NI 43-101 Technical Report

Preliminary Economic Assessment

Čoka Rakita Project

Eastern Serbia

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1 EXECUTIVE SUMMARY

1.1 Introduction

Dundee Precious Metals Inc. (DPM) is an international gold mining company with headquarters in Toronto, Canada, operations in eastern Europe (Bulgaria) and southern Africa (Namibia), and ongoing exploration and development projects in Serbia and in Ecuador.

The Čoka Rakita Project (Čoka Rakita, or the Project) is an exploration project which is 100% owned by DPM. The Project is located in eastern Serbia, about 25 km from Bor, a copper mining centre with a population of about 40,000, and also about 3 km from DPM's Timok Gold Project.

In 2023, DPM retained DRA and other consultants with the mandate to perform a Preliminary Economic Assessment (PEA) of the Project, which is summarised in this Technical Report (the Report). In particular, the PEA and the Report aim to support disclosure of the results of developmental work performed for the Project.

This Report also summarises the results of the maiden Mineral Resource Estimate (MRE) for the Project which has been completed in accordance with National Instrument 43-101 – Standards of Disclosure for Mineral Projects (NI 43-101) and Form 43-101F1 requirements. The maiden MRE for the Project has been defined and classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

DPM engaged Environmental Resources Management Limited (ERM, trading as CSA Global) to complete the maiden MRE for the Project. The maiden MRE was first reported publicly in a DPM news release dated 11 December 2023, which reported an Inferred Mineral Resource of 9.79 Mt at a grade of 5.67 g/t Au for 1.78 Moz. It is reported within a volume, at a 2 g/t Au cutoff which satisfies the requirements of reasonable prospects for eventual economic extraction (RPEEE) by demonstration of the spatial continuity of the mineralisation within a potentially mineable shape.

1.2 Property Description, Ownership, Location and Access

The Project is located in eastern Serbia, approximately 25 km northwest from the town of Bor, a centre for copper mining and smelting in Serbia with a population of about 40,000. The Project comprises one (1) exploration license – Čoka Rakita license (the License) – which was granted on 12 October 2022 to Crni Vrh Resources, a wholly owned subsidiary of DPM. The license area is 14 km² and is issued for three (3) years, with a series of renewals possible for a total potential term of (8) eight years. DPM has an expenditure commitment of €40,229,787 or US $43,678,284 and must meet 75% of this commitment to be eligible to renew the license. The obligations of the license holder are to:

- Complete the submitted and approved work program;

- Provide annual exploration activity reports to the Serbian Ministry of Mining and Energy (MoM&E); and
- Advance the geological knowledge of the Project.

The Serbian government levies a royalty of 5% of Net Smelter Return (NSR) for production of metallic raw materials and a royalty for exploration conducted approximating €88/km^2 or US$95.5/km^2 of the exploration area. There are no other royalties, back-in rights, payments, or other agreements and encumbrances to which the Project is subject. DPM is required to remedy drill roads and pads once drilling is completed unless other agreements are made with the surface landowner. There are no other known environmental liabilities to which the Project is subject.

The Qualified Persons (QPs) are not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project.

The Project is accessible by regional asphalt roads between Bor, Žagubica, Krepoljin, and Zlot, and well-developed unpaved forestry roads. Bor is accessible via the national highway grid, state and paved roads. The Project area is characterised by moderate continental climate, with some influence of high mountainous climate. Winters are long and cold, with abundant snow cover, and summers are usually hot. Access to the Project is possible throughout the year with no seasonal shutdowns of drilling required. Operating mines in the region also do not have seasonal shutdowns.

1.3 History

Prior to DPM, only state-funded exploration is recorded on the Property. State-funded exploration efforts focused on the Dumitru Potok porphyry copper prospect, which is located approximately 1.5 km to the northeast of the Čoka Rakita license. Exploration efforts outlined weak porphyry copper mineralisation which was tested via means of underground drifting and a network of vertical surface drillholes. No historical records exist of the work undertaken.

No other private companies have historically explored on the Čoka Rakita license. DPM has been active in minerals exploration in Serbia since 2004 and acquired several exploration licenses and concessions between 2004 and 2010.

In July 2010, Avala Resources Ltd. (Avala) acquired Avala Resources d.o.o. (formerly named Dundee Plemeniti Metali d.o.o.) from DPM through a reverse takeover transaction, pursuant to which DPM retained an interest in the licenses, by acquiring a 51% share in Avala. In April 2016, DPM subsequently completed the acquisition of the 49% of Avala that it did not own, effectively re-acquiring full ownership of the Property. In November 2021 Avala Resources d.o.o. changed its name to DPM Avala d.o.o.

During 2022, the Potaj Čuka Tisnica license area was decreased, and a portion of the relinquished land was re-applied for by Crni Vrh Resources d.o.o. (Crni Vrh Resources), a subsidiary of DPM, which was granted the Čoka Rakita license on October 12, 2023.

1.4 Geological Setting, Mineralisation, and Deposit Types

Gold-rich skarn mineralisation is hosted within carbonate-rich sandstones and conglomerates, located on the hanging wall of a sill-like body and abutting a monzonite intrusive body to the west. The mineralisation forms a shallow-dipping tabular mineralised body located between 250 m and 450 m below surface, measuring 650 m long, up to 350 m wide, and with variable thickness from less than 20 m in the margins to more than 100 m in the core of the mineralised zone. Coarse gold is often observed in areas of intense retrograde skarn alternation and is found mainly in proximity to syn-mineral diorites within the higher-grade core of the deposit. The current MRE has been prepared on the portion of the Project where gold-rich skarn mineralisation occurs.

1.5 Exploration

Much of the non-drilling exploration conducted on the Project to date has engaged tools that target shallow mineralisation, rather than the deeper skarn mineralisation which is the subject of this Technical Report. Programs of soil sampling, trenching and channelling and geophysical surveys have been completed on the Project.

Geophysical surveys including Versatile Time Domain Electromagnetic (VTEM), Induced Polarisation (IP), electromagnetic response and magnetic signal (TMI), gravity and ground radiometric surveys have been conducted over the Project and neighbouring licenses. These have been used to develop the lithological and structural understanding of the Project and have identified various anomalies.

Soil sampling between 2007 and 2009 identified a series of gold in soil anomalies which were followed up by drilling. 2,592 soil samples have been collected on the license. Trenching (622 m) and channelling (5,163 m) was conducted in 2007-2008 and 2015-2016. These programs identified shallow, structurally controlled, epiclastic breccia hosted gold mineralisation which was found to be highly complex and had poor metallurgical characteristics.

In 2023, a magnetotelluric survey was undertaken over an area of the Project where numerous conductive targets were identified and selected anomalies that may represent deep manto or skarn type mineralisation and this will be tested in future drilling campaigns.

A base geodesic operational network within the Project area has been established that covers the entire area. Drone topographic mapping was carried out and a Digital Terrain Model (DTM) with a resolution of 80*160 cm was generated over the whole area. A detailed Digital Elevation Model

(DEM) has been created by DPM with filtering applied to remove the impacts of vegetation with a final resolution of 2 m grid size.

1.6 Drilling

A total of 173 drillholes for 80,723 m have been drilled since 2008, with the majority drilled since 2021. The drilling has been predominantly diamond (101 for 59,298 m) and diamond tail (24 for 13,469 m) with 48 reverse circulation (RC) drill holes drilled for 7,957 m. RC drilling was completed during 2008 but did not reach the required depth to intercept gold bearing skarn mineralisation and as such, has not been used for grade and mineral resource estimation purposes, however logging data has been used to inform the geological model. RC drilling has more recently been used as pre-collars for diamond tails.

The vast majority of core diameter in the mineralised zones is HQ3 (61.1 mm), and recovery is >98%. Procedures are detailed in DPM's Exploration Procedures Manual (2018). Collar locations are picked up using Total Station or Differential Global Positioning System (DGPS), and downhole surveyed using a Devi Tool digital multi-shot camera or a Devico gyroscope tool, providing measurements every 3 m downhole. Core processing involves photography, logging (geology, structural and geotechnical) and sampled based on sample intervals provided by the Project Geologist. Half core is sampled consistently along sample lines a few centimetres from the orientation line.

Diamond drill holes were included in the estimation of the MRE. The current drillhole spacing within the mineralised domains is approximately 30 m x 30 m in the core of the system, with an up to 60 m x 60 m grid on the periphery.

1.7 Sampling, Analysis and Data Verification

During the period under review, sample analyses were completed at Genalysis Perth, Australia (GEN_PE), ALS Vancouver, British Columbia, Canada (ALS_VA), SGS Bor (SGS_BO), SGS Chelopech (SGS_CH), and ALS Bor (ALS_BO). These laboratories are certified to ISO-standards and are independent of DPM.

Gold grades within skarn domains used in the MRE have been determined systematically using a screen fire assaying technique, which is preferred for mineralisation with coarse gold, and fire assay in approximately 13% of the dataset.

Quality Assurance and Quality Control (QA/QC) were implemented to provide confidence that sample results are reliable, accurate, and precise. Blank material with no mineralised material value, site-specific certified reference material (CRM), site field duplicates and internal (preparation laboratory) duplicates were used as quality control material to monitor accuracy, precision and contamination.

The QA/QC procedures implemented are adequate to assess the accuracy and precision of the assay results obtained. Blank results show no significant indications of contamination. No fatal flaws were noted with the accuracy results. Bias and failures were noted in individual CRMs, but this was not systemic (some bias is positive and some negative). Precision for diamond drill samples was acceptable. Sampling procedures are appropriate and adequate security exists to minimise the risk of contamination or inappropriate mixing of samples.

1.8 Mineral Processing and Metallurgical Testing

In 2021, Wardell Armstrong International performed exploratory testing on five (5) samples from the Čoka Rakita property provided by DPM to investigate their amenability to gravity concentration, cyanidation, and flotation.

Table 1.1 summarises the results of the program indicating that a flowsheet comprising flotation followed by leaching of the float tails can yield gold recoveries exceeding 90%. The highest recovery was achieved for sample Met Ra P05 with a gold recovery of 96.7%.

Table 1.1 – Wardell Armstrong Test Program Summary

Lithology	Sample ID	Head Au Assay (g/t)	Gold Recovery (%)					
			Gravity Conc.	Whole Material Leach	Rougher Flotation	Cleaner Flotation	Flotation Tailings Leach	Flotation + Tails Leach
Retrograde Exoskarn	Met Ra P01	2.68	50.7	88.3	75.1	73.7	73.2	92.9
Phyllic Exoskarn	Met Ra P03	3.91	57.2	88.7	75.8	73.5	74.8	93.3
Retrograde Skarn	Met Ra P05	18.54	63.6	93.0	82.3	81.3	82.3	96.7
Retrograde Endoskarn/ Potassic Porphyry	Met Ra P02	0.55	40.5	88.0	79.5	77.2	76.7	94.7
Epiclastic-Volcaniclastic	Met Ra P04	2.36	35.6	81.7	55.2	51.3	79.7	90.1

Source: Wardell Armstrong, 2021

The BaseMet Labs testwork program in 2023 was performed on three composite samples, representing low, medium, and high gold grades. The BaseMet Labs test program was more comprehensive than the exploratory Wardell Armstrong program and covered many processing aspects including sample mineralogy, gravity concentration, flotation, cyanidation, sedimentation as well as filtration. Extensive testing was conducted to explore and optimise test conditions. The testing culminated in a set of locked cycle tests (LCTs) which simulates the selected flowsheet and employs optimised test conditions.

Table 1.2 summarises the gold recoveries achieved during these LCTs and compares it with a scenario where the tails from this LCT was subsequently leached using cyanide and, finally, against a control test which simply comprised direct cyanidation of the original sample. The result shows that cyanidation of the flotation tailings consistently produced the best gold recoveries. However, DPM did not consider a flowsheet using cyanide appropriate for this project, hence the selected flowsheet uses gravity concentration followed by flotation.

