%	1.36%	1.24%	1.23%	1.22%
Run-of-mine Gold Grade	gpt	1.92	0.00	0.00	3.15	6.27	4.09	2.82	2.85	2.34	1.73	1.22	1.12	0.75	0.61	0.48	0.45	0.42
Run-of-mine Arsenic Grade	%	1.71%	0.00%	0.00%	1.57%	2.22%	2.25%	2.17%	1.94%	1.88%	1.69%	1.41%	1.43%	1.32%	1.43%	1.46%	1.42%	1.24%
Run-of-mine Iron Grade	%	11.92%	0.00%	0.00%	21.88%	21.63%	18.76%	15.90%	13.81%	12.15%	10.70%	9.99%	9.46%	8.61%	8.27%	8.71%	9.23%	8.50%
Contained Copper Metal	kt	913	0	0	0	59	160	161	129	102	83	59	57	42	30	19	12	1
Contained Gold Metal	koz	1,890	0	0	1	133	336	308	290	242	187	121	117	71	45	24	14	2
Waste	kt	1,985	255	137	462	336	85	61	46	162	291	52	34	27	22	15	0	0
Total Material from UG	kt	31,603	255	137	468	975	2,560	3,341	3,108	3,276	3,557	3,027	3,173	2,889	2,254	1,521	950	112

Note:

The first three years are pre-production years and do not have any ring production.

The years (Y1 to Y14) are based on Project Development which include pre-production and LoM production periods.

The production schedule is not truncated to account for sub-economic tonnage and grade at the end of the mine life due to reduced throughput and overall grade.

The truncated production is part of the economic model and is represented in the Mineral Reserve Statement (Section 15).

16.5 Mine Infrastructure

The mine infrastructure required to support operation of the Timok mine can be further subdivided as follows:

- Surface Mine Infrastructure.
- Underground Mine Infrastructure.

16.5.1 Surface Mine Infrastructure

The major surface mine infrastructure required to support the operation of the Timok Mine will include:

- Mine Portal
- Overland Conveyor System
- Water Storage
- Surface Pipelines
- Fans for Main Ventilation System

The surface mine infrastructure also includes the following facilities described in detail in Section 18.1:

- Mine Portal Area Support Facilities
- Mine Maintenance and Warehouse Facilities
- Concrete/Shotcrete Batch Plant
- Fuel and Lubricant Storage Facilities

16.5.1.1 Mine Portal Area

Timok will be accessible via a dual decline with respective portals. The box cut portion of the explorations decline will incorporate corrugated steel arch plates which will allow the boxcut to be backfilled. During operation there is an access portal to the mine and the second portal will house the underground conveyor.

16.5.1.2 Overland Conveyor System

The surface infrastructure for the conveyor system consists of an overland conveyor and a fire suppression system at the drive end (portal) and at the head end (mill) of the conveyor.

16.5.1.2.1 Overland Conveyor Description

The discharge from the underground decline conveyor (CV-02) loads onto the overland conveyor (CV-01) located at the surface adjacent to the mine portal area using a rock box style transfer chute.

The overland conveyor utilizes a 1,050-mm wide steel cord belt operating at 3.5 m/s with a design capacity of 800 t/hr and is powered by a single 630-kW, four-pole motor with variable voltage variable frequency drives for starting and stopping control. The drive is located at ground level at a head drive station along with a gravity take-up system to manage belt tensions under all operating conditions.

The overland conveyor alignment is straight with an overall length of 2,000 m and lift of 154 m with a maximum inclination of 14° (25% grade) when rising to the plant feed bins.

The overland conveyor requires a civil earthworks formation along the majority of the conveyor length to accommodate the conveyor ground modules as well as a single lane unsealed maintenance access roadway suitable for conventional maintenance vehicles and light trucks.

The overland conveyor is mostly mounted upon rigid low-level ground based modular structures with hood covers to minimize snow and water ingress onto the belt and to protect the belt from ultraviolet sun damage. An access and maintenance road parallels the CV-01 conveyor from the mill to the mine portal. Crossovers are provided to allow access to the non-road side for maintenance every 300 m. The ground based modules utilize precast concrete sleeper footings. When the conveyor rises to the process plant feed bins the conveyor is elevated above the ground on steel truss and trestle structures with a walkway provided along one side for maintenance access.

The overland conveyor utilizes conventional belt rubber compounds as the requirements for fire resistant and anti-static performance are assumed to not apply. The use of low rolling resistance belt rubber compounds is also not warranted due to the relatively short length and low tonnage.

16.5.1.2.2 Overland Conveyor Route

The overland conveyor was deemed from the outset to be a straight conveyor with no horizontal curvature necessary. This was due to the relatively short conveyor length and absence of existing infrastructure or land ownership constraints between the mine portal and process plant location.

Whilst the mine portal location had been fixed due to prior mine planning and development approvals, the location of the process plant and specifically the plant feed bins were able to be relocated.

The initial process plant location defined by Ausenco resulted in the overland conveyor alignment passing along the side of a natural gully for a majority of the conveyor length as well as crossing a narrow side gully adjacent to the process plant. This required significant earthworks and additional elevated conveyor truss structures to negotiate as well as preventing the maintenance access road to follow the full length of the conveyor unimpeded.

In conjunction with the study team, the process plant location was moved in a northwesterly direction by approximately 330 m (whilst maintaining the overall conveyor length) which resulted in a longer portion of the overland conveyor alignment outside of the gulley and effectively eliminated the narrow steep-sided side gully crossing adjacent to the process plant. The optimized overland conveyor route resulted in a 50% reduction in bulk earthworks and 120 m less elevated conveyor truss structure.

16.5.1.2.3 Protective Devices

The overland conveyor system includes personnel and equipment protection devices. Nip points at all conveyor head ends, tail ends, drive areas, and pulleys are guarded to prevent injury. Emergency pull cords and emergency stop push bottoms are installed in various locations in the event of an emergency. Warning sirens will sound to alert personnel in the area before the conveyor starts, and conveyor belt slip and blocked chute detection is included in the design to protect personnel and the equipment.

The overland conveyor system utilizes conventional belt rubber compounds as the requirements for fire resistant and anti-static performance were assumed to not apply; however, the conveyor drive ends will have fire suppression as described in the following section.

16.5.1.2.4 Fire Suppression

The conveyor exiting at the portal will include fire suppression for the drive end located on surface. The fire suppression system will be fed from the fire water tank on surface, and will be identical to the systems used on the underground conveyors. A vertical in-line pump will be used to ensure approximately 31.5 L/sec (500 gpm) of fire water is supplied to the fire suppression unit at approximately 550 kPa (80 PSIG). Refer to Section 16.5.2.4.2 for a detailed description of the overall conveyor fire suppression system.

16.5.1.3 Water Storage

Service water and fire water tanks will be located on surface near the mine offices located at the portal. Both the service water tank and the fire water tank will be fed from the same DN150 (6”) water line which will be fed by the mill. The water supply to these tanks is included as part of the mill complex design. Both the service water and fire water tanks will be situated on a concrete pad.

The contractor selected to construct the decline will provide both a service water tank and fire water tank for the development of the decline. The assumption is that these tanks will be insufficient to support mine development and production. As such, Hatch has sized permanent service water and fire water tanks that will be installed once development of the decline is complete. Consideration should be made to having the decline contractor install service water and fire water tanks large enough to support mine development and production instead of installing two sets of tanks.

The permanent service water tank will have a capacity of 630,000L. Since temperatures can drop below 0°C, the tank will include an immersion heater. Since process water will always be in constant motion, the immersion heater will likely only be required during mine shutdowns. Heat tracing on the tank inlet and outlet will be incorporated and activated when temperatures drop below freezing.

The permanent fire water tank will have a capacity of 120,000L and will be independent from the service water tank to provide redundancy and to comply with Serbian regulations. Since the water in the tank will remain stagnant for long periods of time, the tank will be insulated and will include an immersion heater which will be activated when temperatures drop below freezing.

16.5.1.4 Surface Pipelines

A single DN150 (6”) clean water pipeline will run from the mill complex to the service water and fire water tanks located near the portal. The first section of the pipeline will run along the overland conveyor and will be insulated and heat traced. The next section of pipeline will run below surface in a trench between the overland conveyor and the service and fire water tanks. A DN150 (6”) service water pipeline and a DN150 (6”) fire water pipeline will run below surface in a trench between the surface water tanks and the portal.

Two DN150 (6”) dewatering pipelines will run from the +220 level underground pump station to the contact pond at the mill site. These pipelines will run along the overland conveyor on surface and will be insulated and heat traced.

All pipe lines will be constructed of Victaulic grooved schedule 40 piping. Since mine ventilation air will not be heated, all exposed surface piping and a portion of the underground piping in the dual decline will be insulated and heat traced.

16.5.1.5 Ventilation System

The ventilation system design for the Project consists of two separate air streams or splits. The first air stream ventilates the access and egress/conveyor declines, providing fresh air to the access decline and is exhausted from the mine via the conveyor decline. Additionally, production airflow is provided to the active levels via a system of fresh air raises (FAR) that provide air directly to each level in parallel. A series of return air raises (RAR) remove air directly from the levels and remove it from the mine without passing it over any other active areas.

Airflow quantities are controlled at the fresh air raise and return air raise regulators to ensure that each level receives sufficient airflow for the equipment/activities required, with any leakage moving from the level to the ramp. Having two separate fresh air supplies within the mine provides an added layer of security to the mine operations and will greatly assist in emergency egress should it become necessary for any reason.

The total required airflow for the Project was based on the Serbian regulation of 4 m^3/min per kilowatt of rated engine power (0.067 m^3/s per kW).

In accordance with recommended procedures for determining the airflow requirements based on the heat produced by diesel equipment, a second airflow determination was made based on the engine packages for the two most significant pieces of equipment expected to operate at Timok; the 30-t haul truck and the 7.0 m^3/14-t LHD.

The airflow distribution for the mine is based upon the diesel equipment that will be operating during the various mining activities (e.g. development, production, etc.). A total of 35 m^3/s will be required for each active level based on the expected diesel emissions; however, based on the heat produced by the equipment the ventilation system will need to provide approximately 40 m^3/s of fresh air to each heading. As with the total mine airflow requirement, the greater of these two amounts was utilized in developing the auxiliary ventilation system design.

In order to identify the maximum demand for ventilation, a time-phased ventilation model was constructed to represent the development of the mine for each year of the Project Development which includes the mine production period. The mine life is based on the commencement of production tonnes to the mill. Based upon the combinations of equipment utilization and physical mine development, three points of the mine life-cycle were identified as critical for ventilation system planning:

- Year 4, when the dual declines are being developed and before the ventilation raises have been completed.
- Year 10, when equipment usage is at its maximum.
- Year 11 (and afterwards), when the physical extents of mine development are reached.

The access decline will be utilized as a main intake airway and will be used as access for personnel, equipment, supplies and services. The egress decline will be utilized as a return airway and will provide a secondary egress in case of emergency. The egress decline will be utilized as a conveyor decline during the production stage of the mine. The development of the dual declines in parallel will provide significant benefits for ventilation efficiencies and safety. Haulage can be expedited and safety improved by allowing one-way haul traffic in the declines. A fan installation at the egress/conveyor portal will provide approximately 100 m^3/s of fresh air for the circuit, while ventilation at the faces will be provided by a 1.37-m diameter auxiliary duct that delivers 36 m^3/s to each face.

Based upon the results of the climate modeling performed, no mechanical refrigeration will be required at Timok. The surface ambient temperature record for Bor also shows that the mean temperature for the area never falls below freezing, and that minimum temperatures do not fall below -4°C at any time. As a result, it is not thought that air heating will be a significant concern at Timok. However, unknown environmental factors may make air heating at the mine necessary, particularly if there is significant groundwater intrusion near the fresh air raise.

16.5.1.5.1 Main Ventilation Fans

The main ventilation fans have been sized using the SRK mine design, equipment numbers and equipment schedule. The resulting ventilation analysis is summarized in Table 16.5. The pressures in the table below are for mine losses only, and do not incorporate fan system losses. These additional losses were incorporated when sizing the fresh air and return air fans and motors, and are based on vendor supplied information. The surface fan motors will include VFD’s (variable frequency drives) to enable variable speed operation. In the event of an underground emergency the surface fans can be reversed to provide partial flow in the opposite direction.

Table 16.5: Ventillation Fan Parameters

Fan Installation	Quantity (m^3/s)	Pressure (Pa)
Fresh Air Raise	300	3,225
Return Air Raise	300	1,750

16.5.1.5.2 Fresh Air Raise Fans

The fresh air raise will be fed using two fans in a parallel arrangement. The fans will be located approximately 500 meters west of the concentrator area.

Due to the large size of fan motors required, the fans are to be Arrangement 3 (Air Movement and Control Association [AMCA] Standard), meaning that the drive motor will be located external to the fan. It was determined that the fresh air fan motors are to be sized at 1,750 hp (1,304 KW) each. The intake of each fan will include a plenum building equipped with a monorail to install and service the fan motor inside. The plenum buildings will include intake silencers to reduce the noise level in the surrounding area. The fan discharge ducting will include dampers to isolate a fan in the event of scheduled maintenance or equipment failure. Due to the temperate climate, mine air heating and cooling will not be included. The discharge from both fans will ducted into a single five meter diameter fresh air raise. A container style building will house the electrical infrastructure for the two fans. An ethyl mercaptan injection (stench gas) injection system will be located at the fresh air raise collar and will be used to alert underground workers of an emergency.

16.5.1.5.3 Return Air Raise Fans

The return air raise will be exhausted using two fans in a parallel arrangement. The fans will be located adjacent to the long term core sample storage area.

The fans are to be Arrangement 3 (AMCA standard), such that the motor will be located external to the fan. It was determined that the return air fan motors are to be sized at 1,200 hp (895 KW) each. The fan motors will be surrounded by a platform which will enable maintenance personnel to access them. The motors will be rated for outdoor use, and will be shielded from precipitation. No provision has been made for a building overtop of the fan motors. Installation and complete replacement of the motors will be completed using a mobile crane. The ducting from the return air raise to each fan will include dampers to isolate a fan in the event of scheduled maintenance or equipment failure. Exhaust from each fan will be routed through a silencer before being discharged out of a vertical stack. The silencers and exhaust stacks will both contribute to reducing the noise level in the surrounding area. Exhaust air will be routed through a basic demister (mist remover) to capture a portion of the water droplets in the airstream before it is dispersed into the atmosphere. A container style building will house the electrical infrastructure for the two fans.

16.5.1.5.4 Portal Exhaust Fan

In addition to the fans on top of the return air raise, a small exhaust fan will be located at the conveyor portal. The portal exhaust fan will be sized by SRK and specified and installed by the decline contractor.

16.5.2 Underground Mine Infrastructure

The major underground mine infrastructure required to support the operation of the Timok Mine will include:

- **Ore and Waste Handling** - dumps, passes, rock breaker stations, feeders, crushers.
- **Underground Conveyors** - conveyors, drive units, take-up units, transfer chutes.
- **Mine Dewatering** - pump stations, sumps, piping.
- **Mine Services,** service water reticulation, fire suppression systems, compressed air, ballast production, shotcrete production and handling.
- **Mine Support Facilities** - satellite maintenance shop, refuge stations, sanitation cut-outs, permanent explosives and detonator magazine bays, consumable mining material storage, secondary egress escapeway, ventilation doors, regulators and ore pass covers.
- Underground Mobile Equipment.

- Underground Mine Electrical Power Distribution – transformer, switchgear, MCC, cables.
- **Controls and Automation,** process control systems, control room, network general arrangement, communications and mine systems, wired and wireless network infrastructure, voice communication, emergency warning system, blasting system, etc.

16.5.2.1 Ore and Waste Handling

16.5.2.1.1 Ore and Waste Circuit Summary

Sublevels will be developed from the main access decline and subsidiary decline at 20 m vertical intervals to create a total of 17 production levels. Each production level will have access to two ore passes and one waste pass. Blasted material will be mucked from draw points using 7.0 m^3 LHDs and dumped into the appropriate ore/waste pass. Each dump includes a grizzly to prevent oversize from entering the pass, and will be serviced by a mobile rock breaker. The ore/waste passes will transfer material from the production levels to one of the re-muck levels located above each primary crusher room. Two crusher rooms will be constructed, one for the upper mine on the -275 m level, and the other for the lower mine on the -435 m level. Only one crusher will be operational at a time.

As there is no direct feed from the ore/waste pass system to the rockbreaker station, 8.6 m^3 LHDs will be used to re-muck material from the bottom of the ore/waste passes to the rockbreaker station on the re-muck level. The rockbreaker station will include a grizzly and permanent rockbreaker. Material that passes through the grizzly will drop into a bin which will connect to the crusher room below. Flow from the bin will be regulated with a press frame and chain control assembly. Material will be directed onto a vibratory grizzly feeder which will separate out fines and discharge oversize into the primary jaw crusher. Once crushed, the material will discharge onto the crusher discharge conveyor below for transport to surface. Please refer to Table 16.6 for a range of the anticipated ore and waste properties.

Table 16.6: Anticipated Ore and Waste Properties

Ore Properties	Unit	Min	Max
In-Situ Density (Wet)	t/m^3	3.2	4.7
Bulk Density (Broken)	t/m^3	1.9	3.1
Uniaxial Compressive Strength	MPa	80	115
Swell Factor	%	70	50
Moisture Content (by Weight)	%	3	7
Waste Rock Properties	**Unit**	**Min**	**Max**
In-Situ Density (Wet)	t/m^3	2.4	2.4
Bulk Density (Broken)	t/m^3	1.4	1.6
Uniaxial Compressive Strength	MPa	20	80
Swell Factor	%	70	50
Moisture Content (by Weight)	%	3	7

16.5.2.1.2 Ore and Waste Dump Stations

Blasted material will be mucked from draw points using 7.0-m^3 LHDs and dumped into the ore/waste pass system. Each level will have two ore dump stations and one waste dump station.

Each dump station will be equipped with a grizzly panel with 1,500 mm x 750 mm openings to prevent oversize material from entering the passes. A raised concrete lip extending across the grizzly opening will be used to prevent LHDs from driving onto the grizzly. Each dump station will incorporate two drop-in grizzly panels. The dump stations will include a 5 tonne lifting lug and fall arrest tie-off points to assist with grizzly panel installation and removal.

Permanent rock breakers will not be installed at these dump stations. Instead secondary breakage of material will be done using a mobile rock breaker and/or a blockholer as required.

16.5.2.1.3 Ore and Waste Passes

Each production level will have access to two ore passes and one waste pass. The ore/waste passes will transfer material from the production levels to one of the re-muck levels located above each primary crusher room. The dump stations on each level will connect to the ore and waste passes using finger raises. As per the mine layout, the production levels are segmented into an upper mine section and a lower mine section. The upper section contains two ore passes and a waste pass that feed onto the re-muck level located above the upper -275 m level crusher. The lower section contains two ore passes and a waste pass that feed onto the re-muck level located above the lower -435 m level crusher. The upper ore/waste passes do not connect to the lower ore/waste passes.

The raise-bored ore and waste passes will be 3 m in diameter and will be inclined. The ore and waste passes will not be lined. The relatively small 3 m diameter passes were selected due to concerns with ground conditions. The finger raises will be 2.0 m x 2.0 m, and will not be lined. Consideration should be given in the next stage of study to either enlarge the passes or decrease the level dump grizzly openings in order to reduce the likelihood of hang-ups.

The bottom of each ore and waste pass will be open. The passes are to be left sufficiently full so that there will not be an open brow. In order to prevent a pass from being emptied, a measuring device and indicator light will be employed to inform the LHD operator if the pass can be pulled. If a pass is to be emptied, the LHD will be operated remotely when mucking.

16.5.2.1.4 Rockbreaker Stations

Two rockbreaker stations with grizzlies, permanent rockbreakers and picking booms will be utilized. One rockbreaker station will feed the upper -275 m level crusher, and the other station will feed the -435 m level crusher.

As there is no direct feed from the ore/waste pass system to the rockbreaker stations, 8.6 m³ LHDs will be used to re-muck material from the bottom of the ore/waste passes to the appropriate rockbreaker station. Material that passes through the rockbreaker station grizzly will drop into a bin which connects to the crusher room below.

Each station will be equipped with a grizzly, rock breaker, picking boom and local operators booth. The grizzly will have 750 mm x 750 mm openings, and will be used to prevent oversized material from reaching the crusher. The picking boom will be used to remove tramp metal. A removable bumper extending across the grizzly opening will be used to prevent LHDs from driving onto the grizzly. Each rockbreaker station will incorporate two drop-in grizzly panels. The rock breakers and picking booms will be hydraulically powered and will be operable either remotely, or locally from the operator’s booth. A 5 tonne lifting lug and fall arrest tie-off points have been included to assist with grizzly panel installation and removal.

In the next stage of study, consideration should be given to a direct feed arrangement, in which one or more ore of the ore passes feeds directly into the rockreaker station. A direct feed arrangement would reduce the amount of material that has to be re-mucked.

16.5.2.1.5 Crusher Bins

A single bin will be located above each primary crusher room. Each bin will transfer material from the re-muck level to the associated crusher room approximately 20 m below. The bins will each be 5.4 m x 5.4 m in size. The approximate bin capacities are as follows:

Table 16.7: Approximate Bin Capacities

Material	Approx. Bin Capacity (each)
Ore	1,108 – 1,808 tonnes
Waste	817 – 933 tonnes

As these bins will encounter high wear, they are to be fully supported and lined with hardened shotcrete. Consideration in the next stage of the study should be given to lining the bins with steel to increase bin life. Since a single bin for ore and waste will feed each crusher, operations will have to be cognisant when switching between ore and waste in order to minimize dilution. In order to prevent wear on the feeder infrastructure below, the bins are to be kept at 20% capacity. A level indicator will be utilized to assist with this. Consideration should be given to increasing the capacity of these bins to provide greater operational flexibility.

16.5.2.1.6 Primary Crusher Feeders

Flow from each crusher feed bin will be regulated with a press frame and chain control assembly. The bottom of each bin will contain a steel lined throat that will direct material towards the press frame. The press frame and the chains will regulate the flow of material out of the bin. Blast ports will be located above the press frame to enable quick clearing of hang-ups. A 5-tonne monorail and hoist will enable installation and replacement of the press frame, as it will be located above the crusher room bridge crane. A steel chute equipped with liners will be used to contain the flow of material from the press frame, and direct it onto the vibratory grizzly feeder.

The vibratory grizzly feeder will contain scalping bars which will separate out fines from the crusher feed. The oversized material will be fed into the jaw crusher where it will be sized. A chute located under the scalping bars will direct fines onto the crusher discharge conveyor below. Steel skirt walls equipped with liners will be located on either side of the feeder to contain the material. The top of the feeder will be left open to enable tramp steel to be removed.

Several feeder configurations were considered. All three options are fed from a chute with a press frame and chain control assembly to regulate flow. A flooded apron feeder arrangement was not considered as the press frame was requested. Ross chain feeders were also not considered. The vibratory grizzly feeder was ultimately selected as it enables the crusher to meet it's design requirements, is the most economical solution, and is regarded as a common configuration. The following is an overview of the three feeder options considered:

Vibratory Grizzly Feeder with Scalping

- Recommended by Metso and regarded as a common configuration.
- The most cost-effective option as the feeder and scalper are in one unit.
- Scalping decreases the chance of packing and increases the efficiency of the crusher. It enables the crusher to meet its throughput requirements while ensuring the topsize does not exceed the conveyor limitations.
- Not great with wet/sticky material, not as heavy duty as an apron feeder, and more sensitive to large loads because the unit vibrates.

Apron Feeder to Scalper

- Includes the benefits of scalping as mentioned above.
- The apron feeder can handle wet/sticky material and has slightly better control over the feed rate under varying conditions.

- The most expensive option as it requires an apron feeder, scalper unit, and dribble chute/conveyor for fines. Potentially requires a larger crusher room excavation. Generally higher operating costs.
- Potential for debris to get hung up in the grizzlies.

Apron Feeder Direct to the Jaw Crusher (No scalping)

- The simplest installation.
- Apron feeder can handle wet/sticky material and has slightly better control over the feed rate under varying conditions.
- No scalping requires the recommended CSS (closed side setting) to be 1” larger than the crusher minimum to prevent packing. The Metso C130 crusher selected cannot crush the material small enough and at the required capacity when scalping isn’t implemented.
- Upsizing to the larger Metso C150 crusher does not solve this problem as the minimum CSS increases on this unit. When adding 1” to the minimum CSS to account for no scalping, the topsize exceeds the conveyor limitations.
- Not suitable in this application given the current parameters.

16.5.2.1.7 Primary Crushers

Timok will have two underground primary crushers. One crusher will be located on the -275 m level, and the second crusher will be located on the -435 m level. Both primary crushers will be jaw crushers. Only one crusher will be operational at a time.

Oversize material on the vibratory grizzly feeder will discharge into the primary crusher. Once crushed, the material will discharge onto the crusher discharge conveyor for transport to surface. A chute equipped with liners will ensure material and dust is contained, and that material is correctly loaded onto the conveyor belt. The chute will incorporate a partial dead bed to minimize the amount of material directly impacting the conveyor belt. An impact bed or series of impact idlers will be located underneath the conveyor belt at the conveyor discharge point to protect the belt from damage. During the next stage of the study, the option of installing a vibratory feeder below the crusher to feed the conveyor belt should be explored.

Both crushers will be identical in size, and for the purpose of this study Metso C130 jaw crushers were selected. The nominal capacity of each crusher will be 468 tph (tonnes per hour), however each crusher will have a design capacity of 585 tph. These capacities are overall, and account for scalped material throughput. Each crusher will be capable of reducing feed material of 750 mm minus to 220 mm minus. The 750 mm minus is driven by the size of the rockbreaker station grizzly openings. The 220 mm minus requirement is driven by the CDI conveyor system which has a design capacity of 800 tph, and has a maximum allowable lump size of 220 mm.

The nominal capacity of the crusher was provided as an input from the client. When using the anticipated ore production of 3.25 Mtpa and assuming 360 operating days, this equates to the crusher being operational approximately 19.3 hours per day when running at nominal capacity, or 15.4 hours per day when running at design capacity. Since the crusher will also be processing waste, additional run time will be required to crush the waste which has an anticipated nominal production of 220 Ktpa.

Each crusher room will include a 30 tonne bridge crane to enable installation and service of the jaw crusher and vibratory grizzly feeder. A dust collection system will ducted to the crusher discharge chute to reduce the amount of airborne dust in the crusher room. An operators cab will provide local controls for the press frame, vibratory feeder, and crusher. The cab will include an air filtration unit and air conditioner.

Four platform levels will enable operation and maintenance personnel to access infrastructure as required. The ‘crusher level’ platform will provide access to the crusher motor and drive belt. The ‘feeder level’ platform will provide access to the top of the crusher, the vibratory feeder, and a portion of the chute. The operators cab will be situated on this level as it is the most central location. The ‘upper level’ platform will provide access to the press frame and the upper portion of the chute. The ‘top level’ platform will provide access to the blast ports and the upper portion of the press frame.

16.5.2.1.8 Ore and Waste Storage

No allowance has been made for dedicated underground ore and waste bins. Storage for waste material will be in level re-mucks. Temporary storage for waste material will be in the waste pass system. Temporary storage for ore will be in the ore pass system and in the bins that feed the crushers. In order to avoid hang-ups, the passes are only designated as temporary storage locations.

16.5.2.2 Underground Conveyors

The underground conveyor system was designed by Conveyor Dynamics Inc. (CDI) in conjunction with constraints identified by the clients and SRK, the mine development team.

16.5.2.2.1 Material Flow Sheet

The underground conveyor system requires a series of conveyors to transport the material from the two underground primary crushing stations. The lower crusher station discharges onto a crusher discharge conveyor, which loads onto a series of switchback conveyors CV-07, CV-06 and CV-05, to the upper crusher station. The upper crusher station discharges onto the crusher discharge conveyor, which loads onto conveyor CV-03 that rises to load onto the main decline conveyor CV-02, which in turn transports material to the main portal area at the surface. The decline conveyor discharges onto the overland conveyor, CV-01, which travels over the natural terrain to the process plant and discharges into the plant feed bins.

The conveyor system has been designed to cater for either 100% upper or 100% lower crusher station operation. A combination of upper and lower crusher station operation could be accommodated with proportional discharge to ensure the conveyor system is not overloaded.

16.5.2.2.2 Crusher Discharge Conveyors CD-01 and CD-02

The upper and lower crusher stations discharge onto the respective discharge conveyors CD-01 and CD-02 via a rock box style transfer chute. Each conveyor includes a belt weightometer for process control of the crusher discharge rate and a tramp iron magnet to remove potentially damaging foreign material.

The crusher discharge conveyors utilize a wider 1,200-mm wide PN-600 fabric belt operating at a slower 2 m/s to minimize wear. These conveyors have a design capacity of 800 t/hr and each conveyor is powered by a single 75-kW, four-pole motor with variable voltage variable frequency drive controlled starting and stopping. The drive is located at the head pulley and a fixed take-up system is located at the tail end.

The crusher discharge conveyors are straight with an overall length of 50 m and lift of 3 m at an inclination of 4° to achieve the required transfer height. They are mounted upon elevated modular structures mounted from the tunnel floor with a walkway provided along one side for maintenance access.

16.5.2.2.3 Underground Conveyor CV-07

The lower crusher station discharge conveyor discharges onto underground conveyor CV-07 which commences the incline to the upper crusher station level.

Underground conveyor CV-07 utilizes a 1,050-mm wide ST-1100 steel cord belt operating at 3.5 m/s with a design capacity of 800 t/hr and is powered by a single 400-kW, four-pole motor with a variable voltage variable frequency drive that starts and stops the motor. The drive is located at the head pulley and a gravity take-up system is located at the tail end to manage belt tensions under all operating conditions.

Underground conveyor CV-07 is straight with an overall length of 565 m and lift of 80 m at an inclination of 8° (14% grade). The conveyor is primarily mounted on a suspended conveyor structure from the roof (back) of the tunnel along the far-right hand side when viewed in the direction of conveyor travel. This allows sufficient room for vehicles to travel alongside the conveyor for access and maintenance purposes.

16.5.2.2.4 Transfer Conveyors CV-06 and CV-04

The transfer conveyors are short linking conveyors that transfer the material between the main decline conveyor flights and are arranged in a switchback arrangement. Physical geometry and chamber excavation volume prevents the main decline conveyors from directly loading from one to another.

The transfer conveyors utilize a wider 1,200-mm wide PN-600 fabric belt operating at a slower 2.0 m/s to minimize wear with a design capacity of 800 t/hr. Each transfer conveyor is powered by a single 75-kW, four-pole motor with a variable voltage variable frequency drive that starts and stops the motor. The drive is located at the head pulley and a fixed take-up system is located at the tail end.

The transfer conveyors are straight with an overall length of 28 m and lift of 4 m at an inclination of 8° to achieve the required transfer height. They are mounted upon elevated modular structures mounted from the tunnel floor with a walkway provided along one side for maintenance access.

16.5.2.2.5 Underground Conveyor CV-05

The discharge from the transfer conveyor CV-06 loads onto the underground conveyor CV-05 which travels up the decline to connect with CV-03 close to the upper crusher station level.

Underground conveyor CV-05 utilizes a 1,050-mm wide ST-1100 steel cord belt operating at 3.5 m/s with a design capacity of 800 t/hr and is powered by a single 400-kW, four-pole motor with variable voltage variable frequency drives that starts and stops the conveyor. The drive is located at the head pulley and a gravity take-up system is located at the tail end to manage belt tensions under all operating conditions

Underground conveyor CV-05 is straight with an overall length of 694 m and lift of 97 m at an inclination of 8° (14% grade). The conveyor is primarily mounted on a suspended conveyor structure from the roof (back) of the tunnel along the far-right hand side when viewed in the direction of conveyor travel. This allows sufficient room for vehicles to travel alongside the conveyor for access and maintenance purposes.

16.5.2.2.6 Underground Conveyor CV-03

The discharge from both the transfer conveyor CV-04 and the lower crusher station discharge conveyor load onto the Underground conveyor CV-03 which travels up the decline to connect with CV-02 half way to the surface portal.

Underground conveyor CV-03 utilizes a 1,050-mm wide ST-1100 steel cord belt operating at 3.5 m/s with a design capacity of 800 t/hr and is powered by a single 400-kW, four-pole motor with variable voltage variable frequency that starts and stops the conveyor. The drive is located at the head pulley and a gravity take-up system is located at the tail end to manage belt tensions under all operating conditions Underground conveyor CV-03 is straight with an overall length of 753 m and lift of 105 m at an inclination of 8° (14% grade).

Underground conveyor CV-03 is primarily mounted on a suspended conveyor structure from the roof (back) of the tunnel along the far-right hand side when viewed in the direction of conveyor travel. This allows sufficient room for vehicles to travel alongside the conveyor for access and maintenance purposes.

16.5.2.2.7 Decline Conveyor CV-02

The discharge from underground conveyor CV-03 loads directly onto the decline conveyor CV-02 which finalizes the lift to the surface portal.

Decline conveyor CV-02 utilizes a 1,050-mm wide ST-1100 steel cord belt operating at 3.5 m/s with a design capacity of 800 t/hr and is powered by dual 650-kW, four-pole motor with variable voltage variable frequency drives that starts and stops the conveyor. The dual drives are located at ground level outside the portal at the head drive station. A gravity take-up system is located at the tail end to manage belt tensions under all operating conditions.

Decline conveyor CV-02 is straight with an overall length of 3,210 m and lift of 445 m at an inclination of 8° (14% grade).

Decline conveyor CV-02 is primarily mounted on a suspended conveyor structure from the roof (back) of the tunnel along the far-right hand side when viewed in the direction of conveyor travel. This allows sufficient room for vehicles to travel alongside the conveyor for access and maintenance purposes. When the conveyor reaches the surface portal the conveyor is elevated above the ground on steel truss and trestle structures, with a walkway provided along one side for maintenance access. This allows a 5-m height clearance for vehicles passing under the conveyor to the main access portal area.

16.5.2.2.8 Conveyor System Alignment

The underground conveyor system alignment has been designed with several constraints. Firstly, the portal location and the exploration dual decline were both fixed due to prior mine planning and development; therefore determining the alignment of the discharge conveyor at the portal and the overland conveyor. The upper and lower crusher stations were defined by the mineral deposit and mine plan, which also indicated the alignment of the conveyor system. Another constraint was the maximum decline inclination of 14%.

In conjunction with the SRK mine development team, the underground conveyor system alignment and arrangement was selected to meet the above constraints whilst minimizing the conveyor system length and providing access to the lower crusher station.

16.5.2.2.9 Underground Conveyor Structure

The majority of the underground conveyor system utilizes a suspended conveyor structure from the roof (back) of the tunnel along the far-right hand side when viewed in the direction of conveyor travel (see Figure 16.5 and Figure 16.6).

Figure 16.5: Typical Underground Suspended Conveyor Structure

The suspended conveyor structure utilizes dual stringer sections to which the carry and return idlers mount and provides a suspension point for the chain suspension system. The suspension system connects to affixing points on roof bolts which are arranged in pairs nominally every 4 m along the length of the tunnel. Turnbuckles and diagonal bracing chains are incorporated to provide sufficient stability and adjustability.

A cross member section above the conveyor between the suspension chains provides both support for a continuous cable ladder and a fixing point for light fittings which are required along the full length of the underground conveyor system.

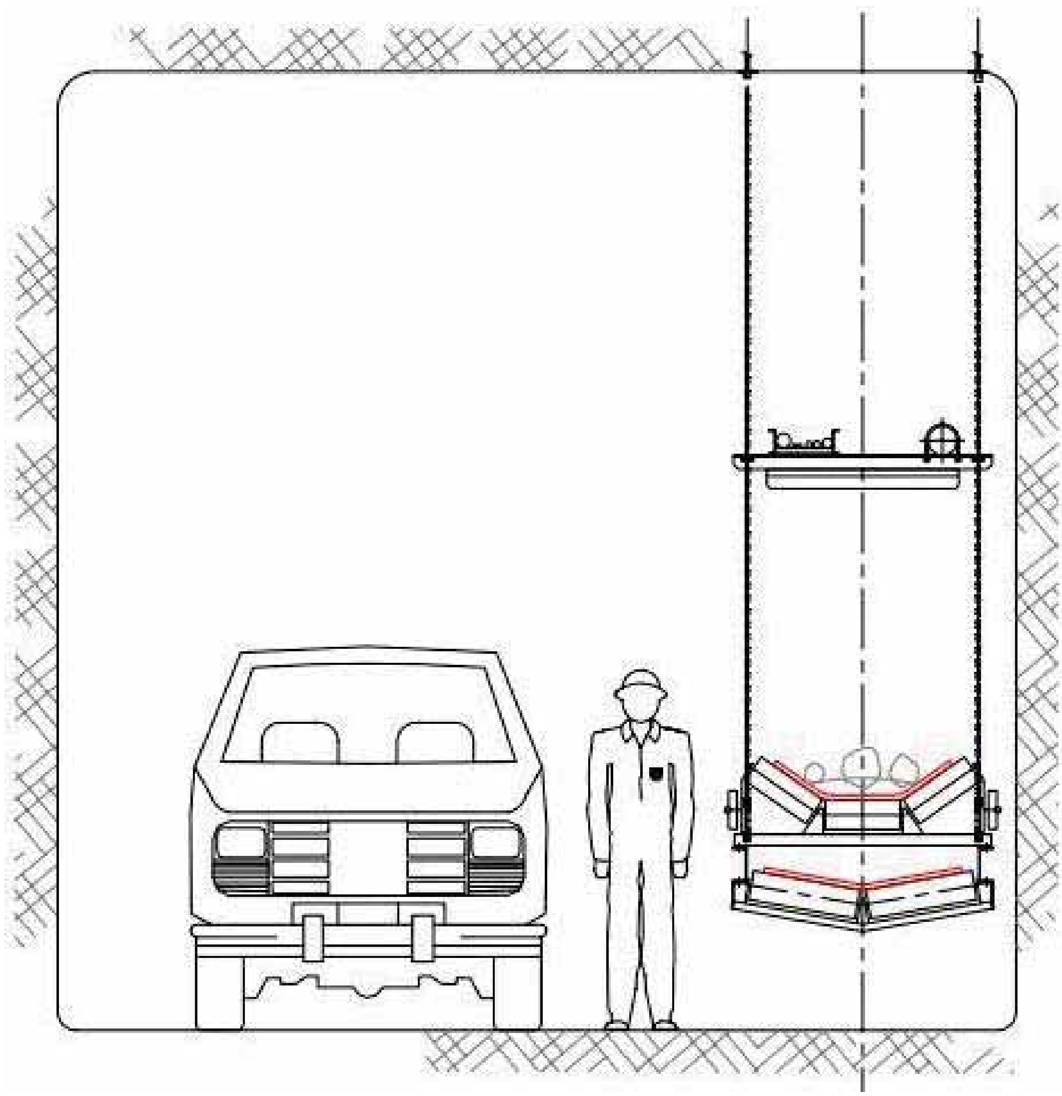


Figure 16.6: Timok Typical Underground Conveyor Section

16.5.2.2.10 Protective Devices

The underground conveyor system includes personnel and equipment protection devices. Nip points at all conveyor head ends, tail ends, drive areas, and pulleys are guarded to prevent injury. Emergency pull cords and emergency stop push bottoms are installed in various locations in the event of an emergency. Warning sirens will sound to alert personnel in the area before the conveyor starts, and conveyor belt slip and blocked chute detection is included in the design to protect personnel and the equipment.

The underground conveyor system utilizes conventional belt rubber compounds as the requirements for fire resistant and anti-static performance were assumed to not apply. A fire suppression system has been included at each conveyor drive end. See Section 16.5.2.4.2 for a description of the system.

16.5.2.3 Mine Dewatering

The mine dewatering system will consist of a contact water pond, pump stations, level sumps, and a ramp bottom sump. The mine water will drain to level sumps, which are gravity fed through boreholes until reaching a sump that pumps to a pumping station. There will be a total of five areas for pumping stations along the conveyor decline which will pump “cascade style” up to the contact water pond located at the mill.

16.5.2.3.1 Pump Skid Arrangements

The pump stations discussed below will be equipped with a prefabricated two-piece pump skid. The reservoir will occupy the largest portion of the skid and will have a live capacity of 45 m^3. A ladder will be installed to perform a visual check of the inside of the tank. The reservoir will be supplied with an optional overflow connection and transfer trough for diverting overflow and transfer water, respectively, to a secondary tank. The second portion of the skid will house the two pumps, motors, agitator and required electrical cabinets. This secondary skid arrangement will mount to the front of the reservoir or to the side to allow modularity depending on the pumping requirements at each station. An external jet pump-style agitator was chosen in order to eliminate the requirement for a catwalk over the tank. Although each pump will be identical, the motor rating will vary depending on the location of each station.

16.5.2.3.2 Pump Stations

Each pump station will be comprised of two pump skids with two pumps per skid, for a total of four pumps. A total of five pump stations will be needed once the mine reaches full capacity, with an identical layout. A single skid will be used for standard duty while the second skid will remain on standby to cover off maintenance requirements or additional pumping requirements during peak periods. In other words, during peak periods, it is expected that all pumps will be operational. The desired capacity of 60L/sec will be met utilizing a single skid of pumps such that when both skids are operational, the capacity will temporarily be doubled. The pump stations will be fully automated and transportable so that they can be easily moved while the mine is being developed.

All pumps will be capable of pumping dirty water with up to 5% solids by weight and the pump stations will be equipped with agitators as required to re-suspend settled solids. Average solids content will be estimated at <2%. See Table 16.8 below for a summary of the pump sizes based on the provided design parameters.

Table 16.8: Design Parameters and Pump Sizes for Dewatering System.

Location (m level)	Pipe Diameter (mm)	Design Capacity (L/sec)	Pump Size kW (hp) x 2 pumps
+220	150	60	224 (300)
+20	150	60	224 (300)
-120	150	60	150 (200)
-280	150	60	150 (200)
-400	150	60	112 (150)

Since there are two skids per pump station, two lines of carbon steel piping will be installed to pump from one pump station to the other up ramp and from the mine portal to the mill site. Two lines are used to add redundancy to the design and maintain pipe velocity to move entrained solids. Underground carbon steel piping will be hung from the back using chains while surface piping will be installed alongside the overland conveyor. HDPE piping was not selected based on it’s lower pressure ratings.

16.5.2.3.3 Level Sumps

There will be sumps at each level which will drain through 200 mm (8”) uncased boreholes. There are two types of sumps. The majority of the sumps will have a borehore draining into a sump box that will drain to a sump on the next level. The other type of sump drains into a sump box that is then pumped to the nearest pump station.

Water from the sumps located above the -260 level will drain to a collection sump on the -260 level. Water from this collection sump will then report to the pump station on the -280 level. Water from the sumps on the -280 level and below will drain to a collection sump on the -400 level. Water from this collection sump will then report to the ramp bottom sump before being pumped up to the -400 pump station. Applicable level sumps will contain an 11 kW (15 hp) submersible pump and the ramp bottom sump will have a 30 kW (40 hp) submersible pump.

16.5.2.4 Mine Services

16.5.2.4.1 Service Water

Service water will be supplied to all underground mine workings within the Timok Mine. It will be used primarily for drilling, dust control, and washing. The service water distribution system for the Timok mine will consist of a service water tank (covered in Section 16.5.1.3) underground distribution piping, and pressure reducing valve (PRV) stations.

Pressure reducing valve (PRV) stations will be installed at regular intervals to reduce water pressure to 552 kPa (80 psi). The maximum usable water pressure will be limited to 690 kPa (100 psi) for safety reasons.

Distribution piping and PRV stations were sized to accommodate maximum design flow rates. All primary PRV stations will be setup with two PRV's in parallel. Only one PRV branch will be operational at a time, however this setup provides redundancy and enables maintenance to be performed on the system without interrupting production. All secondary PRV stations used for takeoff lines will only incorporate a single PRV.

An estimated average service water consumption rate of of 21.2 L/s (336 gpm) was provided by the client. A maximum mine wide service water consumption rate of 31.5 L/sec (500 gpm) was selected for design purposes. The maximum service water consumption on a single level was estimated to be 16 L/sec (250 gpm). The maximum service water consumption for a single cross-cut or take-off line was estimated to be 6.3 L/sec (100 gpm). As such, primary distribution piping was sized at DN150 (6"), level distribution piping was sized at DN100 (4"), and cross-cut/take-off distribution piping was sized at DN50 (2").

The surface service water tank was sized to sustain the average water consumption rate for a period of eight (8) hours.

Service water will be distributed to three independent sections of the mine so that as the active mining area progresses deeper, service water supply to upper levels can be shut-off without affecting lower levels. Where deemed feasible, the service water lines will be connected level to level using the escapeway system. Although service water will be initially run to each level during development using the ramp, it may be advantageous to switch the feed to a larger permanent pipeline in the escapeway system once initial level development is complete. It will be easier to run temporary DN100 (4") piping down the ramp for development use than to permanently install DN150 (6") piping down the ramp. Locating the permanent pipeline in the escapeway system will ensure the piping and pressure reducing valve (PRV) stations are protected from mobile equipment. Further investigation regarding the feed to each level should be conducted during the next stage of study. Above all else, the service water piping should be located such that sub-zero temperatures are avoided.

HDPE pipe has not been considered as it is not suitable for the higher pressures experienced between the PRV stations. In addition, HDPE piping requires more supports than steel piping, likely offsetting any potential cost savings in this application. HDPE piping may offer some benefit when used as level piping, where pressures will be lower and piping will be smaller. For the sake of this study all piping has been assumed to be Victaulic grooved steel piping. The use of HDPE piping downstream of level PRV stations should be explored during the next stage of study.

16.5.2.4.2 Fire Suppression

Fire suppression systems or appropriate fire extinguishers will be present throughout the mine. All conveyor drive ends will feature water based fire suppression systems. The remaining at risk areas in the mine such as satellite maintenance shops and pump stations will be protected using hand held fire extinguishers. The satellite maintenance shops will include fire doors at all entrances as an extra precaution. If a fire were to break out, the fire doors would enable the area to be sealed off from the rest of the mine. In addition, all mobile equipment will be equipped with on board fire suppression systems.