Table 1.2 – BaseMet Labs Test Program (2023) Results Summary

Sample ID	Test Procedure	Head Au Grade (g/t)	Au Recovery (%)	Tails Au Grade (g/t)
MetCRA23-01	Gravity -> Flotation Locked Cycle Test (LCT)	3.33	88.1	0.41
	Gravity-> LCT->Tails Cyanidation	3.33	92.4	0.12
	Direct Cyanidation	3.44	88.4	0.40
MetCRA23-02	Gravity -> Flotation Locked Cycle Test (LCT)	5.93	84.2	0.94
	Gravity-> LCT->Tails Cyanidation	5.93	92.4	0.19
	Direct Cyanidation	6.36	92.1	0.51
MetCRA23-03	Gravity -> Flotation Locked Cycle Test (LCT)	11.0	90.9	1.04
	Gravity-> LCT->Tails Cyanidation	11.0	96.2	0.18
	Direct Cyanidation	10.5	95.4	0.48

Source: BaseMet, 2023

Mineralogical examination of the samples shows that the material consists mostly of garnet (close to 50% w/w) with equal parts of quartz, calcite and feldspar making up most of the remainder. Pyrite was the most prominent sulphide mineral at around 4% of the total mass.

Comminution testing showed that the material can be classified as moderately hard and moderately abrasive. The Axb value average for the three samples tested was 54.2. The average abrasion index (Ai) was 0.138 g.

The series of gravity concentration followed by rougher flotation at different grinds showed that overall gold recoveries do not improve when the grind size was decreased below 80% passing 53 µm.

Analysis of the concentrate sample showed that arsenic is present but below the 2,000 ppm threshold for triggering penalties.

Based on test results the following criteria were adopted with respect to gold recoveries:

- Gravity Recovery = 41 * Gold Head Grade ^ 0.14
- Recovery from gravity tails to flotation rougher concentrate = 88%
- Recovery from flotation rougher concentrate to second cleaner concentrate = 91%

The dilution tests performed to evaluate Jameson cell performance were mostly inconclusive due to the nugget effect, and thus not considered in the trade-off study final selection, and some retesting is included in the recommendations. The dilution procedure was preceded by gravity concentration which yielded significantly variable overall gold recoveries.

Sedimentation testing indicated that both the flotation concentrate as well as the tailings samples are amenable to dewatering by a conventional thickener. Dynamic tests showed that a 1 tph/m^2 loading rate can be used for thickener sizing and can be expected to produce a dense underflow with the solids content exceeding 60% w/w solids. The flocculant addition used during these tests were 50 g/t and 20 g/t for the tailings sample and concentrate samples respectively.

Preliminary filtration testing showed that the tailings sample can be dewatered to 23% moisture by pressing only but that air blowing is required to reduce the moisture content to around 18%. The concentrate sample filtered very well by pressing only reducing the moisture content to 13%.

Environmental testing of the tailings sample yielded a neutralising potential of 107 kg $CaCO_3$/t and an acid generating potential of 2.5 kg $CaCO_3$/t only. The implication is that the TSF does not require lining; however, the design includes provisions for a lined TSF.

Variability testing is recommended for the next phase to improve our understanding of the inherent variability within the deposit and to refine design criteria.

Another important recommendation is to perform comparative flotation testing on a bulk sample which has been subjected to a gravity concentration procedure before it is split into subsamples. This will remove the coarse gold nuggets before the comparative tests are performed and is expected to thereby eliminate the significant variation in overall recoveries.

1.9 Mineral Resource Estimate

DPM implemented an acQuire Geological Information Management Systems (GIMS) for managing all the drillholes and sampling data. The data export supplied undergoes further validation when imported into a relational database using Simple Query Language (SQL). The validated dataset is then exported and used for the MRE review. During the upload process, the data is subject to further validations.

Mineral resource domains were created within volumes of moderate to intense skarn alteration and guided by grade composites over 3 m true thickness averaging 1 g/t Au cut-off value. Detailed lithology and structural models were developed and used to constrain domain extents, as well as to interpret post-mineralisation diorite sills which cut across the mineralisation. Two mineralisation domains were created, with waste domains including late-stage intrusions modelled and estimated separately. Samples were composited to 1 m, which was the dominant sampling length. Top cuts were applied, with gold values exceeding 70 g/t and 24 g/t being capped at those values for the footwall and hanging-wall domains respectively. Semi-variograms were modelled on top cut data for gold and silver and were characterised by moderate nuggets and short to moderate ranges. In-situ dry bulk density (density) was estimated into mineralisation and geological domains using Inverse Distance Weighting Squared (IDW^2).

Gold and silver grades were estimated into mineralisation domains into 10 m x 10 m x 10 m (X x Y x Z) blocks using Ordinary Kriging and with hard boundaries imposed between all domains, for both mineralisation and waste. Optimal block size was informed by Kriging Neighbourhood Analysis. A three-phase search strategy was used with increasing ranges. The Mineral Resource Estimate (MRE) satisfies RPEEE by demonstrating the spatial continuity of the mineralisation based on a 2 g/t Au reporting cut-off grade and smoothed out volumes created by Datamine's Mineable Shape Optimiser (MSO) and was classified as Inferred Mineral Resources.

The MRE was prepared by the QP author, Ms. Maria O'Connor, MAIG, with an effective date of 16 November 2023. The MRE tabulation constrained within the smoothed-out MSO volumes is presented in Table 1.3. Summary assumptions used in RPEEE calculations are presented in the notes of Table 1.3 and are discussed in more detail in Section 14.17.

Table 1.3 – Čoka Rakita MRE Using Underground Mining Scenario

Čoka Rakita Mineral Resource Estimate
Effective Date of 16 November 2023

Mineral Resource category	**Tonnes** (Mt)	**Gold Grade** (g/t)	**Contained Gold** (koz)	**Silver Grade** (g/t)	**Contained Silver** (koz)
Inferred	9.79	5.67	1,783	1.21	382

Notes:

- The cut-off value of 2 g/t assumes US$1,700/oz gold price, 90% gold recovery, 10% dilution, US$79/t operating cost (mining, process and G&A costs), US$7/t sustaining capital cost, as well as offsite and royalty costs.
- Mineral Resources are reported within smoothed MSO underground mining shapes generated at a 2 g/t Au cut-off and a minimum width constraint of 5.0 m x 5.0 m x 2.5 m, to ensure Mineral Resources meet RPEEE. The smoothing process allows for blocks below the cut-off to be included within the final shapes in order to emulate the internal dilution that would be experienced during underground mining as per CIM Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines prepared by the CIM Mineral Resource and Mineral Reserve Committee and adopted by the CIM Council on 29 November 2019.
- The QP author is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that might materially affect the estimate of Mineral Resources, other than those specified below and in Table 1.4.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Figures have been rounded to reflect that this is an estimate and totals may not match the sum of all components.

Factors that may materially impact the Mineral Resource include:

- Changes to price assumptions and input values for mining, processing, general and administrative (G&A) costs and metallurgical recovery and other mining assumptions used to constrain the Mineral Resource.
- Changes to the deposit-scale interpretations of mineralisation geometry and continuity.
- The MRE is very sensitive to the choice of top cuts; therefore, changes to those values would impact the grade and tonnage above the cut-off of the MRE.

- Change to estimation methodology (e.g., to model the high-grade tail) may change tonnage and grade estimates.

The qualitative risk assessment is presented in Table 1.4. The overall risk to the Čoka Rakita MRE is reflected in the current resource classification as Inferred Mineral Resources and is considered moderate, which is consistent with the early-stage nature of the Project.

Table 1.4 – Qualitative Risk Assessment

Factor	Risk	Comment
Sample collection, preparation and assaying	Low to moderate	There are written procedures and data management practices in place. The nature of coarse gold means there is an inherent higher risk relating to and risk associated with sample preparation and analysis, but this is mitigated by the analysis for the vast majority of samples being screen fire assay which requires larger volumes. The majority of the gold is associated with finer fractions, but coarse gold is associated with higher grades.
QA/QC	Moderate	While screen metallics testing is the preferred method for analysing high gold grades in coarse gold environments, the nature of SFA means that direct quality control is less possible than it is for other methodologies. Quality control review has been performed on FA and has indicated no material issues of concern. Insertion of blind standards, duplicates and blanks is recommended.
Geological model	Moderate	Uncertainty in accuracy of location of late-stage intrusives modelled. The fact that core can look very similar in terms of skarnification and intensity of alteration but have different grade character across short distances is notable.
Mineralisation model	Moderate to High	The nature of coarse gold means there is an inherent uncertainty in its location and grade since it can be missed in half-core sampling, and variability at close ranges can be high. The mineralisation has been constrained within moderately to intensely skarnified S1/S2 material and guided by grade composites generated at 1 g/t Au. It is important to retain the geological basis of the interpretation and not be guided only by grade since level of selectivity can be low in this kind of environment. This risk can be mitigated by using larger diameter core barrels such as HQ or PQ to collect more sample for assay analyses and a better representative sample. This can also be mitigated through a bulk sample using closely spaced PQ cores.
Treatment of outliers (grade caps)	Moderate	The MRE is very sensitive to the choice of grade cap. Given the early stage of the Project and broad drill spacing, a relatively conservative grade cap was applied, which cuts 2% of the data and 30% of the metal. When data is top cut (at 70 g/t Au for the largest domain), variograms indicate nuggets that are moderate and not extreme, indicating grade continuity is not extremely low and grade variability is not extremely high.
Location of post-mineralisation intrusives	Moderate	The location of post-mineralisation intrusives represents a low volume but precise location is uncertain based on current broad drill spacing.
Grade estimate	Moderate	The grade estimate has been intentionally smoothed to reflect the uncertainty of the location of coarse gold. Sensitivity to grade estimation methodology is recommended to assess methodology for improved modelling of the high-grade tail.

Factor	Risk	Comment
Tonnage estimate	Low	Density estimate is considered low risk. Volume estimate is moderate risk, associated with uncertainty in the mineralisation model but not unreasonably so, considering the stage of resource development and level of classification.
Permitting risk	Low to Moderate	A potential risk to the Project is associated with permitting delays. Such delays caused by potential changes to Serbian regulations to align with EU Law, regulator delay, public challenge to the Spatial Plan or EIA and administrative appeals. Similar risks have been experienced by other private sector mining projects permitted in Serbia.
Overall rating	Moderate	The current MRE carries a moderate level of uncertainty and risk which is reflected in its classification as Inferred Mineral Resources.

1.10 Mining Methods

1.10.1 General Description of the Deposit and Mining Project

The targeted production rate for the underground mine is approximately 2,300 tpd, which aligns with the process plant capacity of 850 kt annually. The host rock of the deposit is a calcareous clastic sedimentary rock, with mineralisation occurring in skarn-altered calcareous sandstone. The quality of the host rock in the hanging wall generally ranges from fair to good. However, the footwall of the deposit exhibits poor ground conditions. Consequently, DPM plans to position the mine's development and underground infrastructure on the hanging wall side of the deposit.

1.10.2 Mining Method

DPM plans to mine the deposit with sublevel stoping. Most of the mineralised material on each sublevel will be mined with parallel longitudinal stopes extending along the strike of the deposit. Each series of longitudinal stopes requires a production drive on the upper sublevel for drilling and loading explosives and a production drive on the lower sublevel for mucking blasted mineralised material. Once longitudinal stoping is completed on a sublevel, the remaining mineralisation adjacent to a crosscut will be recovered as a series of transverse stopes retreating towards the hangingwall.

The sublevels will be mined in an ascending sequence. The longholes will be drilled in rings angled downwards as inverted fans from the upper production drive and loaded with bulk emulsion explosive. After each stope blast, an LHD will muck the broken mineralised material from the lower production drive of the stope. Mine trucks will haul the material to surface via the spiral ramp and haulage decline. Primary stopes will be backfilled with paste fill, and secondary stopes with either paste fill or rock fill (i.e., development waste).

1.10.3 UNDERGROUND GEOTECHNICAL ASSESSMENT

The underground geotechnical data was limited therefore WSP defined the main rock masses into simple categories that were used for stope stability and ground support assessments. Rock masses were defined as Good quality (Q'>20) to Fair, with limited sections of Poor (Q'<4) quality rock mass. These poorer quality zones are limited in size and location (<1 m to several metres in core length) and will need to be better defined as the Project advances.

Based on these rock mass quality ranges, a longitudinal stope configuration of 20 m high x 20 m wide x 15 m strike Length was considered stable for Good and Fair rock mass quality. In areas where Poor quality rock masses are encountered, additional ground support will be required in the back and hanging wall. Dilution estimates for stopes located in Fair to Good rock mass is estimated at < 0.5 m to 1.0 m, and in Poor rock mass, dilution is estimated at >2 m in stope back, hanging wall and footwall.

The ground support assessment was completed based on a range of underground excavation sizes (4.5 m to 5.5 m) and compared against Potvin (2017) empirical charts, RocScience Unwedge assessment and DPM's current ground support systems employed at the Chelopech Mine, with similar ground qualities to Čoka Rakita Project. Two (2) ground support types were provided one for Good/fair and one for Poor rock mass qualities for both primary support and secondary support. The support systems recommended were based on DPM's support systems at the Chelopech Mine, except for the addition of resin grouted-rebar in permanent excavations.

A simple review was completed of the spiral ramp location (stand-off distance) versus the deposit thickness at five (5) vertical locations. In the widest part of the deposit (80 m to 105 m wide), the spiral ramp is approximately located 30 m to 74 m away in the hanging wall. In these locations there is a potential for increased mining-induced stresses in the spiral ramp as the deposit is fully mined out. These induced stresses could impact the stability of the spiral ramp. Moving the ramp further from the deposit and/or increasing ground support in these locations are recommended methods to reduce potential instabilities. As the Project advances, numerical modelling assessments and trade-off studies will be completed to better understand these potential impacts and if mitigation measures are required.

An empirical crown pillar review (Scaled Span) was completed based on a crown pillar thickness of 162 m high x 17 m wide x 50 m strike length. This crown pillar configuration has a Scaled Span Probability of Failure of < 0.5% and is considered long-term stable.

A temporary sill pillar is planned to separate the upper part of the deposit from the lower part, which will be mined later in the Project schedule. A two-dimensional numerical model was completed using the temporary sill pillar configuration of 40 m high x 25 m wide. A review of the differential stress levels after the upper and lower deposits are mined, indicates the potential for bursting and spalling to occur in the temporary sill pillar. Recommended methods of mitigating the impact of these induced mining stresses are increasing ground support requirements in the sill pillar access drifts, modifying the extraction ratio of the sill pillar and/or modifying the mining sequence. Detailed numerical modelling (3D) is recommended at the next study stage of the Project.

1.10.4 Hydrogeology

Currently, there is insufficient site-specific hydrogeological data for the Project. The groundwater conditions outlined in the PEA are based on studies from the Timok project, located 3 km northwest of the Čoka Rakita Project area. While this information offers a foundational understanding of the anticipated groundwater conditions at the Čoka Rakita Project site, it is crucial to confirm the hydrogeological data during the upcoming hydrogeological and geotechnical field investigations.

1.10.5 Mine Design

The stopes will be 20 m wide (measured perpendicularly to the production drive) and 20 m high. They will be 15 m long and mined in a retreating sequence. The longholes will be 89 mm in diameter and will be drilled in rings with a 2.1 m toe spacing and a 2.1 m burden.

The declines will have a 5.5 m x 6.0 m (width x height) cross-section, and the production drives will measure 4.5 m x 5.0 m in profile. The spiral ramp, level access headings, and crosscuts will be driven at 5.5 m x 5.5 m.

1.10.6 Mine Access and Underground Facilities

The deposit will be accessed by driving twin declines from the east side of the mineralisation. The decline designated as the haulage decline will provide access to the underground for personnel and equipment and will also function as the return-air exhaust for the mine ventilation system. The second decline will serve as the fresh-air intake for the mine ventilation.

The development of a spiral ramp will commence from the end of the decline designated for haulage. It will provide access to the lower levels of the mine and enable the transport of mineralised material and waste to surface. Additionally, level access headings will be developed off the spiral ramp to access the sublevels for mining the deposit. These sublevels will be spaced at 20 m intervals. The ventilation raises connecting the sublevels will be equipped with ladderways, providing an emergency escape alternative to the spiral ramp. Eight (8) portable mine rescue chambers will be strategically positioned throughout the mine.

1.10.7 Mine Equipment

The following list details the maximum requirement for mobile mining equipment, which are all diesel-powered units (battery-electric equipment is recommended for consideration in future project phases):

- Four (4) ea. development jumbos, two (2) boom;

- Two (2) ea. production drill rigs, one tophammer, the other in-the-hole (ITH);
- Two (2) ea. emulsion chargers;
- Six (6) ea. mine trucks, 45 t payload;
- Four (4) ea. LHDs, 15 t tramming capacity;
- One (1) ea. cable bolter;
- One (1) ea. shotcrete sprayer; and
- Two (2) ea. transmixers, 5.6 m³ capacity.

1.10.8 Mine Personnel

The underground mine will reach its peak workforce of 327 employees in Year 2 of the life of mine (LOM) plan. The mine will operate on three (3) 7-hour shifts, the workday length stipulated by Serbian legislation. The workforce will be organised into four (4) rotating groups, each working three (3) shifts followed by a period of days off. As per the mine plan, all development and production activities will be carried out by DPM employees, except raise boring and drilling for service boreholes, which will be contracted out.

1.10.9 Pre-Production Schedule

This pre-production phase will commence in the year before initiating the twin declines. The principal activities will include excavating portals and procuring essential mining equipment. Year -2 of the schedule will focus on driving the twin declines and starting the spiral ramp, with additional progress achieved in developing ventilation drifts, level access headings, and infrastructure excavations. In Year -1, while development will continue in the spiral ramp, ventilation drifts, level access headings, and infrastructure excavations, the emphasis will shift towards advancing crosscuts and production drives in the deposit. Development in waste and mineralisation will continue during Year 1, and stope production will ramp up with commercial production attained in Q2.

1.10.10 Life-of-Mine Plan

The LOM plan extends over 10 years, during which the mine will produce mineralised material at full capacity for eight years. At the end of Year -1, stope production will commence on the 440 sublevel, situated approximately two-thirds down the depth of the deposit. The deposit will be mined in two (2) phases. Initially, the upper zone of the deposit will be mined to deliver mineralised material to the processing plant per the planned schedule. From the 440 sublevel, stoping operations will progressively advance to higher levels in an ascending sequence. The mine will operate at full output during Year 2, producing slightly over 850 kt of mineralised material. In Year 4, sublevel stoping will commence on the 360 level, following the completion of the spiral ramp. Production in the lower zone will progressively supplant that of the upper zone. The mine will continue operating at full capacity from Years 5 to 9. In Year 10, the final year of the LOM plan, production will decrease to approximately 58% of full capacity.

Mine development will progress at an average rate of approximately 4,000 m/year during Years -2 to 2, decreasing to 3,290 m in Year 3. From Years 4 to 8, it will average around 2,000 m annually and cease entirely during the final two (2) years of the mine life. Year -2 will focus on driving the twin declines, the spiral ramp, and level access headings, with additional progress in developing ventilation drifts and infrastructure excavations. In Year -1, the emphasis will shift to include crosscuts and production drives within the deposit. The spiral ramp will remain inactive during Years 1 and 2. However, it will resume development in Year 3, reaching the lowest level of the mine in Year 4. From Years 5 to 8, development will focus on crosscuts and production drives required for stoping operations. All development activities will be completed by the end of Year 8, with no further advance scheduled during the mine's final two (2) years of operation.

1.10.11 Mining Ventilation

Air requirements were determined using the standards and regulations in use in Quebec (CANMET) and Ontario (0.06 m³/s per kW of diesel engine). Total air requirements amount to 261 m³/s, with a minimum of 22 m³/s supplied to a heading.

The mine ventilation layout was modelled with Ventsim™ using the air requirements from a preliminary equipment list. The modelled air requirement is 320 m³/s.

The list of fans at the mine consists of a 1100 kW main fan located near the intake decline entrance, 8 x 110 kW fans distributed over the production levels, 4 x 55 kW fans, and 4 x 20 kW fans for cut-outs.

1.10.12 Mine Backfill

The paste backfill underground distribution system design is driven by the Mine Design and the results obtained from laboratory testing on the Mill tailings, for which testing is yet to be done. Once testing is completed, its results determine the paste characteristics. In particular, the testing determines the pipeline friction losses expected in the distribution system, which in turn establishes the anticipated pipeline pressures and pipeline materials of construction. The pipeline friction losses also serve in the design of the paste pumps required at the paste plant.

Since tailings testing has not yet been done, WSP's database of results on tailings with similar characteristics was used to determine that portion of the underground distribution system design parameters.

The distribution system design will be refined by completing a hydraulic flow model of the entire distribution system, which will better define the underground system design parameters in future phases of the Project.

The paste backfill assessment conducted in this phase of the Project provides a relative indication of the system requirements based on the expected level of accuracy.

1.10.13 Mine Dewatering

The quantities of water considered in the dewatering assessment originate from the following sources:

- Ground water seepage;
- Paste backfill contained water release and distribution system flushing; and
- Water required to conduct mining operations, i.e., for drilling, dust suppression when mucking, and other ancillary water consumption.

Geotechnical drilling has not yet taken place to better determine the groundwater seepage, therefore the hydrogeological team provided estimates based on data from the DPM Timok Mine. Once the geotechnical drilling is completed for the Project, the water seepage estimates will be revised for the dewatering portion if necessary.

Paste backfill contained water release characteristics were estimated based on WSP's database of testing results on tailings of similar characteristics to determine the contained water release of the deposited paste. This estimate will be revised once tailings laboratory testing is completed in subsequent phases of the Project.

Water leaving the mine in mineralised material and waste, as well as the ventilation system or quantities that are generated from diesel fuel combustion emissions, were not considered in this phase of the Project. These elements will be discussed and included in the dewatering assessment if required in future phases of the Project.

In this phase of the Project, carbon steel pipe was considered for use in the decline. In future phases of the Project, an evaluation on the use of HDPE pipe in the decline will be conducted.

The dewatering assessment conducted in this phase of the Project serves to provide a relative indication of the system requirements based on the expected level of accuracy.

1.10.14 Electrical and Communications

Power for the estimated 4 MVA of underground mine loads will be supplied from a surface substation. This substation and associated distribution network steps down the site transmission voltage to the distribution voltage of 10 kV and delivers it to the portal. Cabling will be routed down the declines with two (2) mine power feeders that provide a redundant connection for improved reliability. This redundancy is especially important for critical ventilation and dewatering loads.

At each load centre that supplies power to fixed loads such as mine dewatering and ventilation, an electrical bay (EBay) will be established. Each EBay will house a 10 kV Ring Main Unit (RMU) switchgear that provides feed-through switches for the backbone feeders that drop down to each level and breakers to supply the transformers for 1000 V or 400 V utilisation. The 1000 V output of the transformer will supply power to fixed infrastructure and mining equipment. This voltage was selected to reduce cable costs and the number of EBays required to supply the mining equipment.

Fixed-speed pumps will be controlled by pump controllers or soft starters, and variable-flow fans will be controlled by VSDs. Consideration should be given to supplying fans and pumps at standard voltages, such as 400 V, 690 V or 6000 V to simplify sourcing of equipment.