A central fire water system will supply water to the conveyor drive end fire suppression systems. A heated and insulated 120,000 litre fire water tank (covered in Section 16.5.1.3), will be located on surface to provide a dedicated supply of fire water to the mine and to comply with Serbian regulations. Fire water piping will run down the conveyor decline, with pressure reducing valve stations located at regular intervals.

Each conveyor drive end will feature a deluge style fire suppression system and a fire hose. With a deluge style fire suppression system, the sprinkler heads remain in an open position and are connected to the water supply line through an electronic valve that is only opened when a control unit detects a fire (or heat or smoke). The advantage to a deluge style system is that it ensures the sprinkler piping remains dry, and is not prone to freezing. When compared to a dry pipe fire suppression system, a deluge system has a quicker activation time and is better suited to rapidly growing and spreading fires. The deluge system activates all sprinklers simultaneously, providing large quantities of water over a specified area.

16.5.2.4.3 Compressed Air

The Timok mine will not be equipped with a traditional compressed air distribution system. Mobile equipment requiring compressed air for operation will be purchased with on-board compressors. Due to the limited requirement for continuous compressed air, portable compressors will be used for any work requiring compressed air while local compressors will be used for the maintenance facilities.

16.5.2.4.4 Ballast Production and Delivery System

Road ballast is to be stockpiled on surface and delivered underground using haul trucks. The road ballast is to be purchased from an offsite facility. Alternatively a contractor could be brought on site to process ROM waste material using contractor supplied equipment. No provisions for a road ballast crusher have been made.

16.5.2.4.5 Concrete/Shotcrete Production and Handling

A batch plant located on surface will produce both wet concrete and shotcrete. Batches of concrete and shotcrete will be trucked underground using a series of transmixers. Please refer to report Section 18.1 - Surface Infrastructure for details regarding the batch plant design and location.

16.5.2.5 Mine Support Facilities

16.5.2.5.1 Satellite Maintenance Shop

Two satellite maintenance shops are to be constructed underground, one adjacent to the travel way decline at approximately the -170 m level, and the other on -320 m level. These satellite shops will be used to handle basic underground maintenance. All major repair work will be completed in a larger maintenance facility on surface or outsourced to an off-site facility.

Each shop will have a 150 mm thick concrete floor throughout, and will be equipped with lighting in all bays and travel ways. A fire door and personnel door arrangement will be located at each entrance, and an air compressor will service the service bay and crane bay. All mobile equipment bays were sized to fit the largest piece of equipment, a 30 t haul truck or a 17 t LHD.

Each underground satellite maintenance shop will comprise of the following bays:

- **Service Bay** - The service bay will feature a ramp that will allow access to the undercarriage of mobile equipment.
- **Crane Bay** - The crane bay will feature a bridge crane spanning the width and running the length of the bay. For the purpose of this study, a 25 tonne crane was sized.
- **Wash Bay** - The wash bay will contain a hot water pressure washer which will be fed from the mine service water system. The pressure washer will be located at the back end of the wash bay, along with a storage area for cleaning detergent. A strip curtain will separate the wash bay from the adjacent cross cut. At the front end of the wash bay, a floor drain spanning the width of the bay that will drain into a sump. The sump will include a submersible pump which will pump waste water though an oil/water separator and to the level sump. A back mounted fall arrest system will enable personnel to safety wash equipment.

- **Office** - An office will be located in the cross cut between the wash bay and the service bay. It will be a small, self-contained room with enough space for two desks. It will also contain a phone and a computer.
- **Tool Crib** - The cross cut between the storage area and the crane bay will be used as a tool crib. It will feature an expanded metal wall and locking doors so that tools and equipment can be secured when not in use.
- **Storage Area** – The storage area will contain steel racking and will be used to store common mobile equipment spares and consumables. The storage area will include a wire screen wall and locking door to enable the area to be secured.
- **Laydown Area** – The laydown area will include a raised concrete pad that will be used for temporary material storage.

Fixed plant equipment maintenance personnel will be based out of maintenance shops located at the mill site. Electricians and millwrights servicing underground fixed plant equipment will be equipped with service trucks. No provisions for underground millwright, electrical, or welding shops have been made.

A temporary shop on surface near the portal is to be setup by the contractor selected to construct the decline. This shop will be used to service the contractor's equipment during initial development, and will be in addition to the permanent maintenance shop on surface.

16.5.2.5.2 Refuge Stations

Refuge stations will be used to provide a safe location for underground personnel to gather in the event of an emergency. Two permanent refuge stations will be located underground, one adjacent to the travel way decline on approximately the -200 m level, and the other on the -380 m level. Due to the expanse of the mine, five portable refuge stations will be used to supplement the permanent refuge stations. As there will be approximately three active levels in the mine at a time, these portable refuge stations will be moved as required to support the active mining areas. As Serbian regulations regarding refuge stations do not exist, Ontario regulations were followed for the design and positioning of refuge stations.

For the purpose of this study, the total maximum underground refuge capacity was estimated to be 70 personnel. A permanent refuge station (30 personnel capacity) in conjunction with the five portable refuge stations (8 personnel capacity each), equates to a total capacity of 70 personnel. The second permanent refuge station will not come online until later in the mine life.

The permanent refuge stations will double as lunchrooms and will be sized to provide 30 personnel with a minimum of 12 hours' supply of air. Since a central compressed air system will not be implemented, breathing-grade air and oxygen cylinders will be used. Local breathing-grade air cylinders will enable personnel to manually purge the refuge station air lock of contaminants, and will be used to maintain positive pressure in the refuge station. Local oxygen cylinders in conjunction with an oxygen delivery system will be used to maintain the correct level of oxygen in the refuge station . A carbon monoxide and carbon dioxide scrubber will be used to remove contaminants from the air. The scrubber and oxygen delivery system will include a back-up battery providing at least 12 hours of back-up power in the event of a power outage. A sodium chlorate oxygen candle will be used as a backup source of oxygen.

A breathing-grade air compressor located on surface in the mine rescue room will be used to fill the breathing air cylinders. The oxygen cylinders will be filled off site as required. An additional set of filled breathing air and oxygen cylinders will be maintained on site to ensure cylinders can be quickly swapped out after a test or underground emergency. Please note that the drawings developed do not reflect this setup, however costing was carried for the arrangement described.

In order to ensure comfort and safety, the permanent refuge stations will be equipped with both heating and air conditioning units. In order to ensure the HVAC equipment remains functional in the event of a power failure, the permanent refuge stations will be connected to the backup mine power circuit. Drinking water will be supplied using water jugs delivered from surface. Since the refuge station will double as a lunch room, it will include additional amenities such as a wash basin with hot water and a refrigerator.

Portable refuge stations will be self-contained units operating with no tie-ins to water or compressed air. Permanent power will be provided to the refuge stations for operation of the AC units, heaters and CO_2 scrubbers but the stations will also be equipped with batteries in the event of power loss. A total of five units, each sized to accommodate eight personnel will be employed and moved as required. Portable refuge stations are to be located so that the maximum length of time to walk to a refuge station is less than 30 minutes. All portable refuge stations will be installed in excavations equipped with power, a graded ballast floor, jersey barriers, and lighting.

16.5.2.5.3 Sanitation Cut-Out

Sanitation cut-outs will be located adjacent to the permanent refuge stations, crusher rooms, satellite maintenance shops, and select portable refuge stations. Sanitation bays will be equipped with a portable composting mine toilet, a sink with hot and cold running water using a local electric water heater, a graded concrete floor, overhead lighting, and a process water hose for clean-up requirements. The waste from the composting mine toilets must be removed and transported to surface on occasion.

16.5.2.5.4 Permanent Explosives and Detonator Magazine Bays

Permanent explosives and detonator magazines have been designed using select requirements from both Serbian and Canadian regulations. The designs consist of a mixture of both Serbian and Canadian requirements as opposed to strictly following Serbian regulations. A permanent underground magazine complex has been designed to contain two explosives magazines and two detonator magazines. Provisions have not been made to store explosives and detonators on surface.

As per Serbian regulations, dual magazines are required in order to hold a sufficient supply of explosives to sustain operations for a period of one week. In keeping with Serbian regulations, several other distinct design features have been incorporated into the magazine complex. A small excavation at the entrance to the magazine complex will house an attendant tasked with controlling access to the magazines. Lockable steel doors will be used to prevent unauthorized access to the explosives and detonator magazines. The magazine complex layout itself is designed in such a manner that three right angle turns must be navigated before either of the explosives or detonator bays can be accessed. Excavations opposite to each bay and access drift called ‘antechamber’s’ have been included as per Serbian regulations. Each magazine bay will be ventilated, with exhaust being directed to a return air raise.

As per Ontario regulations, the explosives magazines and detonator magazines are separated at least 20 m from each other, and at least 60 m from refuge stations, transformer vaults, electrical substations, and fuel storage areas. The magazines feature explosion-proof overhead lighting, intrinsically safe electrical systems, and adequate ventilation.

The explosives magazines will feature storage pads for bulk explosives totes, a monorail for unloading and loading of the totes, and a graded concrete floor. Bulk explosives will be manufactured off site, and delivered underground using totes.

The detonator magazines will feature timber storage shelves and a graded concrete floor.

16.5.2.5.5 Consumable Mining Material Storage

Additional re-mucks have been added in the vicinity of the entrance to each level for storage. Materials such as ground support bolts, screen, resin, grout, vent ducting, auxiliary fans, and mine services material will be stored in these re-mucks during development and production.

16.5.2.5.6 Secondary Egress Escapeway

Two separate means of egress to surface will be provided from all working areas of the mine. The primary means of egress will be through the main ramp and through the access decline. The secondary means egress will be through a series of escapeways located in the fresh air raise system and through the conveyor decline. Although the declines are in close proximity to each other, they will be physically separated and will have opposing directions of airflow.

This arrangement has been verified to be acceptable and in compliance with local regulations by SRK.

The level-to-level escapeways will be through the secondary fresh air raises connecting the various levels. Since there are multiple sections of fresh air raise connecting the levels together, there will be multiple sections of level-to-level escapeway. The first section will go from level -80 to -100, the second from levels -100 to -180, the third from levels -180 to -260, the fourth from levels -260 to -340 and the fifth from levels -340 to -400.

The escapeways will consist of steel ladders with regularly spaced platforms. The platforms will be the full width of the raise and will be anchored to the raise wall. The ladders and platforms will be equipped with handrails, safety cages, and swing gates. Bollards on each access level have been added for safety, however may not be required as most levels will have a ventilation bulkhead with a fan and access door prior to the escapeway.

16.5.2.6 Underground Mobile Equipment

The purchase of a permanent mining equipment fleet will be required for the mine production activities performed by the owner.

Mobile equipment costs were developed from estimated fleet requirements and vendor budgetary quotations.

Equipment life-cycle operating hours were based on manufacturer recommendations and SRK project experience. The recommended life-cycle operating hours were used to calculate equipment replacement requirements.

Table 16.9: Underground Mine Mobile Equipment

Equipment Description
Drilling
Drill Jumbo, two-boom
Rockbolter
Cablebolter
Production Drill
Aries ITH Drill & Reamer
Development & Production
Emulsion Loader, Devt.
Shotcrete Sprayer
Transmixer
Emulsion Loader, Prod.
Mobile Rockbreaker

Equipment Description
Blockholer
Loading & Hauling
LHD, 5.4-m^3, 10-t
LHD, 7.0-m^3, 14-t
LHD, 8.6-m^3, 17-t
LHD, 3.7-m^3, 6.7-t
Haulage Truck, 30-t
Haulage Truck, 40-t
Mine Maintenance
Grader
Scissor Lift
Scissor Lift Attachments
Boom Truck
Flat Deck Truck
Toyota Flat Deck Truck
Mechanics Truck
Fuel/Lube Truck
Water Sprayer
Mine Services
Personnel Carrier, 16 per.
Personnel Carrier, 8 per.
Supervisor Vehicle
Forklift/Telehandler
Explosives Truck
Septic Vacuum Truck
Cassette Prime Mover
Cassettes Attachments

16.5.2.7 Underground Mine Electrical Power Distribution

The major underground electrical distribution facilities required to support the mine operation will consist of the following components:

- Underground switching substation EL+20 – 2600-ER-102.
- Underground switching substation EL-120 – 2600-ER-101.

- Underground switching substation EL-280 – 2600-ER-103.
- Underground switching substation EL-440 – 2600-ER-104.
- Underground portable mine power center EL+220 – 2600-PMPC-103 & 2600-PMPC-113.
- Underground portable mine power center EL+20 – 2600-PMPC-102 & 2600-PMPC-112.
- Underground portable mine power center EL-120 – 2600-PMPC-101.
- Underground portable mine power center EL-160 – 2600-PMPC-104.
- Underground portable mine power center EL-280 – 2600-PMPC-105 & 2600-PMPC-106.
- Underground portable mine power center EL-360 – 2600-PMPC-108.
- Underground portable mine power center EL-400 – 2600-PMPC-107.

The underground power distribution follows a dual feed redundant power supply strategy. Each main feed and all main distribution equipment are sized to accommodate the entire underground mine loads. The distribution voltage is defined as 35 kV as selected by Nevsun. The switching substations complete with secondary selective systems will consist of two single-ended 35 kV switchgears with normally open tie breakers. These switching substations will be located on each major mine level. A portable mine power center (PMPC), which consists of a 35 kV fusible load interrupter switch, a step-down dry-type transformer and a Motor Control Center, will be located at each major infrastructure installation. The distribution voltage will be reduced to the utilization voltage required by the end user equipment.

The main underground power supply of 35 kV will be provided from the surface overhead line system, extending to the surface return air raise and fresh air raise substations identified as 2600-ER-005/004. For each feed, a 35 kV mine power feeder cable will be connected to the load side of a 3-phase pole-mounted-fusible-disconnect-switch and routed to the ventilation raise. The 35 kV mine power feeder will transition to a 35 kV shaft cable via a junction box at the collar of the raise. The shaft cable will be anchored at the collar and dangle down through the raise. The shaft cable in the fresh air ventilation raise is about 400 m long, while the cable in the return air ventilation raise is about 500 m long. Each shaft cable will then re-transition back to a mine power feeder cable through another junction box located at the bottom of the raise. The cable will extend from the junction box to the first switching substation identified as 2600-ER-101 at EL-120.

The vertical shaft cable that is used in the ventilation raises will be used for all vertical cable installations including borehole installations. Due to ground support not being installed in the ventilation raises, the traditional installation technique of utilizing shaft cable supports is not possible, thereby requiring the need for a single-point-suspension installation. As a result, the cable will be supported solely at the collar elevation with the remainder of the cable left free-hanging down the raise. According to technical literature on Nexans Powerline Shaft Cable, the free hanging depth of a shaft cable is a maximum of 1,000 m with a security factor of 3 or a maximum of 600 m with a security factor of 5.

While using the shaft cable significantly reduces the overall mine cable length, the self-supporting cable is approximately 50% more expensive than the standard mine power feeder cable used for lateral installations.

The basic design concept to size electrical equipment and assign respective loads was adapted to minimize the damage caused by potential arc flash incidents. As a result, the 400 V Motor Control Center on the PMPC's has been selected with a bus rating of 1,600 A and 50 kA symmetrical short circuit capacity. Furthermore, the upstream secondary distribution transformers were selected with a 1 MVA capacity in order to limit the transformer fault contribution during an electrical fault downstream.

All major Electrical equipment and cables will be standardized in order to simplify installations and maintenance while reducing spare requirements.

All major Electrical equipment will also be interchangeable. The switchgear selected for all switching substations is the S&C Vista 35 kV underground distribution SF6 6-way switchgear (IEC standard), where each way will be supplied with a 630 A circuit breaker. The selected S&C Vista unit is a more compact unit, requiring less footprint area when compared to other conventional 35 kV switchgears. Two types of PMPC's will be used; one with a capacity of 1 MVA and the other with a capacity of 750 kVA. As for the 35 kV main run cables, two cable sizes will be used, a 120 mm^2 mine power feeder cable for all PMPC's and a 185 mm^2 mine power feeder cable/mine shaft cable for the 35 kV distribution underground.

For mine equipment which have their electrical motors geographically located on surface (e.g. overland conveyor 2402-CV-01, decline conveyor 2402-CV-02, fresh air fans 2201-FAA-001/002 and return air fans 2202-FAA-001/002 etc.), these electrical loads will be fed from the surface substation and will not be included in this section.

It is assumed that there will be four active mine levels at any given-time during mine development and operation. Each mine level will have three headings and three dumps. For the ventilation electrical load requirements, it is estimated that on each active level one large auxiliary fan (rated at 50 kW) will be used for the main levels, and six smaller auxiliary fans (each rated at 30 kW) will be used in headings and dumps. It is assumed that the auxiliary fans will be operated continuously on all active levels. The fans will be powered from the nearest PMPC. The Electrical loads attributed to these fans are insignificant in comparison to the spare capacity of each PMPC transformer.

16.5.2.7.1 Electrical Facilities on EL-120

The main 35 kV underground power feeders will run in the messenger cable hanger system for approximately 1 km and then connect to the main switchgears 2600-SWG-101 and 102 located in the main switching substation 2600-ER-101 on EL-120.

One feeder from 2600-SWG-101 connects to a PMPC, 2600-PMPC-101, dedicated to the Dewatering Pump Station, 2701-PPK-003 on EL-120, 300 meters away from the Switching Substation 2600-ER-101. The dewatering pump PMPC consists of a 35 kV load interrupter switch, a 1 MVA dry type transformer, and a 400 V MCC. The low voltage MCC feeds four dewatering pumps at the pump station on this level, each rated 380 V, 150 kW. Each pump motor will utilize a soft starter to limit the inrush current. It has been assumed that the pump motors will be on continuous duty and run at full capacity, without standby, during peak spring dewatering periods. The PMPC's and the soft starters will be located inside an Electrical cut-out in close proximity to the pump skid. The general maintenance and cut-out service equipment will also be fed from the same MCC.

One feeder from switchgear 2600-SWG-102 will connect to PMPC 2600-PMPC-104, which is dedicated to the Conveyor Station 2402-CV-03 on EL-160 located 600 m away from the Switching Substation 2600-ER-101 through the decline. The conveyor PMPC consists of a 35 kV load interrupter switch, a 1 MVA dry type transformer, and a 400 V MCC. The low voltage MCC will feed the conveyor motor. The motor will be rated at 380 V, 400 kW and controlled by a VFD. All conveyor motors are assumed to be continuous loads and run as required. The PMPC and the VFD will be located inside an Electrical cut-out located in close proximity to the head of conveyor CV-03 where the motor is located. The general maintenance and cut-out service equipment will also be fed from the same MCC.

Two feeders, one from each switchgear, 2600-SWG-101 and 102, will feed the switching substation on EL+20. Another two feeders from the same two switchgears, 2600-SWG-101 and 102, will feed the switching substation on EL-280.

Two normally open circuit breakers, one from each switchgear 2600-SWG-101 and 102, will be dedicated as tie breakers connecting both switchgears in 2600-ER-101 for switching purposes. These two ties will be closed to maintain downstream operation when one upstream power supply is shut down due to an equipment failure or as a requirement from operation and maintenance.

16.5.2.7.2 Electrical Facilities on EL+20

The 35 kV mine power feeder cables from 2600-ER-101 will run in the messenger cable support system for approximately 900 m along the decline and connect to the switching switchgears 2600-SWG-103 and 104 located in the switching substation 2600-ER-102 at EL+20.

One feeder from switchgear 2600-SWG-104 will connect to PMPC 2600-PMPC-102, which is dedicated to two dewatering pumps in the Dewatering Pump Station, 2701-PPK-002 on EL+20 located 100 m away from the Switching Substation 2600-ER-102 along the decline. Another PMPC, 2600-PMPC-112, dedicated to the other two dewatering pumps in the same Dewatering Pump Station 2701-PPK-002 on EL+20, is fed via a 35 kV splitter junction box connected with PMPC-102. Each dewatering pump PMPC will be comprised of a 35 kV load interrupter switch, a 750 kVA dry type transformer, and a 400 V MCC. The low voltage MCCs will feed four dewatering pumps at the pump station on this level, each rated at 380 V, 225 kW. Each pump motor will utilize a soft starter to limit the inrush current. It has been assumed that the pump motors will be on continuous duty and run at full capacity, without standby, during peak spring dewatering periods. The PMPCs and soft starters will be located inside an Electrical cut-out in close proximity to the pump skid. The general maintenance and cut-out service equipment will also fed from the same MCC.

One feeder from switchgear 2600-SWG-103 will connect to PMPC 2600-PMPC-103 located on EL+220, 1,400 m away from Switching Substation 2600-ER-102 along the decline.

Two normally open circuit breakers, one from each switchgear 2600-SWG-101 and 102 will be dedicated as the tie breakers connecting both switchgears in 2600-ER-101 for switching purposes. These two ties will be closed to maintain downstream operation when one upstream power supply is shut down due to an equipment failure or as a requirement from operation and maintenance.

16.5.2.7.3 Electrical Facilities on EL+220

The feeder from 2600-SWG-103 will connect to PMPC 2600-PMPC-103, dedicated to two dewatering pumps in the Dewatering Pump Station, 2701-PPK-002 on EL+220. Another PMPC, 2600-PMPC-113, dedicated to the other two dewatering pumps in the Dewatering Pump Station 2701-PPK-001 on EL+220, is fed via a 35 kV splitter junction box connected with PMPC-103. Each dewatering pump PMPC is comprised of a 35 kV load interrupter switch, a 750 kVA dry type transformer, and a 400 V MCC. The low voltage MCCs will feed four dewatering pumps at the pump station on this level, each rated at 380 V, 225 kW. Each pump motor will utilize a soft starter to limit the inrush current. It has been assumed that the pump motors will be on continuous duty and run at full capacity, without standby, during peak spring dewatering periods. The PMPC’s and soft starters will be located inside an Electrical cut-out in close proximity to the pump skid. The general maintenance and cut-out service equipment will also fed from the same MCC.

16.5.2.7.4 Electrical Facilities on EL-280

The two 35 kV mine power feeder cables from 2600-ER-101 will run in the messenger cable support system for approximately 600 m along the travel way to the return air raise on EL-180. The 35 kV mine power feeder will transition to a 35 kV shaft cable via a junction box at the collar of the raise. The shaft cable will be anchored at the collar and dangle down through the raise approximately 100 meters. The shaft cable will then re-transition back to a mine power feeder cable through another junction box located at the bottom of the raise on EL-260. From the junction box, the cable will follow the messenger cable system along the conveyor decline approximately 150 m to the switching substation 2600-ER-103 at EL-280. From this location, the cables will connect to the switching switchgears 2600-SWG-105 and 106.

One feeder from 2600-SWG-105 will connect to PMPC 2600-PMPC-105 which is dedicated to the Dewatering Pump Station 2701-PPK-004 on EL-280 and also located 200 m away from the Switching Substation 2600-ER-103. Each dewatering pump PMPC will be comprised of a 35 kV load interrupter switch, a 750 kVA dry type transformer, and a 400 V MCC. The low voltage MCCs will feed four dewatering pumps at the pump station on this level, each rated at 380 V, 225 kW. Each pump motor will utilize a soft starter to limit the inrush current. It has been assumed that the pump motors will be on continuous duty and run at full capacity, without standby, during peak spring dewatering periods. The PMPC's and soft starters will be located inside an Electrical cut-out in close proximity to the pump skid. The general maintenance and cut-out service equipment will also fed from the same MCC.

One feeder from 2600-SWG-106 will connect to PMPC 2600-PMPC-106 which is dedicated to the Conveyor Station 2402-CV-04/05 & CD-01 on EL-280 and located 200 m away from the Switching Substation 2600-ER-103 through the decline. The conveyor PMPC will be comprised of a 35 kV load interrupter switch, a 1 MVA dry type transformer, and a 400 V MCC. The low voltage MCC will feed the conveyor motors which are all rated at 380 V. The power rating for each conveyor motor will be as follows: 400 kW for CV-05, 75 kW for CV-04 and 75 kW for CD-01. All motors will be controlled by VFDs. It is also assumed that all conveyor motors will operate continuously. The PMPC and the VFD will be located inside an Electrical cut-out close to the head of the conveyors where the motors are located. The general maintenance and cut-out service equipment will also be fed from the same MCC.

Two feeders, one from each switchgear 2600-SWG-105 and 106, will feed the switching substation on EL-440.

Two normally open circuit breakers, one from each switchgear 2600-SWG-105 and 106 will be dedicated as tie breakers connecting both switchgears in 2600-ER-103 for switching purposes. These two ties will be closed to maintain downstream operation when one upstream power supply is shut down due to an equipment failure or as a requirement from operation and maintenance.

16.5.2.7.5 Electrical Facilities on EL-440

The two 35 kV mine power feeder cables from 2600-ER-103 will run in the messenger cable support system for approximately 50 m along the decline to the borehole on EL-280. The

35 kV mine power feeder will transition to a 35 kV shaft cable via a junction box at the collar of the borehole. The shaft cable will be anchored at the collar and dangle down through the borehole for approximately 200 meters. The shaft cable will then re-transition back to a mine power feeder cable through another junction box located at the bottom of the borehole on EL-440. From the junction box, the cable will follow the messenger cable system along the conveyor decline approximately 50 m to the switching substation 2600-ER-104 at EL-440. From this location, the cables will connect to the switching switchgears 2600-SWG-107 and 108.

One feeder from 2600-SWG-107 will connect to PMPC 2600-PMPC-107 which is dedicated to the Dewatering Pump Station 2701-PPK-005 and Discharge Conveyor CD-02 located on EL-400. The PMPC will be located approximately 200 m away from the Switching Substation 2600-ER-104. The PMPC will be comprised of a 35 kV load interrupter switch, a 750 kVA dry type transformer, and a 400 V MCC. The low voltage MCCs will feed four dewatering pumps at the pump station on this level, each rated at 380 V, 115 kW. Each pump motor will utilize a soft starter to limit the inrush current. It has been assumed that the pump motors will be on continuous duty and run at full capacity, without standby, during peak spring dewatering periods. The discharge conveyor CD-02 will feature a motor rated at 380 V, 75 kW and controlled by a VFD. It is also assumed that all conveyor motors will operate continuously. The PMPC, the VFD and the soft starters will be located inside an Electrical cut-out close to these facilities. The general maintenance and cut-out service equipment will also fed from the same MCC.

One feeder from 2600-SWG-108 connects to PMPC 2600-PMPC-108 which is dedicated to Conveyor 2402-CV-06/07 on EL-360 and located 500 m away from the Switching Substation 2600-ER-104 along the conveyor decline. The conveyor PMPC will be comprised of a 35 kV load interrupter switch, a 1 MVA dry type transformer, and a 400 V MCC. The low voltage MCC will feed the conveyor motors which are all rated at 380 V. The power rating for each conveyor motor will be as follows: 400 kW for CV-07 and 75 kW for CV-06. All motors are controlled by VFDs. It is also assumed that all conveyor motors will operate continuously. The PMPC and the VFDs will be located inside an Electrical cut-out close to the head of conveyors where the motors are located. The general maintenance and cut-out service equipment is also fed from the same MCC.

Two normally open circuit breakers, one from each switchgear 2600-SWG-107 and 108 will be dedicated as tie breakers connecting both switchgears in 2600-ER-104 for switching purposes. These two ties will be closed to maintain downstream operation when one upstream power supply is shut down due to an equipment failure or as a requirement from operation and maintenance.

16.5.2.8 Controls and Automation

16.5.2.8.1 Process Control System

The automation of mine equipment and systems will reside in programmable logic controllers (PLCs) and will interface with the operator through human-machine interfaces (HMI).

The communications between controllers and the HMI will be established via a combination of Fiber/Copper cabling and associated hardware.

16.5.2.8.2 Control Room and Stations

The Timok mine control stations will consist of:

- A central control room (CCR) located at the Timok mine portal, which will be used for underground operation, control and dispatch of equipment.
- Local field control stations including rock breaker control booths, ore sizer control rooms and other field HMI booths.
- The control room should be equipped with a communications system for communications with personnel U/G.

Each process control system (PCS) station will be completely self-contained with all functional capabilities such that individual stations will continue to operate should other stations fail.

The mine equipment connected to the process control system will be normally controlled from the CCR. A minimum of three HMI units will be located in the CCR.

The rock breaker stations will be controlled remotely from the Timok mine CCR. Control stations will be designed to allow simultaneous operation of the two rock breaker stations and will be designed as the primary means of controlling the rock breaker stations. The control stations will be adjacent to the CCR.

A field rock breaker control station will be installed in each ore/waste dump area. It will allow full control of the rock breaker station and be located close to the rock breaker control station such that observation of outside controlled equipment.

The field HMI stations will be strategically located based on logical mine process areas. Site standard will be used for booth design. Field HMI stations will be fitted with an industrial keyboard, industrial mouse or trackball, flat screen monitor, and interface to PCS.

16.5.2.8.3 Network General Arrangement

The Timok underground mine will be connected via a private network to the mill site main control room and main office.. A fibre optic cable run will be deployed from portal entrance underground along the conveyor decline throughout the mine, and return to surface along the travelway decline, creating a ring topology for redundancy.

Fibre interface panels (FIPs) will be installed at key locations underground for the ore/waste handling and dewatering. FIPs will also be located at the portal and surface fans.

A fibre optic cable will follow the internal haulage ramp to service each mine level and support wireless network. Redundancy for through the escapeway must be reviewed.

16.5.2.8.4 Communication and Mines Systems

16.5.2.8.4.1 Wired Network Infrastructure

The Timok mine site will have standalone network infrastructures. The wired network infrastructure will be divided in the following networks:

- **Process control local area network (LAN)** – The process network will host PLC intercommunication, remote I/O and HMI communications.
- **Business LAN** – The business network will host all other services to support operations.

All main process control LAN nodes will have a physically redundant link to another node where achievable. The process network and business network will reside on two physically segregated LANs. The business LAN will interface to the process network via a firewall. Critical underground network infrastructure (i.e., refuge stations) shall have redundant communications paths and shall be powered via a UPS.

VLANs (virtual LANs) will be used to segregate traffic within each LAN. These VLAN’s can be used for specific functions, such as blasting, micro-seismic, tracking, and video. This will be further defined in detailed engineering.

16.5.2.8.4.2 Wireless Infrastructure

The wireless Wi-Fi infrastructure will provide service for the following applications:

- **Business LAN and voice/video LAN –** Density of access points will be defined based on traffic such that areas of high traffic shall have higher density of service. Hotspots will be installed in selected areas such as refuge stations, underground control rooms, or garage. The access points will support several connections in line with the maximum plausible number of concurrent connections.
- **Mobile equipment connectivity (LHD remote control) for which a dedicated high-fidelity wireless infrastructure will be designed** – Access point layout, number of concurrent connection and bandwidth requirements will be defined based on selected LHD remote control platform. The Wi-Fi design will include sufficient bandwidth for video and data, multiple simultaneous LHD connection and seamless access point handover.

The wireless protocol will be Wi-Fi (IEEE 802.11) and will be selected to be compatible with all underground wireless applications.

16.5.2.8.5 VHF Infrastructure

A wireless VHF infrastructure (leaky feeder) will be designed to support central blasting and underground radio communication.

16.5.2.8.6 Voice Communication

As discussed above, the radio system will use the leaky feeder infrastructure. Two-way radio communication system will provide multi-channel radio communication. The radio communication system is to be compatible with existing mine site radio system.

Communications between surface and underground shall be done via VoIP phones. The digital desk sets will be powered by Power over Ethernet (PoE) and communicate over the network. Handheld or fixed IP phones will be available in refuge stations, HMI rooms, offices and underground shops and electrical substations.

A two-wire mine “hardened” analogue phone system will be deployed to provide backup communication in the event of an emergency or loss of the network. The phones will be installed in relatively the same location as IP phones, including but not be limited to refuge stations and substations.

Communication systems redundancy will be as shown in Table 16.10.

Table 16.10: Underground Communication Fault Tolerance

Component	Level of Fault Tolerance
Telephone	Primary – VoIP
Telephone	Secondary – two-wire analog phone
Radio	Leaky feeder

16.5.2.8.7 Site Access Control

A vendor system for site access control will be used. The vendor will be identified in the feasibility study and used in detailed design.

16.5.2.8.8 Underground Emergency Warning System

An ethyl mercaptan gas system will be used to alarm workers in the event of an emergency. This emergency system will be delivered via the ventilation system.

16.5.2.8.9 Blasting System

Development blasting will be conducted electrically via the leaky feeder system or via Wi-Fi on a dedicated VLAN. A secondary blasting system will support production blasting using smart detonators. The selected system will provide feedback on the condition of the blast and will be designed to meet the requirements of the mining method in use.

16.5.2.8.10 Micro-seismic System

The micro-seismic system at Timok will be designed to monitor the ground conditions. The system will utilize the communication infrastructure on a dedicated VLAN. The system will be a vendor-provided solution for the Timok mine. Design of the micro-seismic system is to include recorder and listening devices and time-stamping hardware.

16.5.2.8.11 Personnel/Equipment Tracking

Underground personnel and equipment tracking will used. This system will tie into the ventilation system for ventilation on demand. The vendor will be identified and selected in the feasibility study and used in detailed design.

16.5.2.8.12 Closed-Circuit Television System

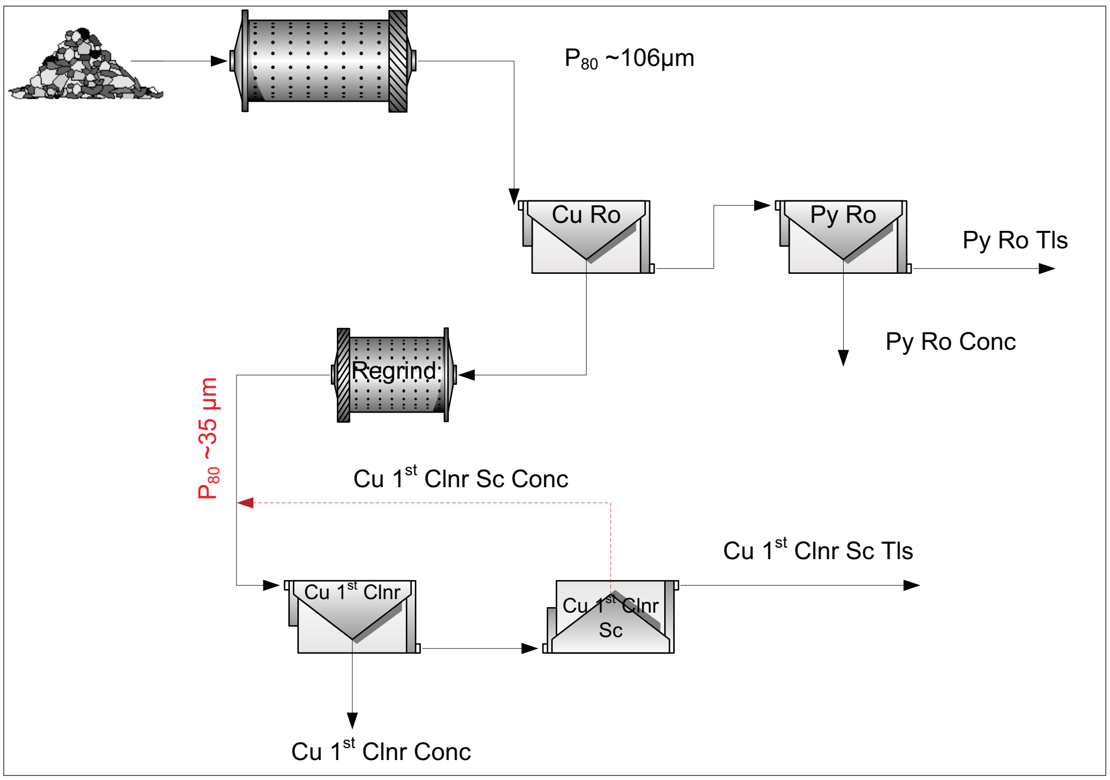
A closed-circuit television (CCTV) system will be provided to visually monitor the various process areas. The basis of the design will be cameras dedicated to individual mine areas and key equipment, which may be viewed from the main Timok mine CCR or field HMI stations where images from individual cameras will be able to be recorded. Video cameras will be connected to the business LAN through a dedicated VLAN. PoE-IP cameras will be used. The vendor will be identified and selected in the feasibility study and used in detailed design.

17. Recovery Methods

17.1 Summary

The copper mineralogy of the ore consists primarily of covellite with lesser enargite. The flowsheet was initially designed with the ability to generate two concentrates: a low (<0.5% As) arsenic content and elevated (>0.5% As) arsenic content. This same general flowsheet, utilized in a more simplified manner, has the capacity to produce a single bulk concentrate. Single bulk concentrate production is the basis of this study.

A schematic representation of the single bulk concentrate flowsheet is shown Figure 17.1, note that in the design for the PFS (PFS, 2018) the pyrite rougher is removed and all of the pyrite goes to the bulk tailings.



Source: SGS 2017

Figure 17.1: Bulk Concentrate Process

17.2 Upper Zone Flowsheet Selection

The process plant design is based on a combination of metallurgical test work, mine production plan and in-house information. Where necessary, benchmarking has been used to support the design.

The Timok process plant includes the following unit processes and associated facilities:

- Primary crushing located underground.
- Overland conveying of crushed process feed.
- Coarse feed storage bins and reclaim.
- SAB grinding circuit.
- Copper flotation comprising rougher flotation, concentrate regrind and two stage cleaning.
- Copper concentrate thickening and filtration.
- Copper concentrate load out and storage for each concentrate.
- Tailings storage and water reclaim.
- Effluent treatment.
- Reagents storage and distribution (including lime slaking, flotation reagents, water treatment and flocculant).
- Grinding media storage and addition.
- Water services (including fresh water, fire water, gland water, cooling water and process water).
- Potable water treatment and distribution.
- Air services (including high pressure air and low-pressure process air).
- Plant control rooms.
- Possible future equipment for flotation of a separate high arsenic complex copper concentrate.

17.3 Upper Zone Process Design

The following sections outline the basis of process design for the overall plant including key criteria, operating schedule and availability, and throughput.

17.3.1 Design Criteria

Key design criteria used in the plant design are summarized in Table 17.1.

Table 17.1: Key Design Criteria

Parameter	Units	Value
Plant capacity, 2021 to 2023	tpa	3,250,000
Flotation feed size, P80	µm	75
Rougher feed density, nominal	% solids (w/w)	33
Cleaner feed density, nominal	% solids (w/w)	25
Concentrate thickeners underflow density	% solids (w/w)	60
Concentrate filter cake moistures	% solids (w/w)	10

17.3.2 Operating Schedule and Availability

The plant operating schedule and availability is based on three 8-hour shifts per day for 365 days per year (i.e. total available hours per year is 8,760). Operating availability criteria (as a percentage of total available hours) used for plant design are summarized in Table 17.2.

Table 17.2: Plant Operating Availability Summary

Description	Units	Value
Crusher operating availability	%	75.0
Grinding and flotation operating availability	%	92.0
Concentrate filter operating availability	%	80.0
Overall processing and tailings systems	%	92.0

17.4 Upper Zone Process Plant Description

The process plant is designed to treat nominally 8,900 tonnes per day (equivalent to 3.25 million tonnes per year) and produce concentrates. The basis of design and description for major plant equipment and unit processes is summarized in the following sections.

17.4.1 Primary Crushed Ore Delivery and Storage

The primary crusher is located underground. Process feed is crushed to minus 150 mm in a jaw crusher which is fed by a vibrating grizzly feeder. The material is then conveyed via decline conveyor to the mill.

The crushed process feed is transferred to the process plant via an overland conveyor. The overland conveyor discharges onto the top of the coarse feed storage bins via a shuttle conveyor. Discharge can be to either of the two coarse feed storage bins or via a chute to a stockpile. Each coarse feed bin has 4,500 t live capacity and the two bins combined provide 24 hours storage capacity for the grinding circuit.

17.4.2 Primary Grinding Circuit

The grinding line consists of a single variable speed SAG mill, followed by a single ball mill operating in closed circuit with a cyclone cluster. The product from the grinding circuit (cyclone overflow) has a typical size of 80% passing 75 µm. SAG mill pebbles are recycled to the SAG mill feed conveyor.

The SAG mill feed conveyor discharges mineralized material, along with pebble recycle and grinding media, into the feed chute of the SAG mill together with mill feed dilution water and lime slurry. The SAG mill is fitted with discharge grates to retain grinding media and larger pebbles while allowing smaller particles to discharge from the mill. SAG mill grinding media is also added to the SAG mill feed chute with a 1-t kibble with a false bottom.

The SAG mill trommel undersize gravitates to the primary cyclone feed hopper where it is combined with the discharge from the ball mill. The slurry is transported to a single cyclone cluster using two variable-speed cyclone feed pumps (duty and stand-by).

Dilution process water is added to the cyclone feed hopper before the slurry is pumped to the cyclone cluster for classification. Lime slurry is added to the hopper in order to increase pH to approximately 10. Coarse particles report to the cyclone underflow and are directed to the ball mill feed chute via a boil box. The cyclone overflow stream gravitates to the vibrating trash screen via a cross-stream sampler.

SAG mill balls are added via a ball addition system and bunker adjacent to the SAG mill feed conveyor. A separate ball storage bunker is provided for the ball mill. The ball mill has a dedicated ball charging system.

A SAG mill feed chute removal system and a ball mill feed chute removal system are used to service the mills. A liner handler is provided for each mill.

There is provision to install a vibrating screen at the SAG mill discharge in case pebble crushing is required in the future if process feed hardness increases sufficiently.

17.4.3 Copper Flotation and Regrind

The copper flotation circuit consists of a vibrating trash screen, conditioner tank, single train of rougher flotation cells, rougher concentrate regrind and two stages of cleaner flotation.

For the initial two years of production rougher flotation alone produces an acceptable concentrate grade because of the high grade feed. Then a first stage cleaner is required to be installed and later a second stage cleaner as the feed grade declines. The description contained here is for the final complete flotation plant after all stages have been installed.

Cyclone overflow gravitates over a vibrating trash screen with the undersize reporting to the conditioning tank. Slurry pH is adjusted to the required value (pH 10.0) with slaked lime at the conditioner tank with the option for pH adjustment at each rougher cell. MIBC (methyl isobutyl carbinol) frother and copper collector Aerofloat 211 is added to each feed stream.

Flotation feed reports to a single rougher flotation bank consisting of six forced air mechanical flotation tank cells.

The rougher flotation cells, except for the first two years of operation, produce a low-grade copper concentrate that requires further liberation and upgrading to remove arsenic. Copper rougher concentrate is pumped to the regrind mill.

The copper rougher tailings stream is pumped to the tailings storage after passing through a cross-cut sampler.

Copper rougher concentrate is reground in a vertical ball mill operating in closed circuit with small diameter cyclones. Copper rougher concentrate is reground to 80% passing 25 μm. Regrind cyclone overflow gravitates to the copper cleaner flotation circuit. Regrind cyclone underflow gravitates to the regrind mill.

The copper cleaner circuit consists of two stages of counter-current cleaning and one bank of copper cleaner scavenger flotation cells. There is provision for a bank of copper-arsenic retreatment flotation cells to produce a high arsenic complex copper concentrate and low arsenic clean copper concentrate in the future.

The first copper cleaner stage consists of seven tank cells. Cleaner feed is conditioned with slaked lime to adjust pH to 11.0. Collector and frother are added to the first tank cell in the first cleaner stage and only frother and lime are added to the downstream cleaner stages.

Concentrate recovered from the first copper cleaner bank is pumped to the second copper cleaner bank for further upgrading. The tailings are pumped to the copper cleaner scavenger flotation cells.

The copper cleaner scavenger flotation cells recover a low-grade concentrate that is pumped to the copper conditioning tank or in the future to copper-arsenic retreatment flotation cells if installed. The cleaner scavenger tailings stream is pumped to the copper rougher tailings tank after passing through a cross-cut sampler.

The second copper cleaner flotation stage concentrate is pumped and split to two copper concentrate thickener feed tanks for dewatering. The tailings from the second copper cleaner cells are pumped to the head of first copper cleaner flotation cells.

There is provision for copper-arsenic retreatment flotation cells in the future, if a two-concentrate approach is determined to be better in future studies. Complex concentrate would be pumped to one of the copper concentrate thickener feed tanks for dewatering. The tailings from the copper-arsenic retreatment flotation cells would be pumped to the head of copper cleaner scavenger flotation cells.

An on-stream analyzer is used to monitor copper and arsenic levels in the feed, major concentrate and tailings streams to allow operators to optimize reagent additions and flotation performance.

17.4.4 Copper Concentrate Thickening and Filtration

Copper concentrates are dewatered using two thickeners and filters. Each copper concentrate thickening and filtration circuit consists of a single 15-m diameter high rate thickener and one Outotec Larox PF60M132 pressure filter. Should a separate complex concentrate be produced in the future, one circuit would be dedicated to the complex concentrate and one for the conventional concentrate.

The concentrate from the second copper cleaner flotation cells is pumped and split into two copper concentrate thickeners.

Flocculant is added to the thickener feed streams to enhance settling. The thickener overflows report to the copper rougher tailings pumpbox. Copper concentrate solids settle and are collected at the underflow at a density of 60% solids. The thickener underflow streams are pumped to two dedicated agitated storage tanks, one tank per filter, by centrifugal pumps (one operating, one standby per thickener).

The storage tanks provide 24 hours surge capacity allowing filter maintenance to be conducted without affecting process plant throughput. The filter feed is pumped to a pressure filter that produces a filter cake of 10% moisture. The copper filter cake is discharged by gravity to a storage stockpile. Filtrate, cloth wash and flushing water is discharged to the filtration area sump pump which returns it to the copper concentrate thickener.

During the two highest quarters of production in the first two years of operation, the concentrate pressure filters must operate at the design availability for this time. Concentrate filtration tests during future studies will provide better definition of filtration equipment requirements.

17.4.5 *Effluent Treatment*

Although submergence of pyrite in the tailings storage is expected to prevent acidification, water from the tailings impoundment pond may become acidic over time. Acidic water from this pond along with contaminated contact water will be put through an effluent treatment plant. The primary purpose of the effluent treatment plan will be: a) treatment of mine water for direct discharge, should there be a surplus and/or b) gradual reduction in stored water for subsequent discharge.

The effluent treatment plant includes a high-density sludge process using slaked lime. After four reaction tanks, treated effluent gravitates to a clarifier. Clarifier underflow is recycled to the reaction tanks. Clarifier overflow is passed through an Ultrafilter. If discharge to the environment is required, a reverse osmosis module will be employed to remove residual dissolved salts. A pond will be constructed between the WTP and the RO plant to provide some retention time between then for equilibration.

17.4.6 *Reagents and Consumables*

Major process reagents are received and stored on site. Dedicated mixing, storage and dosing facilities are provided for each reagent.

17.4.6.1 *Lime*

Lime is used to increase slurry pH and subsequently depress pyrite in copper flotation. It is also used in the effluent treatment circuit.

The quick lime slaking systems consist of a vendor-supplied proprietary slaking system comprising storage silo, feeders, vertical slaking mill and a storage tank. Quick lime is delivered to site in 30-t bulk road tankers and unloaded pneumatically to a 200-t storage silo at the lime slaking plant.

A lime circulating pump and pressurized ring main is used to deliver lime slurry to plant dosing points. Pinch valves are used to control lime addition at each dosing point.

17.4.6.2 *Copper Collector*

The copper collector is a Cytec product, Aerofloat 211, which is an alkyl dithiophosphate. The reagent is delivered as a liquid in 1.0-m^3 plastic totes. The totes are transferred to a storage tank. The storage tank is connected to a fixed manifold and collector is pumped to addition points via dedicated dosing pumps.

17.4.6.3 *Potassium Amyl Xanthate*

Originally specified as the pyrite collector but potentially required in the second copper cleaner stage. PAX will come in 750-kg bulk bags and will be dissolved in water to a concentrate of 20% w/v. The PAX solution is transferred to a storage tank and then metered to pyrite flotation with a dedicated diaphragm type pump.

17.4.6.4 *Frother*

MIBC is used to provide a stable froth in the copper flotation cells. The frother is delivered as a liquid in 1.0-m^3 plastic totes. The totes are transferred to a storage tank. The storage tank is connected to a fixed manifold and frother is pumped to addition points via dedicated dosing pumps.

17.4.6.5 *Flocculant*

Flocculant is used as a settling aid in the two concentrate thickeners and flotation tailings thickener. The flocculant mixing systems consist of a vendor-supplied proprietary mixing system comprising storage bin, screw feeder, blower, auto jet wet mixer, mixing tank, storage tank and dosing pumps.

Flocculant is delivered as a dry powder in 25-kg bags and is transferred to a hopper and blown into the wetting head to produce a 0.25% w/v flocculant solution. Flocculant is mixed for 30 minutes in an agitated tank and transferred to a storage tank. Dedicated dosing pumps deliver flocculant to the respective thickeners.

17.4.7 *Water Services*

17.4.7.1 *Process Service Water*

Two separate raw water tanks are provided, one serves as a raw water tank and the second as a fire water tank as required by Serbian regulations.

The fire water system comprises an electric fire water pump and a jockey pump to maintain pressure in the fire water line. A diesel-powered fire water pump provides back-up in the event of power loss. Fire water is reticulated to fire hydrants and hose reels via a dedicated fire water main.

Process service water is also used to supply the following services:

1. PAX and copper sulphate preparation.
2. Gland water.
3. Make-up water for the plant process water requirements.

Water is aged in the bulk tailings pond to allow thiosalts to decompose. Process water is reclaimed water from the bulk tailings ponds and treated water from the effluent treatment plant.

17.4.7.2 Potable Water

Potable water is distributed for general use in the plant, the camp and the plant safety shower system from the potable water storage tank.

17.4.8 Air Services

Low pressure air for the copper flotation circuit is supplied by two blowers in duty/standby arrangement.

Two separate air compressors (one duty and one standby) provide high pressure air for plant instruments and general service points. Compressed air is dried and filtered to instrument air quality prior to storage in the instrument air receiver and subsequent distribution.

18. Project Infrastructure

18.1 Surface Infrastructure

18.1.1 Overview

Surface infrastructure includes both site infrastructure, (within the property boundaries of the Timok site), and off-site infrastructure, (facilities outside of the property boundaries).

Site infrastructure consists of roads, drainage, security fencing, non-process buildings, electrical power supply and distribution (see Section 18.4), piped utilities, water treatment, and a shotcrete batch plant. In terms of location, site infrastructure encompasses the mill site facilities and the surface facilities located at the mine portal. Mine infrastructure and the tailings storage facility are discussed in other sections of this report.

Off-site infrastructure includes tie-ins to water sources, and facilities required for transportation of the concentrate product to an ocean port (see Section 18.5).

18.1.2 Site Infrastructure

18.1.2.1 Non-Process Buildings and Facilities – Mill Site Area

18.1.2.1.1 Gate Houses

The site will be fenced to prevent unauthorized access. Controlled access through gates will occur at the following locations:

- The main gate house will control access to the site from the direction of Highway E37. The gate house will have facilities for:
  - Security.
  - Turnstiles.
  - Shipping office.
  - Truck scale control.
- An unmanned gate operated from the main gate house will control access to the mill site for vehicles entering and exiting the site from the public road joining the portal and concentrator areas.

18.1.2.1.2 Administration Building

The administration building will provide work space for the management and administration staff. It will contain:

- Reception Area.
- Management Offices (3).

- Safety Management.
- Employee Offices or Cubicles.
- Meeting Rooms.
- Printer and Copier Room.
- Washrooms.
- Data Center.
- Electrical Room.
- Janitor Room.
- Lunch Room.
- HVAC Room.

18.1.2.1.3 Fire and Medical Facilities

The fire and medical facility at the process plant area will be located adjacent to the maintenance building and will consist of a first aid room, ambulance bay, fire truck bay, and safety equipment room, plus a safety induction and training room.

18.1.2.1.4 Mine Dry

The mine dry will be located in the process plant area and will contain the change room and shower facilities for both the mine workers and process plant workers. Workers will walk between the mine dry and the main gate house. Workers will be transported by bus between the mine dry and the mine portal. Facilities will include:

- Men’s clean side lockers (300 lockers).
- Men’s dirty side lockers (300 lockers).
- Women’s clean side lockers (60 lockers).
- Women’s dirty side lockers (60 lockers).
- Showers.
- Toilets.
- Laundry Room.
- Assembly Area.
- Electrical and Mechanical Rooms.

18.1.2.1.5 Maintenance and Warehouse Facilities

The maintenance and warehouse facilities will consist of the vehicle maintenance shop, mechanical maintenance shop, electrical maintenance shop and warehouse storage located in adjoining areas of a common building.

The vehicle maintenance shop will contain the facilities for maintaining the heavy vehicles from the mine and process plant. Maintenance on light vehicles will be contracted to off-site local shops. Facilities will include:

- Three heavy vehicle service bays, sized to service CAT AD45B Haul Trucks and CAT R2900G Underground Mining Loaders.
- Vehicle wash bay with sump and oil skimmer.
- Wash water tank and pump room.
- Lube oil storage.

General mechanical/electrical maintenance shops will include:

- Mechanical repair area.
- Welding area.
- Electrical and instrumentation repair area.
- Tool crib.
- Meeting rooms and offices.
- Building mechanical and electrical services room.
- Lunchroom.

Warehouse facilities will include:

- General warehouse area.
- High value storage area.

18.1.2.1.6 Fuel and Lubricant Storage and Vehicle Fueling

Facilities for vehicle fueling will include:

- Double wall fuel storage tank (60,000 litre capacity).
- Fuel pumps for re-filling vehicles.
- Concrete pad with curb.
- Sump with oil skimmer.

- Bulk and drum oil storage area.

Serbian underground regulations preclude the storage of significant volumes of fuel and oil underground. As a consequence, all underground equipment will be field fueled and oiled using mobile fuel and oil service vehicles. A separate purpose designed fuel and oil facility will be constructed close to the surface maintenance shop which will hold up to 60,000 liters of diesel in double walled storage tanks as part curbed concrete area that will include surface fueling facilities for mill mobile equipment and to replenish the mobile fuel trucks.

Figure 18.1: Aboveground Storage Tanks (ASTs) (Example)

ASTs can be supplied in sizes up to 75,000 gallons. They are available as double-wall tanks.

Similarly, all oils will be stored in cubes in a containment area adjacent to the fuel storage, and at the vehicle maintenance shop.

18.1.2.1.7 Long Term Core Storage

Long term core storage will be at the existing compacted earth, fenced area east of the vent shaft return system.

Pulp reject storage area will be in containers on the concrete foundations of the current core store and building east of the vent shaft return system.

18.1.2.1.8 Laydown Area for Spares

An outdoor laydown area for storage of spares will be provided on a paved area adjacent to the maintenance building.

18.1.2.1.9 Truck Wheel Wash

Trucks shipping copper concentrate product will drive through a wheel wash facility to remove any concentrate picked up in the product loading area. Facilities for the truck wheel wash will include:

- Pumps, headers, spray nozzles.
- Containment walls, catchment basin and sump.

18.1.2.1.10 Truck Scale

A truck scale will be used to record the weight of copper concentrate product contained in each shipment. The Truck scale will be controlled and recorded in the Gate House.

18.1.2.1.11 Rotainer Storage Area

A paved storage area for holding a buffer of rotainers will be provided on site. The size of the retainer area in the Mill Site is large enough for up to 1,000 rotainers stacked 4-high.

18.1.2.1.12 Shotcrete and Concrete Batch Plant

The shotcrete and concrete batch plant will be located at the mill site area.

Rakita's Mine Plan is predicated on a formal set of Manager's Support Rules which combines the use of rock bolts, steel mesh and shotcrete, or sprayed concrete, over almost all of the mine openings. The use of in cycle steel fibre shotcrete, combined where necessary with mesh and bolts, will provided sufficient stability to the surrounding rocks to allow mining of the massive sulphide mineralization. Both dry shotcrete systems, which use a dry, pre-mixed shotcrete powder, purchased at significant expense, in 1 tonne bags and shipped to site and wet shotcrete systems using pre-mixed and batched wet shotcrete, mixed on surface and taken underground by transmixer or pumped down a slick pipe are available. However, dry mix shotcrete, while convenient for small batch application, is very expensive and labour intensive and as a consequence Rakita has opted for a wet mix batch shotcrete plant.

Rakita will therefore purchase and operate an on-site batch plant to produce wet shotcrete, and concrete for use underground. The batch plant will be installed in the mill yard. This batch plant will also be capable of producing ready mix concrete and as such should be purchased early in the construction cycle to help with concrete production during construction and to provide good quality shotcrete for underground development. A photo of a typical semi portable batch plant is shown below.

Figure 18.2: Concrete Batch Plant (Example)

Wet-mix shotcrete would then be batched at the mill site and transported underground using a trans-mixer attached to a tractor. Aggregate for the shotcrete plant will be sourced locally and screened to specification for the shotcrete mix. Cement is delivered to site in 30-t bulk road tankers and unloaded pneumatically to a storage silo. There are a number of cement suppliers within 100 km of the mine site and bulk cement is readily available. Silca fume will also be purchased in bulk and stored in blown silos on site. Steel fiber will be added to the mix as required and is readily available in Serbia.

The wet-mix shotcrete plant will require minus 10 mm – down, size aggregate, plus additional minus 2 mm – down, aggregate mix, (approx. 11,500 m^3) depending of size distribution, that must be crushed and screened. There are multiple local contractors capable of providing aggregate of this type. Aggregate may be required to be heated in winter to provide good quality shotcrete. As a consequence, a simple drum style heater will be added for the aggregate in winter, burning propane.

The shotcrete plant will require a loader to feed aggregate into the plant.

The plant will use recycled water from the mill water supply. No water will be taken from or discharged to the environment.

The shotcrete plant will operate on day and night shift 365 days per year and will be operated by Mine staff. Two workers are required to operate the plant; one in the control room and another on a loader to feed aggregate to the plant.

Transport of the shotcrete underground will require the use of transmixers, similar to the MacLean TM-3 Transmixer tractor or equivalent to transport shotcrete from the batch plant to underground. A photo of the MacLean TM-3 is shown in Figure 18.3.

Figure 18.3: MacLean TM-3 Transmixer Tractor (Example)

18.1.2.1.13 Site Roads

Paved surfaces will be provided around the process plant building, the maintenance building, the mine dry, the office building, and the concentrate shipping area.

Other roads will be granular base roads, capped with crushed stone. These are:

- From Hwy E37 to the concentrator.
- From concentrator to fresh air vent shaft.
- From Hwy E37 to return air vent shaft (partly existing).
- From concentrator to portal area.
- From concentrator to existing public road south-east of the concentrator.

18.1.2.2 Non-Process Buildings and Facilities – Portal Area

18.1.2.2.1 Gate House

The portal area will be fenced.

The portal area will have a gate house with one security guard to control access for vehicles entering and exiting the portal area from the public road joining the mill site and portal areas.

An unmanned gate will provide access to the portal area from Highway E394. This gate is for infrequent and emergency access only.

18.1.2.2.2 Fire and Medical Facilities

The fire and medical facility at the mine portal area will consist of a first aid room, a mine rescue truck bay, an ambulance bay, a first aid room, and a muster station area.

18.1.2.2.3 Offices

Mine office buildings will consist of mine superintendent office, supervisor offices, cubicle workstation area, meeting room, training room, and toilets.

The office building will also house a control room for underground operations.

18.1.2.2.4 Portal Lamp Room

The portal lamp room will contain mine lamp equipment for underground workers to pick up on their way from the bus to the mine.

18.1.2.2.5 Core Handling Building

Existing buildings for core storage, cutting and sampling, logging, and short term storage will be relocated from their existing location to the portal area. A new refrigerated store will be constructed to store high sulphide core samples in this area.

18.1.2.2.6 Utilities

The portal area will contain:

- Fire water tank and pumps.
- Potable water tank and distribution pumps.
- Fresh water tank and distribution pumps.

18.1.3 Off-Site Infrastructure

18.1.3.1 Water Supply

18.1.3.1.1 Process Service Water

Process service water requirements are met using mine water, water reclaimed from the tailings storage facility and site contact run-off water.

18.1.3.1.2 Potable Water

Potable water is supplied from an existing connection to the municipal potable water supply located near the core sample storage area.

18.2 Waste Management

18.2.1 General

Knight Piésold Ltd. (KP) completed the waste management design for the Project. The design is based on 29.6 million tonnes of ore to be processed at a maximum rate of 8,900 tonnes per day using conventional flotation. Approximately 10% of the ore will be concentrate and the remaining material will be tailings that are deposited in the Tailings Storage Facility (TSF).

Waste rock will be stored in designated stockpile areas within in the TSF catchment. The waste rock generated over the life of mine includes non-potentially acid generating (NPAG) and potentially acid generating (PAG) material.

18.2.2 TSF Alternatives Assessment

A tailings technology and TSF location alternatives assessment was carried out utilizing information from previous siting studies, recent project data, and preliminary environmental and socioeconomic baseline data. The alternatives assessment considered four candidate locations and three different tailings technologies:

- Conventional slurry tailings (low solids content).
- Paste tailings (higher solids content).
- Filtered tailings (high solids content, dewatered).

Conventional slurry tailings management was identified as the preferred tailings technology. Conventional slurry tailings require less processing and can be more easily transported using pipelines and centrifugal pumps, eliminating the need for positive displacement pumps, conveyors, or trucks for transportation to the TSF. Conventional slurry also provides a better opportunity for management of potential acid rock drainage (ARD) and metal leaching (ML) with proper operation of the slurry discharge and water pond management systems.

Four alternative TSF locations were evaluated. A preferred location was chosen based on a scoping level Multiple Accounts Assessment approach that considered technical, economic, environmental and socio-economic categories in the ranking and evaluation process.

18.2.3 Site Conditions

Regional Geology

The Project is in an area of gently sloping grassy hillsides with brush and light forest cover. Geologically, the region is part of the Tethyan Belt, which hosts Mesozoic to Cenozoic subduction-related base metal deposits from Romania and Serbia through to Turkey and Iran. The Bor cluster of deposits is hosted by Upper Cretaceous andesites and volcaniclastics that continue at least 5 km south to Timok, where the Cretaceous unit is overlain by a Miocene basin containing clastic sediments that can be hundreds of metres thick.

Local Geology

Geotechnical and hydrogeological site investigations were completed in 2017 to support the PFS design of the waste management facilities. The results of the site investigation indicate the following:

- Overburden thickness ranged from 0.7 to 16.0 m and consists predominantly of medium to high plasticity clay.
- Bedrock consists of interbedded layers of clastic sediments (claystone, sandstone, siltstone, and conglomerate).
- Average hydraulic conductivity in the TSF bedrock ranges from $9x10^{-7}$ to $1x10^{-6}$ m/s.
- Measured groundwater levels ranged from 6 m to 34 m below ground surface in the area of the TSF embankment.
- Field estimates of unconfined compressive strength (UCS) for weathered bedrock units encountered within the TSF area typically ranged from 3 to 9 MPa.
- The rock mass rating (RMR) of bedrock throughout the Project area ranged from 36 to 57, typically indicating a rock mass designation of Poor to Fair.

Climate and Extreme Weather Events

The regional climate in the Project area is moderate-continental with local variations. All data are either from the Serbian Government, or from a study of climatic and hydrologic parameters produced by the University of Belgrade in 2016. Table 18.1 summarizes climatic and hydrologic parameters from this study.

Table 18.1: Climatic and Hydrologic Parameters

Parameter		Value
Average Annual Temperature (°C)		10°C
Maximum Annual Temperature (°C)		21.4°C
Minimum Annual Temperature (°C)		-9.3°C
Mean Annual Lake Evaporation (mm)		786 mm
Mean Annual Precipitation (MAP) (mm)		675 mm
24-Hour Rainfall Events	1 in 100 Year return period	111 mm
	1 in 200 Year return period	137 mm
	1 in 1,000 Year return period	169 mm
24-Hr Probable Maximum Precipitation (PMP)		477 mm
1 in 100 Year Snowpack (Snow Water Equivalent (SWE))		600 mm

Seismicity

Seismic ground motion parameters were estimated using probabilistic seismic hazard analyses from the European Facility for Earthquake Hazard and Risk (EFEHR) web platform. Mean peak ground acceleration (PGA) hazard values were determined for earthquakes with return periods ranging from 100 to 4,975 years and extrapolated to a corresponding return period of 10,000 years. The recommended PGA values for various return periods are summarized in Table 18.2.

Table 18.2: Recommended Peak Horizontal Ground Accelerations (PGA)

Return Period (Years)	Peak Ground Acceleration, PGA (g)
100	0.03
475	0.07
975	0.10
2,475	0.16
4,975	0.22
10,000	0.32

18.2.4 Design Basis, Standards and Criteria

Design Basis

The mine will operate with a maximum mill throughput of 8,900 tonnes per day (tpd) and 8,300 tpd will be tailings (after removing concentrate). The TSF is required to store all tailings generated from milling, the design storm event, operating pond volume and freeboard. The design basis for the TSF is summarized in Table 18.3.

Table 18.3: Design Basis

Description	Value
Mill Throughput (Maximum rate)	8,900 tpd
Tailings Generated (Maximum rate)	8,300 tpd
Tailings Total Tonnes	26.9 Mt
Settled Tailings Dry Density	1.3 – 1.6 t/m^3
Tailings Volume	18.7 Mm^3
Operating Pond Volume (Production)	0.4 to 1.6 Mm^3
Design Storm Storage	• Years PP-1 to P1: 1.5 Mm^3 (1:200 year 24-hour event) • Years P2 to Closure: 3.8 Mm^3 (diverted potion 0.8 Mm^3) total PMF = 4.6 Mm^3
Freeboard Allowance	3 m

Standards and Criteria

The TSF embankment was classified according to the Canadian Dam Association Dam Safety Guidelines (CDA, 2014) to confirm the appropriate design flood and earthquake events. The classification considers loss of life, environmental and cultural values, and infrastructure and economics and an Extreme classification was selected for the design. Based on this, the Probable Maximum Flood (PMF) was assigned for the Inflow Design Flood (IDF). The Earthquake Design Ground Motion (EDGM) selection is also governed by the Extreme classification. The EDGM is the 1/10,000 year event or the Maximum Credible Earthquake (MCE).

18.2.5 TSF Design

The principal objective for the TSF design is to provide secure and stable storage for mine waste and water that does not adversely affect the regional groundwater and surface water during operations and post closure. The design of the TSF has taken into account the following requirements:

- Permanent, secure and total confinement of all solid waste materials within an engineered disposal facility.
- Control, collection and removal of free draining liquids from the tailings during operations for recycling as process water to the maximum practical extent.
- Management of surface water.
- Staged development of the facility.
- Ongoing monitoring and assessment.
- Appropriate closure measures .

The general arrangement for the TSF at the end of operations is shown on Figure 18.4.

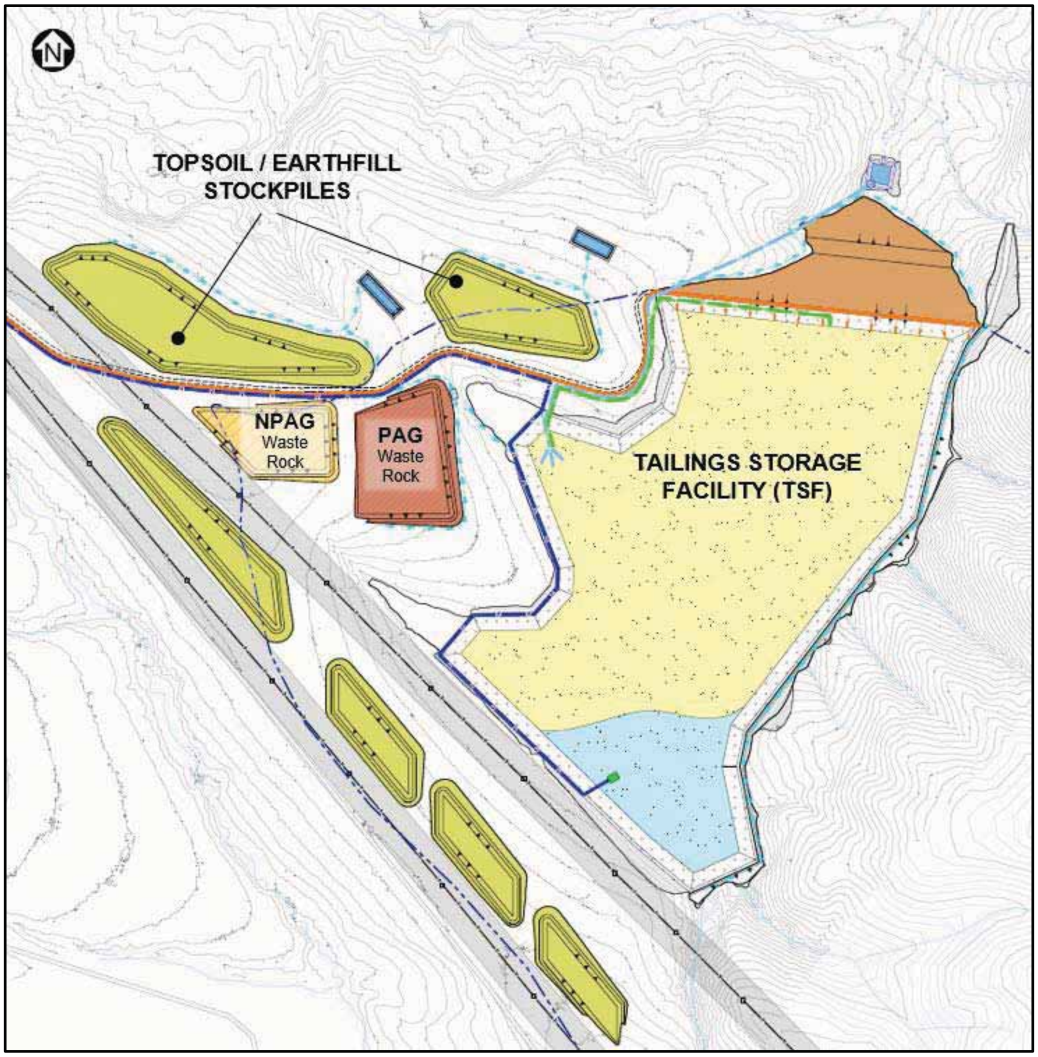


Figure 18.4: Ultimate (Stage 3) TSF General Arrangement

Construction Staging and Filling Schedule

A TSF basin filling schedule was developed to present the storage demand over the life of mine. The filling schedule includes storage for tailings and water, freeboard, and a provision to allow for co-disposal of the PAG waste rock (if needed). The filling schedule also addresses the storage of water for mill start-up.

The TSF embankment staging is shown on the filling schedule on Figure 18.5. The PFS level basin shaping and embankment staging incorporates three stages over the life of mine.

Subsequent designs will consider further basin and embankment construction staging optimizations.

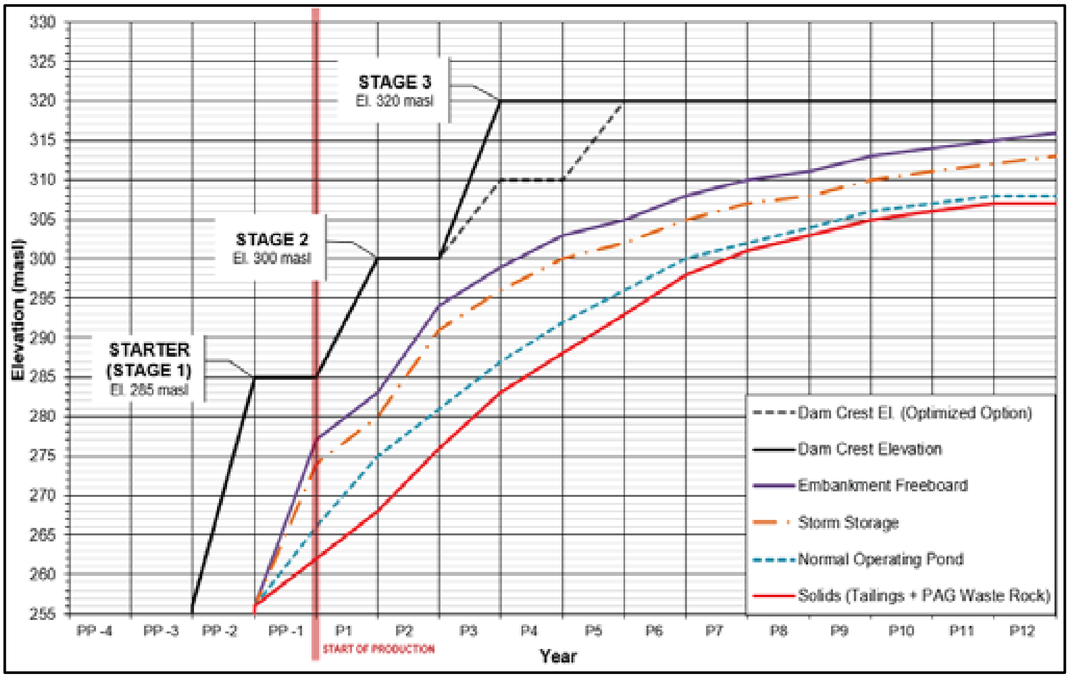


Figure 18.5: TSF Filling Schedule

Embankment Cross-Section

The TSF embankment will be constructed from local materials excavated during basin shaping. Embankment expansions will be completed using the downstream construction method. Each embankment stage is constructed with 3H:1V slopes. There are 8 m benches at each of the stage raises. The final embankment has a minimum crest width of 10 m. A 40 m wide downstream buttress is included for each of the stages. The overall slopes of the final embankment are 3.3H:1V upstream and 3.5H:1V downstream. The maximum height of the TSF embankment is approximately 90 m, measured from the crest to the downstream toe of the embankment (in the Ultimate Stage 3 configuration).

The embankment stages include a buttress and shear key at the downstream toe for stability. Overburden in the embankment footprints may be more than 20 m thick. Most of the overburden will be removed from the embankment footprint.

A cross-section through the embankment is presented on Figure 18.6.

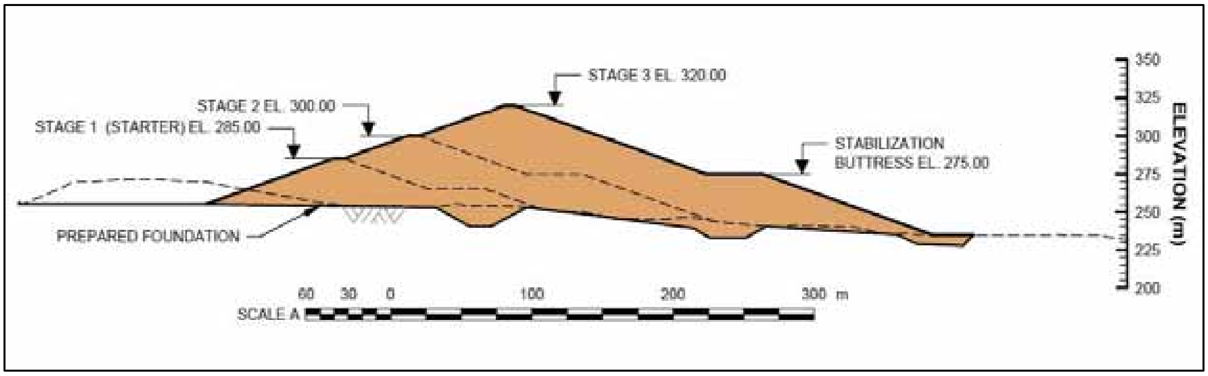


Figure 18.6: TSF Embankment Cross-Section

Foundation Preparation and Basin Shaping

The footprint of the TSF area will be cleared of trees and stripped of topsoil. The basin will be prepared (shaped and graded) to provide the required storage capacity and sub-grade for placement of a High Density Polyethylene (HDPE) geomembrane liner. The material excavated from within the basin will be used as embankment construction material and stockpiled for the TSF closure cover.

Liner System

The TSF basin, including the upstream embankment face, will be lined with HDPE geomembrane. A non-woven geotextile will be placed below the geomembrane, on top of prepared subgrade, as an additional measure to protect the liner. Details of the liner system are shown on Figure 18.7.

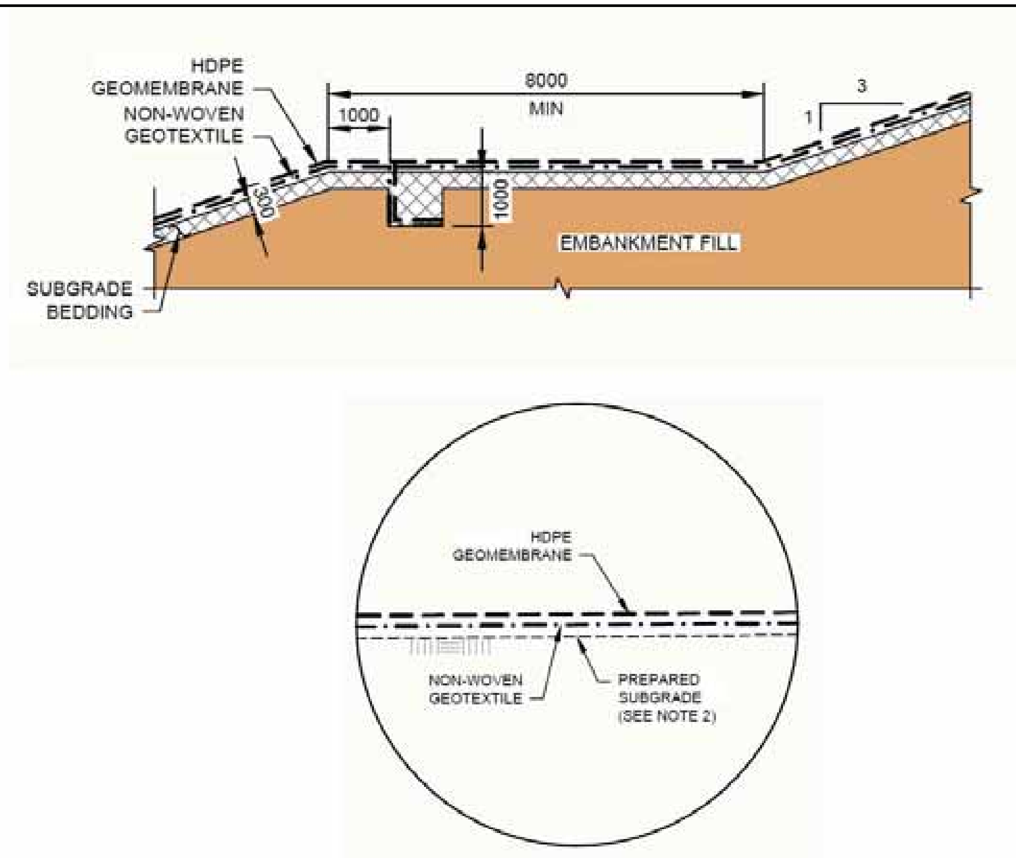


Figure 18.7: TSF Liner System Details

Foundation Drain

A Foundation Drain will be installed within the footprint of the TSF, below the liner system, to collect groundwater flows, potential seepage and infiltration through the TSF embankment. The Foundation Drain comprises a perforated drain pipe surrounded by drain gravel, in a drain trench. The drain will discharge to the Seepage Collection Pond downstream of the TSF embankment. Water collected in the pond will be pumped back to the TSF. The Seepage Collection Pond will also be lined with HDPE geomembrane. Foundation Drain details are shown on Figure 18.8.

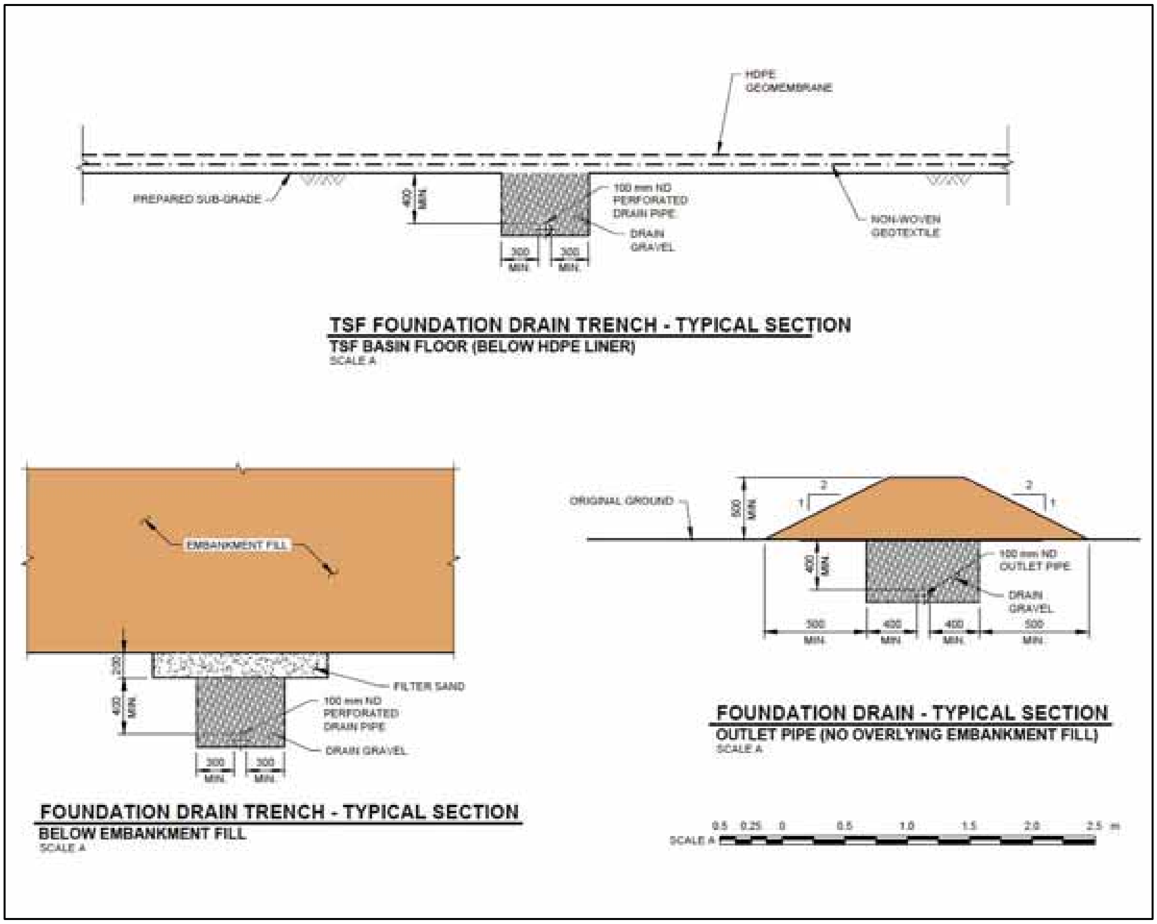


Figure 18.8: Foundation Drain Details

Basin Underdrain

A Basin Underdrain will be installed above the HDPE geomembrane on the lower part of the TSF Starter (Stage 1) basin floor to promote tailings consolidation. The Basin Underdrain will discharge to an underdrain sump at the upstream toe of the embankment. The sump will be lined with HDPE geomembrane and filled with drain gravel. A submersible pump and riser pipe will be used to collect and transfer the water from the underdrain system to the supernatant pond. The Basin Underdrain layout is shown on Figure 18.9.

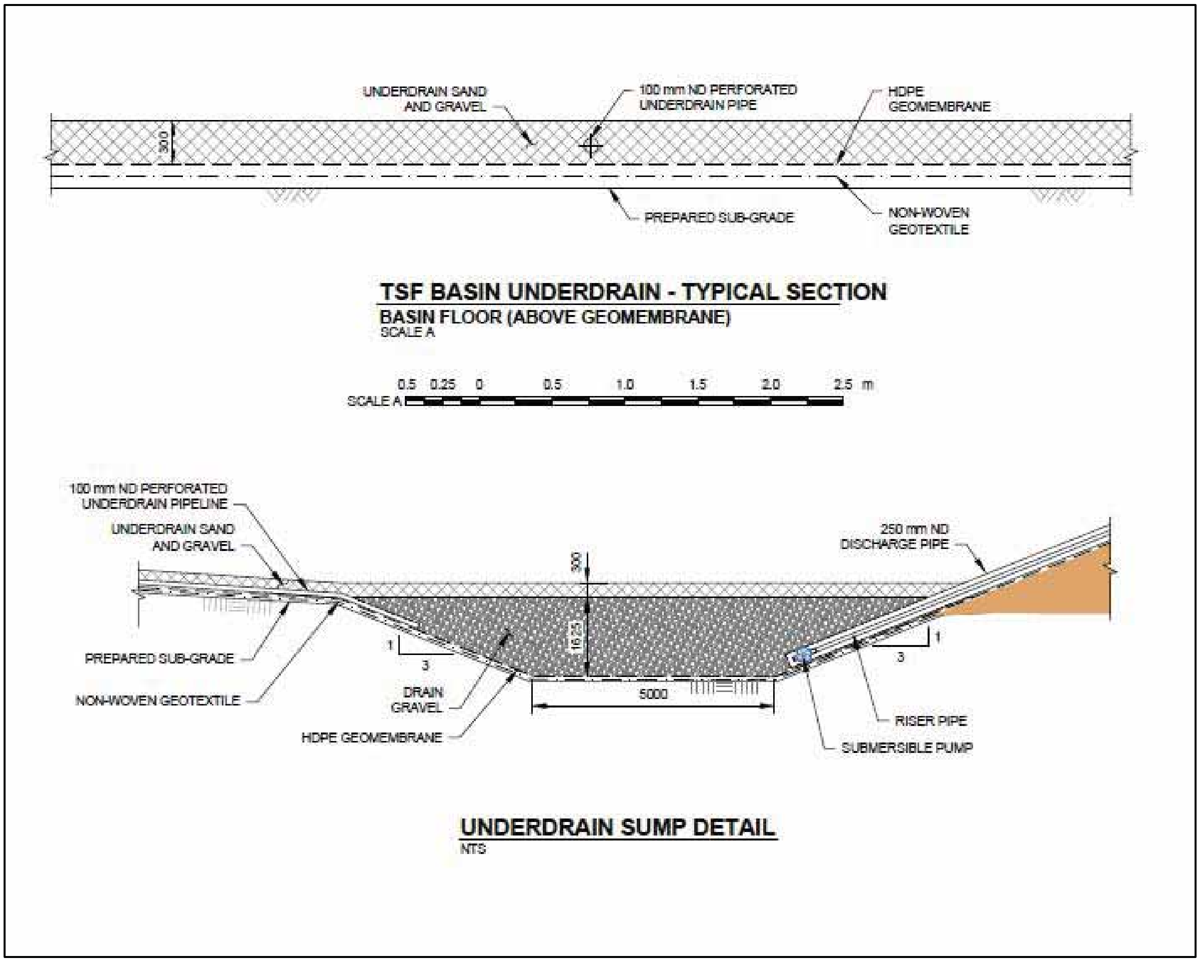


Figure 18.9: TSF Basin Underdrain Details

Mechanical Systems

Management of tailings slurry, reclaim water and potential seepage at the TSF requires the use of various pumps and pipelines, as described below.

Tailings Delivery System:

Tailings are delivered as a conventional slurry with a solids content of approximately 31% (by weight). The slurry will be conveyed from the Plant Site in a single ND 450 mm DR11 HDPE pipeline. Slurry will be discharged from the TSF embankment through spigots spaced at regular intervals and a supernatant pond will develop on the south side of the facility.

Reclaim Water System:

The TSF reclaim water system will convey water from the TSF back to the Plant Site. A floating pump barge will be located on the south side of the TSF. Water will be pumped from the barge to the Plant Site in a single ND 600 mm DR17 HDPE pipeline.

Basin Underdrain Pipeline:

The Basin Underdrain is located on the low part of the TSF basin, above the geomembrane liner. The water recovery system consists of a pump installed in a perforated HDPE pipe in a sump at the upstream toe of the TSF embankment. The HDPE pipe will be an HDPE pipeline that extends up the face of the embankment to the crest. Drain rock placed around and over the pipe creates a sump collection zone to promote drainage of water to the pump. Water recovered in the underdrain system will be discharged to the supernatant pond.

TSF Seepage Collection System:

Seepage from the TSF will drain by gravity to the Seepage Collection Pond via the Foundation Drain. The Seepage Collection Pond is located immediately downstream of the TSF and a pump system will be provided to return any collected seepage back to the TSF.

Instrumentation and Monitoring

Instrumentation will be installed in the TSF embankment and foundation, and monitored during construction and operations to assess performance and to identify any conditions that may differ from those assumed during design and analysis. Amendments to the ongoing designs, operating strategies and/or remediation work can be implemented to respond to changing conditions, should the need arise.

18.2.6 Waste Rock Storage

Approximately 2 million tonnes of waste rock will be generated over the pre-production and production years. Two designated waste rock storage areas (for NPAG and PAG waste rock) are located west of the TSF, within the TSF catchment.

The NPAG storage area will be gently graded, sloping toward the lined TSF impoundment. All surface runoff and snowmelt from this area will report to the TSF via surface ditching. The NPAG storage area has two 8 m high benches and an 8 m wide access ramp. It includes surface grading for runoff management.

The PAG storage area will also be gently graded toward the TSF impoundment. It will have a geomembrane lined surface with a protective sand cover to minimize the risk of liner damage during rock placement. The PAG storage area has two 8 m high benches and an 8 m wide access ramp. It also includes perimeter ditches to collect runoff from the PAG waste rock and allow water monitoring prior to discharge to the lined TSF impoundment via geomembrane lined surface trenches.

18.2.7 TSF Area Water Management

A water balance and water management plan were developed for the TSF area, as described below.

TSF Water Balance

The TSF water balance incorporated average climatic conditions and monthly climate input data. The supernatant pond, runoff from the TSF basin and waste rock storage areas and underground inflows are major contributing water sources and will be used to the maximum extent to reduce reliance on non-contact and off-site water sources. The TSF water balance was as an input to the overall site wide water balance.

The water balance model results indicate that the TSF will generally operate in a surplus condition under average climatic conditions. Key results of the water balance are as follows:

- The TSF supernatant pond volume fluctuates over the life of mine, ranging from approx. 0.4 Mm^3 to 1.6 Mm^3.
- Process water requirements are generally balanced by the input of water from the underground mine works, snowmelt and runoff in the TSF catchments.
- External make-up water is not needed under average conditions.
- At end of production, approximately 1.5 Mm^3 of water will need to be treated prior to discharge, to allow passive closure. It may be possible to reduce this amount by optimizing inputs and outputs over the last few years of operations.

TSF Water Management Plan

A water management plan was developed to control water in the TSF area during construction, operations, closure and post-closure. The key objectives of the water management plan are:

- Minimize the extent of land disturbance.
- Minimize contact water by isolating development areas with diversion/collection ditches.
- Collect and contain mine contact water and treat as needed.
- Collect water in the TSF for use during operations.

The water management structures are described in the supporting reports and drawings.

18.2.8 Closure and Reclamation

TSF closure will be completed in a manner that will satisfy physical stability, chemical stability and meet environmental and social targets. The primary objective of the closure and reclamation activities will be to return the area to a self-sustaining state, while managing surface water and protecting the downstream environment. This will be accomplished during both active and passive closure phases. Closure is discussed in Section 20.

18.3 Water Management

A site wide water balance was carried out under average climatic data, to assess the seasonal water volume variations over the life of mine and the performance of the site water management facilities. The balance includes pre-production and production period providing annual water inputs to the water treatment plant. The groundwater and subsidence infiltration water into the underground mine is a major input to the site wide water balance and acts as the main water source for the mill operation and for mine service water.