A fibre optic cable will be routed through the mine to provide a network for process control, wireless communications, vehicle and personnel tracking, surveillance and fire protection.

1.11 Processing and Recovery Methods

Preliminary metallurgical testing showed that the mineralisation is amenable to a flowsheet featuring gravity concentration followed by bulk sulphide flotation. Such a flowsheet has the potential to produce two clean gold concentrates with overall gold recoveries of approximately 90%. The proposed process plant consists of crushing, grinding, gravity concentration, flotation, thickening, filtration and a paste backfill section. The proposed process plant will be producing a gravity concentrate and a bulk flotation concentrate which will be shipped off-site separately. Tailings will be disposed of underground as paste backfill or above ground as filtered cake in a dedicated Tailings Storage Facility (TSF).

For grinding, the base case assumption at the outset of the PEA was two stages with a SAG mill in primary duty followed by a vertical stirred mill in secondary duty. For flotation, the base case assumption was that conventional tank cells would be used in a configuration featuring a rougher stage followed by two cleaning stages.

Preliminary testing showed that the material is of moderate hardness with an average SMC test Axb value of 55. Gravity and flotation recovery testwork indicated an optimum grind of 80% passing 53 µm. A comminution trade-off study was conducted to evaluate the options available to DPM including potential re-use of equipment from another operating mine. The trade-off's main recommendation was to adopt the SAG mill potentially available from DPM's operating mine in a single-stage SAG (SSAG) milling configuration. This SSAG circuit will be closed out by hydrocyclones.

The gravity concentration circuit will be integrated into the grinding circuit with a portion of the cyclone underflow diverted to the parallel gravity scalping screens. Screen oversize as well as centrifugal concentrator tailings will recycle to the grinding circuit. A second trade-off study explored and defined the options for additional beneficiation of the gravity concentrate. For the PEA direct shipping of the primary gravity concentrate was adopted as the base case. Three options were identified for further study in the next phase of the project.

A third trade-off study was conducted to compare flotation technologies. Conventional tank cells, Staged Flotation Reactors (SFRs) and Jameson cells were considered. Some SFRs will be available to Čoka Rakita after decommissioning and refurbishment. The trade-off study recommended the Jameson cell option as it would demand the lowest capital expenditure and smallest footprint.

The nameplate capacity of the treatment plant will be 850,000 dmtpa. The crushing section and filtration sections will operate at a reduced utilisation of 75%, while the grinding and flotation sections will operate 92% of the time.

Cement will be added at 5% to the paste and will be, by far, the most expensive reagent by virtue of its large consumption. Other reagents used in the process include flocculant, PAX, MIBC and collector A3477. Grinding media consumption rates are expected to be low because of the combination of moderate ROM material hardness combined with its below average abrasiveness.

A site wide water balance was constructed and shows that the site will have a positive water balance even during dry years. This is primarily due to the expected large volume of water flux into the underground workings. This water will be pumped to the mine dewatering pond from where it will either be disposed of via the effluent treatment plant or used as make-up in the Processing Plant. Another factor contributing to the positive water balance is that very little water exits the site because all tailings will be disposed of as filtered cake or high density paste.

1.12 Project Infrastructure

The main Project on-site infrastructure components include:

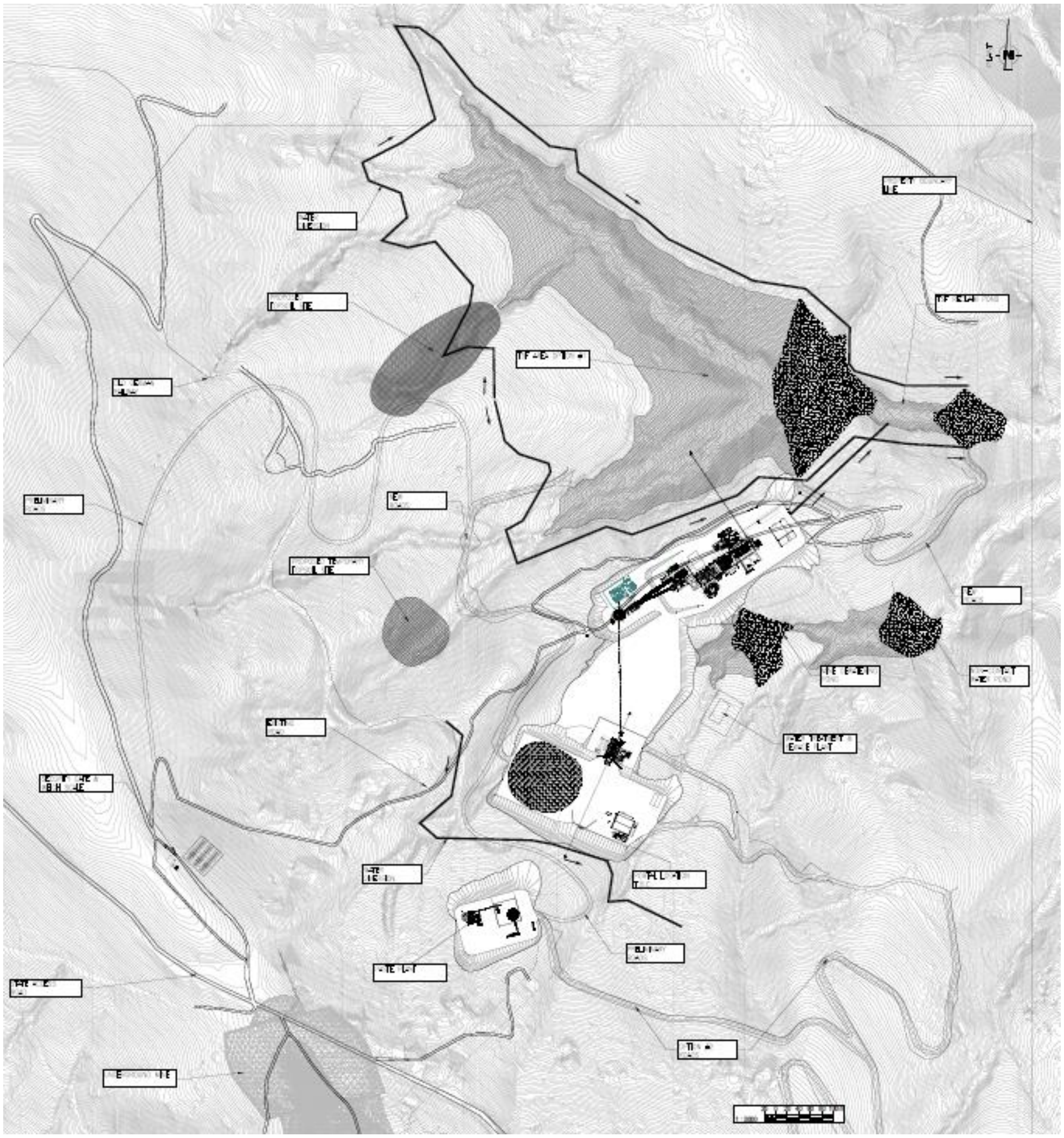
- Roads;
- Terraces;
- Mine and process plant supporting infrastructure;
- Mine waste rock management;
- TSF;
- Water management;
- Power supply and distribution;
- Automation;
- Communication; and
- Services and utilities.

Project infrastructure elements were arranged to optimise the use of available space near the designated TSF and process plant terrace. The ROM stockpile will be located close to the mine portal. Underground development waste rock will be used in the construction of a terrace between the mine ROM terrace and processing facility terrace. Minimal waste haulage is anticipated over LOM.

Mine dewatering will provide process and mine make-up water. There will be three (3) ponds on site; A contact water pond that will be fed by the mine dewatering and management the ROM stockpile terrace area; A non-contact water pond that will be used to store two (2) months of water storage for dry periods and a tailings reclaim water pond that will either provide process water make-up or be treated through an effluent treatment plant and returned to environment. It is assumed that the mine dewatering contact water does not require chemical water treatment, only turbidity treatment via settling, prior to environmental discharge.

Support infrastructure is located on two (2) main terraces namely the ROM stockpile and process facility terraces. The ROM stockpile terrace is where the mining truck shop, mine workshops, fuel storage and mine administration buildings are located. The process facility and associated infrastructure terrace houses the management offices, workshop, substation, and MCCs. There is a third terrace that supports the paste backfill plant and associated infrastructure. An overall site layout is provided for high level context in Figure 1.1 below; details are further explained and illustrated in Section 18.

Figure 1.1 – Site Layout



Source: DRA, 2024

1.12.1 HAUL ROADS AND GEOTECHNICAL

An 8 m wide site access road connects to the main site areas with the national road west of the process and mining facility (Base Case). An option to access from the east is also shown. Site haul roads are designed to connect the crushing, process/administration, and tailings facilities.

An opportunity exists to further optimise haul road construction during the next Project phase by maximising excavation instead of engineered filling during construction, and narrowing the haul road should smaller haul trucks, either owner fleet or contractor mining, be viable.

Geotechnical investigations are planned for the next phase of study to conclude cut and fill design for roads and terracing.

1.12.2 WASTE FACILITIES

The mine development waste will be used to construct a terrace between the process plant area terrace and mining area terrace. There will be little to no mine waste storage on surface for LOM.

The tailings disposal strategy is two-fold:

- A portion of the tailings produced in the mill will be used to backfill underground; and
- A TSF will be constructed to deposit the rest of the tailings on the surface.

Topsoil stockpiles will be generated the initial construction phase of the Project. These topsoil stockpiles will be located above the TSF and used for the rehabilitation of the TSF.

1.12.3 SITE WATER MANAGEMENT AND TREATMENT

The overall water management concept is to divert non-contact water to reduce the amount of contact water to be managed at the Project site and collect the contact water for conveyance to water collection ponds. The contact water collected is used to meet water demands on site, with excess discharged to the receiving environment. A non-contact water collection pond is proposed to provide water supply contingency for the Project. The following facilities are included in the Project site water management plan:

- Non-contact water diversion ditches;
- Contact water collection ditch at the Process Plant terrace;
- Non-Contact Water Pond;
- Contact water collection ponds, including:
  - TSF Reclaim Pond immediately downstream of the TSF dam, and
  - Mine Dewatering Pond.
- Effluent Treatment Facility (ETF).

A deterministic flow model in linked electronic spreadsheets was developed to simulate the site-wide water balance during operations. The flow model was set to simulate the transfer of water between the various mine facilities of the Project site on a monthly basis over a one-year period.

Water balance calculations were performed for three scenarios corresponding to dry, average, and wet annual precipitation conditions. The dry and wet annual precipitation conditions were simulated considering the annual precipitation for a 25-year return period. Estimates of underground mine dewatering were developed by WSP. In agreement with DPM, the lower-case estimate was used for simulation of the dry annual precipitation condition whereas the base case estimate was used for the average and wet annual conditions.

The inflows to the system are contributions from precipitation and groundwater inflow into the Underground Mine. Evaporation from pond surfaces is considered as a loss to the system.

In addition to the flow model simulations conducted for the contact water collection ponds, estimation of water collection in the Non-Contact Water Pond was studied under dry annual precipitation conditions considering various return periods.

1.12.4 Electrical Supply

For supply of electrical power to the Project, the technical and economic feasibility of connecting to the national electrical grid will be explored. Currently two (2) scenarios are possible for this objective:

- The first option is to construct a 35 kV line from the city of Zagubita. The Zagubita substation will need to be expanded to accommodate the necessary equipment.
- The second option is to construct a 110 kV line from the city of Bor. In this case, the respective adjustments will also need to be made to expand the technical capacity of the substation.

The first option was selected as a base case for the PEA.

Other possibilities for energy supply from renewable sources could be explored in future studies to decrease carbon footprint and reduce production costs related to electrical energy consumption.