The water storage in the TSF can be maintained in the higher range reaching the peak value during production ramp-up. Collected water in the TSF can be used as a storage reservoir to meet process plant demand for subsequent production years when process water demand is higher. The TSF water inventory will be decreased as the mine closure period approaches.

18.4 Power/Electrical

18.4.1 Power Supply and Distribution

18.4.1.1 Power Supply

Electrical power to the plant will be supplied from the newly built 110/35 kV utility substation. The new substation will be fed from the nearby 110 kV overhead transmission line "BOR 2 – ZAJECAR 2" through the double circuit transmission line. The 110/35 kV substation and connecting overhead line will be designed and supplied by state owned Transmission System Operator (TSO). Estimated capital costs for the construction of the 110/35 kV are included in the Timok CAPEX and will be covered by the Project.

Total estimated power demand for the plant is expected to be slightly below 34 MVA. The two main step down transformers 110/35 kV, with base capacity of 31.5 MVA, will provide 100% redundant power supply to the main plant substation.

18.4.1.2 Primary Power Distribution

The main 35 kV substation will be fed from the 110 kV utility substation by two 35 kV cable buses. The prefabricated modular building (E-House), with main 35 kV switchgear and all auxiliary equipment inside, will be located adjacent to the processing plant. The main switchgear consists of two section connected by normally open tie-breaker.

Power is distributed from the main plant substation to the key process area substations (Flotation, Grinding and Underground infrastructure) through a system of 35 kV double feeders. Each feeder originates at the different section of the main 35 kV switchgear.

Power will be distributed to the area substations by cables installed in cable tray supported on racks. The area substations are located adjacent to concentrations of load to minimise costs for low voltage cables.

35 kV overhead power lines will be used to supply power to remote loads like underground fresh air and return air ventilation fans. The same lines will be used for surface portion of the feeders supplying power to underground facilities.

The power correction equipment, in form of two harmonic filters, will be installed as part of the main 35 kV plant substation.

18.4.1.3 Secondary Power Distribution

The selected secondary distribution voltage levels for the process plant are 6 kV, 3 phase, 50 Hz for large drives and 400 V, 3 phase, 50 Hz for smaller drives. The secondary distribution system originates at area substations.

The substation step-down transformer ratings have been standardized at 10 MVA for the 35/6 kV distribution system and 1.25 MVA for the 6/0.4 kV distribution system.

18.4.1.4 Standby Power

Standby power for the process plant will be provided from diesel powered generators located at the main 35 kV substation. A total of 3 x 2.3 MW diesel generators are provided. The generator output voltage will be 6 kV.

The purpose of the standby power system is to provide an alternate source of power to critical process equipment and underground mine fans and pumps in the event of failure of the normal supply, allowing for an orderly shutdown of the process plant and to provide the minimum required power in the event of an overall power interruption to the plant. The output from the standby power generators will be connected to a common standby power switchgear assembly.

The standby switchgear assembly will be connected to the main 35 kV switchgear via a step-up transformer. Under normal operating conditions the standby power switchboard will be energized from the main 35 kV supply.

Upon loss of connection to utility power, the diesel generator units will started automatically, connected to the main 35 kV switchgear, and sequence of energizing all area substations started. The Plant Control System (PCS) will control the energization process and will allow only critical process loads to operate from standby power source.

Emergency power will be supplied to small capacity loads from Uninterruptible Power Supply (UPS) systems for the DCS, PLC’s, critical process equipment, essential services and emergency lighting.

18.5 Concentrate Transport

18.5.1 Overview

Timok concentrate will be shipped in sealed half-height shipping containers ("Rotainers") from the plant site via truck and rail to the Bulgarian ocean port of Burgas. A port crane will then hoist the Rotainer into the ship's hold, using a specialized lifting cradle, where it will then be tipped, discharging the concentrate. The empty Rotainer will then be returned to Timok via rail and truck.

This method of shipment, commercially proven for shipping copper concentrates, has zero emissions, no product losses and therefore minimizes environmental risk.

18.5.2 Transport Requirements

A list of concentrate transport requirements are provided in Table 18.4. Note that concentrate production varies considerably over the life of the mine. Transport requirements shown in this section are based on the average of years 2 through 5. After year five (5) the transport requirements decrease significantly.

Table 18.4: Concentrate Transport Requirements

Parameter	Description	Unit	Basis
Production: annual	520,000	dtpy	Dry Basis. Average of Y2-Y5
Production: annual	270,000	dtpy	Dry Basis. Average LOM
Production rate – dry basis	59.4	dtph	Average Y2-Y5 production
Production rate – dry basis	30.8	dtph	Average LOM production
Concentrate Physical Characteristics			
Bulk density (dry basis)	2.2	t/m^3	PDC
Moisture content of product	10%	% wt	Stream tables
Bulk density (wet basis)	2.42	t/m^3	PDC
Transport Requirements (Average based on years 2 through 5)			
Transport tonnage	572,000	tpy	Wet basis
Transport tonnage - monthly	47,667	tpm	Wet basis
Transport tonnage – daily	1,567	tpd	Assumes 7 day/week.

18.5.3 *Concentrate Transport Methodology*

The following transport method is assumed as the basis for the prefeasibility study:

- Rotainers will be used for the “in-country” transport of the concentrate. Rotainers are the same size (footprint) of a standard shipping container but are half of the height. They are equipped with a sealable lid. Refer to Figure 18.10.
- At the Timok mill yard, a reach stacker will load an empty Rotainer onto a truck/trailer.
- A fork lift will remove the lid of the Rotainer.
- The truck will drive into the concentrate load-out building.
- A front-end loader will load the Rotainer with concentrate. After filling, the Rotainer’s lid will be repositioned on the Rotainer and fastened in position.
- Trucks will deliver Rotainers filled with concentrate to the Bor train transhipment yard (a distance of about 10km). A reach stacker will lift the container off the truck and stack the Rotainer in preparation for train transport. At least one complete train load of containers will be stacked in the railyard at any time to speed train turn around.
- The trucks will backhaul empty Rotainers to the plant site.
- Empty Rotainers will either remain on the truck to be re-filled immediately or will be stacked in the mill yard.
- Truck operation will be limited to two, eight hour shifts per day (total of 16 hours/day). Sufficient storage will be provided in the concentrate building to store the concentrate produced during the 8 hour night shift, or additional containers will be stacked full, in the mill yard.
- Rotainers will be loaded onto railcars by reach stackers. Each train will carry a total of 30 Rotainers (maximum train capacity) on 15 railcars.
- Each day, 2½ trains (each carrying 30 Rotainers) will transfer concentrate from Bor to the port of Burgas (Bulgaria), a distance of about 750km. The round trip travel time is expected to be less than 48 hours (including loading and unloading).
- When the train arrives at the port, the Rotainers will be unloaded from the train using a reach stacker and then stacked at a port transhipment area, up to 4 units high.
- Typical ship loads will be 10,000 tonnes. As a consequence, at least 10,000 tonnes will be stored in a stacking area adjacent to the port, prior to the ship docking.
- When the ocean freighter is docked at the berth and is ready for loading:

    - The reach stacker will transfer Rotainers from the stacking area onto jockey trucks for transfer to a position adjacent to the ship.
    - A crane (with a Rotainer lifting bale) will lift a Rotainer into the ship's hold where it will tip the Rotainer, thereby dumping concentrate directly into the hold. The crane may either be a port crane, or the ship's crane, whichever is available.
    - A total of 10,000 tonnes of concentrate (wet basis) will be loaded into the ocean freighter's hold as a single batch. Some ships may take two batches of concentrate depending on the customer.
    - Empty Rotainers will be returned to the dock. A jockey truck will transfer the empty containers back to the stacking area, where the reach stacker will stack the empty Rotainers in preparation for loading on railcars.
- The ocean freighter will transfer the concentrate to the applicable port. Bulk unloaders at the delivery port will be used to offload the concentrate for transport to the metallurgical process plant.
- Empty Rotainers will be backhauled by train to the Bor train station and then by truck to the Timok site. On the train the empty Rotainers will be stacked two high, leaving room on the same train for back haul freight, in shipping containers, if necessary.

Figure 18.10: Rotainer (Example)

18.5.4 Rotainer Information

Rotainer information and quantities are provided in Table 18.5.

Table 18.5: Rotainer Information

Parameter	Description	Units	Basis
Capacity (wet concentrate)	21.5	t	Rakita - Serbian Regulations
Dimensions			
width	2.44	m	
length	6.05	M	
height	1.45	M	
Tare weight	2.8	t	Rotainer
Maximum weight - loaded	24.3	t	Calculated
Number of Rotainers shipped/day	73	units	Assumes 7day/week
Number of Rotainers Required			
At Mill Site	146	units	Capacity for two days
At Rail Station	60	units	Two train loads
In transit between Bor and Port	219	units	Based on 48 round trip travel time and 50% extra
At Port	698	units	50% more than one ship load
Total number of Rotainers	1,122	units	Sum of above

18.5.5 Trucking Assumptions and Basis

Trucks will haul concentrate (in Rotainers) from Timok to the Bor rail station. The trucks will operate 16 hours per day. During the night shift concentrate will accumulate in the loadout building or will be stored in filled containers on the mill site.

Table 18.6: Trucking Requirements

Parameter	Description	Units	Basis
Nominal number of trucks/day	73	/day	Calculated
Distance from Timok to Bor train station	10	km	
Daily Trucking Hours of Operation	16	/day	Assumption
Number of trucks/hour (day shift only)	4.6	/hr	Calculated
Number of trucks required	4.1	trucks	- average speed 50 km/hr - 20 minute/load at rail station - 10 minutes/load at plant
Concentrate bulk storage capacity at site for non-trucking hours	522	t	Calculated
	216	m^3	

The truck route between Timok and the Bor railyard in shown in Figure 18.12. A photo of the Bor railyard and the proposed Rotainer loading area shown in presented in Figure 18.13.

Figure 18.11: Truck Transport of Rotainers (Example)

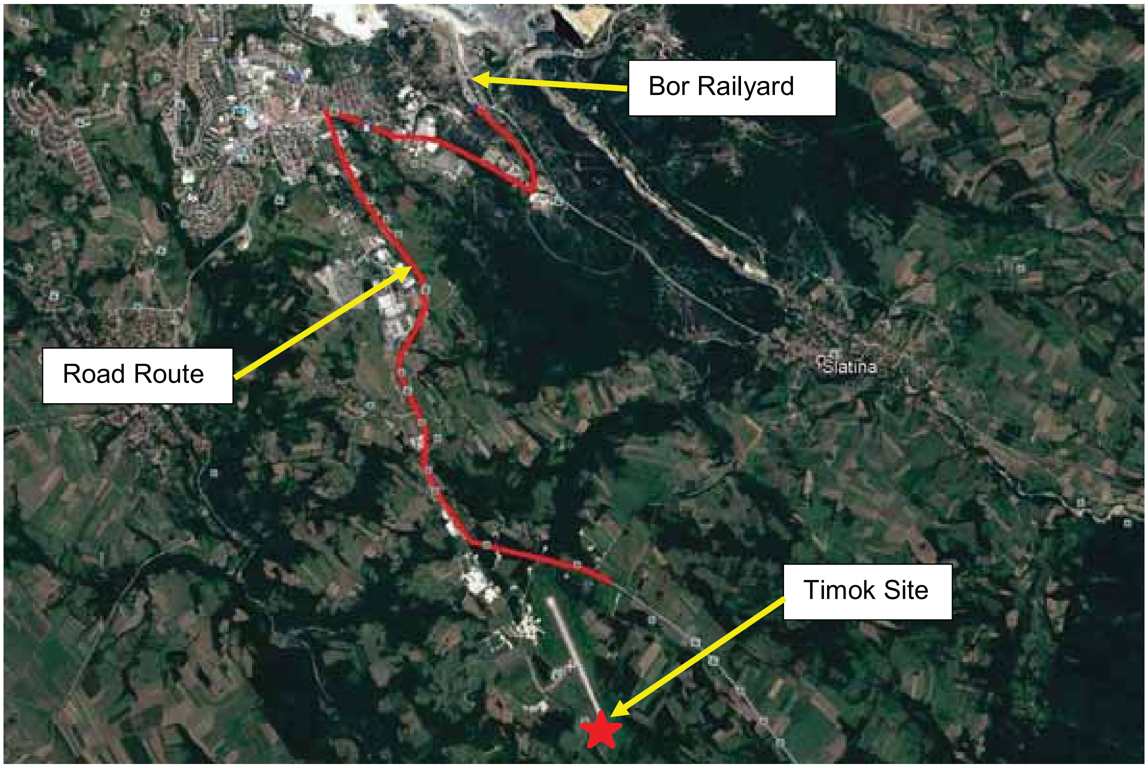


Figure 18.12: Trucking Route from Timok to Bor Railyard

Figure 18.13: Proposed Rotainer Loading Area at Bor Railyard

18.5.6 Rail Assumptions and Basis

A map showing the rail route from Bor to the port of Burgas is provided in Figure 18.14. Train information is provided in Table 18.7. The rail distance between Bor and Burgas is 741 km (one way). Transit time is estimated to be 17 hours in each direction (i.e. 34 hours total) excluding loading and loading time.

Figure 18.14: Map of Rail Route from Bor to Burgas

Table 18.7: Rail Information

Parameter	Description	Units	Basis
Number of Railcars/Train	15	Cars/train	1,100 t/train limit
Number of full Rotainers/railcar	2	Units	Railcar net capacity = 50 t
Number of full Rotainers/train	30	Units/Train	
Distance from Bor to port	741	km	
Train frequency	2.4	/day	Calculated

18.5.7 *Port and Ocean Freighter Assumptions and Basis*

The port of Burgas is a modern Port with two separate basins. Port Terminal Burgas West which handles mixed cargos and containerized deliveries and Port Terminal Burgas East II, which handles bulk cargos, including Cu concentrates, grain and LNG. Both basins are protected by extensive sea walls and the port is capable of handling Panamax size ships, with 16 deep-water berths with a total length of 2.7 km.

The Port is undergoing a program of modernization and has all the equipment that would be necessary to handle the rail hauled concentrate from Timok. Total bulk tonnage in the Burgas East II terminal for 2016 was 1.15M tonnes, out of a total tonnage for all cargos of 2.25M tonnes for East II alone. Burgas East II can handle bulk carriers up to 300 m in length with capacities of 65-80,000 DMT and maximum draft up to 14.8 m.

Burgas East II would be the port of choice for concentrate export, as it already handles import and export of Cu concentrate for others.

Table 18.8: Port Requirements

Parameter	Description	Units	Basis
Freighter capacity – Bulk concentrate	10,000	t	Rotainer quote. Assumes one hold is used. Freighter capacity likely to be >40,000 t total capacity
Ship frequency	6.38	days/ship	Calculated
Number of Rotainers/shipment	465	Units	Calculated
Ship loading time	13.3	hours	Based on Rotainer video
Required Rotainer storage capacity at the port	698	Units	Assumption. 50% more than 1 ship load

Figure 18.15: Aerial View of Burgas Port

Figure 18.16: Ship Loading (Example)

19. Market Studies and Contracts

19.1 Concentrate Marketing

The project will produce a single stream of copper concentrate with an average life of mine grade of 26.2% copper, 5.7g/dmt gold and 1.4% arsenic. The project's treatment and refining charges have been adjusted upwards and arsenic penalties have been added to compensate prospective buyers for the concentrates' arsenic content, particularly later in the mine life. Arsenic levels will be lower in the early years of production allowing for a more diverse customer profile and ease of product placement in the initial years. Apart from elevated levels of arsenic, the production profile of Timok is expected to have low to negligible levels of other impurities that would attract penalties or cause difficulties to market. In the latter years, the higher arsenic bearing material requires a more carefully defined marketing strategy. Nevertheless, a conservative approach has been taken with regards to the NSR value in the PFS (PFS, 2018). It is envisaged that all the copper concentrate produced by the Project will be marketable and sales will be made with penalties for arsenic, consistent with industry norms.

The Company has held preliminary discussions with two potential groups of customers - European and Asian-based smelting companies and concentrate trading companies with blending capabilities, both of whom have expressed interest in procuring the project's concentrate via long term contracts. The marketing cost assumptions are based on these discussions, on the Company's own views and experience in the copper concentrate market, and on a detailed marketing report prepared by Bluequest Resources AG, a specialist in marketing elevated arsenic content concentrates. The Company's marketing assumptions were also peer reviewed by Ocean Partners, the Company's marketing advisor for its Bisha concentrates. Discussions with potential offtakers will continue to be advanced in parallel with the feasibility study.

A well-defined marketing strategy for this concentrate will be a key focus for the Company going forward. Nevsun has not entered into any concentrate sales contracts for this Project and has not committed any tonnages to or fixed any terms with any potential customers. Nevsun's preferred sales strategy is to commit the majority of its production under longer-term contracts with smelters directly. Any uncommitted balance will be placed onto the spot market or under contract to traders. The tonnage allocation to each customer will be driven in part by product requirements, favourable logistics (proximity to Project), counterparty credit risk and diversification, and commercial terms that can be achieved with individual buyers.

For purposes of the PFS (PFS, 2018), Nevsun has modelled the sales of the Project's yearly copper concentrate production to a combination of specialized smelters in Europe and Asia, and traders. In the event of unforeseen capacity constraints at one of these smelters, other smelters have been identified as suitable locations for the Company's production. In determining the treatment and refining charges and associated penalties, the Company has assumed no changes from the current benchmark treatment and refining charges. Total treatment, refining and penalties over the life of the Project are $966M, or $306 per tonne. Transport costs of $141 per tonne reflect the cost of transport from the mine to smelter, associated materials handling costs, weighing, sampling, and assay fees.

Table 19.1 summarizes the key assumptions for the sale of the Project's copper concentrate.

Table 19.1: Key Marketing Assumptions

Item	Units	LOM (USD)
Copper Price	US$/pound Cu	3.15
Gold Price	US$/ounce Au	1300
Copper payable percentage	%	95.88
Gold payable percentage	%	89.1
Total Treatment & Refining Charges, and Arsenic	US$/ dry metric tonne	$306
Transport and Other Selling Costs	US$/ dry metric tonne	$141
Total Realization Costs	US$/ dry metric tonne	$447
TC, RC, Penalties, Transport & Selling Costs	$ Million	1,414

Treatment charges are applied per dry metric tonne of copper concentrate and copper refining charges are calculated per payable pound of copper. In determining the selling costs for this material, assumptions were made as to the arsenic penalties that will need to be applied based on the profile of the concentrate throughout the life of mine. Metal payable percentage is normally used to describe the proportion of metal for which payment will be made and in the case of copper and gold are expected to be in line with market norms.

Ongoing metallurgical and marketing trade-off studies are evaluating whether producing two streams of concentrate will result in lower realization costs than producing a single stream of concentrate.

19.2 Complex Concentrate Market

The copper concentrate market is expected to move into deficit over the next four years as the project is developed. According to the ICSG, China is continuing to expand its smelting capacity, albeit at a slower pace than before. China's copper smelting capacity more than quintupled in the period from 2000 to 2016, and is expected to increase by a further 20% until 2020, accounting for 80% of the forecast world growth in smelting capacity by then. Outside of China, smelting capacity could potentially be added in countries such as India, Indonesia, Kazakhstan, Mexico and Mongolia. With yearly copper production capacity projected to lag the growth in smelter capacity, Timok would be well placed to take advantage of this supply/demand imbalance. A tightening of the market would benefit the project both through reduced benchmark treatment charges and the reduction in penalties for complex concentrates as these concentrates are easier to move in an undersupplied market. The realization costs assumed in the project economics do not include the potential cost savings that could occur if the market tightens.

Additionally, the majority of existing copper mining projects are expected to continue to produce clean concentrates. This contrasts with recently commissioned copper projects, such as Toromocho and Ministro Hales, which resulted in a step change in the level of complex concentrates production. Traders reacted to this increase by developing concentrate blending facilities and smelters reacted by investing in technologies to increase their ability to process complex concentrates. Traders and smelters are now looking for complex material to allow them to utilize these new capabilities and realize additional revenues from arsenic penalties. The relative reduction in supply of complex concentrates to full these facilities may result in compressed arsenic penalties which the project would benefit from but which are not currently assumed in the project's economics. Lastly, any future relaxation of current arsenic import limits on copper concentrates into China, which are currently limited to 0.5% arsenic would materially improve commercial terms and represent increased marketability of Timok's concentrate.

19.3 Marketing Opportunities

As part of the feasibility study, the Company will continue to study options for reducing the project's realization costs. In addition to the single bulk concentrate, the PFS flow sheet allows for the production of a clean copper concentrate with a low level of arsenic and a complex copper concentrate with relatively higher level of arsenic. Ongoing trade-off studies are evaluating whether producing two streams of concentrate will result in lower realization costs than producing a single stream of concentrate. Additionally, the Company is studying potential use of a roaster to decrease the arsenic content in the higher arsenic concentrate via the production of a Calcine. The results of these trade-off studies will determine the final product for inclusion in the feasibility study.

20. Environmental Studies, Permitting and Social or Community Impact

20.1 Environmental

The Project is subject to Serbian environmental legislation covering environmental protection, environmental impact assessment (EIA), water, air quality, noise, waste management, biodiversity and cultural heritage. Serbia is an accession state to the European Union (EU), and as such, it is working to harmonise its environmental legislation with that of the EU. The Project will adhere to Serbian, EU, and the International Finance Corporation (IFC) environmental and social standards. The Serbian EIA regulation differs from the international standards mainly in its requirements for stakeholder engagement, which are significantly more lenient.

The Project has contracted ERM, a UK-based environmental consulting company, to perform EIA activities. ERM is working with Envico, a Serbian environmental consulting company based in Belgrade.

Development of the EIA requires a project description that identifies the project activities during construction, operation, and closure/post-closure. The conceptual level Project design information available at this time has enabled EIA scoping, tentative definition and implementation of baseline data collection, and preliminary impact assessment. Progress on the EIA will continue through the feasibility study phase.

The PFS level engineering includes alternatives analyses to determine the optimum project configuration and design. The selection of preferred design alternatives has been made on the basis of environmental, technical and economic criteria. These alternatives and criteria will be documented in the EIA project description as part of the Feasibility Study process.

Based on the EIA scoping activities, a preliminary internal version of an EIA report has been prepared to identify the likely Project impacts and potential management measures. The intent was to provide early feedback to the Project's engineering team so that design alternatives could be analyzed with consideration of the environmental impacts and associated management alternatives. The objective is to develop a preferred environmental and social management strategy, and prepare and implement the associated environmental management plans.

A range of environmental considerations have been identified based on the preliminary environmental impact assessment. As is often the case with mining projects, water supply and water quality are the most significant environmental issues. Other issues with potentially significant environmental management implications include noise emissions during portal construction and the deterioration or destruction of habitat of listed species.

The Ministry of Environment ruling that the exploration decline development can proceed without an EIA was accompanied by a series of environmental management requirements as conditions of approval. Prior to starting construction activities at the exploration decline portal site, the Project will have environmental management plans in place with specific procedures established, and the resources to implement them, in order to comply with these permit conditions.

As part of the Serbian spatial planning permitting process, the author of the new spatial plan the Institute of Architecture and Urbanism (IAUS), will conduct a separate Strategic Environmental Assessment (SEA) for the area covering the Spatial Plan. The SEA is a broad assessment of the environmental impact of the Project on the spatial plan area.

20.2 Social

The social baseline for impact assessment purposes has been established from studies conducted in Bor Municipality, including five settlements surrounding the Project site: Slatina, Brestovać, Metovnića, Ostrelj, and Sarbanovać. The studies were undertaken in October 2015 and February 2017. Data were collected through interviews with individuals who had specific interest or knowledge of the study area such as local healthcare professionals and government officials. Focus group discussions were undertaken with groups of men and women separately in all settlements, as well as separate discussions with farmers, fishermen and beekeepers. The social baseline covers the following specific topics: demographics; gender equality; education; health; economy; employment; working conditions; land ownership; ecosystem services; traffic; and transportation infrastructure.

A preliminary social impact assessment has been performed. The significance of potential project effects has been evaluated for a range of social impacts, and recommended management measures have been identified to mitigate negative impacts and enhance positive ones. The following specific types of potential social impacts have been evaluated:

- Physical and economic displacement from Project land acquisition.
- Loss of priority ecosystem services (mainly water, fish and woodland).
- Economic growth.
- Local employment and skills enhancement.
- Local procurement.
- Loss of employment and business opportunities upon construction completion and mine closure.
- Population influx and change to demographics.
- Working conditions.

- Occupational health and safety.
- Community grievance over unmet expectations.
- Vehicular traffic congestion.
- Traffic safety.
- Degradation of roads.
- Disturbance from dust and noise.
- Risk of accidents due to trespassing on site.
- Conflict between the community and security providers.

20.3 Permitting

20.3.1 Project Approvals/Permitting

The subject of this section is primarily the requirements for permit approvals and Project documentation for all the stages, from project conception to operating permit.

All capital projects in Serbia require Government approvals. Smaller projects are approved by local governing bodies (municipality). Larger projects and/or projects significant to the Republic (of Serbia) are approved by the appropriate Ministry having jurisdiction. For the Timok Project, the Ministry of Mining and Energy will have jurisdiction over the underground mine and all other surface mine related infrastructure that will be situated in the future mining field, and the Ministry of Infrastructure, Construction and Transportation will have jurisdiction over surface infrastructure that will be used as a mine support objects (administrative buildings, etc.), roads and other transport related infrastructure in the Project's proximity. After consultation with various international and domestic mining companies, international organizations such as World Bank, etc., Government of Serbia adopted new mining laws in December 2015. The new mining law provides improvements in protecting private investors in the mining sector in Serbia and easier creation of modern mining projects.

20.3.1.1 Current Status of Permits at Timok

At the current time Rakita is operating on the Timok Project under the auspices of the Brestovać-Metovnica exploration licence. This exploration permit is valid until 19/04/2020 with the ability to extend the exploration permit for two subsequent periods of three and two years respectively (Article 38. Law on Mining and Geology). As part of the exploration program Rakita has planned to complete 100,000 m of exploration drilling during the three-year period of the license with approximately 87,000 m already drilled. Mining law requires at least 75% completion during the three-year period of the license. The Ministry of Mining and Energy have granted to Rakita on February 27th, 2018 an Annex of the geology exploration program to include the construction of a defined "mining object"; an exploration decline. The exploration decline will be used to access the ore body and take bulk samples necessary for geotechnical, metallurgy and other testing as well as underground drilling. The Annex of the exploration program covers the construction of a portal complex and the completion of twin 5m*5m exploration declines, 2.6 km long, to be driven down dip at 14 degrees, plus three tunnel cross cuts, 220 m, 110 m, and 70 m in lengths to access the orebody for exploration purposes.

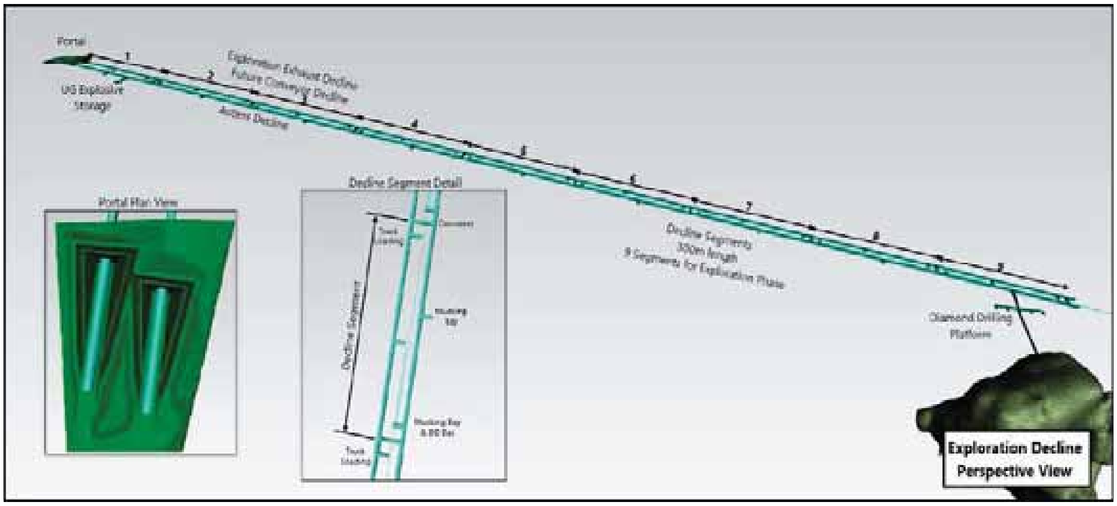


Figure 20.1: Exploration Decline

The Decline complex will allow the taking of a bulk sample from the orebody and will eventually form the primary access and conveyor hoisting facility for the underground mine when developed.

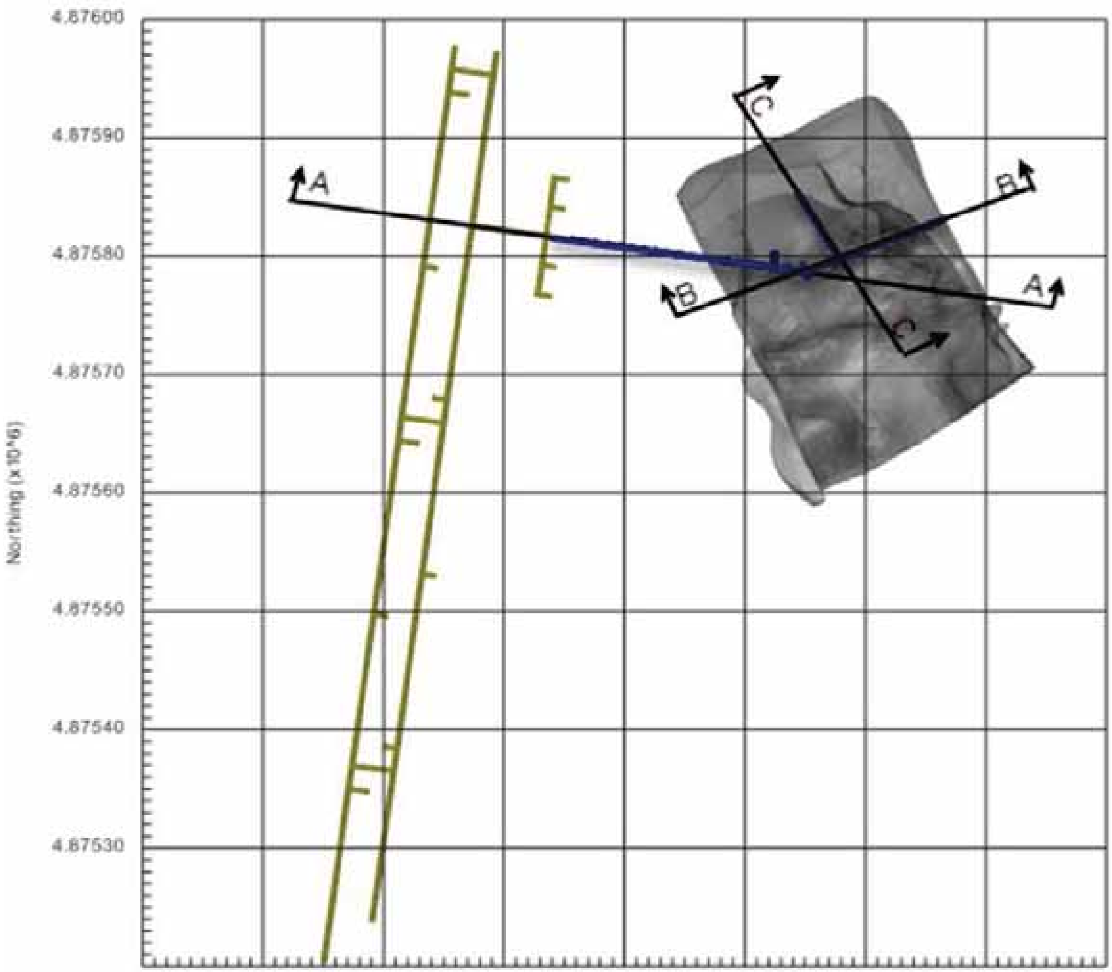


Figure 20.2: Exploration Decline (Plan View)

Completion of these two declines and remaining access tunnels is expected to take 24 - 28 months, with planned start of preparatory work and construction in Q2 2018.

20.3.1.2 *Procedure to Obtain all Permits Required for Development and Operation of the Timok Project*

Figure 20.3 shows the current position of the Permit Application procedure for the development and operation of a mine at Timok. The diagram attempts to simplify this process; however, a more detailed description of the permitting process is outlined below.

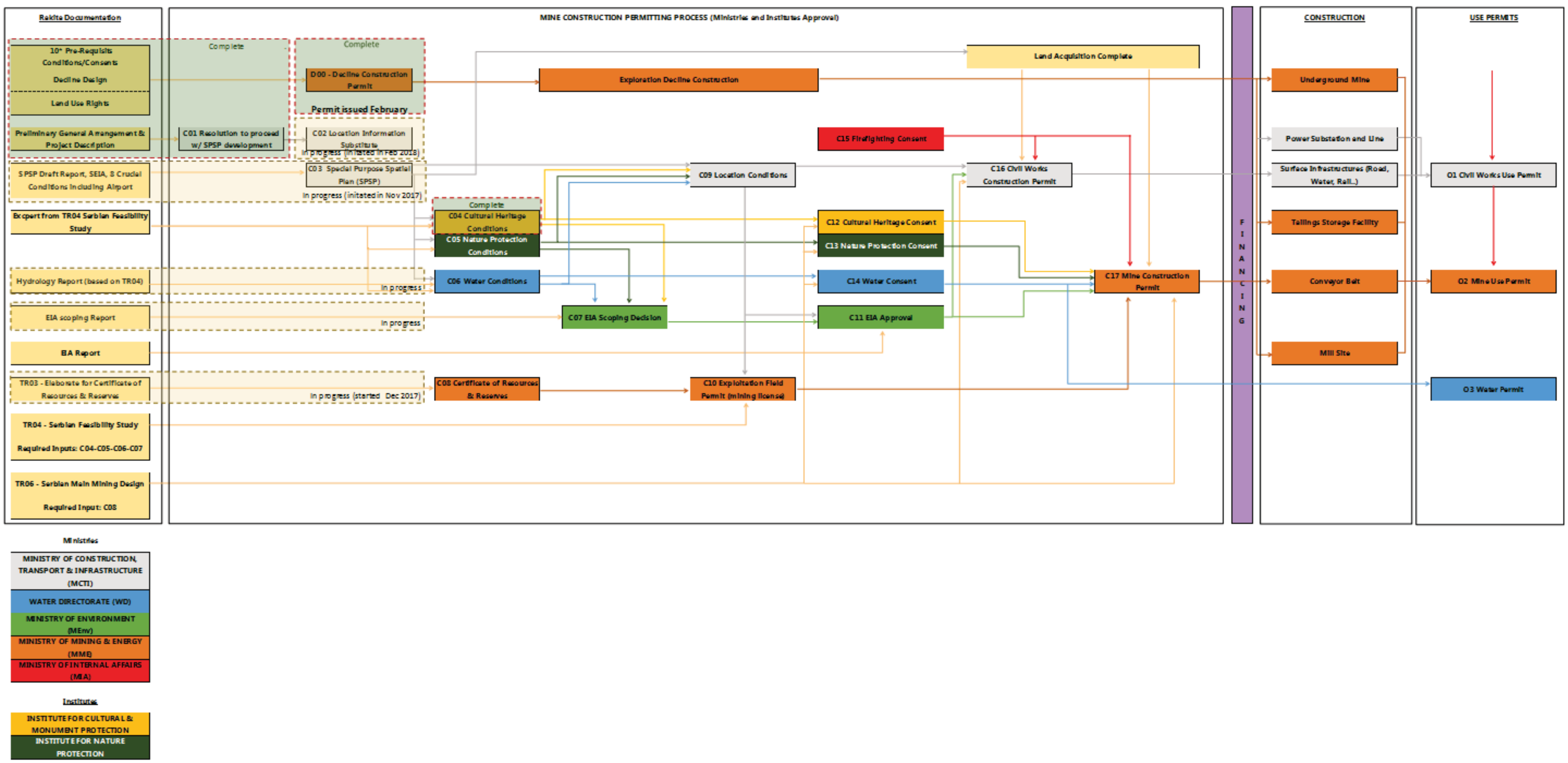


Figure 20.3: Timok Permitting Application Procedure

The procedures for project approval under the various Ministries differ, but are summarized as follows:

20.3.1.3 Ministry of Infrastructure, Construction and Transportation

The Ministry of Infrastructure, Construction and Transportation (MICT) will require the preparation on a number of documents and completion of a number of activities including:

1) **Spatial Plans.** In Serbia, there are three levels of spatial plans: State, Regional and Local. Spatial plans govern special and urban planning for the area in the order of importance starting from the state spatial plan. In order to designate the area for special purpose, such as mine exploitation field, a company can request that MICT creates Special Purpose Spatial Plan (SPSP). Once the SPSP is adopted by the Government of the Republic of Serbia at the recommendation of MICT, it becomes the governing Special Plan for the area. SPSP's cover both public and private land and will result in changed conditions and purpose for the land resulting in its subsequent designation for future industrial use. This allows the Company to change location conditions for the land from agricultural, construction and forestry to an industrial (mine exploitation) purpose. Location Conditions are an important document, issued by local or state authorities in charge of spatial planning, that is required during the application for construction and mine field permits.

   At the request of Rakita in Q2 of 2017, MICT have issued an ordinance that approves the creation of an SPSP for a future mine exploitation field 'Čukaru Peki'. The process of creation of the SPSP is currently under way, with MICT appointing Belgrade's Institute for Architecture and Urbanism (IAUS) as the author of the plan.

   During initial public hearings only Bor Municipality and local Boy Scouts of Serbia chapter have submitted comments and conditions for the creation of the SPSP. After the initial public hearing, MICT have asked for comments and conditions from 45 different stakeholders such as government agencies, ministries, public institutes, state and private companies. So far 42 stakeholders have responded with only eight conditions considered to require the response mainly involving large infrastructure (Bor sports airport, roads, power transmission, etc.) that is included in the SPSP's zone.

   Part of the SPSP is the development of a Strategic Environmental Impact Assessment (SEIA) of the Project on the SPSP area. This is a lower level study than EIA and it focuses on the general impact that the Project will have on the environment. IAUS, as the author of the plan, and ENVICO, Rakita's environmental permitting consultant, is currently working on the creation of the SEIA. The SEIA is expected to be completed by end of Q1 2018.

2) **SPSP purpose.** Once adopted by the government, the SPSP will allow the company to apply for necessary construction and mining permits within the SPSP zone. Within the

zone, land will be designated for the use for the Project. The SPSP will make possible to convert parcels within the zone (after the payment of necessary taxes) into land for a different purpose, i.e. converting agriculture land to land for industrial purpose. In order to have the SPSP approved the Company will need to get approval from all necessary utility and infrastructure (roads, etc.) maintenance companies for conditions to connect the project with their networks.

3) **Land Purchase.** The adoption of an SPSP is expected to require the resolution of the property rights regarding the Bor Airport. Additionally, in order to obtain a permit for the construction of a mine issued by Ministry of Mining and Energy, Rakita will require proof of the right of ownership, or the right to use the land, or to have a lease agreement and/or consent, or the servitude rights obtained for the land on which the construction of mining facilities and mining works will happen in advance of development; except in the case of exploitation of reserves of mineral raw materials that are of strategic importance for the Republic of Serbia, for which a special act of the Government on the determination of the public interest for a period of five years of exploitation is needed.

20.3.1.4 Ministry of Mining and Energy (MME)

The Ministry of Mining and Energy (MME) will require the preparation of a number of documents and completion of a number of activities including:

- **Exploration Permit.** The Project is currently operating under an Exploration permit granted on 19/04/2017 that can be extended for two more periods of three and two years respectively. This permit allows for the drilling of exploration holes, testing and building of mine objects as part of the exploration program (exploration decline complex) and carrying out of exploration activities and other related tasks.
- **Exploration Decline Complex.** The decline complex will be constructed under an Annex to the current exploration license as this development is for exploration purposes only. The permit for the decline construction has been received by Rakita on February 27, 2018.

- **Elaborate and State Certificate of Resources and Reserves. (SCORR).** The Elaborat is the highest geological document in Serbia, where a company presents to the State Commission the results of its exploration program. Once the Elaborat is approved, the State Commission recommends to the government the issuance of the SCORR. This SCORR is then an integral part of the application for the Exploitation Field Permit as well as to estimate State's amount of royalties that will be collected in the future. It also forms the basis for the Serbian Feasibility Study and for the Exploitation Field license submission, along with a number of other studies. Rakita is currently involved in creation of an Elaborat to support the SCORR application for the whole Project area and including both the Upper Zone and the Lower Zone orebodies. The Elaborat submission is expected to be completed in mid-2018.
- **Approval for Exploitation Field Permit (Mining License)** is one of the most important permits in any mining project and is required when the project owner had finalized geological survey, declared mineral resources and reserves, and entered the exploitation stage. The application submittal includes the SCORR and other documents that are not related to engineering (deliverables), including the Serbian Feasibility Study (packaged per Serbian regulatory requirements). The Approval for Exploitation Field/Exploitation is valid for two years during which the Permit for Construction of Mine Objects is obtained.
- **Mine Construction Permit** submittal includes the Main Mining Project, that is intended for construction, together with other defined "mining projects" and includes detailed design, plus a review of the mine development technology and method. All mining projects are subject to Technical Review by an independent third party (same analogy as the project for construction permit under the Ministry of Infrastructure). Submittal for a Mine Construction Permit also includes the approvals of a comprehensive EIA study including water, cultural heritage and also a separate report on fire protection.
- **Underground Permit to Operate** on completion of construction is also conditional on passing a technical inspection of the completed facility or its area. Technical Examination of a mine object will determine if the mine is ready for use. MME organizes Technical Examination from the pool of Serbian licensed companies, and Nevsun/Rakita bears the cost of these examinations.

20.3.1.5 Permitting Strategy

Rakita's strategy is to work with the various stakeholders in Serbia on determining the appropriate jurisdictions and the exact requirements for each permitting step, in terms of licensing requirements and, the content of submittal packages. It is expected that through constant and transparent communication with authorities, regulatory bodies and other stakeholders overall permitting process timing will be optimized.

20.3.2 Licensing Requirements – Engineering and Construction Companies

Engineering design for capital investments, as well as construction management is regulated in Serbia, i.e. the engineering company or individual requires a specific license for each discipline, and further to that, licenses need to be specific for the industry field and the type of facility being engineered and built. These licenses are regulated separately by two Ministries, and are not interchangeable. Licenses are issued by each appropriate engineering chamber and for example those individuals that pass the mining regulations would earn mining engineering licenses.

20.3.2.1 Licenses under the authority of Ministry of Infrastructure, Construction and Transportation (MICT)

License requirements apply to individuals (individual licenses) and to the companies (also known as ‘great licenses’) engaged in engineering and construction.

Each individual engineering discipline (mechanical, electrical, civil and structural, process, metallurgy, architectural, geotechnical, transportation infrastructure etc.) has a list of licenses, properly numbered (300’s) and with narrowly defined scope permitted under a particular license. The regulations also define which University diplomas, together with appropriate work experience are required for each license.

Likewise, each individual’s construction license (same disciplines as engineering) narrowly defines the scope of facility construction permitted under each license. They are numbered in 400’s series, and are similarly (although not exactly) defined as engineering licenses.

The individual engineering and construction licenses are issued by the Chamber of Engineers, under the jurisdiction of Ministry of Infrastructure and Construction. An individual candidate has to pass a state board exam in order to qualify.

Company licenses (great licenses) are issued directly by the Ministry of Infrastructure and Construction. As for individual licenses, company licenses are split into engineering and construction licenses. The numbering system used differs from the system used for individual licenses. The register is first divided by type of facility: for example, licenses for dams and accumulations (tailings storage would likely require this type of license), licenses for the nuclear industry, licenses for the chemical industry and metallurgy, licenses for thermal power plants, etc. Each of these groups is further divided by discipline or specialty basis; (for example, thermal power plant engineering is split into a number of licenses –structures, electrical installations, automation etc.).

20.3.2.2 Licenses Under the Authority of Ministry of Mining and Energy

The licensing related to mine facilities is regulated under a separate jurisdiction (Ministry of Mining and Energy). These licenses are mandatory and defined for the individual design engineers responsible for each part of the specific mining designs, and for the engineering companies involved. The corresponding Law on Mining and Exploration mentions that adequate experience and licenses are required for mining projects. The Union of Engineers and Technicians is issuing license for engineers of different categories that are allowed to design projects for mining purposes. The required education and years of experience for persons supervising or operating mines are defined, along with the qualifying ‘state exam and a requirement to speak fluent Serbian.

20.3.3 Engineering Resources

Considering the specific requirements in Serbia for approvals of technical documentation and designs at various stages, it is imperative to optimize the plan for engineering production, to satisfy all requirements, specifically local requirements in Serbia, as well as meeting the content and form that are standard in mature mining jurisdictions (Northern Europe, North America, Australia, etc.). It is therefore important that local engineering resources must be used at all stages of the final mine and mill design, including licensed Serbian engineers, and licensed Serbian engineering companies.

Under the expectation that both Ministries have jurisdiction over the parts of the Project to be engineered and constructed (to be confirmed), there will be Project packages required for permitting and approvals during the Project External Feasibility stage including, but not exclusively:

- A Serbian FS for Exploitation Permit.
- A conceptual solution package For Location Conditions and Conceptual Project for revision.

Considering that these are not as extensive as the packages that follow, (in particular the Project for Construction Permit of the surface facilities and the Main Mining Project), participation of Serbian licensed engineers and Serbian Licensed Engineering entities will be limited to setting the design criteria for engineering, in terms of the mandated technical standards and regulations, and to preparing the packages for permits submissions.

The role of Serbian licensed engineers will increase significantly during the detailed engineering and execution phase, and in that regard a more complex approach is required, including an engineering team with embedded Serbian engineers and, or Serbian licensed entities, working together with the EPCM team from the start. The exact assignment of responsibilities needs to be carefully planned during the external FS process.

20.3.3.1 Environmental Impact Studies

The Project is subject to Serbian environmental legislation covering environmental protection, environmental impact assessment (EIA), water, air quality, noise, waste management, biodiversity and cultural heritage. Serbia is an accession state to the European Union, and as such, it is working to harmonise its environmental legislation with that of the European Union.

The Project will adhere to Serbian, European Union, and International Finance Corporation environmental and social standards. The Serbian EIA regulation differs from international standards mainly in its requirements for stakeholder engagement, which in Serbia are generally more lenient. The Project will follow the more stringent international stakeholder engagement standard, wherever this is the case.

The basic engineering design now underway as part of the external PFS and FS process, includes alternatives analyses to determine the optimum project configuration and design. The selection of preferred design alternatives will be made on the basis of environmental, technical, social and economic criteria, and documented in the EIA project description.

A recent Ministry of Environment ruling that the exploration decline development can proceed without an EIA was accompanied by a series of environmental management requirements for the decline development, as conditions of approval. Prior to starting construction activities at the exploration decline portal site, the Project should have environmental management plans in place with specific procedures established, and the resources to implement them, in order to comply with these permit conditions.

As is often the case with mining projects, for the exploration decline development water supply, water disposal and water quality are the most significant environmental issues for the decline site and Rakita has designed facilities to deal with both Contact and Non-Contact water prior to discharge to the receiving environment. Other issues with potentially significant environmental management implications include noise emissions during portal construction and the potential for deterioration or destruction of habitat of listed species.

As part of the main Project permitting process, Rakita will be expected to complete an independent Environmental Impact Assessment and further ground water studies. In addition, Rakita is expected to produce an Environmental Protection Plan, focusing on the potential effects of the Project on the land, air and water and fauna and flora in the Project area and the net effect of the Project on the surrounding populace. To this end, Rakita engaged a number of internationally recognized outside consultants, supported by ERM (UK) ltd, Envico in Serbia, the University of Belgrade and other specialist consultants both in Serbia and overseas, to carry out an extensive series of baseline studies.

These studies started in 2015 and are currently ongoing and include items such as:

- Baseline Surface water monitoring throughout the Project area, covering both quality and quantity.
- Baseline Ground water monitoring throughout the Project area covering both quality and quantity.
- Humidity cell test work to determine the quality of ground water during mine operations and post mine closure.

- Air quality monitoring around the Project site.
- Surveys of Fauna and Flora over areas to be disturbed by the Project.
- Soil sampling.
- Noise pollution monitoring.
- Community and Social Impact Studies.

These studies are ongoing and will continue through much of the development and subsequent production period.

20.3.3.1.1 Social Impact

The social baseline for impact assessment purposes has been established from studies conducted in Bor Municipality, including five settlements surrounding the Project site. The social baseline covers the following specific topics: demographics, gender equality, education, health, economy, employment, working conditions, land ownership, ecosystem services, traffic, and transportation infrastructure.

The Project has conducted a preliminary social impact assessment. Potential Project effects have been evaluated for a range of social impacts, and recommended management measures have been identified to mitigate negative impacts and enhance positive ones. This work is ongoing.

20.3.3.1.2 Closure Plan

The purpose of a closure plan is to transition the planned Project site from an industrial mining operation to a post-closure state that is acceptable to local property owners and communities for the long term. For PFS level closure planning purposes, closure design requirements are categorized in three main categories: physical stability, chemical stability and social acceptance.

These three categories cover aspects of safety, environmental performance and matters of interest to local communities. The following provides a general sense of how these categories are applied to the current closure design, which focuses on physical and chemical stability:

- Physical Stability: Includes closure designs that focus on the geotechnical stability of facilities and surfaces prone to wind and water erosion or surfaces that would cause adverse effects if they were to erode or be transported by the wind (e.g. tailings and contaminated wastes). Physical stability also refers to removal or repurposing of infrastructure remaining after the mine closes (e.g. buildings, roads, tailings storage facilities and dams).
- Chemical Stability: Includes closure designs that focus on limiting risks to the environment and human health and safety by addressing long-term containment of mine

wastes (e.g. tailings, waste rock, by-products from concentration and water treatment plant sludge) and limiting leaching from these wastes to surface and ground waters.

- Social Acceptance: addresses matters of interest or concern for local communities (e.g. the expected post closure land use; long-term socio-economic benefits (e.g. employment, transfer of ownership of assets or infrastructure that can be of benefit to the community).

The current Project design within the PFS (PFS, 2018), aims to proactively design for closure of the Project and to meet the standards of Serbia in regard to stability and harm mitigation post closure. This project design continues to be refined as more information becomes available.

20.4 Land Acquisition

Rakita is engaged in a land purchase program to acquire properties expected to be adversely impacted by the Project. This program is being undertaken in parallel with the intention to purchase private and State-owned lands which are currently classified for use as Agricultural, Forestry, or Construction. In relation to private lands, Rakita is conducting a willing buyer/willing seller land acquisition process. The first step in this process is to identify each individual plot plan that is expected to be disturbed, including location which includes dimensions thereof. Once property ownership is confirmed, a socio-economic survey of the related stakeholder(s) of the property is conducted. The survey includes the identification of potentially vulnerable individuals, people/families who may need to be resettled, existing buildings, structures, and improvements. Properties are then evaluated by independent, licensed evaluators, and an offer is presented to the applicable property owner(s). To date, Rakita has evaluated all private properties included in the Project footprint, and have provided offers to 98% of the identified property owners. As of February 28, 2018, Rakita owns approximately 43% of the private lands within the Project footprint which are expected to be required to construct the operational mine.

20.5 Closure Planning

20.5.1 Introduction

Closure planning at a conceptual level provides the basis for further development of the decommissioning and rehabilitation strategies over the life of the Project. Closure planning in Serbia is regulated under the *Law on Mining and Geological Explorations* (Official Gazette of RS, No. 101/15) entered into force in December 2016. Abandoned mines, temporary suspension, planned permanent suspension, and rehabilitation and reclamation of the mine operations are addressed in Articles 146 through 153 of the *Law on Mining and Geological Explorations*, however environmental protection is not regulated in detail, and reference is made to the *Law on Environmental Protection* (Official Gazette of RS, No. 43/2011). The *Law on Environmental Protection* regulates environmental protection through the provision of limitations and monitoring requirements. The CCP also aligns with international standards and guidelines, such as those of the European Union, International Financial Institutions (IFIs), International Finance Corporation (IFC), Cultural Heritage Conventions, and ICMM, as applicable.

20.5.2 Conceptual Closure Plan

A Conceptual Closure Plan (CCP) was developed during the PFS to provide preliminary closure methodologies and activities to address the physical stability, chemical stability, and the protection of the natural and social environment in which the Project exists. A comprehensive CCP (CCCP) will be developed during FS using information obtained from the EIA consultants (ongoing natural and social environmental studies), and the FS engineering team (FS studies) to complement the Project definition developed.

The CCP is based on the current Project schedule which identifies a LoM (or operations phase) of approximately 10-12 years, followed by a 2-year closure phase, and a subsequent post-closure/monitoring phase.

The primary focus of the CCP is final closure, although implementation of closure activities may occur at various times throughout the life of a Project to either reduce the cost of closure at end of LoM (progressive reclamation) or as a result of economic circumstances. The closure activities related to progressive reclamation, care and maintenance, and early closure are also briefly discussed in the CCP.

The objectives of closure planning identified in the CCP include:

- Protection of the health and safety of employees, contractors, community neighbors, and the public.
- Protect the natural environment during closure.
- Achieve physical stability of the Project domains.
- Return the site to a condition that generally conforms to the surrounding terrain and does not impose safety concerns to the public.

The Project was divided into geographical domains (Mine, Process Plant, TSF) to assist in establishing the rehabilitation methodologies. General closure measures identified in the CCP include, but are not limited to:

- Decommissioning, decontamination, and removal of mobile equipment, machinery, surface building and associated infrastructure and foundations.
- Utilities, such as electrical power, ventilation, lighting and mine safety equipment, will be maintained as long as necessary for safe demobilization.

- Collection, safe storage, and removal of all waste (hazardous and non-hazardous) off-site in a timely manner.
- Removal of culverts and bridges will be removed.
- Grading of areas to match, as close as possible, the surrounding landscape.
- Revegetation of the areas to meet land use end goals, as yet to be determined.

The mine is generally the first Project facility to enter the closure phase immediately following the removal of the last ore from the underground mine with closure of the remaining facilities generally following the movement of the last ore through the process. Domain specific closure measures include:

20.5.2.1 Mine Site (Portal and Subsidence Areas)

It is anticipated that an area of subsidence will occur above the mine site and thus access will be controlled through fencing and signage to this area, and the mine portal area. The declines and box cuts will be closed, portals capped and sealed; and all hazardous material collected, stored and shipped off-site. Closure of the remaining facilities generally follows the movement of the last ore through the processing facilities.

20.5.2.2 Process Plant

Diesel fuel services at the Process Plant will be required for most of the active closure phase, and will be one of the last areas decommissioned. The main electrical power supply will remain in use during the decommissioning phase of closure as long as the power demand warrants. The water treatment plant will be retained for the majority of the active decommissioning phase (2 years). All site ponds will be decommissioned and backfilled.

20.5.2.3 TSF

Active closure of the TSF is anticipated to occur over the initial 2-year period following cessation of mine operations. A cover system will be installed following the dewatering of the TSF and consist of a Low-Density Polyethylene (LDPE) geomembrane liner and composite soil cover system. The cover system is designed to create a landform, which is vegetated and graded to drain surface runoff towards the closure channels. Embankment material above the tailings level will be removed and will be used as non-acid generating cover material to develop the final TSF landform. No long-term maintenance is expected once the landform is established.

20.5.3 Stakeholder Engagement

The potential may exist for the ownership of a number of Project facilities and infrastructure to be transferred, or relinquished, to the local communities, or government stakeholders, should an end use be determined through stakeholder consultation. The Project facilities and infrastructure to be relinquished at closure have not been identified at this time. Stakeholder consultation will continue throughout the Project planning and operations phases to identify any potential future uses and the CCP updated, as required.

20.5.4 Environmental Monitoring and Maintenance

A number of facilities and structures will be retained during active and post-closure (or a portion thereof) to enable monitoring or inspection/maintenance activities to be undertaken. An environmental monitoring and maintenance program is a key component in evaluating the performance of mine and process facility closure activities, and is conducted to assess the physical, chemical, and biological stability of the rehabilitated mine landforms and, where necessary, proactively identify areas where maintenance is required. The environmental monitoring and maintenance program is intended to:

- Confirm whether the site completion criteria have been achieved.
- Ensure the closure activities are progressing satisfactorily towards meeting these criteria and attaining close out status.

Final site decommissioning will only take place following the complete consolidation of the tailings.

20.5.5 Conceptual Closure Cost Estimate

The closure cost estimate was developed using a site-specific costing model developed by Hatch. The costing model was developed using Microsoft Excel with a comprehensive work item list that addresses all Project-related activity areas and infrastructure related to the Project. The closure cost model is made up of a detailed direct cost estimate for each of the reclamation activities identified for each project component in addition to monitoring and post-closure costs and indirect costs.

The total estimated cost to close the Project is currently estimated at $48M.

21. Capital and Operating Costs

21.1 Capital Cost Estimate

The purpose of this section is to outline the methodology by which the capital cost estimate (CAPEX) was developed for Nevsun's Timok Project.

The CAPEX was developed by a team of engineers and cost estimators from SRK (mine development), Knight Piésold (tailings and waste rock storage facilities), Hatch (mine infrastructure, concentrator and surface infrastructure) and Nevsun (Owner's costs). The team worked together to coordinate their individual cost estimates and then reviewed the integrated document.

Hatch had the overall responsibility for integrating the work of the individual companies into a single, comprehensive and consistent capital cost estimate. Hatch was not responsible for work completed by others.

21.1.1 Estimate Summary

The Project's capital costs are divided into two main categories:

- Pre Sanction Date expenditures (US$144 million)
- Post Sanction Date expenditures (US$574 million)

The "Sanction Date" is defined as Q3 2020. This is the date before which the Owner will complete exploration work, including the exploration decline complex, as well as acquire all land, required permits and will obtain financing to allow construction of the mine, surface facilities and infrastructure to proceed to completion without constraint.

A summary of the capital cost estimate according to major work breakdown structure (WBS) is presented in Table 21.1. Summaries of direct costs and indirect costs for the post Sanction Date expenditures are presented in Table 21.2 and Table 21.3 respectively.

Table 21.1: Level 1 CAPEX Summary

WBS	Description	Estimated Cost (USD)
Pre Sanction Date Expenditures		
	Owner's Project Development Costs	70,928,000
	Decline Development (Under Exploration License)	42,962,000
	Total Pre Sanction Date Expenditures	**113,890,000**
Post Sanction Date Expenditures		
	Direct Costs	
1000	Site Development	49,142,000
2000	Mining	159,798,000
3000	Concentrator	99,109,000
4000	Pre-Production Operating & Maintenance	17,194,000
5000	Tailings, Waste Rock and Reclaim Water Management	46,866,000
	Subtotal Direct Costs	**372,109,000**
9000	**Indirect Costs**	**105,882,000**
	Contingency	**95,598,000**
	Total Post Sanction Date Expenditures	**573,589,000**
Total Pre-Operational Expenditures		**687,479,000**

Table 21.2: Post Sanction Date Direct Cost Summary

WBS	Description	Estimated Cost (USD)
1000	**Site Development**	**49,142,000**
1100	Site Preparation	5,517,000
1200	Site Infrastructure and Facilities	14,732,000
1300	Site Utilities	13,374,000
1400	Off-site Infrastructure and Facilities	11,497,000
1500	Off-site Utilities	4,022,000
2000	**Mining**	**159,798,000**
2100	Development	70,353,000
2200	Ventilation Systems	14,226,000
2300	ROM Ore & Waste Handling	2,644,000
2400	Crushing and Conveying	15,154,000

WBS	Description	Estimated Cost (USD)
2500	Mobile Equipment	30,040,000
2600	Power Distribution and Communications	15,126,000
2700	Dewatering	4,242,000
2800	Service Water Distribution	4,926,000
2900	Mine Infrastructure	3,087,000
3000	**Concentrator**	**99,109,000**
3100	Crushed Ore Storage	7,148,000
3200	Grinding	18,345,000
3300	Flotation	24,799,000
3400	Concentrate Thickening and Filtration	8,994,000
3800	Process Building and Utilities	39,823,000
4000	**Pre-Production Operating & Maintenance**	**17,194,000**
5000	**Tailings, Waste Rock and Reclaim Water Management**	**46,866,000**
5100	Site Preparation	3,763,000
5200	Tailings Storage Facility	34,583,000
5300	Major Water Diversion Structures	1,003,000
5400	Waste Rock Storage	6,877,000
5500	Haul Road (From Hwy 37 to TSF)	640,000
	Total Post Sanction Date Direct Costs	**372,109,000**

Table 21.3: Post Sanction Date Indirect Cost Summary

WBS	Description	Estimated Cost (USD)
9100	Camp and Catering	1,867,000
9200	Site Indirects (Temporary Facilities and Services)	7,632,000
9300	Freight	11,010,000
9400	Spare Parts	4,245,000
9500	EPCM Cost	37,275,000
9600	Third Party Consultants	1,558,000
9700	Pre-Operational Testing	5,664,000
9900	Owner Execution Costs	36,631,000
	Total Post Sanction Date Indirect Costs	**105,882,000**

21.1.2 General

21.1.2.1 Estimate Class

The CAPEX was prepared in accordance with guidelines established by the Association for the Advancement of Cost Engineering (AACE) for a Class 4 (equipment factored) estimate. The anticipated level of accuracy is -20% to +25%.

An equipment factored estimate is based on limited information. The level of Project definition is generally between 1% - 5% of engineering with preliminary designs, configurations, equipment sizing and facility locations established. The estimating method is generally based on using mechanical supply costs and factoring the remainder of the estimate. Use of the equipment factored estimate is limited to the part of the plant in which the equipment forms a major part of the direct field costs. It applies to facilities where the costs of installation and bulk materials relative to the cost of major equipment are reasonably predictable as a factor of major equipment supply cost.

Equipment factored estimating is not applicable to facilities such as the mine, mine infrastructure, tailings storage, etc. Estimates for these areas are based on preliminary quantity take-offs or factoring based on some other design parameter.

21.1.2.2 Estimate Coding

Outlined below are the main coding structures that were used in developing the PFS capital cost estimate.

21.1.2.2.1 Work Breakdown Structure (WBS)

A WBS is a project-oriented hierarchy of geographic work areas.

The WBS is incorporated into various Project documents including equipment numbering, drawings, requests for quotation, project execution planning/scheduling, the CAPEX, etc. It's a key device used to integrate the Project deliverables.

21.1.2.2.2 Commodity Codes

Commodity Codes are used to collect the estimate items into groups of a similar nature or discipline, and are summarized in Table 21.4.

The standard Commodity Code is an alpha character which is directly aligned with the Project standard discipline descriptions. The Commodity Code is defined as the first character of the "Price Code" and indicates the discipline to which the Price Code belongs.

Table 21.4: Commodity Codes

Commodity Code	Description
A	Site Development
B	Mining
C	Concrete
D	Roadwork
E	Earthworks
F	Architectural and Pre-Engineered Buildings
J	Control & Instrumentation
L	Electrical Equipment, Cable Ladders, Wiring
M	Mechanical Equipment
N	Mechanical Plate Work & Tanks
O	Mobile Equipment
P	Pipe Work & Fittings
Q	Refractory & Insulation
R	Cable Ladder, Tray & Conduit
S	Structural Steel
V	Owner’s Cost
W	Wire & Cable
X	Multi-Discipline
Y	Indirect
Z	Contingency

21.1.2.2.3 Resource Codes

Resource Codes are used to collect the resources based estimate items into groups, and are summarized in Table 21.5.

Table 21.5: Resource Codes

Resource Code	Description
L	Labour
M	Material
E	Equipment
S	Subcontract
Y	Indirect

21.1.2.3 Estimating Software

Estimating software was utilized as per company preferences for specific areas of the Project. The estimate information was transferred onto an Excel template and then integrated into a single CAPEX file.

The estimating software that was used by each company is listed below in Table 21.6.

Table 21.6: Estimate Software

Company	Area Description	Estimating Software
SRK	Mine	Excel
Knight Piésold	Waste Storage and Management	Excel
Hatch	Mine infrastructure	Hard Dollar
	Process Plant and Surface Infrastructure	Excel
Nevsun	Owner’s Costs	Excel

21.1.2.4 Terminology

The capital cost estimate consists of four major cost groupings: Direct Costs, Indirect Costs, Contingency and Owner’s Costs, as described below. The definitions provided in the following subsections are generic. Specific requirements for the Timok CAPEX are described in Sections 21.1.3, 21.1.4 and 21.1.5.

21.1.2.4.1 Direct Costs

Direct costs are the costs of all equipment and bulk materials, together with construction and installation costs for all permanent facilities. Examples of direct costs include, but are not limited to the following:

- Supply, assembly and installation of permanent equipment.
- Supply, fabrication and installation of bulk materials.
- Supplemental resources for equipment and bulk material installation, such as labour and construction equipment.
- Site preparations (bulk earthworks) and the construction of roads and storm water ditching.
- Supply, fabrication and erection of permanent buildings and associated services.
- Supply, fabrication, erection of utilities and distribution systems.
- Process control systems including software programming and DCS/HMI configuration costs.
- Pre-production costs.

- Contractor's distributable costs such as mobilization and demobilization, overheads and profit, supervision, general construction equipment including construction cranes, small tools and consumables used in construction, etc.

21.1.2.4.2 Indirect Costs

Indirect costs include the following:

- Any applicable temporary construction facilities including temporary worker lodgings/services, secure lay-down areas, warehouses, etc.
- Temporary construction services including IT, catering, camp and office cleaning services, worker transportation to the job site, etc.
- Fuel, electrical energy and water required for construction or pre-operational testing.
- Freight and logistics.
- Vendor representatives.
- First fills of materials such as transformer oil, lubricants and other items that are not consumed by the process.
- Start-up/commissioning spares, capital spares and two-year's operating spares
- Engineering, procurement and construction management services (including travel expenses).
- Third party engineering and other services.
- Pre-operational testing services, including associated materials.

21.1.2.4.3 Contingency

Contingency included in the capital cost estimate is an allowance for normal and expected items of work which have to be performed within the defined scope of work and project execution plan as covered by the CAPEX, but which could not be explicitly foreseen or described at the time the estimate was completed. The contingency amount is an integral part of the cost estimate, and it should be assumed that contingency will be spent in completing the Project. Contingency does not cover potential scope changes, price escalation, currency fluctuations. Also, contingency does not include allowances for Project "event" risks such as labour unrest, blockades, adverse market conditions, force majeure, or any of the items that are specifically excluded from the CAPEX (see Section 21.1.3.2).

21.1.2.4.4 Owner's Costs

Owner's costs include those tasks that will be managed directly by the Owner. A list of typical Owner's costs are listed below:

- Site security and first aid services.
- Owner's management, general and administrative (G&A) costs including statutory supervision of Contractors and others, required by Law – for example U/G mine management.
- Taxes and duties.
- Recruitment and training costs (for operations).
- Technology studies and further metallurgical test work.
- Exploration, drilling and assaying activities (excluding geotechnical).
- Environmental studies and reports.
- Third-party due diligence reviews and consulting services.
- Insurances.
- Land acquisition and right of way.
- In-process inventory.
- Working capital.
- Performance bond premiums.
- Environmental and ecological cost issues.
- Start-up, hot commissioning and production ramp up costs.
- Owner's contingency.

21.1.3 Qualifications, Terminology, Structure and Exclusions

21.1.3.1 Qualifications

21.1.3.1.1 Level of Project Definition

The estimate was prepared by mining, process and discipline engineers and cost estimators. Prefeasibility level documents used to generate the capital cost estimate include:

- Process Plant geotechnical conditions have been assumed since information is not available specifically in the proposed process plant location. Data from the mine portal and the TSF was considered for the assumed geotechnical conditions (to be confirmed during the feasibility study).
- Terrain (topographic) model: LiDAR survey data (with 1 m vertical contours) is available for the Project area.
- 3D model of the mine and mine infrastructure.

- 3D model of the tailings storage facilities.
- Process deliverables including metallurgical test work, block flow diagrams (BFDs), process flow diagrams (PFDs), process design criteria (PDC) and mass/energy balances.
- Preliminary site plans, area plot plans and general arrangement drawings of the process plant.
- Mechanical equipment list (MEL) including assumed electrical loads.
- Single line diagrams (SLD).
- Firm price quotes for the portal area site preparation, mine decline and budget quotations for the design/supply of major mechanical equipment.
- Material take-offs (MTOs) for mine development, mine infrastructure, site preparation and TSF bulk earthworks as well as process plant concrete, structural steel and architecture (i.e. building siding and roofing).
- Factored process plant utilities including electrical power distribution and controls/instrumentation.
- Unit pricing for bulk commodities (concrete, steel) and site labour based on in-house data for the Serbian market.
- Factored indirect cost estimates.
- An assigned contingency based on a review of the quality of the available information by the Project team.

21.1.3.1.2 Capital Project versus Operations Transition

The capital Project is considered to include tasks up until the process plant/concentrator is ready to receive and start processing ore. Mine operating costs incurred prior to this time will be capitalized and included in the CAPEX. After this time costs will be considered operating costs (OPEX) or sustaining capital costs (SUSEX).

21.1.3.1.3 Pricing

None of the pricing for commodities, the design/supply of equipment, or construction contracts are based on binding quotations. Firm price bids have been received for the preparation and construction of the twin decline portals and the driving of the initial exploration declines. Budget quotations were received from vendors for major equipment packages (refer to Section 21.1.4).

Equipment and applicable materials will need to be CE certified. The Hatch cost database is not based on CE cost data. The cost impact is not expected to be significant but this will need to be investigated during the feasibility phase of the Project.

21.1.3.1.4 Estimate Base Date, Currency and Foreign Exchange

The estimate base date is the first quarter of 2018 (Q1-2018). No provision was included for escalation beyond this date.

The CAPEX base currency is United States dollars (USD).

Costs submitted in other currencies were converted to USD according to the published exchange rates for Q1 2018 shown in Table 21.7. No provision was included for currency fluctuation or any fees applicable to currency exchange.

Table 21.7: Project Exchange Rates

Country	Currency	Code	Inverse	Per USD
USA	Dollar	USD	1.00	1.00
South Africa	Rand	ZAR	13	0.08
Serbia	Dinar	RSD	0.0098	101.83
European Union	Euro	EUR	1.18	0.85
Australia	Dollar	AUD	0.77	1.29
Canada	Dollar	CAD	0.78	1.29
China	Renminbi	CNY	0.15	6.62

21.1.3.2 Estimate Exclusions

The following items are excluded from the CAPEX:

- Sustaining capital costs and closure costs. The CAPEX is limited to costs to construct the facility up to the point of concentrator start-up. Sustaining capital costs, including costs for mine operation/development, tailings lifts, etc., that occur after the concentrator begins operation are excluded from this CAPEX, but were estimated and included in the PFS Final Report and economic analysis on a separate line.
- Operating costs
- Impacts of foreign currency exchange rate variations.
- Allowances for any changes to the scope of the Project.
- Allowances for either:
    - General project risks that could affect any project (such as variations in market conditions, that could affect equipment, commodities and/or labour costs, labour unrest, disputes with local residents including local indigenous groups, geotechnical or process related design issues, delays due to the late receipt of equipment or materials, poor performance by contractors, force majeure, etc.).
    - Risks that are specific to this Project.

- Allowance for the risks associated with the Serbian political, legal or regulatory environment, including:
  - The risk of changes to any laws, regulations, rules or policies in Serbia, or the governmental or judicial interpretation thereof.
  - The risk of Nevsun failing to comply with any such laws, regulations, rules or policies and the costs of any resulting penalties, fines, suits, etc.
  - The risk of Nevsun not being able to obtain or maintain any permits, licenses and other authorizations required for the Project.
- Costs associated with lost time due abnormal weather events.
- Working capital, sustaining capital and or facility closure costs.
- Operational spare parts.

21.1.4 Direct Costs

Generally, this section refers to post Sanction Date direct costs but some references to the decline direct costs (pre-Sanction Date) are included.

21.1.4.1 Site Development

- Bulk earthworks quantities were estimated based on the terrain model (existing site conditions) and the prepared site (including “benches” and storm water ponds) created in the 3D model for the process plant and other surface facilities. Unit costs are based on in-house data adjusted for Serbian market conditions.
- Quantities for site fencing were estimated from the site plan and unit costs ($/m) applied for supply and installation based on in-house data.
- Costs for the supply and installation of non-process buildings, such as administration buildings, gate/security houses, warehouses, laboratory, cafeteria, etc., are based on building area (footprint, m^2) as taken from the site plan. All-in unit costs ($\$/m^2$) were applied based on building type, costs obtained by Rakita for similar structures, and/or in-house data, adjusted as per Serbian market conditions.
- Costs for off-site and on-site roads are based on road surface area (extracted from the site plan) and an assumed unit cost ($\$/m^2$) based on in-house data.
- Costs for the power supply are based on the length of the power transmission lines and a unit cost ($/km) for the applicable voltage based on in-house data. The cost for the main substation (110 kV/35 kV) was estimated based on capacity (i.e. $/MW). The substation all-in unit cost is based on in-house data.

- Cost allowances for other utilities (water treatment, effluent treatment, fuel and oil storage, etc.) were generated based on capacity requirements and in-house unit cost data.
- Costs for raw and potable water supplies were estimated based on the estimated pipe sizes (based on flow requirements) and pipe lengths (from the site plan).
- Costs for concentrate handling equipment were based on a budget quote provided by Rotainer, with quantities adjusted to meet specific requirements of the Project. Allowances were included for rail and port trans-shipment facilities based on preliminary site assessments completed by Rakita personnel and sketches developed by Hatch.

21.1.4.2 Mine

It is assumed that initial mine development will be completed by a contractor. The contractor will continue with main access and infrastructure capital development and the owner crews will be responsible for mine operating development during production.

The contractor will use his own equipment fleet and personnel for capital development. Rakita will purchase their own mobile equipment for mine production, operating development and mine services. Rakita will purchase their own equipment and will use their own crews during production.

21.1.4.2.1 Summary of Assumptions for Estimate

It will take approximately four years for initial development of the Timok underground mine including two years for pre Sanction Date exploration development.

The mining capital cost estimate is based on the following:

- Preliminary Project development plan.
- Contractor quotes for the exploration decline development.
- Mining equipment list.
- Budget quotes for the major mobile equipment obtained from equipment manufacturers
- Contractor equipment lease costs estimated based on the equipment budgetary prices and equipment depreciation periods.
- Contractor labour rates are based on a combination of local labour rates provided by Rakita and information obtained from contractor quotes.
- Budget quotes obtained from mining contractors for vertical development and geotechnical drilling.
- SRK’s in-house database.

- Existing on-site and regional costs provided by Rakita, including:
  - Owner labour rates.
  - Diesel fuel cost.
  - Electrical power cost.

It is assumed that the following work will be completed by a contractor:

- Initial mine development during development of the exploration decline complex, pre Sanction Date; and mine pre-production development; post Sanction Date.
- All vertical development including ventilation raises, ore and waste passes.
- Geotechnical and delineation drilling.

The contractor will provide all labour, equipment and supplies, which are not provided by the owner. A 12% markup has been applied to all contractor expenses, including equipment and supplies, to account for the contractor's profit.

The Owner will purchase its own mobile equipment for mine production, operating development and mine services during mine operating period.

No allowance for escalation, inflation factors or interest during construction are included in the estimate.

21.1.4.2.2 Mine Development

All underground mine development costs were included in the capital cost estimate, except for the exploration decline complex to the top of the ore body, that would be completed prior to the Sanction Date. Also waste and ore draw points, and slot drives, were excluded in mine development costs; being separately included in operating costs as an operating development cost item. However, any operating development completed in the mine pre-production period, was included in the initial capital cost estimate.

Seven current quotes from mining contractors were available from the pre-Sanction Date exploration decline cost estimate. These quotes form one part of the cost database for ongoing capital development. A contractor quote was also used for the vertical development cost estimate. All other underground mine development costs were estimated from first principles and were then compared to SRK's database for mine development by a contractor and to the contractor quotes for the exploration decline development at Timok Project.

Underground capital development cost is based on the mine development schedule and uses existing quotes from mining contractors for the exploration development, plus estimates for owner's costs and includes the following:

- Extension of the Dual-decline exploration development to the first primary crusher to service the operating mine.
- Water management.

Lateral development cost is based on the mine development schedule and unit costs per meter of development estimated from first principles and includes the following:

- Extension of the access and conveyor declines from the end of exploration decline development to the top of the underground crusher
- Level development
- Ventilation drifts connecting level development to the ventilation raises
- Miscellaneous development.
- Vertical development cost are based on the mine development schedule and unit costs per meter of development based on a contractor quote and includes the following:
  - Ventilation raises.
  - Ore and waste pass system.
  - Service holes of large diameter.
- Infrastructure excavations are based on the mine development schedule and costs for excavations estimated from first principles and includes the following:
  - Explosives storage.
  - Underground workshops.
  - Refuge stations.
  - Underground crusher chambers.
  - Sumps.
- Contractor overhead cost includes the following:
  - Contractor mobilisation (additional cost after completion of exploration development including raise bore contractor).
  - Contractor overhead labour cost.
  - Contractor equipment lease.
  - Contractor support equipment and site facilities operating costs.

Contractor's overhead labour cost includes contractor management, supervision, services and maintenance labour, which was not accounted in the direct costs per unit of development.

The contractor will supply their own mobile equipment for mine development. There was no assumption made for equipment purchase. Instead, the lease rates for the contractor equipment were estimated based on the following inputs:

- Equipment budget price.
- Equipment depreciation period, which varies by equipment type.
- 3% interest rate.
- 1.5% insurance.
- 12% contractor markup.

The estimated lease rates were applied to each unit in the equipment fleet required on site during mine development based on quarterly periods.

The operating costs of the contractor's support equipment such as a grader, boom and mechanic trucks, personnel carriers and supervisor vehicles, maintenance and site facilities were not accounted in the direct cost per unit of development. Those costs were estimated based on the assumptions for expected equipment utilisation and included in the contractor's overhead cost on a quarterly period.

The contractor will demobilize from the mine site after completion of the work.

The summary of the Timok underground mine development capital costs is presented in Table 21.8.

Table 21.8: Underground Mine Development Capital Cost Summary

Cost Item	Initial (US$M)	Sustaining (US$M)	LOM Cost (US$M)
Lateral Development	32.05	18.06	50.10
Vertical Development	8.73	7.29	16.02
Infrastructure Excavations	2.98	1.34	4.33
Contractors Overhead	26.59	2.12	28.71
Operating Development (in Capital Period)	10.38	-	10.38
Total Underground Mine Development Capital Cost	**80.73**	**28.81**	**109.54**
Exploration Decline Development (pre Sanction Date)	42.96[1]		42.96

Total Mining Project Development Cost	**123.69**	**28.81**	**152.5**

Note 1: A down payment of US$3 million will be spent on the decline development before June 2018. This amount is considered a sunk cost and is not included in the capital cost estimate.

The initial capital cost for mine development (post the Sanction Date) is estimated at $80.73M including $10.38M of operating development in the pre-production capital period, with sustaining capital thereafter of $28.81M. $42.96M will be spent on the exploration decline development and ore body access for a bulk sample before the Project Sanction Date.

21.1.4.2.3 Mining Mobile Equipment

The purchase of a permanent mining equipment fleet is required for the mine production activities performed by the Owner.

Mobile equipment costs were developed from estimated fleet requirements and vendor budgetary quotations.

Equipment life-cycle operating hours were based on manufacturer recommendations and SRK project experience. The recommended life-cycle operating hours were used to calculate equipment replacement requirements.

The schedule for major equipment rebuilds and replacement was based on equipment operating hours and anticipated equipment life. It was assumed that all equipment will have one major rebuild (60% of initial price) before replacement.

The initial capital cost for mine mobile equipment is estimated at $30.04M, with sustaining capital thereafter of $65.09M. The sustaining capital cost will be spent on purchasing additional equipment, replacement of existing equipment after maximum operating hours for equipment life and for the major equipment rebuilds. The mobile equipment cost also includes face dewatering pumps and auxiliary ventilation fans.

21.1.4.2.4 Mine Operating in Pre-Production Period

The mining operation costs during the Capital period, prior to commercial production, were estimated from first principles and included in the initial Capital costs. The total capitalized operating cost during this period is $14.35M and includes mine operating development cost of $10.38M, which is presented in the mine Capital development cost section above.

The remaining capitalized operating cost includes the following:

- Longhole drilling of $0.34M.
- Longhole blasting of $0.22M.
- Production mucking and re-handling of $0.24M.

- Mine services and maintenance of $1.93M.
- Definition drilling of $1.25M.

21.1.4.3 Mine Infrastructure

- A mechanical equipment list was developed.
- Nevsun has recently received budget price quotations for the supply of several major equipment packages including the Jaw Crusher and Vibrating Feeder. Hatch reviewed these bid(s) and recommended a cost to include in the CAPEX as well as a duration to include in the PEP schedule.
- Hatch worked with CDI to obtain an updated estimate for the cost to supply the decline and overland belt conveyors.
- The supply cost for the balance of mine infrastructure equipment is based on in-house pricing derived from an established database and adjusted as required to suit the Project's particular requirements.
- Equipment installation base hours were estimated based on published or in-house information for similar equipment. Underground productivity factors and labour rates were applied (refer to Section 21.1.4.6).
- Hatch developed preliminary MTOs for concrete (for floors and equipment foundations, etc.), chutes, supporting structural steel, pipe, cable, portable mine substations, etc., based on general arrangement drawings prepared for most infrastructure areas.
- Unit supply costs for bulk materials was based on in-house data with adjustments based on regional information as available
- Installation costs: base installation hours were based on in-house data for similar installations. The base hours were multiplied by an underground productivity factor and hourly labour rate (refer to Section 21.1.4.6 for underground productivity and labour rate information).
- Where general arrangements were not prepared for typical infrastructure or equipment, in-house pricing was utilized.

Note that all excavation costs are included in the mine section of the cost estimate.

21.1.4.4 Process Plant

- A mechanical equipment list was developed based on the process flow diagrams. The list includes the equipment type, main capacity/size parameter(s), materials of construction, etc.
- Nevsun recently received budget price quotations for the supply of several major equipment packages as listed below:

- SAG and Ball Mills.
- Cyclones.
- Concentrate Filters.

Hatch reviewed the bids and recommend costs to include in the CAPEX as well as durations to include in the PEP schedule.

- Other equipment supply costs were estimated based on in-house data adjusted as per equipment capacity/size and escalated as per the price base date. Hatch obtained a budget quotation for flotation equipment.
- Equipment installation costs were estimated based on:
  - Base installation man-hours were estimated based on published and/or in-house metrics including equipment type/size, etc.
  - A surface productivity factor was developed based on site specific factors and regional data (refer to Section 21.1.4.6). This factor was applied to the base installation hours.
  - In-house labour cost information. Hatch estimated the site labour rate based on published wage information, application of burdens, development of construction labour crew rate build ups, estimated equipment costs and estimates of contractor indirect costs. The rate is based on Serbian conditions. The rates include an allowance for PPE, tools and some construction equipment.
- Concrete quantities were estimated based on the preliminary general arrangement drawings. Concrete volumes were estimated based on the areas of buildings and an assumed slab thickness and foundation configuration. Allowances were included for equipment foundations, containment berms, etc.
- Concrete supply and installation costs are based on in-house data for Serbia. The unit cost includes the supply of cement, aggregate, accelerants, a batch plant, formwork, reinforcing steel, embedded plates and anchor bolts, delivery to the workface and placement.
- Steel quantities are based on the preliminary general arrangement drawings. The required steel tonnage was estimated by multiplying building volumes by typical steel densities for similar structures. Pipe rack quantities were estimated by multiplying pipe rack lengths by an assumed linear weight per unit length. Allowances are included for connections and miscellaneous steel.
- Steel supply and installation costs are based on in-house data for Serbia.
- Architectural quantities were developed based on building surface areas and in-house cost data.

- Piping, electrical and controls and instrumentation were factored based on the mechanical supply and installation costs. Factors were derived from recent Projects for similar facilities.

21.1.4.5 Tailings, Waste Rock and Reclaim Water Management Facilities

21.1.4.5.1 Scope Summary

The main items for the TSF and waste rock management are as follows:

- TSF site preparations including tree clearing/logging, stripping and stockpiling of topsoil.
- Stage 1 TSF construction including embankment fills, foundation preparation, liners, foundation and under-drain systems, etc.
- Mechanical systems (pumps, pipelines, etc.) as well as associated power supply.
- Access roads and storm water ditches.

21.1.4.5.2 Cost Basis

- All quantity estimates were based on a collection of design figures/drawings completed using Civil3D or Muck3D software.
- Linear features (roads, channels, right-of-way) quantities were completed using tables in Excel rather than 3D software.
- Earthworks unit rate cost assembly considers:
  - Equipment all in costs based on local Serbian reference where available. In absence of local rates, the BC (Canada) Road Builders and Heavy Construction Association Guidebook (BC Blue Book) were used with consideration for equipment rental rate assumptions agreed to and as specified in the estimate.
  - Equipment rates (all found) include all costs; insurance, fuel, equipment use/rental rate.
  - The labour rate was applied separately of equipment unit costs.

  The earthworks equipment efficiency is dependent on equipment availability, and cycle time assumptions.
- The supply and installation costs of pumps were obtained from similar projects and recent supplier quotes.
- Supply and installation of HDPE pipelines includes consideration of required valves and fittings. Unit costs developed from supplier estimates and the RS Means Heavy Construction Cost Data Book.

- Hatch provided input regarding power transmission line unit cost (US$/km) and road cost (US$/m^2).

21.1.4.5.3 Assumptions and Exclusions

- Any necessary earthworks related to installation of pipelines are covered under the road construction line items.
- All earth fill materials are assumed locally sourced (maximum haul of 1.5 km).
- Earth fill material processing (drain rock, etc.) is based on processing rates from local sources. Canadian (BC rates) were used where Serbian rates were not available.

21.1.4.6 Labour Rate and Productivity

Assessing construction labour rates and productivities in Serbia is challenging. Few industrial capital projects have been completed in Serbia over the past 25 years. As a result, it is difficult to obtain cost data from contractors – particularly in formats with which western companies are familiar.

Serbia contains skilled and unskilled labour at relatively low wage rates. This has created a demand, particularly from other Western European countries, for Serbian workers to work outside Serbia.

Cost information obtained from Serbian construction contractors along with publicly available data on base salaries and statutory cost was used to estimate an all-in site labour rate. This rate was benchmarked against labour rate information from other sources including:

- Labour rate data from the 2011 Flash Smelter project at CMSC Bor, with rates escalated to 2017. This data was obtained by Nevsun from SNC (SNC executed the project). The cost data was escalated to 2017 dollars. A breakdown of the cost data is not available so it's not possible to confirm if this is actually an "all-in rate".
- Firm price bid data obtained by Nevsun in Q4 2017 for the Timok decline contract. Bids were received from seven companies and three bids were short-listed for award.
- Firm price bid data obtained by Nevsun in Q4 2017 for the Timok portal preparation contract.
- All-in rates developed by SRK for the Timok Project (for the PEA).

An all-in labour rate was selected based on the above information.

- Items that are included in the base labour rate are:
  - Fully burdened labour rate (includes vacation pay, payroll taxes, benefits).
  - Insurance.

    - Overtime.
    - Small tools.
    - Site transportation.
    - Consumables.
    - Personal protective equipment.
- General Construction Equipment Rate. This includes:
    - Construction equipment including construction cranes up to 80t capacity.
    - Fuel for equipment operation.
- Contractor Indirect Costs. Items considered to be contractor indirect, or contractor distributable costs include:
    - Contractor temporary facilities.
    - Contractor mobilization and demobilization.
    - Medical checks/immunizations.
    - Warehouse material handling.
    - Training/induction (for construction).
    - Travel.
    - Maintenance, general clean up.
    - Supervision and administration.
    - General office consumables.
    - Overhead and profit.
    - Indirect equipment (site vehicles, trucks, radios).

21.1.4.6.1 Productivity Factor (PF)

Productivity accounts for site specific or project specific issues that affect construction. References information used to develop a PF for the Timok project is listed below:

- Data that Nevsun obtained for the 2011 Flash Smelter Project at CMSC Bor. That project was a brownfield (retrofit) project.
- Productivity factors that Hatch derived from the Q4 2017 Timok decline firm price bids.
- Productivity factors that Hatch derived from a Q4 2017 Timok portal firm price bid.

- The productivity factor developed by SRK for the Timok surface plant for the PEA.
- Compass International published data (PFs between 1.45 and 1.75 for Serbia).

A productivity factor was selected based on the above reference information and with consideration of the greenfield nature of the Timok Project, the relatively small size of the Project and the relatively low complexity.

21.1.4.7 Subcontract Costs

In some cases, direct costs included in the CAPEX don't align with the "equipment supply", "material supply" or "installation" cost categories. In these cases, the costs were listed under the "subcontract" column on the detailed CAPEX spreadsheet.

For instance, in-plant roads and bulk earthworks are normally listed as subcontracts. Costs for the supply and installation of non-process buildings, or field erected tanks, may also be listed in the subcontract column (since contractors' bids typically don't list supply and installation costs separately).

21.1.4.8 Growth Allowances

Growth allowance were included in the base estimate before contingency for surface infrastructure (WBS 1000), mine infrastructure and the process plant (WBS 3000).

21.1.4.8.1 Price Growth

Estimates are generally based on non-binding quotations that rarely take into account final specifications or negotiated commercial terms. A price growth allowance is an amount added to the price of materials or equipment to account for the expected increase in cost from the budget quoted amount and the final purchase order value at the time of equipment delivery.

Price growth allowances generally range depending on the quality of the pricing data received. Most mechanical equipment pricing will require price growth allowances of 5% even where written purchase orders are in place.

The application of price growth for the process plant CAPEX was determined by discipline engineers, estimators and senior project management based on experience and historical database information.

21.1.4.8.2 Design Growth

Previous project experience indicates that there is a gap between what is known and identified during the estimating process and what is expected during the execution or construction phase. A design growth allowance is a subjective amount added to the MTO based on the degree of engineering completed and a comparison to historical experience of the expected quantity.

Design growth was added to the base estimate in the form of an increase to the design quantity or MTO quantity. The addition of growth is intended to allow for the most likely quantity and therefore cost in the estimate.

21.1.4.8.3 Waste Allowance

A construction waste allowance is intended to cover items such as concrete over pour during construction, or wasted cable from reels not cut to exact size. Wastage allowances are added to the estimated cost and are not added to the estimate quantity because these quantities will not be installed.

The price growth, design growth, and waste allowance included in the estimate are shown in Table 21.9.

Table 21.9: Growth Allowance Criteria by Commodity

Commodity	Design Growth	Price Growth	Waste Allowance
Earthworks	15%	0%	5%
Concrete	15%	0%	5%
Structural steel	10%	0%	5%
Duct work and plate work	5%	0%	5%
Architectural	10%	0%	5%
Equipment	0%	10%	0%
Piping bulks	15%	0%	5%
Electrical equipment	0%	5%	0%
Electrical cable	10%	0%	10%

21.1.5 Indirect Costs

21.1.5.1 Summary and Basis

Indirect costs were factored. The factors were developed based on historical information from similar projects, adjusted to account for site specific factors. The indirect factors were developed for each of the major WBS areas (Site Development, Mining, Mining Infrastructure, Concentrator and Tailings) by the responsible parties.

The Indirect Costs and their bases are presented in Table 21.10.