In cases of emergency, two (2) diesel generation plants, each with an effective capacity of 1.6 MW, will be available to supply the mine and critical processes within the plant.

1.12.5 Site Facilities and Services

Site wide services include non-contact water supply, potable water supply, compressed air, fire water systems, sewage treatment, water treatment and waste disposal.

1.13 Market Studies and Contracts

No specific market studies have been commissioned by DPM nor its affiliates or consultants, in relation to this Project. However, for the purposes of the financial evaluation, DPM has selected a gold price of US $1,700 per oz.

There are no material contracts or agreements in place as of the effective date of this Report. DPM has not hedged, nor committed any of its production pursuant to an off-take agreement.

1.14 Environmental Studies, Permitting and Social or Community Impact

The proposed Project is located in Serbia, and as such will be permitted to operate and regulated by Serbian authorities to Serbian standards.

Mining structures are permitted under the Law on Mining and Geological Explorations whilst supporting and auxiliary structures such as roads and administrative buildings are separately permitted under the Law on Planning and Construction. There is a range of other approvals and permissions required under ministries including the Ministry of Agriculture, Forestry and Water Management (MAFWM), Ministry of Environmental Protection (MEP), Ministry of Interior (MI), and Institute for the Preservation of Cultural Heritage and the Institute for Nature Conservation of Serbia.

The permitting system is undergoing change as part of Serbia's planned accession to the European Union (EU). DPM has sought to minimise permitting risks by engaging with regulators and aligning the Project with EU requirements and good international practice, such as the performance requirements of the EBRD Environmental and Social Policy and World Bank Group Environmental, Health and Safety Guidelines (International Finance Corporation World Bank Group, 2007). DPM has committed to develop all of its projects to comply with EBRD Environmental and Social Policy.

A Spatial Plan will be required for the area of Čoka Rakita to authorise a change in land use to mining. The Spatial Plan is a statutory legal document which sets out the development context for proposed land use and related infrastructure.

The key environmental and social risks are similar to those associated with other gold mining projects and include safeguarding rivers and groundwater, the impact from loss of riverine habitat on biodiversity, dewatering and diversions during operations affecting springs, wells and streams, including in adjacent catchments to the south, and economic displacement associated with land acquisition. Completion of the baseline survey program is required to fully understand and manage these risks.

The TSF will be situated over the Dumitrov Stream, while the process plant, UG mine portal, ROM pad and paste backfill plant will be located on elevated ground within the headwaters of the Ogašu Lu Gjori Stream. The mine water management plan (Section 18.10) outlines the measures embedded in the design to avoid or mitigate impacts. The headwater streams in this area are highly channelised in steep valleys, mitigation measures include contact water collection ponds and non-contact water diversion channels. The main mitigation will be to separate non-contact water by diversions. The non-contact water collected in the non-contact water pond will be settled prior to discharge into Ogašu Lu Gjori Stream, a tributary of the Lipa River.

All contact water from mine dewatering and surface runoff from process areas and the TSF will be collected in lined ponds and ultimately directed to the ETP. Treated water will be used as freshwater make up in the process plant and any excess water will be discharged to the environment.

The Project site lies outside existing water supply protection zones. Abstraction of groundwater to supply the Project and the dewatering required for underground mining is likely to affect community water use, agriculture and habitats, including possible impacts to springs, wells and streams in the Crna Reka catchment to the south. The proposed water management system for the Project is described in Section 18.10.

The closest protected area is the Nature Park Kučaj – Beljanica (located within the Crna Reka catchment 5 km from the nearest project infrastructure) which is currently in the process of being upgraded to National Park status. It is not anticipated that the protected sites identified will be impacted by the Project. However, downstream effects including the potential for a reduction in water flows caused by underground mine dewatering will need to be assessed.

DPM has worked to establish good relationships with the local community since 2007 and communications are managed through a DPM's communication plan. DPM expanded its resources and conducted training in 2019 to facilitate transparent and meaningful community engagement. The Project team maintains a map of stakeholders and has earmarked vulnerable groups which will require targeted engagement.

The Project is located in a rural, hilly area with steep valleys, characterised by seasonally grazed pastures, woodlands and isolated farms and houses. Land acquisition is required for the development of the Project, access to the site and the construction of associated facilities. Gaining access to this land requires purchasing land from private and public owners. The Project will be required to develop a Land Acquisition Strategy in line with national legislation and international good practice, to manage this complex and sensitive process.

Of the rock generated during mine development 60% is planned to be used as mine backfill, and the remainder will be used to construct an operational platform in the process area. This terrace platform will be lined and run-off directed to lined ponds and ultimately on to the ETP. There will be no other waste rock facilities required by the operation or in construction. Furthermore, DPM targets to store 40% of the filtered and cemented tailings underground with the remainder stored in a TSF at surface. The TSF will be a dry stack facility progressively reclaimed during operations and fully covered with a soil growth medium and vegetated during the closure phase.

The Project will advance closure planning and costing exercises throughout the planning stages and operational life, with the next revision at the Environmental Impact Assessment (EIA) stage.

An EIA will be required. There are risks to the Project associated with permitting delays, such delays may be caused by changes to the Serbian regulations to align with EU Law, regulator delay, public challenge to the Spatial Plan or EIA and administrative appeals. DPM plans to work closely with regulators to understand priorities, anticipate changes and to provide support as few private sector mining projects have passed through the full permitting process.

1.15 Capital and Operating Cost Estimates

The capital cost estimate (Capex) and the operating cost estimate (Opex) were compiled by DRA and based on the scope of work described in this Report.

DRA developed the Capex and Opex for the process plant, plant infrastructure and on-site infrastructure for the Project scope described in this Report. Other Capex and Opex inputs were received as follows:

Table 1.5 – Other Capex/Opex Responsibilities

Consultant	Scope
DPM	Provided Owner's G&A Costs, Taxes and Duties; and Closure Costs
WSP	Prepared underground mining and mining infrastructure costs for initial and sustaining Capex and Opex
SLR	Prepared construction quantities for the Filtered TSF, Water Management, and DRA applied rates to complete the Capex
RMS	Prepared Capex and Opex for Paste Backfill Plant, where DRA only estimated bulk earthworks, concrete, and structural steel

1.15.1 Capital Cost Estimate

All costs are expressed in United States Dollars (US$) and are based on Q1 2024 pricing. The estimate is within the range -30%+60% and was prepared in accordance with the AACE International (Association for the Advancement of Cost Engineering) Class 5 estimating standard.

The Capex consists of direct and indirect costs as well as contingency. Provisions for sustaining capital are also included. Table 1.6 presents a summary of the initial Capex and sustaining Capex is distributed over the LOM, separately indicated from the initial Capex. Owner's costs, its contingencies and risk amounts have been included in this Capex.

Table 1.6 – Capital Cost Summary

Area	Description	Initial Capex (US$ M)	Sustaining Capex (US$ M)	Total (US$ M)
2000	Underground Mine	76.2	49.5	125.7
4000	Processing Plant	91.1	1.2	92.3
5000	Filtered Tailings / Water Treatment Facilities	31.3	18.07	49.3
6000	On-Site Infrastructure, Site-Wide General	25.6	0.0	25.6
7000	Off-Site Infrastructure	9.7	0.0	9.67
8000	Operational Readiness	11.5	0.0	11.5
9000	Indirect Costs	52.8	3.4	56.2
9100	Owner's Costs	13.5	0.0	13.5
9170	Closure and Rehabilitation[1]	0.0	19.3	19.3
9900	Project Contingency	69.2	20.3	89.5
	Total Major Area Capex	**380.9**	**111.7**	**492.6**

Numbers may not add due to rounding.
1. Closure costs do not include the non-recoverable VAT of approximately $3 M.

1.15.2 Operating Cost Estimate

A summary of the overall LOM project Opex is presented in Table 1.7. The costs presented exclude pre-production operating cost allowances for mining, process and G&A which are considered in the Capex. Labour rates for this Report were provided by DPM. Direct employment during operations will total approximately 452 people including mine, concentrator, tailings, maintenance, management, and infrastructure.

Table 1.7 – Opex Summary by Major Area

Area	LOM Total Opex (US$ M)	Average Opex (US$/t of material treated)
Mining	294.8	37.13
Processing and Tailings	175.6	22.11
G&A	99.0	12.46
Total	**569.4**	**71.70**

Numbers may not add due to rounding.

1.16 Economic Analysis

The economic analysis of the Project is preliminary in nature and is based on inferred mineral resources, which are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves. As a result, there is no certainty that the 2024 PEA will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The results of the economic analysis are forward-looking information under Canadian securities law. These results rely on inputs that are subject to known and unknown risks, uncertainties, and other factors, which may cause actual results to differ from those presented here.

The economic analysis is based on the discounted cash flow (DCF) method on a pre-tax and after-tax basis at a discount rate of 5%. For the purposes of the evaluation, it is assumed that the operations are established within a single corporate entity. The Project has been evaluated on an unlevered, all-equity basis.

The cash flow model uses inputs from all elements of the Project to provide a comprehensive financial projection for the entire Project, on an annual basis over a 10-year operating life. All prices and costs are in Q1 2024 US dollars. No provision is made for the effects of inflation in this analysis. Table 1.8 provides a summary of the key technical assumptions and cost inputs. At a $1,700 per ounce gold price assumption, the financial results indicate a positive pre-tax Net Present Value (NPV) of $588.2 M at a discount rate of 5%. The pre-tax Internal Rate of Return (IRR) is 33.0% and the payback period is 2.39 years. A forecast of annual cash flow is provided in Table 22.4.

Current legislation in Serbia allows for tax relief for large investments for a maximum period of ten (10) years, subject to certain conditions. DRA has been advised by DPM that the Project is eligible for this tax relief and that the tax relief will be declared by DPM upon the start of production. The effective income tax rate applied is 0% over the project's 10-year mine life. Owing to this tax relief, the after-tax economic metrics are the same as the pre-tax metrics, as shown in Table 1.9.

Table 1.8 – Technical Assumptions and Key Financial Metrics

Description	Unit	Value
Gold Price	US$ per oz	1,700
Government Royalty (NSR)	%	5.0
Mineable Mineral Resource	Mt	7.9
Average Grade Mined (LOM)	g/t	5.68
Annual Throughput	tpa	850,000
Average Grade Processed (LOM)	g/t	5.68
Average Metallurgical Recovery	%	88.8

Description	Unit	Value
Mine Life	years	10
Total Gold Produced (LOM)	M oz	1.3
LOM Gold Payable	%	98.4
Average Annual Gold Production (LOM)	oz	129,000
Average Annual Gold Production (first five years)	oz	164,000
Capital Cost Estimate		
Initial Capital	US$ M	381
Sustaining Capital (LOM)	US$ M	83
Closure Costs[2]	US$ M	28
LOM Operating Unit Costs		
Mining	US$ per tonne processed	37
Processing	US$ per tonne processed	17
Filtered Tailings and Paste Fill	US$ per tonne processed	5
General & Administrative	US$ per tonne processed	13
Total Opex	*US$ per tonne processed*	*72*
LOM Average All-in Sustaining Cost [1]	US$ per gold ounce	715

Numbers may not add precisely due to rounding.
2. Closure costs do not include the non-recoverable VAT of approximately $3 M.

Table 1.9 – Economic Results Summary

Description	Unit	Pre-Tax	After-Tax
Free Cash Flow (LOM) [1]	US$ M	891.2	891.2
NPV @ 5%	US$ M	588.2	588.2
IRR	%	33.0	33.0
Payback Period	Years	2.4	2.4

Numbers may not add due to rounding.

The after-tax results of the sensitivity analysis are shown in Table 1.10 to Table 1.12 and Figures 1.2 and 1.3. The NPV and IRR of the Project are most sensitive to variations in gold price followed by Capex and Opex. The Project retains a positive NPV at the lower limit of the price interval tested.