Table 21.10: Basis of Indirect Costs

Description	Estimated Cost (USD)	Basis
Camp and Catering	1,867,000	Estimated based on the following assumptions: • US$40/man-day for food and lodging • 33% of labour force from outside the region (i.e. 67% local)
Temporary Facilities (shops, ablutions, laydown areas, etc.) Temporary Construction Services (including catering, fuel, power, laundry, communications) Construction Major Equipment, Heavy Cranes, etc.	7,632,000	Estimated based on the following assumptions: • Existing or permanent facilities will be used for the Project (i.e. offices, warehouse, portal offices, etc.) • Allowance of 4% of direct costs included for WBS 1000 and WBS 3000 • No allowance included for WBS 2000 (mining) • Allowance of US$100K for WBS 5000 (tailings) • Allowance of US$1.5M included for heavy craneage
Freight, Duties & Logistics	11,010,000	Estimated based on the following assumptions: • Allowance of 7.5% of equipment and material supply costs for WBS 1000, 3000 and for mine infrastructure • US$692,000 included for mine mobile equipment • No allowance for WBS 5000 (included in direct costs)
Detailed engineering, procurement and construction management (EPCM) services	37,275,000	Estimated based on the following assumptions: • 15% of Direct costs for WBS 1000 • 0% for mine development (included in Owner's costs) • 12% of Direct costs for mine infrastructure • 18% of Directs for WBS 3000

Description	Estimated Cost (USD)	Basis
		• US$4.9M based on estimate developed by KP
Pre-Operational Testing First Fills Vendor Representatives	5,664,000	Estimated based on the following assumptions: • 2.5% of Direct costs for WBS 1000 and WBS 3000 • 2% of Direct costs for mine infrastructure • US$700K for mine development
Third Party Services	1,558,000	Estimated based on the following assumptions: • 1% of Direct costs for WBS 1000 and WBS 3000 • US$50K for mine infrastructure
Spares – Commissioning Spares – Capital	4,245,000	Estimated based on the following assumptions: • 3% of equipment supply costs for WBS 1000, WBS 3000 and for mine infrastructure • US$1.47M for mine mobile equipment spares

21.1.5.1.1 Construction Indirects (Temporary Facilities and Services)

This budget covers the following items:

- TSF construction area development including temporary diversion, laydown/staging areas, material borrow quarries and stockpiles, etc.
- Temporary water.
- Temporary buildings, such as, offices and trailers, ablution blocks and the temporary camp, if required.
- Site office supplies, furniture, janitorial services.
- Communications systems and local area network required to support the site construction activities.
- On-site Third-Party NDT testing and inspection.
- Fuel and/or electrical power to be used during construction.

- Catering, cleaning services.
- Snow removal.
- Heating.

21.1.5.1.2 Construction Mobile Equipment

This budget covers the cost of heavy construction equipment to be purchased or rented directly by the Project (such as heavy lift cranes). The budget includes mobilization, demobilization, fuel and an operator.

Note that the (Direct Cost) labour rate includes costs for PPE, tools, fork trucks, small cranes (<50 tonne capacity), etc.

21.1.5.1.3 Freight, Duties and Logistics

The freight budget covers costs for the transportation of equipment and materials from the anticipated market to the plant site.

Unless provided by vendor, SRK included the cost of freight as 4% of equipment budgetary price to cover freight and on-site assembly.

21.1.5.1.4 Engineering, Procurement & Construction Management (EPCM) Services

The budget for EPCM covers the following costs:

- Detailed engineering.
- Procurement of equipment and materials and field contracts.
- Construction management (including Stage S1 commissioning).
- Project controls/reporting.
- Project administration.
- Office expenses, communication, IT services, etc.
- Travel costs associated with the EPCM team.

21.1.5.1.5 Pre-Operational Testing

The stages of commissioning are shown in Table 21.11.

Table 21.11: Stages of Commissioning

		Mechanical Completion			Commissioning	
Function	Procurement / Construction / Preoperation Team	**Construction**	**Pre-Operations**		**Commissioning**	**Operations (Ramp Up)**
Stage	**FAT**	**S1**	**S2**	**S3**	**S4**	**S5**
Activity	Factory Acceptance Testing	Construction Inspection & Testing	Pre-Operational Equipment Testing	Pre-Operational Systems Testing	Process Commissioning	Ramp-up
Conditions	Temporarily Energized	Not Energized	Energized	Inert materials	Process materials	Process materials
Example Tasks	*IO Interlock Checks Sequence Checks*	*Wiring point to point Vessel leak check Pipe pressure test*	*Motor bump test Vibration check Align conveyor*	*Interlock & Loops check Sequence check Pump flow check*	*Run systems with process materials Run the complete plant*	*Increase throughput to design rate and quality*
Responsible Person	Vendor Project Manager	Construction Manager	Pre-Operational Testing Manager		Commissioning Manager	Operations Manager
Responsible Organisation	**Equipment Vendor**	**Construction Contractors**	**Pre-Operational Testing Team**		**Commissioning Team**	**Owner's Operations Team**
Milestones	Authorized to ship to Site or Module Yard	Transfer to Pre-Operational Testing	Handover to Commissioning		Start of Production	Nameplate Production

This budget covers planning and supervision services for stages S2 and S3 (Pre-Operations) commissioning.

For major mine mobile equipment an allowance of 2% of supply costs was included to cover equipment commissioning and training.

21.1.5.1.6 Spare Parts

Allowances were included in the CAPEX for:

- Start-up & commissioning spares for any spare parts required for start-up.
- Capital spares.

No allowance has been included for operational spares.

For mine mobile equipment, unless provided by the vendor, an allowance has been included for spares as 5% of the major mobile equipment budgetary price to cover initial parts stock.

21.1.5.1.7 First Fills

The budget is to cover costs associated with materials not normally consumed by the process such as:

- Transformer oil.
- Hydraulic fluid.

- Lube oil and grease.

First fills do not include reagents and consumables including fuel and ball charges (these items are included in the operating cost estimate).

21.1.5.1.8 Vendor Representatives

An allowance is included for vendor representatives to be on site during construction and/or pre-operational testing, depending on the nature of the equipment.

In some cases, to fulfill the requirement of equipment manufacturer’s warranties and guarantees, selected manufacturers require their representatives to complete an inspection of their equipment prior to it being placed into operation.

21.1.5.1.9 Third Party Services

An allowance is included for topographic surveys, concrete testing, inspection services as required to support the Project.

21.1.6 Contingency

A contingency of 20% was applied to the Timok Project. This is aligned with AACE guidelines for a Class 4 estimate. A quantitative analysis and contingency simulation was not completed for the PFS.

For Class 3 (or lower) estimates, a quantitative contingency analysis is normally used based on a Monte Carlo simulation.

21.1.7 Owner’s Costs

Owner’s costs were estimated by Nevsun.

21.1.7.1 Pre Sanction Date Owner’s Cost

Pre Sanction Date Owners costs are all costs to develop and maintain the Timok Project and advance it to fully permitted and fully financed state; including:

- Portal site preparation for the Exploration decline complex.
- Development of the exploration declines to the top of the orebody and associated underground drilling and access to the orebody for bulk sampling.
- Condemnation drilling for various aspects of site surface development.
- Site investigation for the tailings dam and main mill site etc.
- Ongoing Environmental background studies.
- Completion of the PFS and the subsequent Feasibility Study (FS).
- Completion of full Serbian permitting to development permit status.

- Purchase of all lands required for construction of the project.
- Consultants and Nevsun Head Office expertise to develop and manage the Project.
- Serbian management of the extensive Site and Project development works.

These costs have been developed from current 2018 Contractor quotes for the diamond drilling, site investigation, Portal Preparation and Exploration Decline work, 2018 wage and consultants costs and current costs incurred by Rakita operations to-date. The estimated Pre-Sanction Owners cost for all these activities between June 2018 and June 2020 is some $113.9M, including 10% contingency.

21.1.7.2 Post Sanction Date Owner's Costs

Post Sanction Date Owner's costs up to Production are all costs incurred by the Owner to control the Timok project and liaise with the EPCM contractor, plus control the underground development and include:

- Rakita Corporate expense in Serbia.
- Nevsun corporate and support expense in Serbia provided by Nevsun head office.
- Formal mining legislative control of all underground works, carried out by underground contractor.
- Ongoing environmental monitoring and other statutory monitoring.
- Early recruitment of mine and mill staff prior to start up to allow for training and to facilitate start up.
- Additional external training experts to carry out continuous improvement of operations during the pre-production and post production periods.
- Start up, hot commissioning and production ramp up costs.
- Other Execution Owner's Costs.

Again, the estimates for these costs were prepared by Nevsun, in conjunction with Rakita and based on current 2018 contractor quotes for the work, current 2018 wage rates at Rakita, current costs of consultants and other experts and other information from the current operations at Rakita. The sum total of the estimated Post-Sanction, pre-Production Owners cost for all these activities between June 2020 and planned Production start-up is $36.6M plus 20% contingency for a total of $44M.

Post start up, additional Owners costs have been included, over and above the calculated G&A and operating costs, for ongoing training and plant hand over during the first year of operations. The cost for this activity is based on current costs and is approximately $4.9M over and above G&A and operating costs. No contingency is included in this amount.

21.2 Operating Cost Estimate

This has been prepared by Nevsun, SRK Consulting, Knight Piésold (KP), Conveyor Dynamics INC.(CDI) and Hatch. The team worked together to coordinate their individual cost estimates and then reviewed the integrated document.

Hatch had overall responsibility for integrating the work of the individual companies into a single, comprehensive and consistent operating cost estimate. Hatch was not responsible for work completed by others.

The operating cost structure developed for the Project has four components: mining, ore re-handling on the surface and processing, water management and Tailings Storage Facility (TSF), plus General and Administrative (G & A) costs. The labour component in each area of the operating costs has been developed separately from a detailed organization plan.

The summary operating costs for the Project used for the evaluation are shown in Table 21.12, and Table 21.13. Figure 21.1, and Figure 21.2 show the variation in OPEX over the 10-year production period.

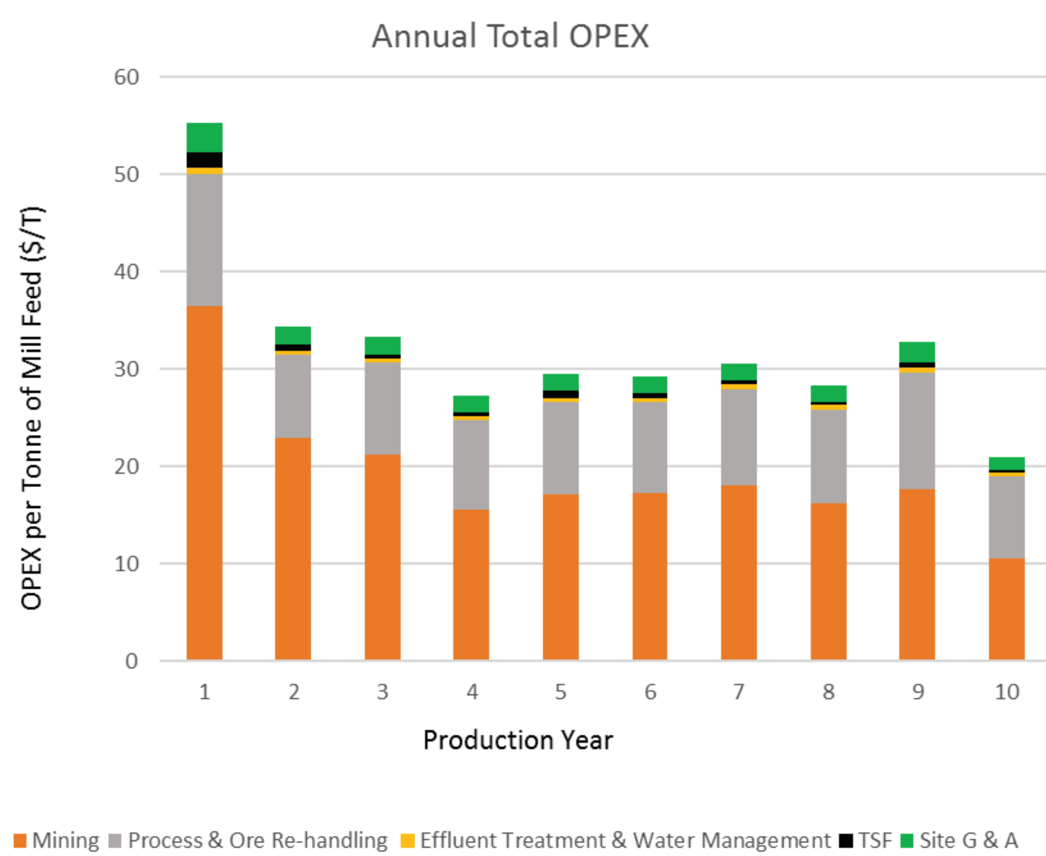
Table 21.12: Overall Site Operating Costs

Description	Total LOM OPEX ($M)	LOM Unit Costs ($/t)
Mining		
Labour	56	2.07
Materials	159	5.87
Equipment	147	5.43
Fuel	31	1.14
Power	48	1.78
Subcontract	9	0.32
Mining Maintenance	1	0.05
Contingency	75	2.76
Total Mining	526	19.41
Processing & Ore Re-handling on the Surface		
Power	77	2.84
Labour	31	1.15
Reagents	43	1.58
Consumables	65	2.40

Description	Total LOM OPEX ($M)	LOM Unit Costs ($/t)
Maintenance	16	0.59
Miscellaneous	6	0.23
Laboratory/Assay Costs	10	0.36
Ore Re-handling and Stockpiling on the Surface	0.4	0.01
Contingency	25	0.92
Total Processing & Ore Re-handling on the Surface	274	10.09
Water Management, Effluent Treatment & Tailings Storage		
Labour	2	0.08
others	23	0.84
Contingency	2	0.09
Total Water Management, Effluent Treatment & Tailings Storage	27	1.01
G & A		
Management & Accounting	7	0.28
Warehouse, Security & IT	1	0.05
Marketing & Supply management	2	0.07
Environmental & Social	6	0.21
Human Resources	2	0.08
Other Costs	29	1.05
Contingency	5	0.17
Total G&A	52	1.91
Total OPEX including Contingency	***879***	***32.42***

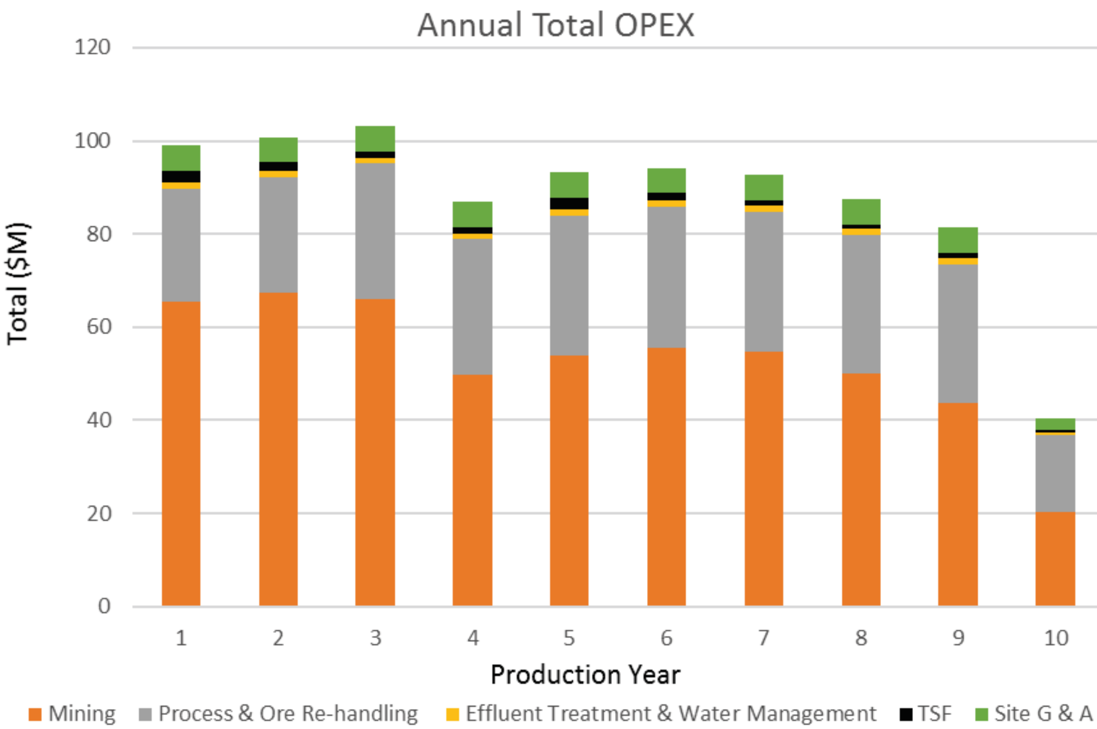
Table 21.13: Site Operating Cost Summary

Operating Cost Summary	LOM ($M)	Unit Costs ($/t)
OPEX - Mining	526	19.41
OPEX – Processing & Ore Re-handling	274	10.09
OPEX - Water Management, Effluent Treatment & TSF	27	1.01
OPEX - G&A	52	1.91
Total OPEX including Contingency	***879***	***32.42***



Note: Production years mean a full year of operation

Figure 21.1: Unit OPEX by Year



Note: Production years mean a full year of operation

Figure 21.2: Total OPEX by Year

21.2.1 Underground Mining Operating Costs

The detailed mining section was provided by SRK Consulting with inputs from Conveyor Dynamics INC (CDI), and Hatch.

21.2.1.1 Summary

Underground mining operating costs were developed based on a quarterly LOM schedule for an owner operating scenario. Productivity, equipment operating hours, labour and supply requirements, and costs were calculated for each cost activity, such as: mine operating development, production drilling and blasting, mucking, truck haulage, secondary breaking, mine services, primary crushing, ventilation, mine dewatering, conveying and maintenance. The cost of mine operating, technical and maintenance staff was estimated as separate cost items based on the staff roster.

The operating costs were estimated using a combination of first principle calculations, experience and factored costs. The underground mining operating cost summary by process is presented in Table 21.14.

A contingency of 16.6% was assumed for all underground mining operating cost items.

Table 21.14: Mining Operating Cost by Process

Description	LOM ($M)	LOM ($/tonne)
Operating Development	94	3.47
Longhole Drilling	57	2.10
Longhole Blasting	46	1.70
Production Mucking	44	1.62
Production Material Re-handling	30	1.10
Trucking	2	0.06
Secondary Breaking	6	0.22
Rehabilitation	6	0.21
Definition Drilling	6	0.22
Mine Services and Maintenance	79	2.90
Mine Technical Staff	7	0.25
Ventilation	21	0.78
Dewatering	5	0.17
Crushing	1	0.03
Conveyors	17	0.64
Rockbreaking	0.2	0.01
Mine Systems	1	0.05
Parts & Materials	31	1.13
Contingency	75	2.76
Total	**526**	**19.41**

Underground mining operating cost summary by cost elements is presented in Table 21.15.

Table 21.15: Mining Operating Cost by Cost Elements

Description	LOM ($M)	LOM ($/tonne)
Labour	56	2.07
Materials	159	5.87
Equipment	147	5.43
Fuel	31	1.14
Power	48	1.78
Subcontract	19	0.32
Mining Maintenance	1	0.05
Contingency	75	2.76
Total Mining	526	19.41

21.2.1.2 Basis of Estimate

The major assumptions used for the cost estimation include:

- Mine operating schedule: three 8-hour shifts per day including 1.5 hours of non-productive time accounting for shift change, equipment fueling and inspection, lunch and breaks during a shift.
- Labour: owner labour rates to be used for estimates were provided by Nevsun based on a 2017 Mining Labour survey and also rates currently paid at another similar sized underground mine in the Balkans.
- Shift rotation: senior staff working 8-hour shifts, five days on and two days off and hourly staff working 8-hour shifts with four crews on rotation.
- Diesel fuel price of $1.18/litre as per current price for fuel delivered to the mine site.
- Electric power price of $0.0675/kWh as per current all-in electricity cost on site.

The input data used for estimation of productivities, equipment and labour requirements, and costs are presented in Table 21.16.

Table 21.16: Shift Schedule Inputs

Operating Factors	Units	Quantity
Days/Year	days/year	365
Mine Operating Days	days/year	365
Mine Production Rate	tonnes/year	3,000,000
Working Days per Week	days/week	7
Shifts per Day	shifts/day	3
Shift Length	hours/shift	8
Shift Change	hours	0.5
Equipment Inspection	hours	0.25
Lunch/Coffee Breaks	hours	0.5
Equipment Parking/Reporting	hours	0.25
Subtotal Non-Productive Time/Shift	hours	1.5
Usable Time/Shift	hours	6.5
Shift Efficiency	%	81%
Usable Minutes per Work Hour	min	50
Operational Efficiency (50 min in hour)	%	83%
Effective Work Time/Shift	hours	5.42
Work Time Efficiency	%	68%

Material costs were based on estimated consumption of consumables and recent supplier's prices for drill and steel supplies, explosives, ground support and services supplies. Consumables costs were increased by 10% to account for miscellaneous use and material wastage. It was assumed that 25% of auxiliary ventilation and electrical cables will be recovered and reused, resulting in cost reduction.

Operating costs of major mobile equipment were estimated based on calculated operating hours and indicative costs for mobile equipment maintenance provided by vendors. The indicative costs from vendors were increased by 15% over vendor's estimates to account for damage not anticipated by those vendors. The operating hours for auxiliary mobile equipment were estimated based on expected equipment utilisation.

Mobile equipment operating costs include maintenance consumables, tires, fuel, lube and power. Those costs are part of mine development, production and services costs.

21.2.1.3 Mining Cost Estimate

The operating development cost was estimated from first principles and included development of waste drawpoints, ore drawpoints and slot drives.

Longhole drilling and blasting costs were estimated from first principles. These include drilling and blasting costs to create initial slots and costs for drilling and blasting of production rings.

Blasted material will be mucked from the drawpoints by 7.0 m^3 LHDs and dumped to the ore pass/waste pass system. Each production level has access to two ore passes and one waste pass. The ore/waste pass system will transfer material from the production levels to a crusher loading level. The mucking costs were estimated based on an average mucking distance on a production level from a drawpoint to the dumping points of the ore/waste pass system.

As there is no direct feed from the ore/waste pass system to the crushers, material will be re-handled from the bottom of the ore/waste pass system to the crusher loading point. 8.6 m^3 LHDs will be used for that purpose. Re-mucking costs were estimated based on average re-muck distances from the ore/waste pass system to the crushers.

Secondary breaking costs were estimated based on the assumed amount of oversize material to be broken to get through the grizzly openings. Secondary breakage activities require one blockholer and one mobile rockbreaker to deal with drawpoint oversize. A stationary rockbreaker will be installed at the dumping point on the top of each crusher chamber.

Trucking costs include process feed and waste trucking from development when the ore/waste pass system cannot be utilized.

It was assumed that geotechnical and definition drilling will be performed by a drilling contractor. The geotechnical and definition drilling costs were estimated based on the assumptions made for the geotechnical and definition drilling requirements and the contractor quotes for drilling.

An allowance for mine rehabilitation was made to account for drawpoint and general mine rehabilitation including installation of bulkheads to isolate mined-out production levels.

The mine services and maintenance costs include operating costs of the service mobile equipment fleet, allowance for mine services and maintenance supplies, maintenance and construction hourly labour, and miscellaneous expenses.

Labour requirements were estimated for each mining activity. Indirect labour costs, which include mine operating, technical and maintenance staff, were estimated based on the staff roster and labour rates provided by Nevsun.

21.2.1.4 Mining Labour Cost Estimate

Mining employees at the Timok operation are divided into two categories: technical staff and hourly operating and maintenance labour.

Mine technical staff include engineering and technical personnel working 8-hour day shifts on a rotation of five days on and two days off.

Most of hourly operating personnel will work 8-hour shifts with a rotation schedule based on three crews on site and one crew off. Hourly personnel include labour for mine development, production, services and maintenance. The mine roster factor of 1.55 was estimated to account for additional labour based on shift days off and a total of 35 days of annual leave including vacation time, sick, training and other duties. These factors were applied to the estimated on-site labour requirements to estimate the total mining labour requirements and labour costs.

A mining contractor will be used for development during the pre-production period to allow time for the owner to recruit staff for the Project. Also, a contractor will be used for all vertical development and major stationary equipment installation such as main ventilation fans, conveyor systems, the ore/waste pass system and primary crushers. Contractor labour and supervisory staff is not included in this section.

The labour cost estimates were based on the labour rates provided by Nevsun and the labour requirements estimated by SRK. The mine roster of hourly labour was estimated on a quarterly basis based on the development and production schedule, mine services and maintenance requirements for crew rotation of three 8-hour shifts per day with three crews working on site and one crew off, plus a shift factor allowance for personnel off sick or on holiday.

21.2.2 Process Plant

21.2.2.1 Summary

Process plant detailed operating costs were provided by Hatch. Processing plant operating expenses include costs associated with all process plant areas, plant maintenance and plant technical services (metallurgy, engineering and laboratory services), and ore-re-handling and stockpiling costs on the surface.

The exact estimation methodology was varied by cost component, but was primarily built from first principles relying on a combination of:

- Mass balance and process design criteria.
- Information from laboratory testing reports.
- Power consumption estimates per motor.
- Hatch in-house knowledge of similar operations.
- Supplier reagent and consumables quotes.

Major categories include the following, which collectively result in a processing cost estimate for each category:

- Power.

- Labour.
- Reagents.
- Consumables.
- Maintenance Materials.
- Assay Laboratory.
- Miscellaneous.
- The estimate base date is Q1 2018. All costs have been estimated to an expected level of - 20%/+25%. A contingency of 10% was included in the operating cost estimate. Exchange rates used were as per those reported in Section 11.
- Costs reported in Table 21.17 reflect plant operation with Life Of Mine (LOM) mill throughput of 27.1 Mt over 10 years of plant operation at 92% operating factor. The estimate covers all processing plant costs expected during the ordinary course of operations for a Project such as this.
- In broad terms, the estimate includes all operating costs associated with the processing of copper-gold material to produce a saleable bulk copper concentrate. The operating cost estimate for the process plant does not include costs associated with mining (including primary crushing), downstream transport, marketing of products or corporate overheads.
- The operating costs have been estimated on a quarterly basis and are linked to the mine production schedule. Process plant operating costs were developed for the processing five “ore grade bins” options defined in Section 13.2.

Table 21.17: Summary of Ore Re-handling, and Process Plant Operating Cost

Operating Cost Summary	LOM ($M)	Unit Costs ($/t)
OPEX - Ore Re-handling & Stockpile on the Surface	0.4	0.01
OPEX - Processing	273	10.08
Total	274	10.09

21.2.2.2 Ore Re-handling and Stockpiling on the Surface

An allowance of $1/t of was made for ore re-handling and stockpiling on the surface. The ore re-handling and stockpiling operating cost was estimated to be approximately $0.4M during the 10-year life of mine.

21.2.2.3 Processing

The forecast average and total key operating cost estimates for LOM process plant at Timok are provided in Table 21.18.

Table 21.18: Operating Cost Summary LOM

Item	LOM ($M)	Average Annual Cost ($)	Annual Cost ($/t of ore)
Power	77	8	2.84
Labour (operating + maintenance)	31	3	1.15
Reagents	43	5	1.58
Consumables	65	7	2.40
Maintenance	16	2	0.59
Miscellaneous	6	1	0.23
Laboratory	10	1	0.36
Contingency	25	3	0.92
Total	273	29	10.08

The overall cash operating cost to process a tonne of plant feed is $10.08 (LOM). Figure 21.3 is a chart illustrating the overall LOM operating cost for the processing of the material.

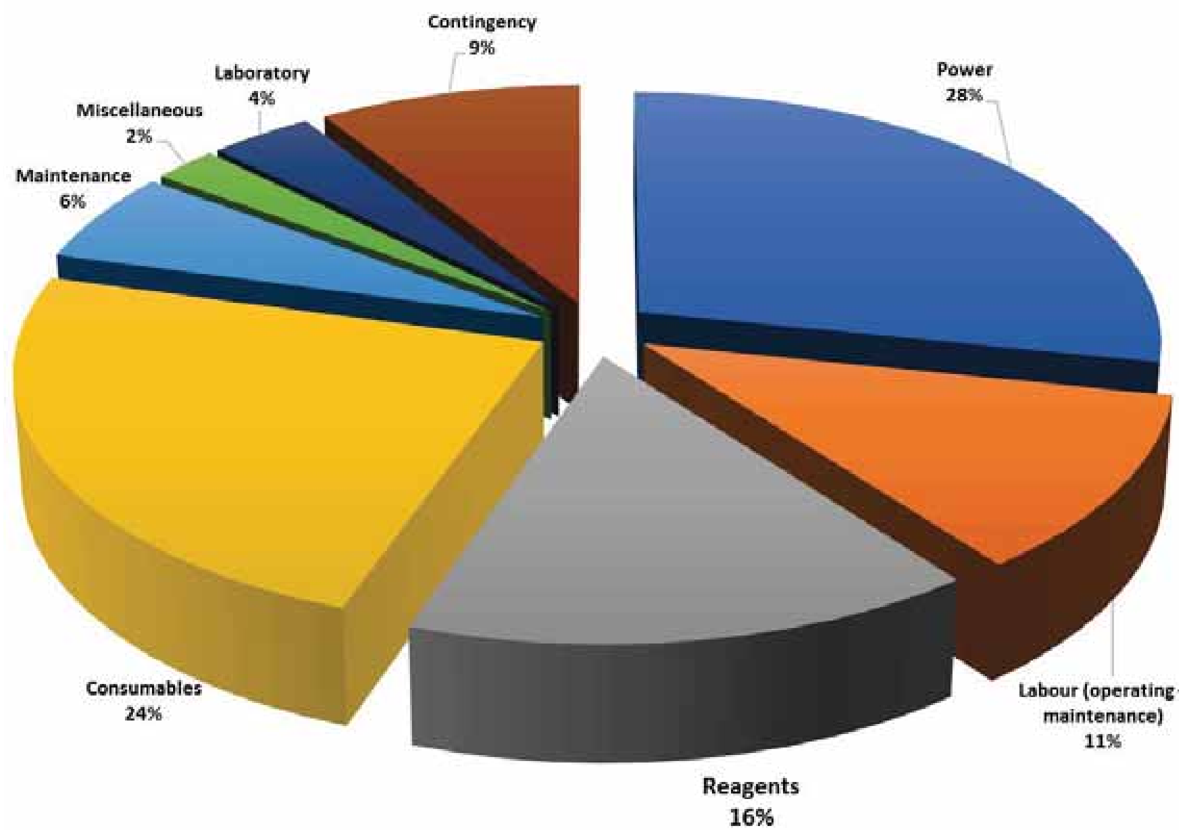


Figure 21.3: Summary of Operating Costs

21.2.2.3.1 Basis of Estimate

The following basis was used to determine the LOM operating costs required for the Timok Project in agreement with the cost definition and estimate methodologies outlined in detail below. This basis considers the development of a facility capable of processing an average of 3,250,000 tonnes per year.

- The operational cost estimates have been broken down into the following five primary cost areas:
  - Crushed ore storage bins.
  - Grinding.

  - Flotation.
  - Concentrate dewatering.
  - General process plant.
- Operating cost was estimated based on a quarterly basis and are linked to the mine production schedule and ore feed grade.
- Commercial production completed in 10 years of operation.
- Operating costs are calculated based on manpower, power consumption, reagent and consumable, process and maintenance consumables, and laboratory costs.
- Labour, and sample analysis costs were assumed to be fixed with all other costs being varied according to the plant throughput, ore feed grade and electrical loads.
- Electric power price of $0.0675/kWh as per current all-in electricity cost on site.

21.2.2.3.2 Labour – Production, Plant Maintenance and Contract

Labour staffing was estimated by benchmarking against similar projects. The labour costs incorporate requirements for plant operation, such as management, metallurgy, operations, maintenance, site services. The total operational labour averages 130 employees. All positions for the processing plant will be filled by local workers.

The salary rates were provided by Nevsun as overall rates, including all burden costs.

21.2.2.3.3 Power

The processing power draw was based on the average power utilization of each motor on the equipment list for the process plant and services. Power will be withdrawn from the local grid to service the facilities. The annual power costs were calculated using a unit price of $0.0675/kWh provided by Nevsun. The power consumption rate for the process plant (not including G&A and underground mining) was estimated as 49 kWh/t.

A chart showing the power allocation based on the process areas is provided in Figure 21.4.

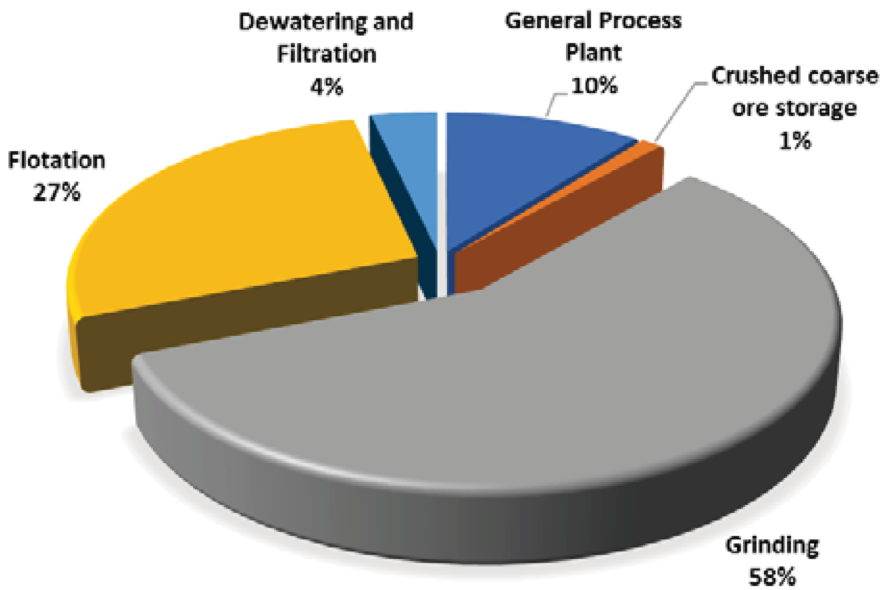


Figure 21.4: Power Allocation Based on Process Areas

21.2.2.3.4 Reagents

Annual operating consumables expenses were calculated from the required reagent consumption (kg/t processed).

Individual reagent consumption rates were estimated based on the metallurgical test work results, Hatch's in-house database and experience, industry practice, and peer-reviewed literature. Reagent unit costs ($/t) were obtained through vendor quotations or recent in-house data. Reagents represent approximately 16% of the total process operating cost.

Table 21.19 depicts the estimated consumption rates for the reagents used in the process plant. Consumptions are for reagents in the form, quality and quantity, they are supplied.

Table 21.19: Process Plant Reagents and Consumption Rates

General Plant Reagent	Consumption Rate (kg/t processed) Average LOM
Quicklime	3.64
Copper Collector (Aerofloat 211)	0.07
Potassium Amyl Xanthate	0.030
Frother (MIBC)	0.11
Copper Concentrate Flocculant	0.025
Antiscalant	0.002

21.2.2.3.5 Consumables

The maintenance consumables include liners and balls for the SAG mill, ball mill, and regrind mill. The ball consumptions for the mills were estimated using:

- Metallurgical testing results (abrasion).
- Orway Mineral Consultants (OMC) report.
- Hatch's in-house calculation methods.
- Forecast total power consumption.

Table 21.20 depicts the various consumption rates for the process plant.

Table 21.20: Process Plant Consumables

Consumable	Consumption Rate					
	Sets/yr	Very High Cu (kg/t milled)	High Cu (kg/t milled)	Medium High Cu (kg/t milled)	Medium Low Cu (kg/t milled)	Low Cu (kg/t milled)
SAG Mill Ball (100 mm)		0.324	0.324	0.324	0.404	0.394
Ball Mill Ball (40 mm)		0.584	0.584	0.584	0.727	0.71
Regrind Mill Ball [1]		1	1	1	1	1
SAG Mill Liner consumption		0.062	0.062	0.062	0.076	0.074
Ball Mill Liner Consumption		0.076	0.076	0.076	0.092	0.09
Regrind Mill Liner Consumption[2]	1.0					

[*] Note: 1 kg/t of rougher flotation concentrate.
2. for all ore feed grade type.

21.2.2.3.6 Maintenance

Maintenance consumable costs were calculated based on total direct costs for each area using a weighted average factor of 4% and are allocated as shown in Figure 21.5.

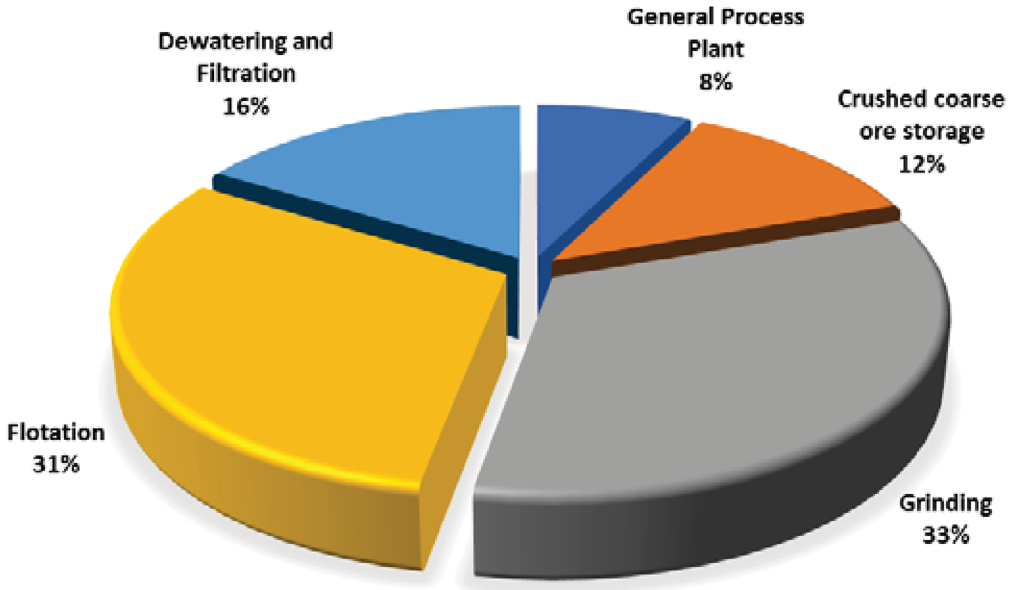


Figure 21.5: Maintenance Cost Breakdown by Weighted Average

21.2.2.3.7 Miscellaneous

Miscellaneous costs include the following:

- Medical/first aid supplied estimated to be $100/person/year.
- Safety clothing/PPE estimated to be $700/person/year.
- Training estimated to be $2000/person/year.
- Safety rewards estimated to be $200/person/year.
- Computers and software licences estimated to be $2000/person/year.

Costs for external consultants is not included.

21.2.2.3.8 Assay Laboratory

The costs for assaying for processing were estimated according to the number of anticipated assays per year and average sample costs. A sample cost of USD 41.6 is estimated to be the average based on a laboratory's services' quote. The assaying costs for geology and mining are not included in this section. Laboratory requirements will be 48 samples per day for the process plant.

21.2.3 Effluent Treatment, Water Management, and Tailings Storage Facility

The life of mine effluent and tailings treatment and water management operating costs were estimated to be approximately $27 M. The LOM average cost is $1.01 per tonne of ore. A contingency of 10% was included in the forecasted estimate.

21.2.3.1 Effluent Treatment and Water Management

Table 21.21 summarizes the effluent treatment plant operating cost by cost elements during life of mine. A contingency of 10% was included in the operating cost estimate.

Table 21.21: Summary of Effluent Treatment Operating Cost Estimate

Item	LOM ($M)	LOM Unit Costs ($/t)
Power	3	0.12
Labour (operation)	0.3	0.01
Reagents	4	0.15
Consumables	1	0.05
Maintenance	2	0.08
Laboratory	0.2	0.01
Fresh Water Cost	0.4	0.01
Contingency	1	0.04
Total	13	0.46

Effluent treatment operating cost was estimated as follows:

- Power consumption was calculated using information from the mechanical equipment list.
- An operator for day shift only was considered.
- Reagent and consumable rates were estimated based on Hatch’s in-house database.
- A factor of 4% applied to the total mechanical equipment cost to estimate maintenance costs.
- Laboratory requirements was assumed to be 4 samples per day for water quality testing.

21.2.3.1.1 Fresh and Potable Water Supply

The water supply operating cost was estimated to be approximately $0.4M during the 10-year life of mine. A contingency of 10% was included in the operating cost estimate. Water supply OPEX estimate includes:

- Purchase of 29,000 m^3/y of potable water from JKP Vodovod Bor. The potable water requirement was extracted from the site water balance.
- Pipeline maintenance.
- Pump(s) maintenance and power.
- Storage tank maintenance.
- Fresh water river intake and discharge structure(s) maintenance.
- Local staff costs.
- Costs for external Serbian and international contractors and consultants.
- All water monitoring costs including costs for local laboratory analysis.

The unit cost of the purchase of potable water is RSD144.72/m^3 ($1.42/$m^3$) provided by Nevsun.

The annual maintenance and operating cost for fresh or make up water of pipelines and pumps (except power) was estimated at a flat rate of 5% of the purchase and installation cost. An exception was made for the potable water supply pipeline from JKP Vodovod Bor, where all maintenance and operating costs are assumed to be already covered in the purchase of the water from the supplier.

All water pumps are powered by mains electricity. Pump efficiencies of 75% and electric motor efficiencies of 90% were assumed. An electricity cost in Year 1 of $0.0675/kWh was used.

21.2.3.2 Tailings Storage Facility (TSF) and Waste Rock Storage

This section detail was provided by Knight Piésold (KP). The life of mine TSF operating costs were estimated to be approximately $15M. The LOM average cost is $0.54 per tonne of ore. A contingency of 10% was included in the operating cost estimate.

The TSF Operating Costs (OPEX) include:

- Electricity supply for the TSF Mechanical Systems.
- Mechanical systems (Delivery, Reclaim and Seepage Pump-back) maintenance.
- Waste Rock transport (Plant site to TSF).

- Access Road (Plant Site to TSF) maintenance.
- Construction Water Management.
- Sub-contracted Engineering Support.

21.2.3.2.1 Access Road Maintenance

Maintenance of the TSF access road is required over the life of mine, as it will be frequently used by waste rock haul trucks, mine service vehicles, and light use vehicles. The maintenance of the access road is provided for as a percentage (%) of the road initial Capital cost estimate (CAPEX). An allowance of 5% of the road CAPEX is applied annually over the operating mine life.

21.2.3.2.2 Construction Water Management

The TSF site is located within a natural valley that has a reporting catchment of approximately 490 hectares. Construction water management will be required over the life of mine to keep runoff away from active construction areas and will include the use of temporary ditches, ponds and erosion and sediment control measures. These measures will be further defined in subsequent design stages. An allowance of 3% of the initial capital cost of bulk earthworks (foundation preparation, embankment construction, disturbance etc.) is applied annually over the operating mine life. This estimate is consistent with typical construction water management costs of Projects of like scale.

21.2.3.2.3 Power Supply and Electricity Costs

Electricity costs over the life of mine are based on the $90.91/MWh/y rate. A fifty percent (50%) contingency was applied to the electrical demand of the tailings delivery, reclaim and seepage pump-back systems.

21.2.3.2.4 Mechanical System Maintenance

Annual maintenance of mechanical systems includes equipment, material, and labour. In absence of a detailed man-hour-loading schedule, the following maintenance allowances are provided:

- Valves, connections and fittings: 5% of total pipe cost included in the CAPEX, applied annually.
- Installation (valves, controls and automation): 5% of total pump cost included in the CAPEX, applied annually.
- Pump maintenance: 10% of CAPEX pump cost applied annually.
- Pipe maintenance: 10% of CAPEX pipe cost applied annually.

21.2.3.2.5 Waste Rock Management

Waste rock will be hauled from the Plant site to the TSF over the operating mine life. The cost of hauling, including loader, haul trucks and operators has been included in the development of the per cubic metre unit rate of waste rock transportation. This transportation cost is included in the OPEX as the work will be performed directly by the Mill. The unit rates include consideration for the Project labour rates and typical all-in costs equipment rate (insurance, fuel, maintenance, etc.). The quantities of Waste Rock are based on the PFS Waste Rock production schedule.

21.2.3.2.6 Engineering Support

The TSF embankment and impoundment will be subject to regular dam safety inspections over the operating mine life. A provisional allowance of approximately $120,000 USD has been applied annually to cover dam safety inspections and independent audits of the TSF.

21.2.3.2.7 Labour Considerations

An operator and a labour per shift was considered for the TSF operation. The labour costs comprise approximately 15% of the total TSF operating costs.

21.2.4 General and Administration (G&A)

G&A detailed was provided by Nevsun. General and Administration (G&A) costs were estimated on a per annum basis. The site is not remote and does not require the construction of a mining camp or other remote-site facilities. G&A costs were generally assumed to be fixed per unit of time (independent of small variations in production rate) and totaled approximately $5.5M per year for each full year of production. A contingency of 10% was included. Table 21.22 summarizes the forecasted G&A operating cost estimate over life of mine.

Table 21.22: General and Administrative Expenses

General & Administrative	LOM ($M)	LOM Unit Costs ($/t)
Management and Accounting	7	0.28
Warehouse, Security & IT	1	0.05
Marketing & Supply management	2	0.07
Environmental & Social	6	0.21
Human Resources	2	0.08
Other Costs	29	1.05
Contingency	5	0.17

Total G&A	52	1.91

21.2.5 Conclusions and Recommendations

The operating cost estimate has been prepared at a level commensurate with a prefeasibility study. Estimating refinement and precision will increase through the feasibility study phase.

22. Economic Analysis

22.1 Introduction

To evaluate the benefits of the required capital investment for the proposed Timok Project, the economics were evaluated based on a detailed estimate of Project future cash flows, assuming the Project is approved and expenditures commence at the Project Sanction Date of early Q3 2020. The model was constructed in Microsoft Excel as a real basis after-tax discounted cash flow (DCF) model in which the capital costs, production, revenues, capital and operating costs, including G&A and a head office component, sustaining capital costs and appropriate taxes and royalties were considered.