[1] All-in sustaining cost and free cash flow are non-GAAP financial measures or ratios and have no standardised meaning under IFRS Accounting Standards ("IFRS") and may not be comparable to similar measures used by other issuers. As the Project is not in production, DPM does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS.

Table 1.10 – Economic Metrics Sensitivity to Variations in the Gold Price

Au Price	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	888.2	738.2	588.2	438.3	288.3
IRR	%	43.5	38.4	33.0	27.1	20.7
Payback	Years	1.94	2.14	2.39	2.72	3.19

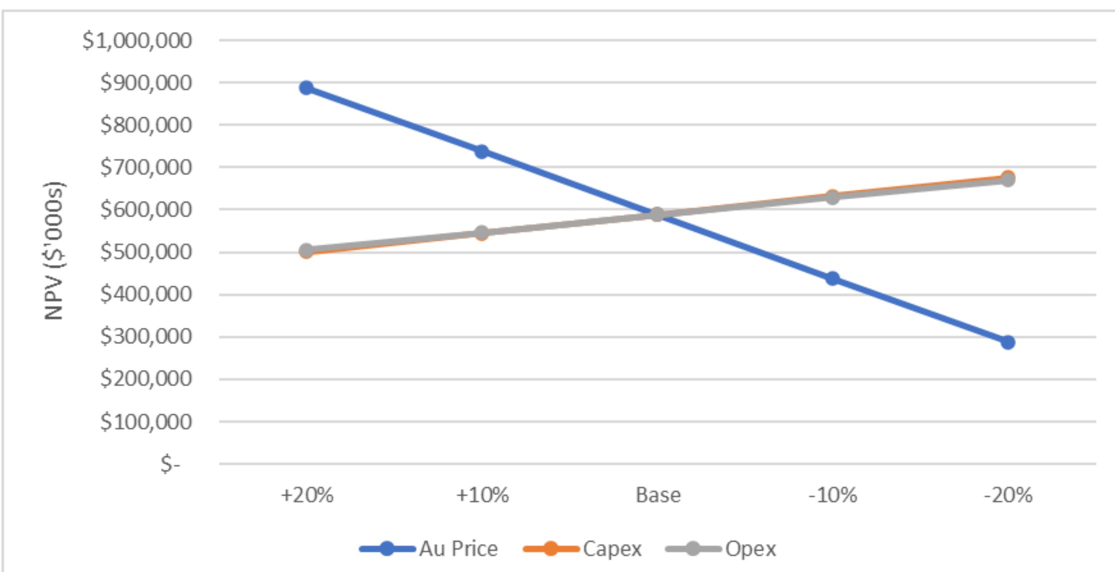
Table 1.11 – Economic Metric Sensitivity to Variations in the Capex

Capex	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	500.8	544.5	588.2	631.9	675.6
IRR	%	26.2	29.3	33.0	37.2	42.2
Payback	Years	2.79	2.59	2.39	2.19	1.99

Table 1.12 – Economic Metric Sensitivity to Variations in the Opex

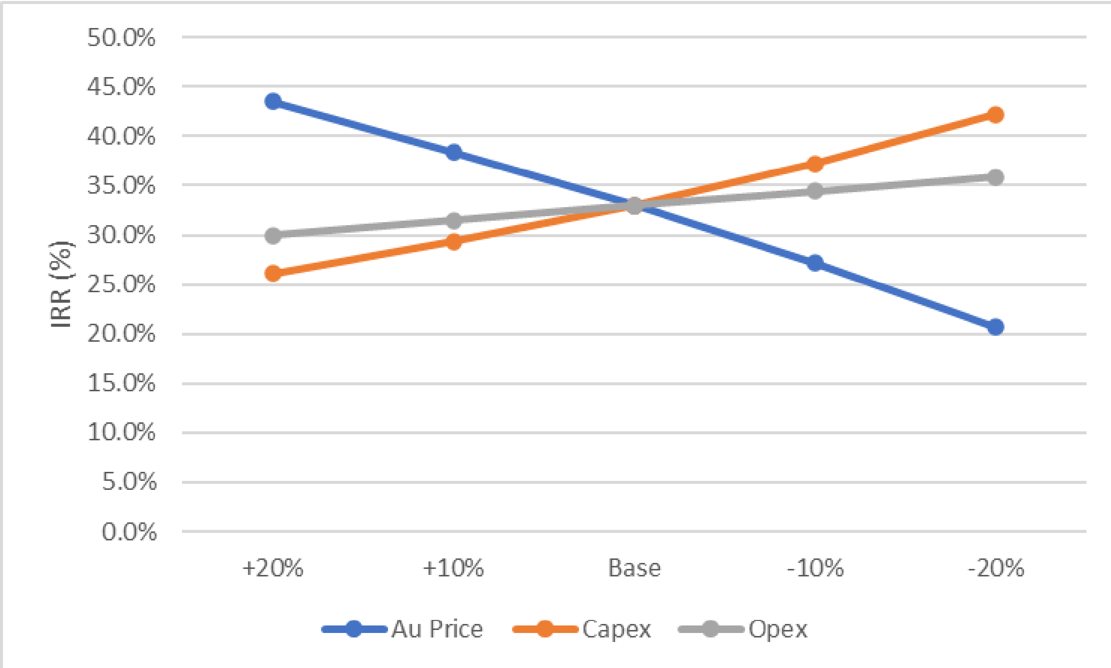
Opex	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	505.9	547.1	588.2	629.4	670.5
IRR	%	30.0	31.5	33.0	34.4	35.9
Payback	Years	2.55	2.46	2.39	2.31	2.24

Figure 1.2 – After Tax NPV 5%: Sensitivity to Capex, Opex and Price



Source: DRA, 2024

Figure 1.3 – After-Tax IRR: Sensitivity to Capex, Opex and Price



Source: DRA, 2024

The sensitivities of the key after-tax economic metrics of the Project were also evaluated at specific gold prices. The results of this analysis are shown in Table 1.13.

Table 1.13 – Gold Price NPV and IRR Sensitivity After-Tax (base case highlighted)

Gold Price	Unit	US$1,500/oz	US$1,700/oz	US$1,900/oz	US$2,100/oz
IRR	%	26.1	33.0	39.3	45.2
NPV at 5.0%	US$ M	411.8	588.2	764.7	941.1

1.17 Interpretation and Conclusions

Full details on interpretations and conclusions described below are provided in Section 25.

1.17.1 Geology and Mineral Resource Estimate

Gold-rich skarn mineralisation is hosted within carbonate-rich sandstones and conglomerates, located on the hanging wall of a sill-like body and abutting a monzonite intrusive body to the west. The mineralisation forms a shallow-dipping tabular mineralised body located between 250 m and 450 m below surface, measuring 650 m long, up to 350 m wide, and with variable thickness from less than 20 m in the margins to more than 100 m in the core of the mineralised zone. Coarse gold is often observed in areas of intense retrograde skarn alternation and is found mainly in proximity to syn-mineral diorites within the higher-grade core of the deposit. The current MRE has been conducted on the portion of the Project where gold-rich skarn mineralisation occurs.

The QP (Ms. Maria O'Connor, MAIG) conducted a personal inspection of the Project on October 3 and 4, 2023 and is of the opinion that the data used and described in this Report is adequate for the purposes of mineral resource estimation of the Project. The QP reviewed the policies and procedures for sample methods, analyses, and transportation, as supplied by DPM and they were found to be in line with CIM exploration best practice guidelines and industry best practice.

The QP is satisfied that the relevant procedures have been followed consistently, all laboratories used for analyses are adequately certified, and are independent of DPM, and that the standards used as part of the QA/QC routine adequately reflect the characteristics of the mineralisation.

The drillhole database was handed over as of 16 November 2023. A total of 173 drillholes totalling 80,723 m were included in the estimation of the MRE. The current drillhole spacing within the mineralised domains is approximately 30 m x 30 m in the core of the system, with an up to 60 m x 60 m grid on the periphery. Gold grades within skarn domains have been determined systematically using a screen fire assaying technique, which is preferred for mineralisation with coarse gold. Grade capping was applied to composites to limit the influence of anomalously high-grade values, resulting in a cut of metal of approximately 30%.

Mineral resource domains were created within volumes of moderate to intense skarn alteration and guided by grade composites generated at a 1 g/t Au cut-off value. Detailed lithology and structural models were developed and used to constrain domain extents, as well as to incorporate post-mineralisation diorite sills which cut across the mineralisation. Block grade estimates have been undertaken for gold, silver, (which are reported here) and copper, sulphur and arsenic (which are used for geometallurgical characterisation) using Ordinary Kriging at a 10 mE x 10 mN x 10 mZ parent block size with sub-celling to honour domain volumes.

A breakeven cut-off value of 2 g/t Au and a minimum width constraint of 5.0 m x 5.0 m x 2.5 m was used to define optimised mineable shapes using Datamine's MSO. These shapes were subsequently smoothed and used to constrain continuous zones of mineralisation for reporting the final Mineral Resource statement.

The application of MSO shapes at the MRE stage provides a robust estimate for the purposes of a future PFS for the Project, and a higher confidence in the potential for the conversion of Mineral Resources into mineable tonnes and grades for the purposes of a mine plan for any future PFS in the next phase of work. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The MSO shapes have been used to ensure the Mineral Resources demonstrate RPEEE.

Material within the reporting MSO constraints (smoothed) was classified as Inferred Mineral Resources according to Mineral Resource confidence categories defined in the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). Data quality and quantity, geological and grade continuity, and confidence in the grade, density and RPEEE criteria were considered when classifying the MRE. Given the relatively continuous and stratified mineralisation style at Čoka Rakita, the QP has reason to expect that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with additional infill drilling.

The qualitative risk assessment is presented in Table 1.4. The overall risk to the Čoka Rakita MRE is reflected in the current resource classification as Inferred Mineral Resources and is considered moderate, which is consistent with the early-stage nature of the Project.

1.17.2 Mining Methods

Conclusions on mining methods are contained in Section 25, for the following aspects:

- Underground geotechnical;
- Hydrogeology;
- Underground mining methods;
- Electrical and communications;
- Mine paste backfill;
- Mine dewatering; and
- Paste backfill.

1.17.3 Environmental Studies, Permitting and Social or Community Impact

Conclusions reached for the Project are contained in Section 25.

1.17.4 Risks

Risk management is an essential part of any project. Risks have the potential to impact a project's timeline, budget, and resources, which can pose challenges for all stakeholders. Efficient risk management involves identification of potential risks, assessment of risk impacts, and development of effective strategies to mitigate them over the life of the Project. In Section 25.7, various potential risks associated with the Project are discussed.

Two (2) distinct risk assessment workshops were conducted in the PEA phase, aimed at identifying and evaluating risks that could affect the success of the Project. During these workshops, each identified risk was evaluated according to its individual likelihood and potential impact. In addition, potential mitigations were established for each risk event, after which risks were re-assessed and scores reevaluated.

1.18 Recommendations

Specific recommendations for the Project are summarised below and full details are provided in Section 26 of the Report.

1.18.1 Proposed Work Program

To ensure the potential viability of the mineral resources, the following activities should be undertaken in the next phase of the Project, leading to the completion of a PFS. Proposed costs for these activities are provided in Section 26.1:

- Resource Drilling and Assays;
- Geotechnical / Hydrological Drilling and Testwork;
- Metallurgical Testwork;
- Trade-Off Studies;
- Further advancement of the mine, process plant, and infrastructure design and engineering;
- Environmental Studies;
- Permitting; and
- PFS Report.

1.18.2 Geology

Work programs for the next phase of the Project are recommended in Section 26.2, for the following aspects:

- Exploration;
- Drilling;
- Geological database;
- Assay QA/QC; and
- Lithogeochemistry.