Pre-sanction costs prior to the mid-2020 valuation, or sanction date, such as exploration costs, (including the exploration decline), Pre-Feasibility and Feasibility study costs, work to support the permitting effort and environmental studies in Serbia were considered sunk and not included in the Project cash flows.

The financial model was set up as a single model with multiple sets of input assumptions that can be varied as needed. Mining and processing were assumed to access the Measured and Indicated Mineral Reserve only, in accordance with National Instrument 43-101 (NI 43-101).

NI 43-101 requires that mineralization in a Reserve must not include Inferred mineralization and must only include mineralization that can be economically mined.

All inferred mineralization that was mined as part of the mine plan for the PFS (PFS, 2018) was given zero grade and treated as waste.

The key model assumptions, financial results, Project returns and cash flows are presented herein in 2018 real terms and in US Dollars.

All economic assessments are calculated at the Timok Project level and therefore, do not include certain costs including corporate office, interest, financing and exploration expenses.

22.2 Summary of Results

Table 22.1 shows the results of the financial analysis.

Table 22.1: Summary of Key Financial Results

Parameter	Unit	Value
Cu Price	$/lb	3.15
Au Price	$/oz	1,300
Project CAPEX	$M	574
Sustaining Capital	$M	239
Closure Costs	$M	48
OPEX	$/t Ore	32.42
Total Cash Costs including. transport and smelting	$/t Ore	93.32
Concentrate Produced	Mt	3.16
C1 Costs	$/lb	0.92
After Tax NPV8%	**$M**	**1,816**
IRR	**%**	**80%**
Payback	years	0.9

Detailed discussion on the basis and key assumptions used in the financial model are presented in subsequent sections.

22.2.1 Project Cash Flows

Figure 22.1 illustrates the breakdown of the undiscounted cash flows over the life of the Project in a waterfall diagram.

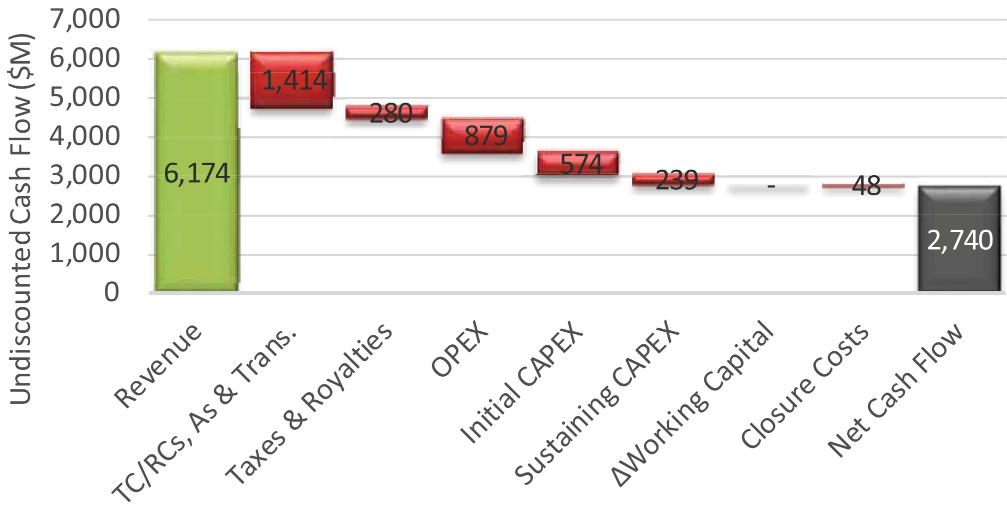


Figure 22.1: Undiscounted Cash Flow Waterfall Diagram

The discounted cash flow waterfall diagram (at 8%) is presented in Figure 22.2.

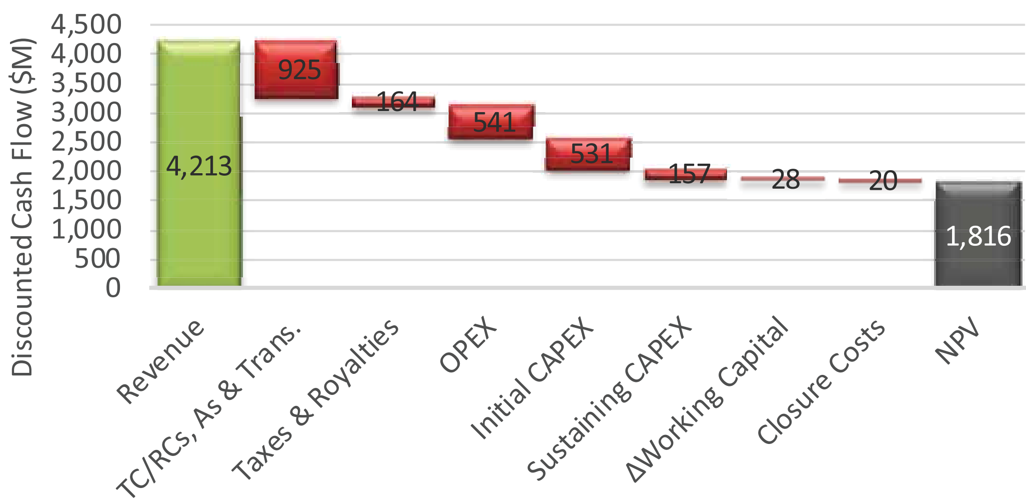


Figure 22.2: Discounted Cash Flow Waterfall Diagram

Free cash flow is presented year on year in Figure 22.3.

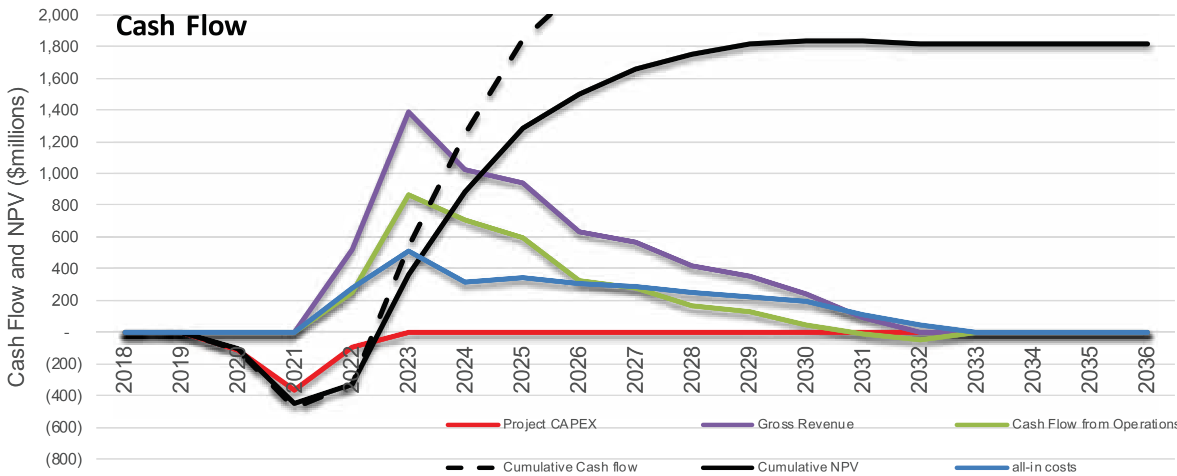


Figure 22.3: Net Cash Flows for the Project and Operations

22.3 Sensitivity Analysis

A sensitivity analysis was conducted on the financial model in order to identify key variables which may significantly impact forecasted returns. Particularly, the analysis focused on metal prices, operating and capital costs. Forecasted sets of key variables were independently varied and the resulting net present value was recorded.

The $NPV_{8\%}$ sensitivity to changes in copper price, head grade, recovery, production rate, gold price, F/X, Project CAPEX, OPEX, are shown in Figure 22.4.

As can be seen, the $NPV_{8\%}$ is most sensitive to changes in copper price and head grade on the upside and recovery on the downside.

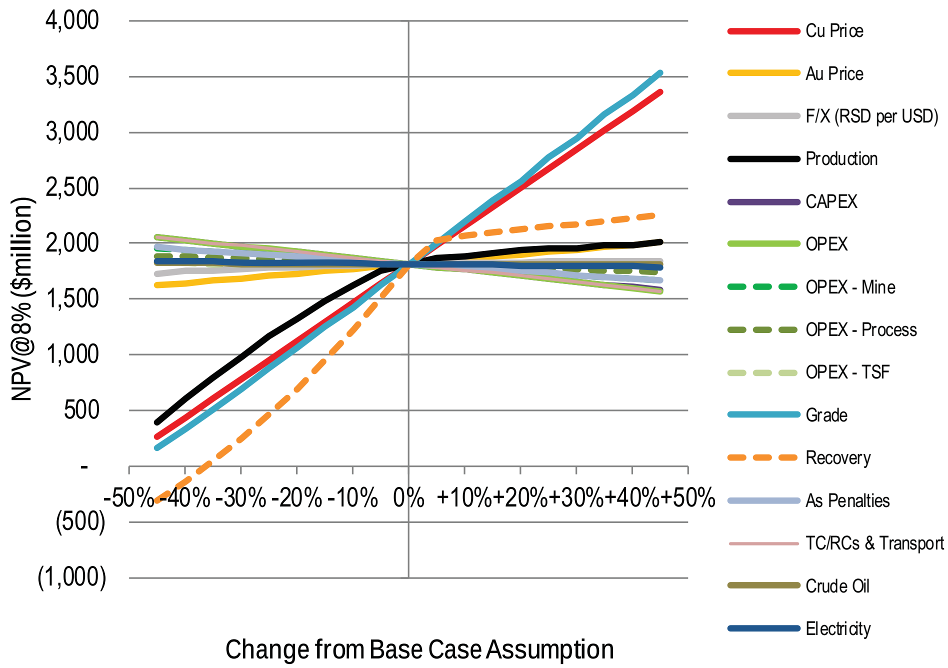


Figure 22.4: Spider Diagram illustrating NPV$_{8\%}$ Sensitivity

The IRR sensitivity to changes in the aforementioned key variables are shown in a spider diagram in Figure 22.5. As can be seen, the IRR is most sensitive to CAPEX, followed by head grade, copper price, production rate, recovery and F/X.

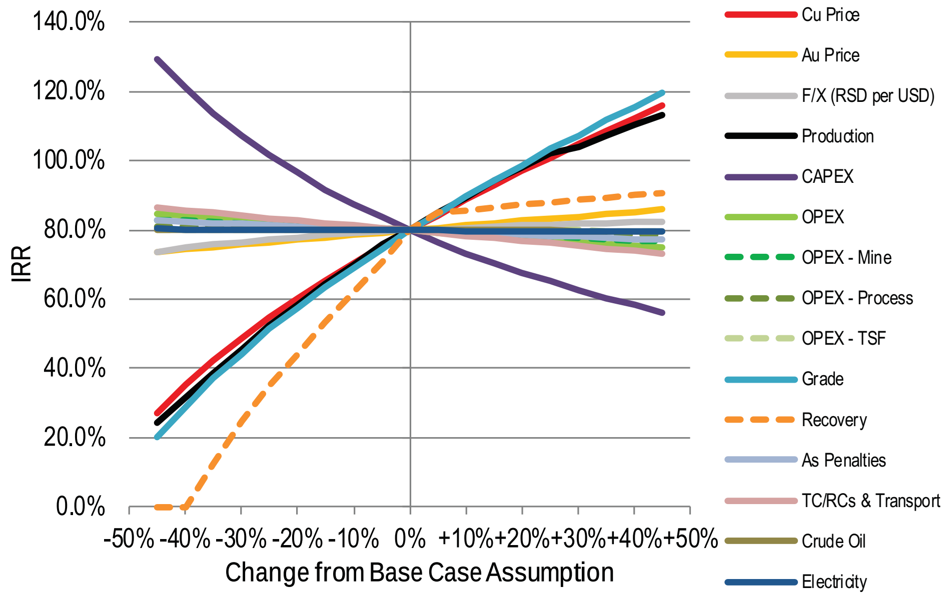


Figure 22.5: Spider Diagram Illustrating IRR Sensitivity

Figure 22.6 illustrates the sensitivity of NPV to the Project discount rate. The Project base case discount rate of 8% has been bolded.

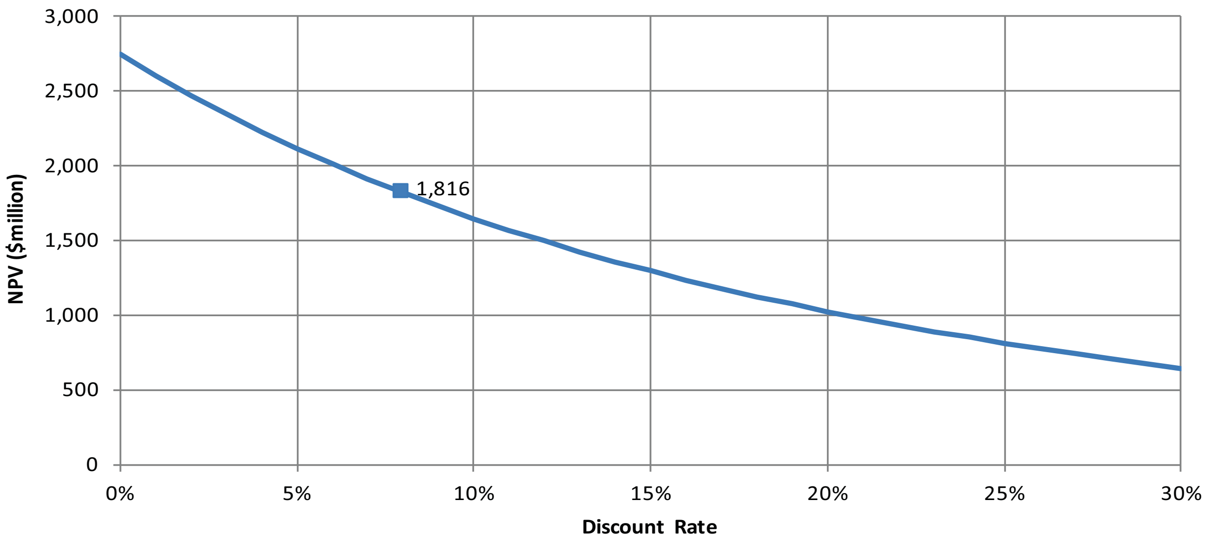


Figure 22.6: NPV Sensitivity to Project Discount Rate

Table 22.2 illustrates the $NPV_{8\%}$ and IRR at various copper price assumptions. The Project base case copper price of $3.15/lb has been bolded.

Table 22.2: Key Financial Results at Various Cu Prices

Cu Price ($/lb)	2.55	3.00	3.15	3.25	3.75
NPV ($M)	1,159	1,652	**1,816**	1,926	2,473
IRR (%)	61%	75%	**80%**	83%	96%
Payback (years)	1.14	0.96	**0.92**	0.89	0.78

22.4 Key Assumptions

The fundamental assumptions used in development of the financial model are shown in Table 22.3.

Table 22.3: Key Financial Model Assumptions

Parameter	Assumption	Description
Units	metric	The model has been constructed using metric tonnes
Valuation Date	30-Jun-20	The analysis is based on a valuation date of Jun 30, 2020. Study, maintenance and exploration incurred costs incurred prior to that date are considered sunk and are not considered in the analysis.
Discount Rate	8%	The financial evaluation has considered 8% as the discount rate. Mid-period discounting has been used in the model. A sensitivity is shown for other discount rates.
Currency	USD	The model has been constructed using US Dollars.
Inflation	Real basis	All projected revenue and costs for the project are assumed to be in 2018 real terms over the DCF time frame, with no inflation applied.
Capital Structure	Unlevered	The calculated financial results assume a project financed entirely on equity. No Interest Payments have been assumed.
Royalty	5%	A 5% NSR royalty has been included.
Income Tax	15% after 10 year holiday	15% Corporate income taxes have been included in the project model after a tax holiday based on a 91% reduction in taxes for 10 years based on guidance from Nevsun and its tax advisers.
Depreciation	15% declining balance	15% declining balance basis depreciation has been utilized in the analysis.
Accounts Receivable	30 days	30 days of total revenue has been assumed for accounts receivable for working capital funding requirements.
Accounts Payable	30 days	30 days of total OPEX has been assumed for accounts payable for working capital funding requirements.
Inventory & Consumables	30 days	30 days of total OPEX has been assumed for inventories and consumables for working capital funding requirements.
Long Term Prices		
Cu	$3.15/lb	Basis for Pricing: Long term Broker forecasts.
Cu	$6614/t	
Au	$1,300/oz.	Basis for Pricing: Long term Broker forecasts.

Please refer to the Section 19 Marketing regarding treatment charges, refining charges, arsenic penalties, transportation and other selling costs.

22.4.1 Production

Figure 22.7 below illustrates the processing production schedule including the tonnes of ore processed, copper grade, payable copper production and arsenic grade of the concentrate:

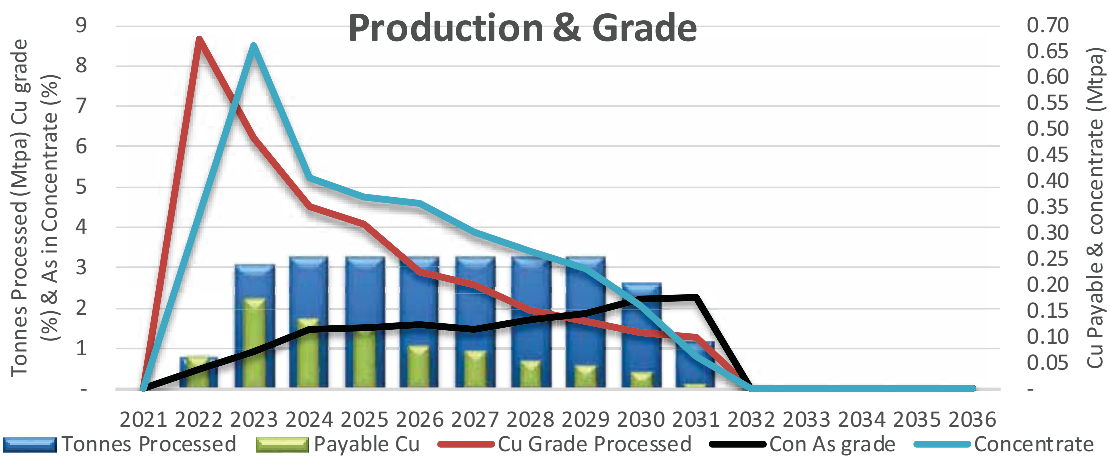


Figure 22.7: Production and Grade Profile

22.4.2 Treatment & Refining Charges and Arsenic Penalties

Treatment & refining charges and arsenic penalties have been considered in the model, in a non-linear manner reflective of the varying arsenic profile in the concentrate. The figure below illustrates the Arsenic Penalties per tonne of concentrate and per pound of copper.

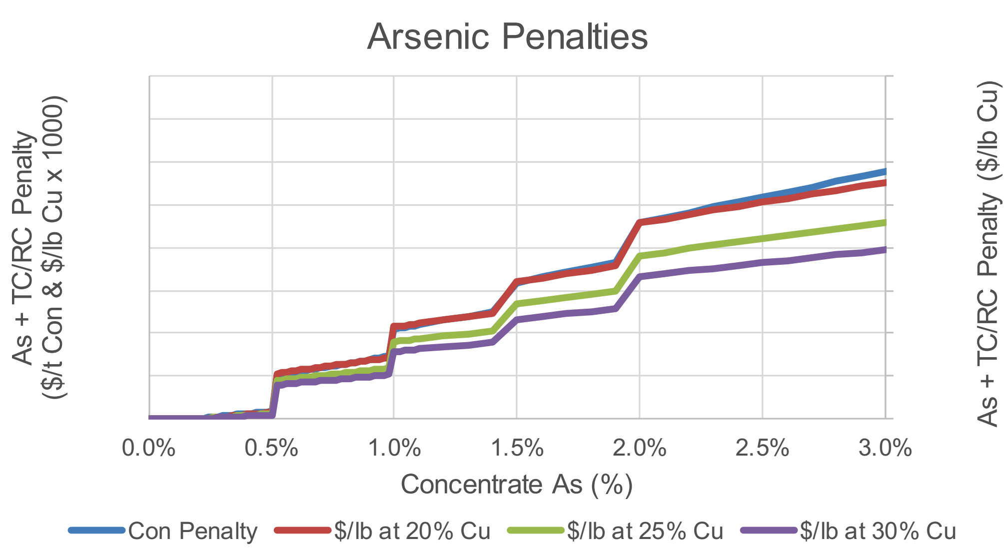


Figure 22.8: Arsenic Penalties

22.5 Detailed Financial Results

Table 22.4 provides detailed outputs of the financial modeling results.

Table 22.4: Financial Results

LOM Totals	Units	LOM Total
Quantity Mined/Processed – Measured & Indicated only	Mt	27.12
Cu Grade	%	3.26%
Au Grade	g/t	2.07
Concentrate tonnes	Kt	3,160
Recovery Cu	%	93%
Recovery Au	%	32%
Payable Cu	Kt	793
Payable Au	Koz	516
Revenue - Cu	$M	5,504
Revenue - Au	$M	671
Total Revenue	$M	6,174
TC/RCs & As Penalties	$M	(966)
Transport	$M	(448)
Royalties	$M	(238)
Net Revenue	$M	4,523
OPEX- Mine	$M	(526)
OPEX - Process	$M	(274)
OPEX - Water and TSF	$M	(27)
OPEX - G&A	$M	(52)
Total OPEX	$M	(879)
EBITDA	$M	3,643
Pre-Sanction CAPEX	$M	-
Project CAPEX	$M	(574)
Total Initial CAPEX	$M	(574)
Sustaining CAPEX	$M	(239)
∆ Working Capital	$M	(0)
Closure Costs	$M	(48)
Pre-Tax Cash Flow	$M	2,782
Taxes	$M	(42)
After-Tax Cash Flow	$M	2,740
After Tax NPV@ 5%	$M	2,112
After Tax NPV@ 8%	$M	1,816
After Tax NPV@ 10%	$M	1,646
After Tax NPV@ 15%	$M	1,293
After Tax NPV@ 20%	$M	1,022
IRR	%	79.8%
Payback	Yrs	0.92

The PFS (PFS, 2018) for Timok Project has indicated an initial estimated capital cost of \$574M and estimated sustaining capital of \$239M (and total estimated operating expenditure of \$879M (with a C1 cost of \$0.92/lb Cu) over the life of the mine, generating an after-tax internal rate of return of 80% and an after-tax $NPV_{8\%}$ of \$1,816M.

22.6 Detailed Project Cash Flows

Based on estimates of revenue, operating costs and capital spending schedule, the after-tax Project cumulative cash flows and cumulative discounted cash flows using an 8% discount rate is illustrated in Figure 22.9.

The buildup of the Project cash flows are provided in Table 22.5.

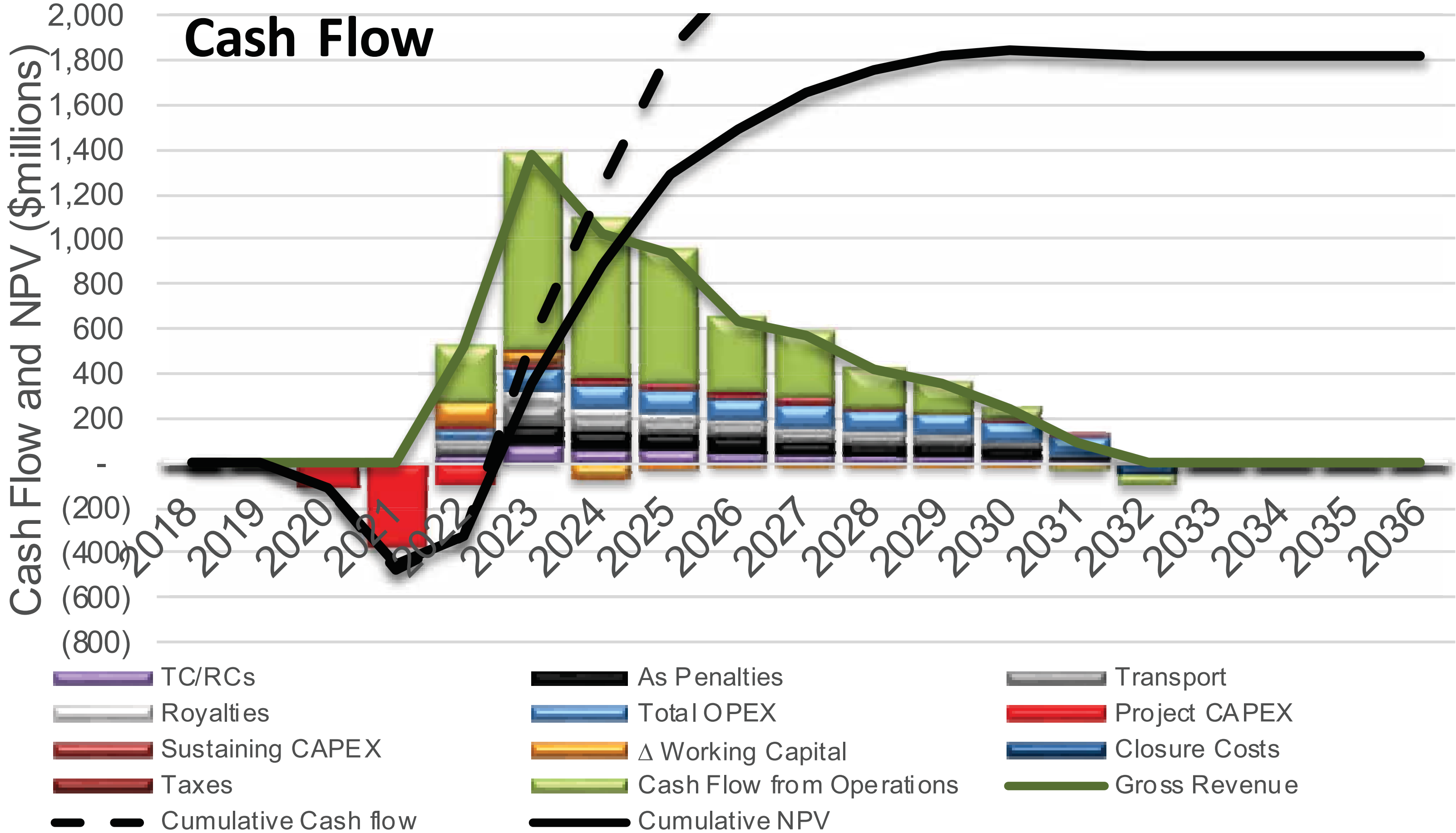


Figure 22.9: Total Project Cash Flows

Table 22.5: Cash Flow Summary

Area	Unit	Total/Avg	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032
Cu Price	$/t	6,945	6,945	6,945	6,945	6,945	6,945	6,945	6,945	6,945	6,945	6,945	6,945	6,945	6,945	6,945	6,945
Au Price	$/t	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Tonnes Mined	Mt	27.1	-	-	0.01	0.09	1.69	2.94	3.09	3.18	3.15	3.22	3.03	3.08	2.48	1.15	-
Cu Grade	%	3.3%	-	-	1.5%	4.4%	8.2%	5.5%	4.4%	4.0%	2.8%	2.2%	1.9%	1.6%	1.4%	1.0%	-
Au Grade	g/t	2.07	-	-	0.79	2.91	5.42	3.31	2.64	2.85	1.95	1.42	1.20	0.94	0.68	0.40	-
As Grade	%	0.17%	-	-	0.04%	0.12%	0.22%	0.22%	0.21%	0.19%	0.19%	0.14%	0.14%	0.14%	0.14%	0.10%	-
Cu Contained	kt	885.4	-	-	0.4	8.4	133.6	160.8	137.3	126.5	89.5	70.0	58.4	50.6	34.9	14.9	-
Au Contained	koz	56,042	-	-	20	554	8,944	9,639	8,184	9,086	6,159	4,623	3,643	2,890	1,676	624	-
Tonnes Processed	Mt	27.1	-	-	-	-	0.77	3.08	3.25	3.25	3.25	3.25	3.25	3.25	2.62	1.15	-
Cu Grade Processed	%	3.26%	-	-	-	-	8.7%	6.2%	4.5%	4.1%	2.9%	2.6%	2.0%	1.7%	1.4%	1.3%	-
Au Grade Processed	g/t	2.07	-	-	-	-	5.77	3.95	2.68	2.85	1.98	1.70	1.21	0.99	0.65	0.54	-
Contained Cu	kt	885	-	-	-	-	66.9	190.8	147.3	133.4	93.8	83.5	63.5	54.3	37.2	14.9	-
Contained Au	koz	1,802	-	-	-	-	143	391	280	298	207	178	127	103	55	20	-
Recovered Cu	kt	827	-	-	-	-	64.2	180.6	138.4	125.1	86.3	77.1	58.1	49.4	33.8	13.5	-
Recovered Au	koz	579	-	-	-	-	87	156	82	84	52	48	31	23	12	4	-
Recovered As	kt	43.6	-	-	-	-	1.6	6.2	6.1	5.7	5.7	4.5	4.5	4.4	3.5	1.5	-
Concentrate Tonnes	kt	3,160	-	-	-	-	337	663	406	371	357	303	266	233	160	64	-
Con Cu grade	%	26%	-	-	-	-	19%	27%	34%	34%	24%	25%	22%	21%	21%	21%	-
Con Au grade	g/t	5.7	-	-	-	-	8.1	7.3	6.3	7.1	4.5	4.9	3.6	3.0	2.3	2.1	-
Con As grade	%	1.4%	-	-	-	-	0.47%	0.93%	1.50%	1.53%	1.60%	1.48%	1.71%	1.88%	2.22%	2.28%	-
Payable Cu	kt	793	-	-	-	-	60.9	173.2	133.5	120.7	82.6	74.0	55.4	47.0	32.2	12.9	-
Payable Au	koz	516	-	-	-	-	78.0	138.2	73.4	76.0	44.7	42.7	27.4	20.2	11.1	4.1	-
Gross Revenue	$M	6,174	-	-	-	-	524.1	1,382.7	1,022.8	937.2	631.8	569.1	420.5	352.9	238.1	95.0	-
TC/RCs & As Penalties	$M	(966)	-	-	-	-	(42)	(169)	(142)	(130)	(122)	(92)	(90)	(84)	(68)	(27)	-
Transport	$M	(448)	-	-	-	-	(39)	(93)	(61)	(56)	(52)	(44)	(38)	(33)	(22)	(9)	-
Royalties	$M	(238)	-	-	-	-	(22.2)	(56.0)	(41.0)	(37.6)	(22.9)	(21.6)	(14.6)	(11.8)	(7.4)	(2.9)	-
OPEX- Mine	$M	(526)	-	-	-	-	(32.3)	(66.2)	(66.7)	(58.0)	(51.4)	(56.7)	(54.6)	(52.5)	(46.1)	(41.9)	-
OPEX - Process	$M	(274)	-	-	-	-	(8.9)	(26.1)	(29.3)	(29.0)	(29.0)	(30.8)	(29.1)	(30.5)	(30.7)	(30.5)	-
OPEX - Water & TSF	$M	(27)	-	-	-	-	(1.1)	(3.0)	(3.1)	(3.1)	(3.1)	(3.1)	(3.1)	(3.1)	(2.7)	(1.9)	-
OPEX - G&A	$M	(52)	-	-	-	-	(2.7)	(5.5)	(5.5)	(5.5)	(5.5)	(5.5)	(5.5)	(5.5)	(5.5)	(5.5)	-
Total OPEX	$M	(879)	-	-	-	-	(45.0)	(100.8)	(104.5)	(95.6)	(89.0)	(96.0)	(92.3)	(91.5)	(84.9)	(79.7)	-
EBITDA	$M	3,643	-	-	-	-	375.8	964.0	674.3	618.4	346.3	314.8	185.7	132.7	55.5	(24.0)	-
Pre-Sanction Spending	$M	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Project CAPEX	$M	(574)	-	-	(111.5)	(366.4)	(95.7)	-	-	-	-	-	-	-	-	-	-
Sustaining CAPEX	$M	(239)	-	-	-	-	(21)	(32)	(30)	(37)	(29)	(47)	(17)	(7)	(15)	(5)	-
∆ Working Capital	$M	0	-	-	-	-	(105)	(52)	69	23	15	15	2	6	8	18	2
Closure Costs	$M	(48)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(48)
Pre-Tax Cash Flow	$M	2,782	-	-	(111)	(366)	154	880	712	605	332	283	170	132	48	(11)	(46)
Taxes	$M	(42)	-	-	-	-	(4)	(12)	(8)	(7)	(4)	(3)	(2)	(1)	(0)	-	-
After Tax Cash Flow	$M	2,740	-	-	(111)	(366)	149	868	704	597	328	279	169	131	48	(11)	(46)
After Tax NPV@ 5%	$M	2,112	-	-	(110)	(348)	131	749	581	469	245	200	114	84	30	(6)	(26)
After Tax NPV@ 8%	$M	1,816	-	-	(109)	(338)	122	687	520	407	208	164	91	66	22	(4)	(19)
After Tax NPV@ 10%	$M	1,646	-	-	(109)	(332)	116	650	484	372	186	145	79	56	19	(3)	(15)
After Tax NPV@ 15%	$M	1,293	-	-	(107)	(317)	102	568	406	298	143	107	55	37	12	(2)	(9)
After Tax NPV@ 20%	$M	1,022	-	-	(106)	(303)	91	499	343	241	111	79	39	25	8	(1)	(5)
IRR	%	79.8%															

23. Adjacent Properties

There are significant exploration assets belonging solely to Nevsun in the adjacent East Timok Nikolicevo and Kraljevica exploration licenses immediately to the east and southeast of the Brestovać-Metovnica license on which exploration is at the exploration and early drilling stage (Figure 23.1). The Nikolicevo, Nikolicevo East and West, and Kraljevica licenses cover a combined area of approximately 164.57 km^2 and remain relatively under-explored (with little drilling). Nevsun considers that the licenses are prospective for both Timok deposit and Bor district style porphyry and HS epithermal massive sulphide targets.

In June 2016, four additional areas surrounding the main licenses were granted to Nevsun; these included Nikolicevo East (20.92 km^2) and West (2.96 km2); Coka Kupjatra East 4.78 km^2) and Tilva Njagra South (2.3 km^2).

Regarding Nevsun's 100%-owned licenses in the TMC (formerly owned by Reservoir Minerals until June 2016), on January 26, 2016, Reservoir Minerals announced that its subsidiaries Tilva (BVI) Inc, and Global Reservoir Minerals (BVI) Inc, had completed all the conditions relating to an earn-in and joint venture agreement signed between Rio Tinto and Reservoir Minerals.

Under the terms of the agreement, Rio Tinto has the option to earn, in stages, up to a 75% interest in Nevsun's four wholly-owned exploration licenses (the Tilva Project) as shown in Figure 23.1.

- Nikolicevo - 70.32 km^2.
- Kraljevica - 70.37 km^2.
- Coka Kupjatra - 40.64 km^2.
- Tilva Njagra - 36.76 km^2.

To achieve this, Rio Tinto will need to sole-fund project expenditures of up to USD$75M. Nevsun (after June 2016) is the Manager of the Tilva Project until Rio Tinto exercises its right to assume the role.

The geology in the East Timok licenses contains the same prospective late Cretaceous andesite volcanic sequence and key metallogenic and structural trends associated with the Timok and Bor deposits. In the Nikolicevo license, Nevsun believes that the alteration assemblages and hydrothermal breccia zones, together with copper mineralization in float and outcrop and the presence of fragmental epiclastic and volcaniclastic beds containing clasts of copper and iron sulphide mineralization and altered volcanics (observed in both outcrop and drillholes), indicate proximity to an as-yet undiscovered HS epithermal mineralization in the license area.

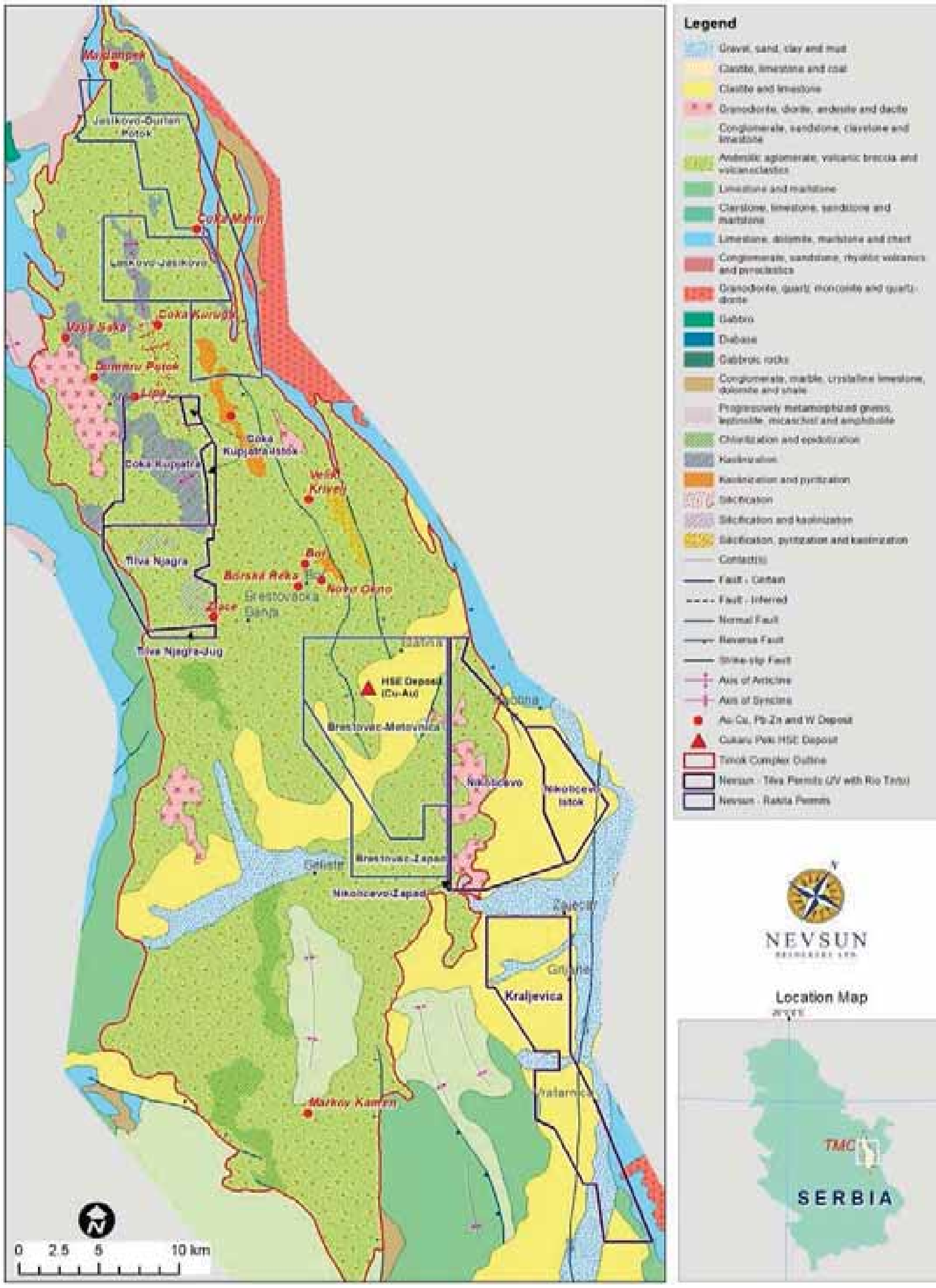


Figure 23.1: Nevsun (former Reservoir Minerals) JV and 100%-Owned Properties, Timok Magmatic Complex

The Tilva-Njagra and the Coka-Kupjatra exploration licenses cover an area of 85.48 km^2 and are located in the western sector of the TMC and Nevsun believes these licenses to be prospective for both epithermal gold and porphyry copper-gold mineralization. Mapping, geochemical (soil and rock chip) sampling and drilling have confirmed the presence of copper and gold mineralization within intense advanced argillic altered basaltic andesitic volcanics and hydrothermal breccias in the Lipa, Coka Kupjatra and Kumstaka prospects. Porphyry style alteration and altered volcanics were observed at the Crni Vrh and Beljevina prospects, respectively. Limited drilling also confirmed the presence of overprinted porphyry style alteration and veining at depth in the Lipa, Coka Kupjatra and Kumstaka prospects. Nevsun believes that there are further porphyry and skarn targets at the Kumstaka-Beljevina and Red River prospects.

24. Other Relevant Data and Information

24.1 Project Execution Plan

The purpose of a Project Execution Plan (PEP) is to provide clear and consistent objectives, strategies and work methods, and thereby a roadmap, for the execution of the Project. The PEP must be tailored to the specific requirements of the Project.

At a prefeasibility stage the PEP (including the associated schedule) is limited to high level issues, such project procurement packaging strategy, construction approach, local/regional capabilities, etc. During subsequent phases the PEP should be significantly expanded including the addition of sections concerning safety, quality, environmental/ community, logistics, workforce planning, project controls/reporting, commissioning, etc.

24.1.1 Logistics and Heavy Lift Transport to Site

A detailed route survey study will need to be completed, including the involvement of a reputable heavy haul transport company. The investigation will consider viable ports, rail transport, and relevant roads to access the site. It is critical to determine what size loads can be brought to site and what restrictions and extra work will be required to identify logistics and transport windows from fabricators and suppliers globally.

The logistics study needs to be completed as early as possible to determine the sizes of the mechanical/structural equipment packages to align with engineering and procurement which is very important and a key to the overall success of the Project.

24.1.2 Project Execution Schedule

24.1.2.1 Introduction

This section describes the parameters and assumptions that were used to develop the Pre-Feasibility Study Schedule for the Timok Pre-Feasibility Study.

The schedule prepared during the Pre-Feasibility Study provides a reference timeline for the execution phase of the Project.

The schedule was developed to achieve the following objectives:

- Establish the overall timeframes for the execution phase of the Project, encompassing the entire scope of work as developed during the Pre-Feasibility Study phase.
- Develop a Level 1 Project Master Schedule, which represents the breakdown of the Project scope of work by phase and area.
- Develop an integrated schedule that is construction driven, i.e. procurement and engineering activities are sequenced and timed to support construction.

- Develop a schedule that considers the technical standards and rulebooks currently prescribed and used in Serbia.
- Develop reasonable and achievable durations for construction activities that are based on monthly percent (%) progress, material and equipment quantities and the associated lead times.
- Identify the critical path and near critical paths associated with the execution phase of the Project.
- Provide a basis on the assumptions upon which the schedule was developed for use in the Pre-Feasibility phase of this study.

The Pre-Feasibility Study Schedule was developed using Primavera P6 software.

24.1.2.2 Key Drivers

The key driver to execute the project is the receipt of all permitting, therefore Hatch has constrained the start of all commitments to a Project Sanction Date, which is considered to be when all permits and financing are in place.

A joint engineering team must be formed with Serbian licensed engineers working with the Owner's team to guarantee that the engineering production and the basis of design type of documents will include all relevant local standards and to generate specific Serbian documents, so the specific permitting package submittal dates may be planned with certainty.

A summary of the project execution schedule is provided in Figure 24.1.

24.1.2.3 Scope of Schedule

The scope of the Pre-Feasibility Study Schedule includes high level activities ranging from detailed engineering, through to procurement (long lead equipment purchasing, and critical contract formation and award), with lead times and delivery, construction and commissioning, associated with the direct scope of facilities.

The mine development schedule is based on information provided by SRK. The schedule for Tailings and Waste Rock Facilities is based on information provided by Knight Piésold.

24.1.2.3.1 Key Assumptions

- Feasibility Study Start Date is 2-April-2018.
- Project Sanction Date (Construction Decision) is 26-Aug-2020 (M1 on ordinal dates). Nevsun is to have applicable permits and financing in place before this date.
- A Basic Engineering Phase is included to accelerate the receipt of inputs critical for detailed engineering that need to be completed before Sanction Date. During this phase the following tasks will be completed:

- Developing structural models and designs in preparation for issuing an RFP.
- Developing RFPs for major equipment (i.e. mills) and issuing for firm price bids.
- Developing P&IDs.
- Completing HAZOPs.
- Completing any additional geotechnical investigation work (i.e. for the process building).
- Developing an Early Works bid package (if one is required) and issuing for firm price bids.
- Receiving bids for long lead equipment and preparing evaluations and recommendations.
- Developing a full PEP and procedural documents to run the project.
- Developing control documents such as the Control CAPEX and detailed execution schedule.

- The Basic Engineering start date is 13-Jan-2020.
- Detailed Engineering will start once Basic Engineering is Complete, by 23-Jun-2020. Duration of detailed engineering is 12.5 months. Any activities beyond this date will transfer to the Field Engineering.
- The EP Contractor is based overseas. The EP Contractor will engage with local Serbian engineering companies which will then submit the final documents to the Mining Law authorities as required.
- The Pre-Feasibility Study Schedule has been developed based on different calendars (Canada and Serbia).
- Receipt of Preliminary Vendor Data: It has been assumed that Preliminary Vendor Data will be received six (4) weeks after PO award, unless specifically advised during the Pre-Feasibility Study.
- Permitting approval is not a constraint to this schedule and permits will be received at Sanction Date.
- Decline Construction is advanced to permit start of underground development as of 26-August-2020.
- TSF Construction for Dam starts on 27-Aug-2020 with contractor’s mobilization.
- Construction of site development (early works) starts on 9-Oct-2020.
- Construction work is based on a single shift, 10 hours day and 6 days per week.

- A six (6) weeks mobilization period has been assumed for each contractor from contract award.
- The schedule is presented in Calendar and Ordinal Dates.
- Weather conditions were taken into consideration.

24.1.2.4 Exclusions

- Contingency.
- Any shift work.
- Permitting submittal and approval.