1.18.3 Mineral Processing and Metallurgical Testing

Proposed future testwork programs are recommended in Section 26.3 for the following aspects:

- Further general process development;
- Treatment of gravity concentrate;
- Variability testing; and
- Additional other testwork.

1.18.4 Mineral Resource Estimate

A small program of very close spaced drilling is recommended to better understand the variability of the mineralisation grade at short distances. Due to the depth of the mineralisation, drilling from surface would be costly and time consuming, but directional drilling using a mother hole may be effective in gaining drill coverage over a small area of the high-grade core of the deposit. This would be used to inform a variability study to be conducted prior to the next MRE update.

Following the close spaced drilling program, the drill spacing density should be reviewed where the highest risk to the resource is with a view to tightening the drill spacing in that area; this is considered a Phase 2 work, contingent on the close spaced drill program being completed.

Improved modelling of the late-stage intrusions continues, involving re-logging in association with lithogeochemical analysis, with this work being undertaken to reduce the uncertainty of the precise location and volume of late-stage intrusions, currently representing approximately 4% of the mineralisation volume.

A review of the sensitivity to grade estimation methodology is recommended to assess methods for modelling the higher grades of the mineralisation.

Drilling is continuing at the Project and the MRE will be updated based on new drilling, with the current mineralisation model being tested and interpretations revised as required, at a suitable point. An updated MRE is considered Phase 2 work.

1.18.5 Mining Methods

Recommendations with respect to mining methods are provided in Section 26.5 for the following:

- Underground mining;
- Mine ventilation;
- Mine backfill;
- Mine dewatering; and
- Paste backfill.

1.18.6 Recovery Methods

Recommendations with respect to recovery methods are provided Section 26.6 for the following:

- Gravity concentration options;
- Comminution options;
- Flotation options; and
- Other recommendations.

1.18.7 Project Infrastructure

Recommendations with respect to Project infrastructure are provided Section 26.7 for the following:

- General site infrastructure development and optimisation;
- Tailings storage facility and water management; and
- Paste backfill.

1.18.8 Market Studies and Contracts

DPM will explore opportunities to enhance the commercial terms of concentrates by engaging with the market sector.

1.18.9 Environmental Studies, Permitting and Social or Community Impact

Recommendations for future Project phases are provided Section 26.9.

1.18.10 Capital Cost Estimate

Recommendations with respect to Capex are provided Section 26.10.

2 INTRODUCTION

DPM is an international gold mining company with headquarters in Toronto, Canada, with operations in eastern Europe (Bulgaria) and southern Africa (Namibia), and ongoing exploration and development projects in Serbia and Ecuador. DPM is listed on the TSX (TSX:DPM), with headquarters at 150 King Street West, Suite 902, Toronto, Ontario M5H 1J9.

This Technical Report has been prepared on behalf of DPM by the Qualified Persons (QPs) indicated in Section 2.3. The purpose of this Report is to support disclosure of the results of developmental work performed for the Project.

Following the completion of the maiden Mineral Resource Estimate (MRE), DPM commissioned DRA Americas Inc (DRA) to lead a team of specialists to develop a PEA to demonstrate the economics of the Project. The Report describes mine infrastructure, a processing plant, and all associated facilities necessary to process 850,000 tpa of mineralised material.

The purpose of the PEA is to assess a preliminary configuration for the mine and processing facilities based on the latest available testwork, as described in Section 13 and the MRE (Section 14). This PEA provides engineering definition through mine design, project infrastructure definition and optimised operational descriptions. The definitions will be followed by the preparation of capital and operating costs and confirmation of project economics. The maiden MRE for the Project has been defined and classified in accordance with the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014).

2.1 Scope of Study

DPM retained various Consultants to participate in and prepare this Report. More specifically, the Consultants listed below provided QPs (as identified in Section 2.3) to participate in and to complete Project work as described below:

- **DRA Americas Inc.** (DRA): review metallurgical testwork, develop design criteria and recovery methods, process plant, and related plant infrastructure, and compile the Report.
- **Environmental Resources Management Ltd.** (ERM): develop the Mineral Resource Estimate (MRE) as well as environmental studies, permitting, and social impact.
- **Responsible Mining Solutions Corp.** (RMS): provide paste backfill design.
- **SLR Consulting Ltd.** (SLR): design tailings management and waste rock facilities, and develop a site wide water balance for the Property.
- **WSP Global. Inc.** (WSP): design the underground mine and related mine infrastructure.

2.2 Sources of Information

DRA is the lead consultant for the PEA and for preparation of this Report, and thereby provided a QP for overall report compilation. This QP collaborated with the QPs from the other consultants (for scope outside of DRA's responsibility), and is responsible for compiling this Report which is inclusive of the work and deliverables prepared by all Consultants.

As described above, this Report relies on various consultants for descriptions of Project elements. The list of consultants above is intended to indicate sources of information for the various Project aspects, and it does not necessarily indicate responsibility.

The Project assessments of the QPs were based on maps, published material, pre-existing reports, Project development work specifically performed by the Consultants and others, and data, professional opinions and published and unpublished material provided by DPM. The QPs reviewed all relevant data provided by DPM and/or its agents. The QPs reviewed and evaluated all information used to prepare this Report and believe that such information is valid and appropriate considering the status of the Project and the purpose for which this Report is prepared. A full listing of references is provided in Section 28.

2.3 List of Qualified Persons

The Qualified Persons (QPs) listed in Table 2.1 are responsible for preparation of this Technical Report, and their certificates are also contained herein which contain further details.

Table 2.1 – Qualified Persons and their Respective Sections

Name	Title, Company	Responsible for Section(s)
Stephan Blaho, P. Eng.	Mining Engineer, WSP Global Inc.	16, and portions of 1, 21, and 25 to 28
Daniel Gagnon, P. Eng.	Senior Vice President of Mining and Geology, DRA Americas Inc.	19 and 22, and portions of 1 and 25 to 28
Kevin Leahy, C. Geol.	Chartered Geologist, ERM Ltd.	20, and portions of 1 and 25 to 28
Marcello Locatelli, P. Eng.	Project Manager, DRA Americas Inc.	2, 3, 15, and 24, portions of 1, 18, 21, 25 to 28, and overall report compilation
Daniel (Niel) Morrison, P. Eng.	Principal Process Engineer, DRA Americas Inc.	13, 17, and portions of 1, 21, and 25 to 28
Maria O'Connor, MAIG	Technical Director Mineral Resources, ERM Ltd.	4 to 12, 14, 23, and portions of 1, and 25 to 28

Name	Title, Company	Responsible for Section(s)
Nino Pavlovic, P. Eng.	Process Engineer, RMS Corp.	Portions of 1, 18, 21, and 25 to 28
Eric Sellars, P. Eng.	Geotechnical Engineer, SLR Consulting Ltd.	Portions of 1, 18, 21, and 25 to 28
Luis Vasquez, P. Eng.	Principal Hydrotechnical Engineer, SLR Consulting Ltd.	Portions of 1, 18, 21, and 25 to 28

2.4 Site Visit

An initial visit to the site at Čoka Rakita, Serbia, and to the DPM Avala d.o.o. office in Bor, Serbia, was undertaken from October 3 to 5, 2023 for the purposes of scope confirmation and site familiarisation, and for evaluation of plant location and infrastructure design work.

Table 2.2 provides details of the personal inspection of the Property by the QPs.

Table 2.2 – Site Visits by QPs

Qualified Person	Company	Date of Site Visit
Stephan Blaho	WSP	October 3 to 5, 2023
Kevin Leahy	ERM	October 9, 2023
Marcello Locatelli	DRA	October 3 to 5, 2023
Maria O'Connor	ERM	October 3 to 4, 2023

2.5 Units and Currency

In this Report, all currency amounts are in US Dollars ($ or US$), unless specifically stated otherwise. Quantities are generally stated in *Système international d'unités* (SI) units as per standard Canadian and international practice, including tonne (t) for mass, and kilometre (km) or metre (m) for distance. Abbreviations used in this Report are listed in Section 27.

2.6 Non-GAAP Financial Measures

Certain financial measures referred to in this Report are not measures recognised under IFRS and are referred to as non-GAAP financial measures or ratios. These measures have no standardised meaning under IFRS and may not be comparable to similar measures presented by other companies. These measures are intended to provide additional information and should not be considered in isolation or as a substitute for measures prepared in accordance with IFRS.

3 RELIANCE ON OTHER EXPERTS

The QPs who prepared this Report relied on information provided by other experts pertaining to specific areas of expertise supporting the Project. The QPs who authored the sections in this Report believe that it is reasonable to rely on these experts, based on the assumption that the experts have the necessary education, professional designations, and relevant experience on matters relevant to the Report.

The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this Report and adjusted information that required amending. A full list of previous reports is provided in Section 28. This Report includes technical information, which required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

This Report has been reviewed for factual errors by DPM. Any changes made as a result of these reviews did not involve any alteration to the conclusions made. Hence, the statement and opinions expressed in this document are given in good faith and in the belief that such statements and opinions are neither false nor misleading at the date of this Report.

The QP for Section 4 (from ERM) has relied upon DPM for information regarding the Project exploration licenses and their current legal status as discussed in Section 4.2 of this Report. The QP has also relied upon DPM's management with regards to the legal status of each exploration license and any royalty agreements as discussed in Section 4.4.

The QP for Section 4 (from ERM) has not independently verified legal ownership of surface title and exploration licenses of the Project beyond information that is publicly available or been provided by DPM. The property description presented in this Report is not intended to represent a legal, or any other opinion as to title ownership.

DRA relied on DPM to describe the following sections:

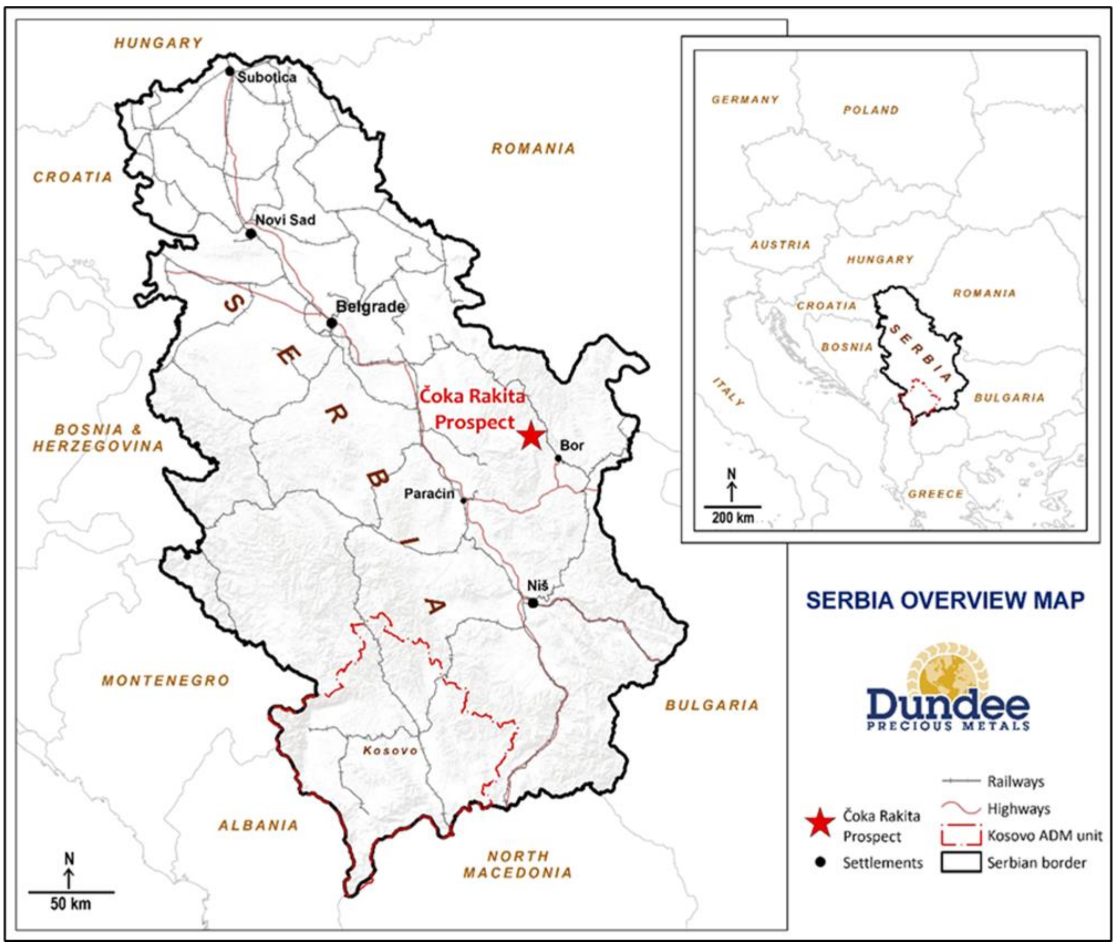
- 19 – Market Studies and Contracts; and
- 22 – Economic Analysis (taxes).