24.1.2.5 Schedule Development Basis

24.1.2.5.1 Schedule Development Overview

The following provides an overview of the Project Master Schedule developed during the Pre-Feasibility Study:

- The Pre-Feasibility Study Schedule has 149 activities and 262 relationships.
- Four activities have constraints:
  - Start of Basic Engineering and Start of TSF Engineering have a Start-on-or-After constraint.
  - Decline Construction Finish has a Finish-on-or-After.
  - Full Production has a Finish-On-or-Before constraint.
- No out-of-sequence activities.
- No Milestone Activities with Invalid Relationships.
- No negative lags.

24.1.2.5.2 Schedule Structure and Coding

The following outlines the structure and coding that was used to develop the Pre-Feasibility Study Schedule for the Project.

24.1.2.5.3 Package Coding

Table 24.1 presents the package coding that was used to develop the Schedule.

Table 24.1: Package Coding

Code	Description
P	Pre-Purchased Equipment (P-Packages)
C	Installation Contract (C-Packages)

24.1.2.5.4 Schedule Calendar

The following Calendars where used to develop the Pre-Feasibility Study Schedule:

- Engineering and Procurement – Work week is five (5) days and eight (8) hours per day. Canadian (Ontario) statutory holidays are observed (considered as non-working days).
- Construction and Commissioning- Work week is six (6) days and ten (10) hours per day. Construction observes Serbian statutory holidays.

24.1.2.5.5 Development Basis by Discipline

Engineering

The engineering schedule was based on the following:

- Receiving vendor data as scheduled is critical for the advancement of engineering. The following typical durations have been used in schedule for the delivery of major equipment vendor data:
- Receipt of Preliminary Vendor Data: It has been assumed that Preliminary Vendor Data will be received four (4) weeks after PO award.
- Receipt of Certified Vendor Data: It has been assumed that Certified Vendor Data will be received ten (10) weeks after PO award.
- Multi-disciplinary engineering activities will be required to proceed in parallel as per standard practice to maintain schedule progress. Critical decisions are to be made on a timely basis, and in accordance with the scheduled dates.

Procurement

The procurement schedule is based on the following:

- Includes a major equipment and critical contract packages. The table below presents all equipment and contract packages currently considered in this phase.

Table 24.2: Phase 1 Procurement of Major Equipment Contract Packages

Description	Fabrication/Mobilization on Site after Award (in weeks)
Long Lead Equipment Packages	48w
CB002 - Bulk Earthworks, Drainage, Site Roads	6w
CC001 - Concrete Supply and Installation	5w
CE004 - Electrical and Instrumentation (E&I) Installation	6w
CM001 - Structural, Mechanical and Piping (SMP) Instal.	6w
CS001 - Process Plant Buildings Supply and Install	21w

- Standard Procurement Cycle for long lead procurement packages is 15 weeks from issue of RFP to Award).
- Procurement Cycle for contracts is 11 weeks from issue of RFP to Award.
- Lead times for packages are based on in house data.

Construction

The construction schedule was developed assuming an integrated EPCM team. The construction schedule is based on the following:

- Contracting strategy for the Project is based on regional Serbian contractors.
- Working day is based on a single ten (10) hour shift, i.e. no night shift.
- Construction is based on a six (6) day work week and ten (10) hours per day. Construction observes Serbian statutory holidays.

Pre-Commissioning and Commissioning

Pre-commissioning and commissioning activities were defined by the team and incorporated into the schedule.

Risk and Contingency Provisions

Currently no risk contingency is included in the schedule.

24.1.2.6 Schedule Summary and Overview

A summary version of the schedule is presented in Figure 24.1.

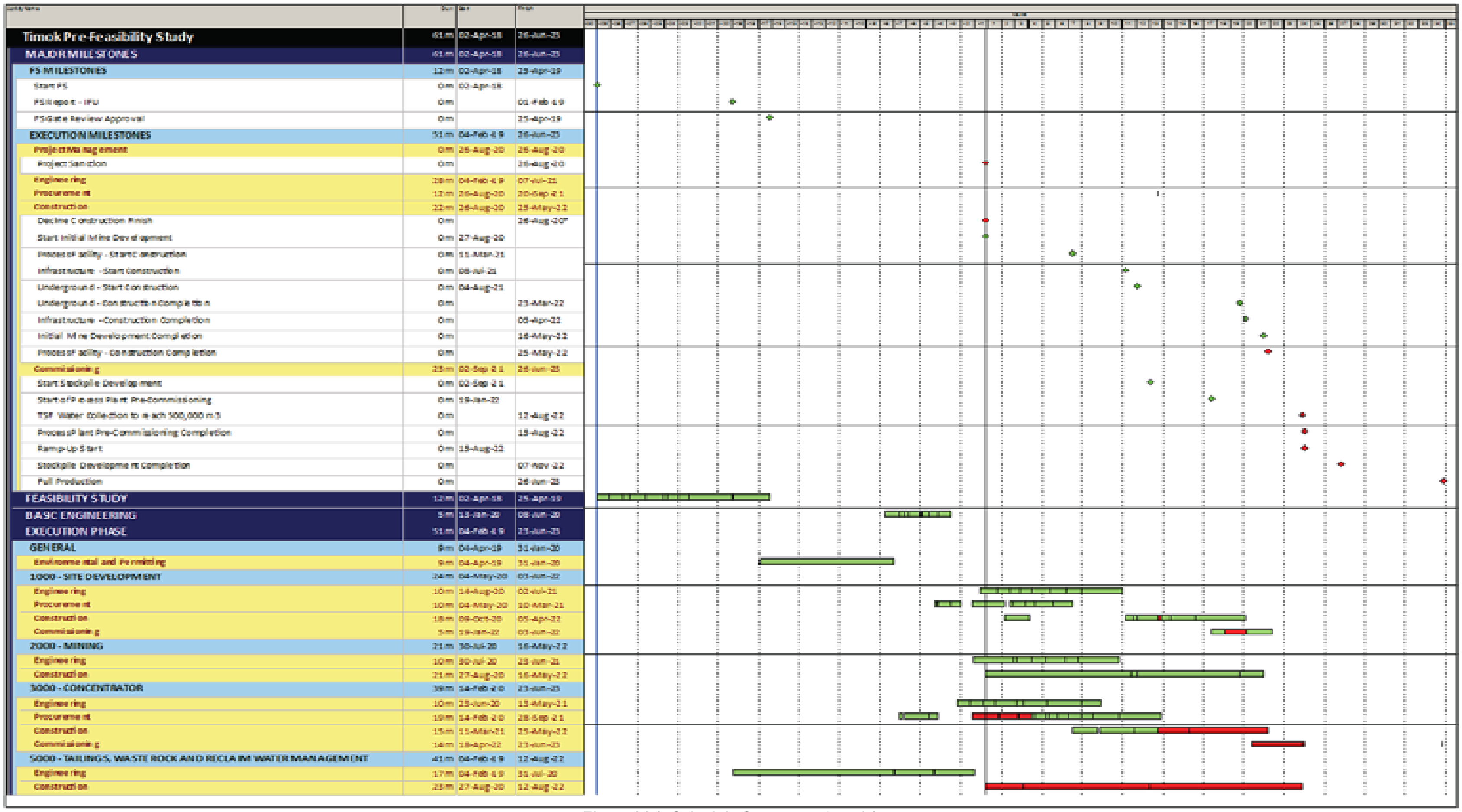


Figure 24.1: Schedule Summary – Level 1

24.1.2.6.1 Key Milestone Dates – (constrained by Sanction Date)

Refer to the table below for project milestone dates:

Table 24.3: Key Milestone Dates

Milestone Description	Start	Finish
Sanction Date	Month 1	
Decline Construction Completed	Month 1	
Detailed Engineering	Month -2	Month 11
Procurement First Commitment	Month 1	
Construction First Commitment	Month 1	
Procurement – First Equipment on Site	Month 13	
Construction - Infrastructure	Month 11	Month 21
Construction – Process Facility	Month 8	Month 21
Construction - TSF	Month 4	Month 15
Water Collection - TSF	Month 13	Month 24
Construction – U/G	Month 13	Month 20
Process Plant – Pre-Commissioning	Month 18	Month 24
Mine Development	Month 1	Month 21
Stockpile Development	Month 13	Month 27
Ramp-Up	Month 24	
Full Production	Month 34	

24.1.2.6.2 Critical Path and Near Critical

The Critical Path includes activities with ten (10) days of float in the schedule. Critical path activities include (refer to Figure 24.2):

- The Sanction Date.
- Award of long lead packages for the Process Plant.
- Process plant construction.
- TSF Water collection.

Activity Name	Dur.	Start	Finish
Timok Pre-Feasibility Study	733d	29-Jul-20	26-Jun-23
MAJOR MILESTONES	713d	26-Aug-20	26-Jun-23
EXECUTION MILESTONES	713d	26-Aug-20	26-Jun-23
Project Management	0d	26-Aug-20	26-Aug-20
Project Sanction	0d		26-Aug-20
Procurement	260d	26-Aug-20	20-Sep-21
Procurement Packages - First Commitment	0d		26-Aug-20
TSF Construction Contract Award	0d		26-Aug-20
Procurement - Critical Equipment/Material Received at Site (RAS)	0d		20-Sep-21
Construction	519d	26-Aug-20	25-May-22
Decline Construction Finish	0d		26-Aug-20*
Process Facility - Construction Completion	0d		25-May-22
Commissioning	272d	12-Aug-22	26-Jun-23
TSF Water Collection to reach 500,000 m3	0d		12-Aug-22
Process Plant Pre-Commissioning Completion	0d		15-Aug-22
Ramp-Up Start	0d	15-Aug-22	
Stockpile Development Completion	0d		07-Nov-22
Full Production	0d		26-Jun-23
EXECUTION PHASE	732d	29-Jul-20	23-Jun-23
1000 - SITE DEVELOPMENT	164d	20-Sep-21	08-Apr-22
Construction	144d	20-Sep-21	16-Mar-22
General Utilities (Air/Water/Effluent Treatment)	144d	20-Sep-21	16-Mar-22
Commissioning	44d	16-Feb-22	08-Apr-22
General Utilities Pre-Operational Testing	44d	16-Feb-22	08-Apr-22
3000 - CONCENTRATOR	732d	29-Jul-20	23-Jun-23
Procurement	280d	29-Jul-20	20-Sep-21
Long Lead Packages	280d	29-Jul-20	20-Sep-21
Long Lead Equipment Packages - Award	20d	29-Jul-20	26-Aug-20
Long Lead Equipment Packages - Preliminary Layout and Load Data	20d	27-Aug-20	24-Sep-20
Long Lead Equipment Packages - All Certified Data	30d	25-Sep-20	06-Nov-20
Long Lead Equipment Packages - Fabrication and Delivery to Site	210d	09-Nov-20	20-Sep-21
Construction	204d	20-Sep-21	25-May-22
Mechanical, Platework and Piping Installation	156d	20-Sep-21	30-Mar-22
Electrical and Instrumentation Installation	145d	29-Nov-21	25-May-22
Commissioning	310d	18-Apr-22	23-Jun-23
Process Plant - Pre-Operational Testing	86d	18-Apr-22	15-Aug-22
Start of Milling	0d	13-Aug-22	
Mill Reaches 100% Throughput	0d		23-Jun-23
5000 - TAILINGS, WASTE ROCK AND RECLAIM WATER MANAGEMENT	487d	27-Aug-20	12-Aug-22
Construction	487d	27-Aug-20	12-Aug-22
Pre-Production - TSF Started Dam Construction Contractor Mobilization	57d	27-Aug-20	17-Nov-20
Pre-Production - TSF Started Dam Construction	245d	18-Nov-20	18-Nov-21
Pre-Production - Water Collection to reach 500,000 m3	240d	01-Sep-21	12-Aug-22

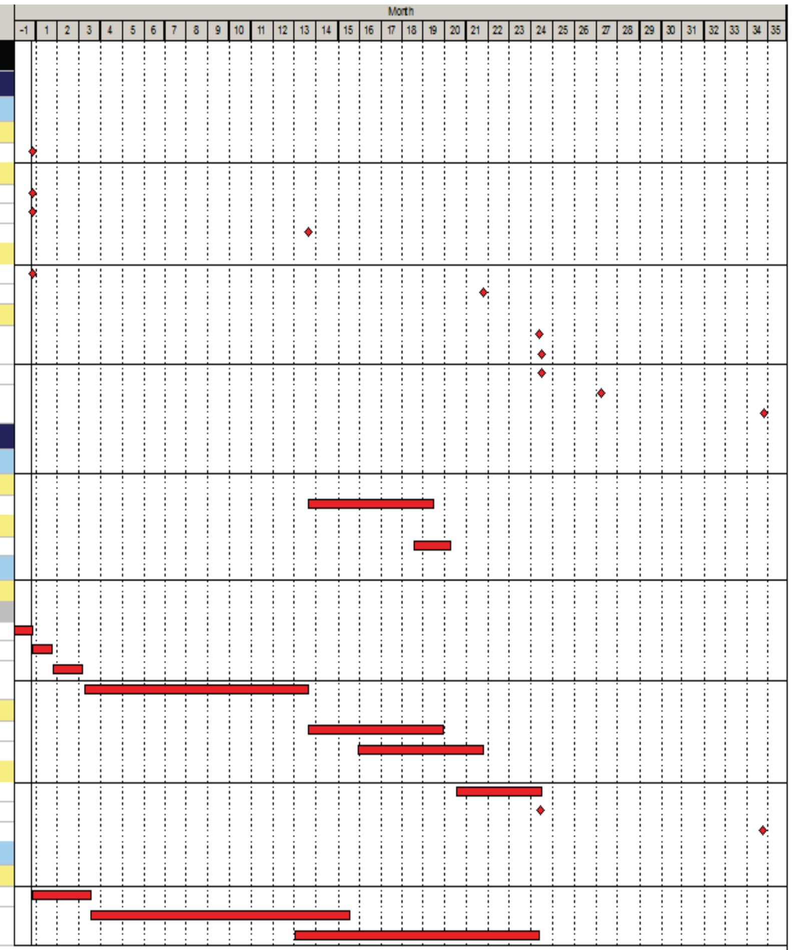


Figure 24.2: Critical Path

24.2 Risks and Opportunities

Effective risk management is integral to the capital investment cycle, from evaluation of a business development opportunity through basic and detailed engineering, Project execution, operations and, ultimately, closure. A structured and thorough understanding of the key risks of the investment allows the Project team to focus their attention and allocate resources effectively.

Aligned with the Nevsun Resources Ltd risk management guidelines and the Hatch risk management framework, the concept of risk during the Pre-Feasibility study considered all aspects related to mining, geology, metallurgy, permits, land acquisition, cost and schedule, environment, communities, health and safety, human resources, Project strategy and economics.

The primary focus of risk management was to update the Project risk register developed in the previous Project phases to ensure that risk information was valid, accurate and complete, and all Project risks (threats and opportunities) which could affect the achievement of the Project and business objectives were identified and evaluated. The risk register update was undertaken with key Project stakeholders from Nevsun, SRK, Knight Piésold and Hatch.

Figure 24.3 and Figure 24.4 provide a summary of the Project risk profile at end of PFS. In summary, ninety-six (96) risks remained open, of which 92 threats and four (4) opportunities.

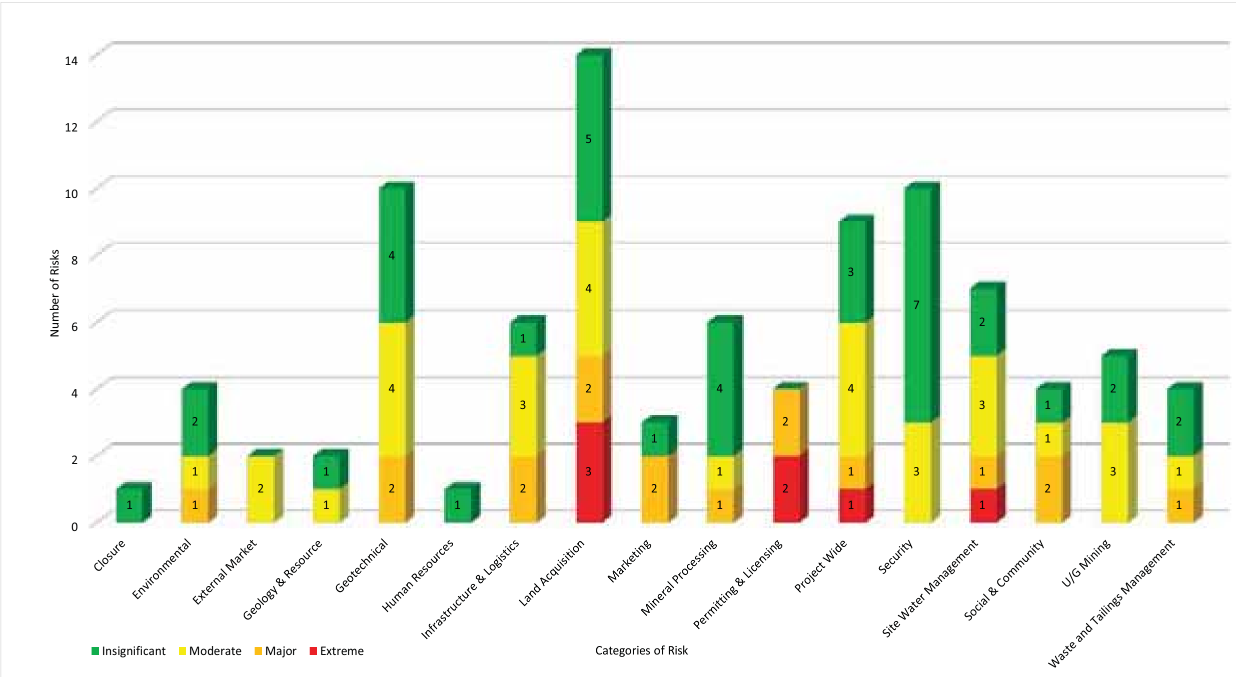


Figure 24.3: Timok PFS Project Risk Profile (Threats)

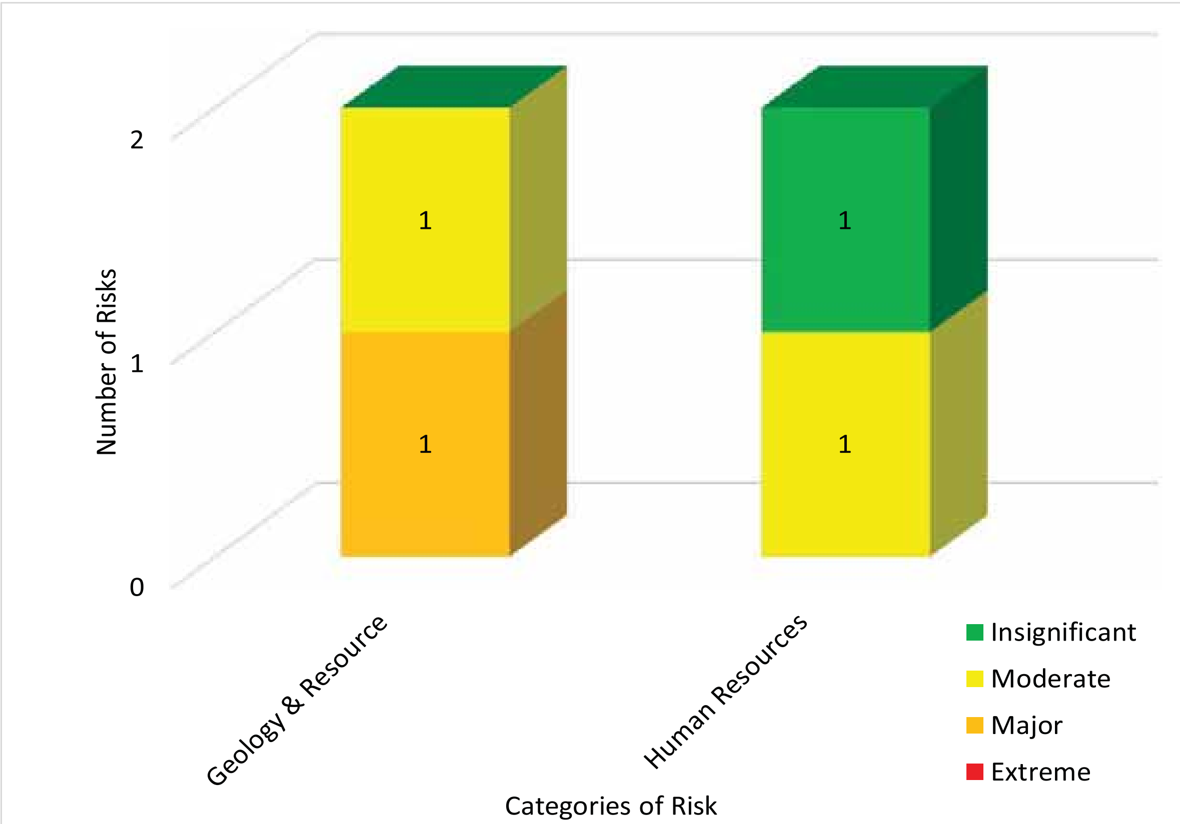


Figure 24.4: Timok PFS Project Risk Profile (Opportunities)

The most significant risks to the Project at end of the Pre-Feasibility study are shown in Table 24.3. These risks are related to land acquisition, permitting and licensing, site water management and engineering.

Note that subsequent to the Risk Assessment Workshop, Nevsun decided to advance the Project sanction date from Q3 2021 to Q3 2020. The impact of this strategy is an increase in the liklihood of the a schedule delay due to not having permits in place before the sanction date. The Project risk profile presented in this document does not reflect the advanced sanction date.

Table 24.4: Extreme Project Risks

Risk Number	Risk Category	Risk Name	Risk Description	Risk Causes	Consequences	Existing Controls	Risk Ranking	Future Mitigation Options
PWD05-D	**Project Wide**	Inability to integrate and satisfy legislated engineering requirements	Failure to produce documentation and stamps as required by local regulations	1. Interpretation of legislation 2. Documents need to be produced in Serbian 3. Uncertainty of what legislation needs to be followed	1. Schedule delays 2. Cost increases 3. Permit delays	1. Preliminary surveys of existing local engineering companies	23	1. Align local and external engineering contractors 2. Form an integrated team
WMG16-D	**Site Water Management**	'Discharge of non-compliant water to Brestovac River	Deterioration in decline water quality on contact with the ore body resulting in inability to direct discharge Brestovac after settlement	1. ARD caused by contact with ore and air 2. Sulphates from drainage of the ore body	1. Current water treatment regime will not handle ore body contaminants	None	23	1. Quantify risk and volume of water from UG workings 2. Apply for construction of water treatment plant if necessary 3. Do not access ore body until WTP is constructed

Risk Number	Risk Category	Risk Name	Risk Description	Risk Causes	Consequences	Existing Controls	Risk Ranking	Future Mitigation Options
PLC21-D	**Permitting & Licensing**	Community opposition to Project	Lack of understanding of the Project impact on the Community causes opposition to the Project	1. Community opposition to the TSF site (particularly Slatina village) 2. Lack of knowledge of the Project impacts on the community 3. Fears about contamination to Brestovac river	1. Delays in SPSP & EIA approvals because of the opposition during the hearings and pressure on authorities 2. Negative community perception creating problems and delays in the Project execution and operation phases 3. Project Sanction Date delay	None	24	1. Develop Stakeholder Management Plan 2. Unroll communication campaign to demonstrate positive Project impacts on community 3. Highlight Project advanced environmental design criteria to alleviate fear of negative environmental impact
PLC41-D	**Permitting & Licensing**	Government support to the Project	While authorities publicly support the Project, on the tactical level there are many obstacles in moving the permit process forward	1. Lack of understanding of Project benefits to country and population 2. Lack of alignment on Project benefits between government and Nevsun	1 Project Sanction Date delay 2. Delay to the Project due to late permit issuance 3. Slow permit processing by authorities	None	24	1. Develop Stakeholder Management Plan 2. Unroll communication campaign to demonstrate positive Project impacts on country 3. Escalate the permit process delays to the highest levels of government

Risk Number	Risk Category	Risk Name	Risk Description	Risk Causes	Consequences	Existing Controls	Risk Ranking	Future Mitigation Options
LAN02-D	**Land Acquisition**	Private Property Ownership Hold-Outs	Unwilling sellers will cause necessity to Expropriate private land	1. Land purchasing Offers are not attractive 2. Speculators purchase land to benefit/hold Project hostage	1. Project permitting delay	1. increased land purchase price and offered bonus	24	1. Use Stakeholder management plan and subsequent communication campaign to project positive impact on the community 2. Design the facility to avoid speculator owned land
LAN23-D	**Land Acquisition**	Airport Purchase	Pre-agreement for airport future needs to be reached prior to SPSP approval. This agreement is expected to outline the future of the airport; outright purchase and decommissioning, or replacement	1. Serbian permitting process 2. SPSP scoping decision requires pre-agreement 3. Expectations of Municipality	1. Permit and Project delays 2. Budget overrun	1. Initiated discussions with municipally and State on airport resolution	25	1. Evaluate preparation of Airport Feasibility Study to facilitate discussions with municipality and State

Risk Number	Risk Category	Risk Name	Risk Description	Risk Causes	Consequences	Existing Controls	Risk Ranking	Future Mitigation Options
LAN25-D	**Land Acquisition**	Speculators	Speculators purchase properties in the Project footprint with unwelcome intentions	1. Land purchasing Offers are not attractive 2. Speculators purchase land to benefit/hold Project hostage 3. Potentially for nationalistic resource beliefs	1. Permit and Project delays 2. Budget overrun	1. increased land purchase price and offered bonus 2. Sped up the offer process 3. Built Tax case against Speculators based on their use of illegal contracts 4. Phantom Zone purchases 5. Designed around many parcels	25	1. Use Stakeholder management plan and subsequent communication campaign to project positive impact on the community 2. Design the facility to avoid speculator owned land 3. Expropriation as a last resort

Where practical, the Project team recommended future mitigation plans to reduce the risks to a level that falls within Nevsun's tolerable limits. The Project team will proactively use the Project risk register and risk profile to ensure their focus and efforts are directed to those areas of uncertainty and risk that need to be improved and managed carefully to enable the business to achieve their strategic goals and objectives.

Document Number	Title	Revision
H356045-0000-140-131-0001	Project Risk Register	0
H356045-0000-140-066-0002	PFS Risk Review Report	0
H356045-0000-140-066-0002-AP0B	Project Risk Matrix	0

24.3 Timok Lower Zone Opportunity

Prior to this study and the drilling program that was completed in February 2018, it was uncertain whether the Lower Zone could be considered as satisfying the criteria of having a reasonable prospect of eventual economic extraction. Now that the Upper Zone has been proven to be economic, it is likely that at least some of the infrastructure could be re-purposed for the Lower Zone, such that there is now a reasonable prospect of economic extraction for at least part of the Lower Zone. As such, Nevsun has completed an initial mineral resource for the porphyry copper portion of the Lower Zone. On the basis of this resource, future work will focus on determining how much of this resource is potentially minable by conducting preliminary scoping and PEA studies.

25. Interpretation and Conclusions

25.1 Geology and Mineral Resources

The Upper Zone of Čukaru Peki is a high sulphidation copper-gold deposit with sufficiently high grades to be an underground mining target, which is at a relatively advanced stage of drilling and geological understanding. Significant infill drilling from surface and updated 3D geological modelling has considerably improved geological confidence, local scale geometry of the mineralization and grade distributions in the Mineral Resource model since the March 2016 statement.

The geological interpretation used to generate the Mineral Resource presented herein is generally considered to be robust; however, there are minor areas of lower geological confidence which may be subject to further revision in the future. In addition, SRK notes there is potential to find further HS and porphyry bodies within the surrounding permit area, which, with additional exploration drilling, may add to the reported Mineral Resource.

SRK considers that the exploration data accumulated by Rakita is generally reliable and suitable for the purpose of the current Mineral Resource estimate.

The Lower Zone of Čukaru Peki is a porphyry copper-gold deposit of sufficient tonnage and grade to be an underground bulk mining target which is at a relatively early stage of drilling and geological understanding. Additional geological drilling and understanding is necessary to better define the mineralization within the Lower Zone. Once a better understanding is achieved through the current re-logging program, a more robust Mineral Resource estimate will be prepared.

25.2 Mining

Based on design progressions and client reviews/discussions, the study authors make the following recommendations, which should be addressed in the next phase of engineering:

25.2.1 Mine Design

- Mine access will be accomplished via dual declines from the south side of the deposit.
- The Upper Zone deposit can be successfully mined using the SLC mining method. The currently defined deposit has the potential to develop a mine with an annual production rate of 3.25 Mt (inclusive of production and development ore). Proper management of the SLC during operation is critical to the success of the mining method.
- Ore will be mined via LHD to an ore pass system, and then re-handled to one of two underground jaw crushers. These crushers will then feed crushed ore to a system of conveyors that will lead to the Process Plant. The two underground jaw crusher chambers will be developed at different stages of mine production.

- A total of 24 km of capital lateral development, 40 km of operating lateral development, and 3 km of vertical development will be required to fully develop the Upper Zone deposit.
- The mine design is based on proven industry standards and productivities achieved at other similar SLC mines around the world. The level of engineering is appropriate to declare Mineral Reserves, once proven economically viable by financial modelling.
- The mine design considers leaving a blanket of ore to buffer the entry of dilution into the cave. This ore blanket must be maintained for the operation as planned to be successful. The planned ore blanket in this study carries grade, thus allowing for “dilution” (material external to the original ring design) to still be value accretive.
- The ventilation design for this project is based on internationally recognized standards and practices in the field of mine ventilation, and in compliance with Serbian mining regulations where applicable. A maximum of 430 m^3/s of unrefrigerated air flow will be required during mining.

25.2.2 Life of Mine Plan

- The Upper Zone will be mined by SLC mining methods over a 10-year mine life (excluding pre-production years) at a maximum rate of about 3.25 Mtpa. In the final years of production, the annual mining rate decreases to account for the reduced extraction rings. As well, due to the nature of SLC, towards the end of mine life the dilution in the mined material increases to the point where the material extracted has insufficient grade to sustain the operation. This has resulted in the 12-year mine life being truncated in the economic model to a 10-year mine life.
- LOM plan dilution (material not included within the original ring design) is estimated at 27%. SRK notes that some of the dilution material carries grade as is above cut-off, but was not included in the original mine design to allow for a dilution blanket above the cave during operations.
- Adherence to the draw strategy is critical for successful extraction of the ore body. Lack of adherence to the draw strategy can result in early entry of dilution lowering the feed grade to the mill and resulting in loss of ore as well as introducing increased risk of mudrush. Early lack of adherence to the strategy is difficult to correct later in mine life and will have a cascading effect to subsequent levels of the mine.
- Poor ground conditions, in particular during the early years of production, could result in slower than anticipated ramp up to full production and increased costs.
- Should the orebody geometry proves to be more complex or smaller than currently modelled, a mine production rate of 3.0 Mtpa may be difficult to achieve at certain stages

in the mine's life The risk of this increases during the later years as the orebody narrows, reducing the footprint of the cave.

25.3 Mineral Processing and Metallurgical Test Work

The 2017 metallurgical program covered flotation optimization, variability testing, testing of simulated annual blends, comminution testing, concentrate and tailings characterization, oxidation and environmental testing and mineralogy. Four trade-off studies were completed to provide guidance on concentrate marketing, gold recovery from pyrite, arsenic reduction in the complex copper concentrate and concentrate transportation.

For much of the recent metallurgical program, it was planned to develop and optimize the flowsheet selected in the 2016 PEA, which produced two copper concentrates, a low arsenic and a complex concentrate. However, during variability testing, it was realized that a significant proportion of the orebody did not respond well to this flowsheet. Testing, and analysis of a flowsheet producing a single bulk concentrate was then carried out, and in conjunction with the project's marketing consultants, a decision was made to base this PEA study on a simplified, more robust flowsheet that yields a single bulk concentrate.

Significant test work and engineering efforts were undertaken to establish an economic route for gold recovery from pyrite. However the low gold content of the pyrite in all but the initial years of production, the up-front capital required to produce a pyrite concentrate and store it separately for later reclaim, and the revenue deferral from this element of the Timok project rendered it uneconomic. It is not proposed to include pyrite recovery and separate storage in the flowsheet for Rakita. Pyrite in the copper tailings will report to the tailings storage facility.

25.4 Tailings and Waste Rock Management

All tailings will be deposited in the TSF. Waste rock will be hauled from the Plant Site and stockpiled in two designated waste rock storage sites within the TSF catchment. The design of the TSF has taken into account the following requirements:

- Permanent, secure and total confinement of all solid waste materials within an engineered disposal facility.
- Control, collection and removal of free draining liquids from the tailings during operations for recycling as process water to the maximum practical extent.
- Management of surface water.
- Staged development of the facility.
- Ongoing monitoring and assessment.
- Appropriate closure measures .

Tailings will be deposited in the TSF as a conventional slurry at a solids content of 31% by mass, with an estimated average settled dry density that ranges from 1.3 to 1.6 tonne/m³. The TSF is designed to accommodate tailings, an operating pond and the Inflow Design Flood (IDF) for the facility.

The TSF embankment construction sequence involves a starter dam with ongoing expansions that utilize the downstream construction method. The TSF basin, including the upstream embankment face, will be lined with geomembrane. A Foundation Drain will be installed within the footprint of the TSF, below the liner system, to collect groundwater flows, potential seepage and infiltration through the TSF embankment. A Basin Underdrain will be installed above the HDPE geomembrane on the lower part of the TSF basin floor to promote tailings consolidation. Water management at the TSF involves reclaim of supernatant water and runoff to the processing plant, with provision for the specified design storm event.

TSF closure will be completed in a manner that will satisfy physical and chemical stability. The primary objective of closure and reclamation will be to return the TSF site to a self-sustaining condition consistent with the local landscape.

The TSF will be designed, constructed, operated and closed to meet local and international standards.

25.5 Marketing

The copper concentrate market is expected to move into deficit over the next four years as the project is developed. The Project has identified smelters, traders, and blenders as potential receivers of the concentrate product. It is reasonable to expect there will be a market for the concentrates produced. The project has also preserved the option to produce a single or dual concentrate and these options will be evaluated in subsequent studies.

25.6 Project Financials

The project is showing robust Economics both with an NPV of $1.8 Billion and IRR of 80% evaluated at the sanction date. Offtake negotiations are ongoing particularly with regard to the arsenic penalties which are subject to increase. Further investigation is planned with regard to a two-phased case whereby the CAPEX is reduced, and producing a less peaked concentrate tonnage profile, thereby reducing the NPV and Capital at risk, however this is unlikely to negatively impact the viability of the project.

25.7 Environmental

The Project is subject to Serbian environmental legislation. Serbia is an accession state to the European Union (EU), and as such, it is working to harmonise its environmental legislation with that of the EU. The Project will adhere to Serbian, EU, and the International Finance Corporation (IFC) environmental and social standards.

A range of environmental considerations have been identified based on the preliminary environmental impact assessment. As is often the case with mining projects, water supply and water quality are the most significant environmental issues. Other issues with potentially significant environmental management implications include noise emissions during portal construction and the deterioration or destruction of habitat of listed species.

Rakita has evaluated all private properties included in the Project footprint, and has provided offers to 98% of the identified property owners. Rakita owns approximately 43% of the private lands within the Project footprint.

26. Recommendations

26.1 Geology and Mineral Resources

SRK considers there to be good potential to further improve confidence in the reported mineral resource at the Čukaru Peki Upper Zone deposit with additional drilling and further modelling work. In relation to drilling and sampling, SRK would recommend the following:

- Conduct infill drilling either from surface or from underground, to achieve 25 m coverage to convert the remaining Inferred resources to Indicated and convert more of the Indicated to Measured resources and to further constrain and geotechnically characterize the steep faults that bound the mineralization.
- Perform geological, structural, alteration and mineralization style modelling incorporating the results of LZ drilling, to help to better refine the geological framework and feeder pathways in the relatively poorly sampled depth extents and margins of the UZ.
- Ensure aqua regia digest is used for arsenic analysis in future exploration programs, given the under-reporting of results using the four-acid digest, which has been corrected for in this estimate.
- Investigate further the significant variance in the results shown for arsenic in CRM RAK4 and RAK5, possibly by additional round-robin analysis and mineralogical study.
- Source and use additional CRMs whose copper and gold grade cover the top-end of the higher range (1 to 15 g/t Au and 2 to 18% Cu) to further add to the confidence in laboratory accuracy at this grade range.
- Whilst SRK has a high overall confidence in the block tonnage and grade estimates in the well drilled parts of the UHG and other domains, (for completeness) select 5 to 10% of the 2011 to 2014 sample pulps from the UHG domain and re-submit these with the current QAQC standards to provide a retrospective validation of the arsenic and high-grade copper assays reported at that time.
- Nevsun proposes to complete two infill drillholes to improve classification in targeted parts of the deposit. These will be completed once access is gained from underground development. There is a further proposal to sample a previously completed metallurgical hole that will also provide infill assay information.
- SRK recommends that, for better understanding of the Lower Zone mineralization, an incline drilling program is conducted once Upper Zone access has been establshed.

26.2 Mining

- Geotechnical conditions relating to the ore passes should be reviewed further at the next stage of study. There is a risk if ore passes are located in poor ground that the ore passes may not be stable for their designed life and therefore may not be practical, resulting in the need for the ore handling strategy to be reconsidered at the next stage of study.

- Additional trade-off studies are recommended to assess the size and locations of the underground crushers, ore handling strategy, ore handling infrastructure options, and waste handling strategy.
- The mine design considers only Measured and Indicated resources. There is significant material above economic cut-off that is classified as Inferred. SRK recommends further investigation of this material to see if it can be converted to Indicated and added to the mine plan. Mine design should take into account this material and preserve the optionality to extract it when in operations.
- As some of the dilution is mineralized and because the there is a dilution blanket left above the first extraction level, SRK agrees to use 150% draw on third level. The impact of high draw rate and dilution assumptions should be modelled in greater detail during the next study. Extraction rates assumed for this study must be revisited in feasibility study and operations to ensure overdraw does not result in loss of the ore blanket.
- Infill drilling is recommended prior to the start of production to better define the shape and grade distribution of the mineralization, to increase the level of confidence in the estimated mineral resources, and to optimize the mine plan (based on the results of the drilling program). Such drilling would also provide better information of the extent and location of any poor quality ground.
- SRK recommends in future stages of study that the final years of production are assessed for economic viability and the mine plan is adjusted according to non-economic production periods.

26.3 Mineral Processing and Metallurgical Test Work

26.3.1 Upper Zone

The following work is recommended during the FS stage:

- Run mini flotation pilot plant at XPS | Expert Process Solutions.
- Resolve the lower copper recovery, using the two-concentrate flowsheet, in the Group 3 variability samples and quantify its significance for the project.
- Further comminution and flotation variability tests, including those in progress at MMI.
- Following further flotation testing of single and multi concentrate production make a final decision on plant configuration and optionality.
- Conduct further concentrate characterization.
- Investigate arsenic removal from the complex concentrate by partial roasting, including stabilization for final storage.

- Finalise the investigation into gold recovery from pyrite under conditions of low-sulfide oxidation using the Albion process.
- Quantify implications of resource oxidation in the broken ore in the underground mine.
- Quantify implications of concentrate heating on concentrate transportation

26.3.2 Lower Zone

Following are the recommendations provided by Aminpro for future test work:

- Perform a comparative check on ore floatability on fresh samples versus reject sample.
- Renew the search for more effective/selective reagents that promote Cu and Au.
- Perform variability test work on rougher feed material from all three ore types to obtain grade vs. floatability information.
- Re-visit the cleaner test work with improved reagent scheme to depress pyrite.

26.4 Surface Infrastructure

As part of the feasibility study, the Company will complete a logistics study to evaluate methods for transporting copper concentrate from the Timok site to a port of export. The study will include the involvement of an international logistics company with experience in Serbia.

The study will assess transport costs for the various transport options as well as the capital cost for any new facilities that may be required for the project including facilities located at the Timok site, along the in-country transport route and at the port of export.

26.5 Tailings Storage Facility

The main recommendations related to the TSF are as follows:

- Conduct additional geotechnical investigations to refine the material parameters used in the TSF stability analysis to better define the stratigraphy of the clay foundation and embankment fill material parameters.
- Conduct a site specific seismic hazard assessment to further define the seismic conditions influencing the TSF.
- Optimize TSF embankment design to include opportunities to reduce basin shaping and embankment quantities, including operational efficiencies and deposition strategies.
- Conduct additional stability analyses to address staged construction geometry and pore pressures.
- Conduct a dam breach analysis to assess the incremental consequences of a potential TSF failure.

- Prepare a site-specific hydrometeorology report to refine the key hydrometeorologic parameters for the project.
- Run the TSF water balance model with variable climate parameters to quantify the effects of a full range of climatic conditions, especially wet and dry situations, and conduct sensitivity analyses with respect to the groundwater inflow from the underground works.
- Incorporate results from geochemical testing into the TSF design and water balance.

26.6 Marketing

The following recommendations are made relating to marketing aspects of the project:

- The PFS flow sheet allows for the production of a single bulk concentrate as well as a two product configuration of a clean copper concentrate (with a low level of arsenic) and a complex copper concentrate (with relatively higher level of arsenic). Ongoing trade-off studies should assess whether producing two streams of concentrate will result in lower realization costs than producing a single stream of concentrate.
- The company should further advance a study on the potential use of a roaster to decrease the arsenic content in the higher arsenic concentrate via the production of a Calcine as the sellable product.

26.7 Project Financials

The following recommendations are made:

- It is recommended that the Project proceed to a Feasibility study stage due to its robust economics.
- Negotiations should continue with arsenic removal enabled smelters to ensure full offtake of concentrate.
- To minimize capital at risk, a two-phased case could be investigated whereby the CAPEX is reduced, and producing a less peaked concentrate tonnage profile, thereby reducing the NPV and Capital at risk. If desirable, a PFS level Investigation of this case is recommended prior to the Feasibility study stage.

26.8 Environmental

The following recommendations are made:

- Develop and implement an environmental management system in compliance with ISO 14001 covering the construction, operating and closure phases of the project. Assign appropriate Rakita personnel and material resources to operate and maintain the environmental management system. Develop and implement the specific environmental management plans (EMPs) focused on the decline construction works as a first priority. Require contractors to develop and implement their own EMPs incorporating the relevant components of Rakita's EMPs as minimum performance standards. Make the foregoing contractually binding on contractors.

- Complete the environmental baseline characterization work needed for environmental impact assessment (EIA).
- Complete a draft EIA Scoping Report in accordance with Serbian and international requirements. Before submitting the Scoping Report to the Serbian regulator, conduct an internal review of the draft Scoping Report with particular focus on the proposed environmental and social management measures and other company commitments. Evaluate environmental and social criteria as part of the engineering alternatives analyses and trade-off studies, and document same in the EIA report.
- Conduct stakeholder engagement.
- It is recommended a pre-agreement for the airport is reached prior to SPSP approval. This agreement is expected to outline the future of the airport; outright purchase and decommissioning, or replacement.

The project is exposed to potential permitting delays. Nevsun will continue updating and implementing the project permitting plan and its impact on project execution. Develop the technical documentation required for the outstanding permit applications. Integrate the permitting schedule into the overall project execution schedule. Maintain relationships with the relevant national ministries and the municipality.

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28. Date and Signature Page

This Technical Report was written by the following "Qualified Persons" who have each provided a signed "Certificate of Qualified Person". The effective date of This Technical Report is June 19, 2018.

Qualified Person	Signature	Date
Jarek Jakubec, CEng	"original signed"	August 7, 2018
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Reviewed by

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Gary MacSporran, P.Eng.
Project Reviewer

All data used as source material plus the text, tables, figures, and attachments of this document have been reviewed and prepared in accordance with generally accepted professional engineering and environmental practice.

Appendix A

List of Abbreviations and Acronyms

List of Abbreviations and Acronyms	
PFS	Preliminary Feasibility Study
RscNSR	resource net smelter return
amsl	above mean sea level
CDI	Conveyor Dynamics Inc.
PEA	Preliminary Economic Analysis
ToS	Trade-off studies
Cu	Copper
Au	Gold
Au-Ag	Gold/silver
Au-Cu	Gold/copper
Pb	Lead
Mtpa	Millions of tonnes per annum
Kt	Thousands of tonnes
Klbs	Thousands of pounds
KOz	Thousands of ounces
As	Arsenic
kV	Kilovolts
Hz	Hertz
CuCov	Covellite
CuEn	Enargite
SLC	Sublevel Caving
SRK	SRK Consulting
TSF	Tailings Storage Facility
IDF	Intensity-Duration-Frequency
AACE	Association for the Advancement of Cost Engineering
WBS	work breakdown structure
OPEX	operating costs
SUSEX	sustaining capital costs
BFDs	block flow diagrams
PFDs	process flow diagrams
PDC	process design criteria
MTOs	Material take-offs
MEL	Mechanical equipment list
SLD	Single line diagrams
TMC	Timok Magmatic Complex
PGA	peak ground accelerations
SPSP	Special Plan for Special Purpose
MICT	Ministry of Infrastructure, Construction and Transportation
IAUS	Institute for Architecture and Urbanism
SEIA	Strategic Environmental Impact Assessment
SCORR	State Certificate of Resources and Reserves.
EIA	environmental impact assessment
ABTS	Apuseni-Banat-Timok-Srednogorie
TMMB	Tethyan Magmatic and Metallogenic Belt
BEM	Balkan Exploration and Mining

List of Abbreviations and Acronyms	
UHG	ultra-high grade
QAQC	quality assurance/quality control
CRM	certified reference material
ETC	European Test Control
LA	lower andesite
UA	Upper andesite sill
MCS	Miocene clastic sediments
SLOS	Sub Level open stoping
TEM	Technical Economic Model
FAR	fresh air raise
RAR	return air raise
AMCA	Air Movement and Control Association
LHD	Load/Haul/Dump (underground ore handling equipment)
PRV	pressure reducing valve
PMPC	portable mine power center
MCC	Motor control centre
CCR	central control room
PCS	process control system
FIPs	Fibre interface panels
PoE	Power over Ethernet
CCTV	Closed-circuit television