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The Project is located in the eastern part of the Republic of Serbia (Serbia), (coordinates 21°54’47.745”E, 44°12’44.787”N, WGS 84 grid system), approximately 270 km southeast of its capital, Belgrade, as shown in Figure 4.1. The main deposits on the Project are located approximately 25 km northwest of the town of Bor, Serbia. Bor is a historical centre for copper mining and smelting in Serbia.

Figure 4.1 – Location Map – Čoka Rakita Project



Source: DPM, 2023

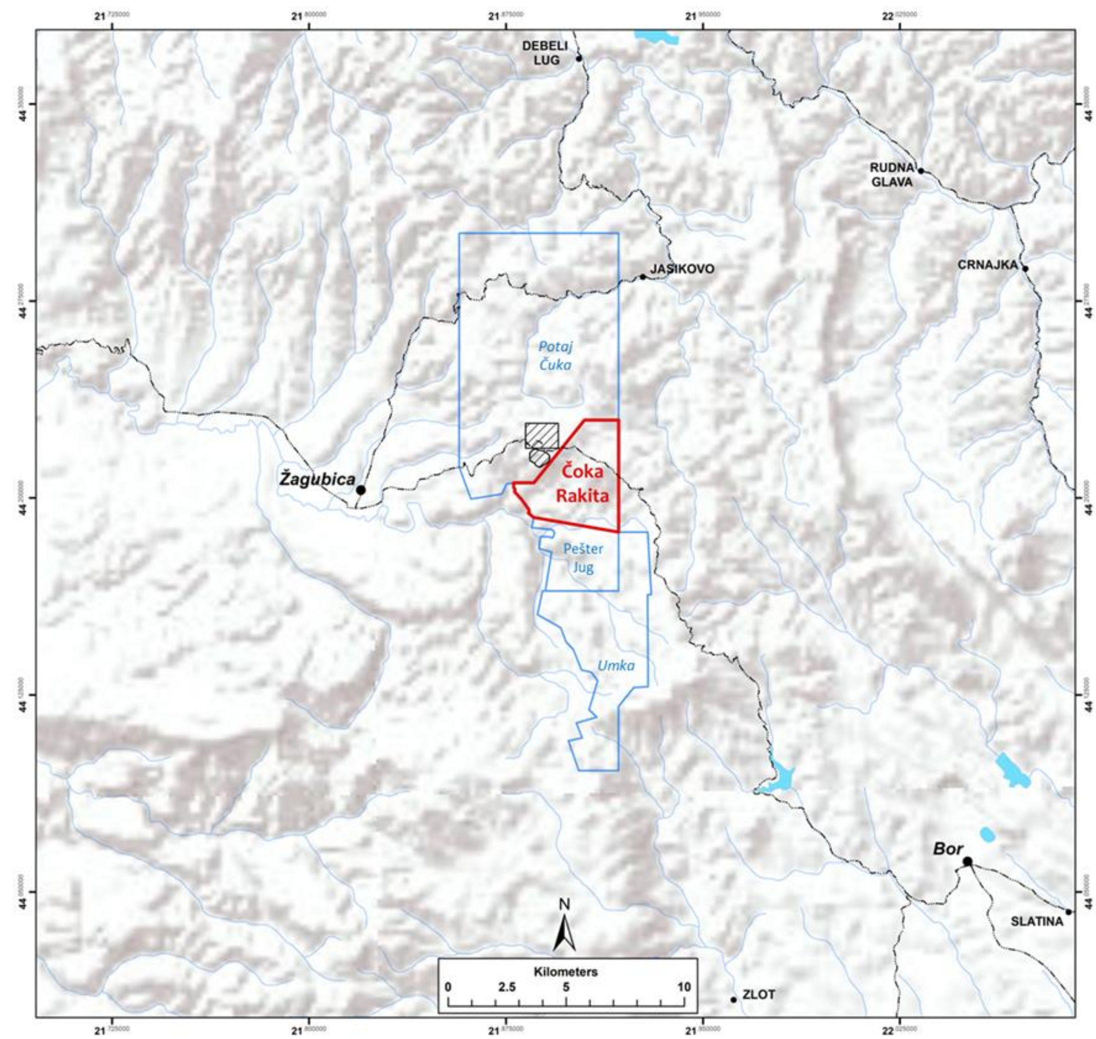
4.2 Mineral Tenure and Surface Rights

The Project is located in the Čoka Rakita exploration license which has an area of 13.81 km². In total, DPM has four (4) exploration licenses (Potaj Čuka, Pešter Jug, Čoka Rakita, and Umka) covering an aggregate area of 109.24 km² in the Bor region. The Potaj Čuka license hosts the Timok

Gold Project, which was advanced to a Prefeasibility Study (PFS) level by DPM as of 2021 and is the subject of a Technical Report which remains current. Section 23 contains more details on the adjacent Timok Gold Project.

During 2022, the Potaj Čuka Tisnica license area was decreased, and a portion of the relinquished land was re-applied for by Crni Vrh Resources, a wholly owned subsidiary of DPM, who was granted the Čoka Rakita license on 12 October 2022. Subsequently, the Potaj Čuka Tisnica license was re-applied for as the Potaj Čuka and Pešter Jug licenses, which were granted to Crni Vrh Resources on 12 October 2023. The exploration licenses and their boundaries are shown in Figure 4.2.

Figure 4.2 – Čoka Rakita Project Exploration License



The other exploration licenses owned by DPM are also shown in blue.
Third-party mining licenses (black with crosshatch). Grid values in WGS 84 Grid System.
Source: DPM, 2024

4.3 Exploration Licenses in Serbia

Exploration licenses in Serbia are currently granted by the Ministry of Mining and Energy (MoM&E) within the Government of Serbia. They are generally issued on an initial three-year basis and are twice renewable for a further period of three years (first renewal), followed by a period of two years (second renewal), for a total potential term of eight years. An integral part of the exploration license application and renewal process is submission of a detailed exploration work program. Supporting documentation is also required from the Institute for the Preservation of Cultural Heritage and the Institute for Nature Conservation of Serbia to ensure that the proposed exploration activity is in accordance with Republic of Serbia's environmental and cultural legislation.

The license permits the license holder the right to complete surface exploration works, which among other things, includes surface drilling, trenching, surface sampling and geophysics during the agreed license period. The obligations of the license holder are to complete the submitted and approved work program, provide annual exploration activity reports to the MoM&E, and advance the geological knowledge of the property.

Exploration licenses can be renewed if the exploration license holder fulfils its obligations, including the completion of at least 75% of the planned work program. The legislation provides for a clear development process, from discovery through to mine development and operation.

To retain the licenses beyond the final two-year extension period, a similar application can be made to request a reservation of the exploration licenses for a further three-year period, during which permitting activities may take place. This phase, termed the retention period, allows the exploration license holder time to prepare technical studies, most notably the development of the Elaborate of Mineral Resources and Mineral Reserves (Elaborate of Reserves) that are required to convert the exploration license to a mining license.

4.3.1 License Ownership and Obligations

The Čoka Rakita exploration license is held by Crni Vrh Resources which is a Serbian corporate entity and a wholly-owned subsidiary of DPM. Details of the exploration license and the expenditure commitments for maintaining the exploration license in good standing are summarised in Table 4.1. DPM expects to fulfil all obligated commitments to maintain the exploration license in good standing until expiry.

The Čoka Rakita exploration license is currently within the first three-year phase. Upon the expiration of the exploration licenses, DPM is entitled to secure mineral rights to the area to allow for permitting activities.

Table 4.1 – Summary of the Čoka Rakita Exploration License

License	License Number	Holder	Initial Grant Date	Expiry Date	Area (km²)	Expenditure Commitment[1] (€)
Čoka Rakita	310-02-00980	Crni Vrh Resources d.o.o.	12 Oct 2022	12 Oct 2025	13.81	40,229,787

[1] Expenditure commitment relates to the full work program (covering the period from the grant date to the expiry date) as submitted to the Serbian MoM&E. DPM is required to meet 75% of this commitment for the license to be eligible for renewal after the expiry date.

Source: DPM, 2023

The Čoka Rakita, Potaj Čuka, and Pešter Jug licenses are currently within the first three-year phase. DPM was granted the Umka exploration license for a second two-year renewal on 19 October 2022. Upon the expiration of the exploration licenses, DPM is entitled to secure mineral rights to the area to allow for permitting activities.

4.4 Royalties

The Serbian government levies a royalty of 5% of NSR for production of metallic raw materials.

In addition to the royalty generated during the production of metallic raw materials, the government also levies a separate royalty for geological exploration applicable throughout the exploration phase. This amounts to approximately €88 per 1 km² of the envisaged exploration area.

There are no other royalties, back-in rights, payments, or other agreements and encumbrances to which the Project is subject.

4.5 Permitting and Environmental Liabilities

DPM is required to remedy drill roads and pads once drilling is completed unless other agreements are made with the surface landowner. There are no other known environmental liabilities to which the Project is subject. No additional permits are required if the work program associated with the license application does not fall below or exceed the proposed work costs by 25%. An addendum must be filed detailing the work program if the 25% tolerance is exceeded.

4.6 Other Significant Factors and Risks

The QP is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The Project is accessible by regional asphalt roads between Bor, Žagubica, Krepoljin, and Zlot, and well-developed unpaved forestry roads. The area is also linked via Bor to Zaječar and Paraćin and via Žagubica to Požarevac (and further to Belgrade). The Project area is 40 km by road from Bor and 9 km by road from Žagubica. A location map of the Property, relative to regional towns and transport connections is shown in Figure 5.1. Bor is accessible via the national highway grid (Paraćin turnpike), leading to paved roads through Boljevac and Petrovac to Bor with State Roads 161 and 164 passing north of the Project area.

The town of Bor is connected by rail to Belgrade (via Požarevac). This same rail network is part of European Transportation Corridor 10, which extends southwards through the Republic of North Macedonia to Greece and the Mediterranean, and also eastwards through Bulgaria to ports on the Black Sea (and further on to Turkey).

Figure 5.1 – Project Location and Surrounding Towns

Source: DPM, 2023

5.2 Climate and Physiography

The Project area is characterised by moderate continental climate, with some influence of high mountainous climate. Winters are long and cold, with abundant snow cover, and summers are usually hot. First seasonal frosts occur in October and the last frosts are in April. Site elevations vary between 600 m and 950 m above mean sea level. Long-term monthly and daily observations from the Crni Vrh weather station located approximately 9.5 km to the southeast at an elevation of 1,037 m represent climate at the upper end of the Project site elevation range. Records indicate the coldest month is January, with an average temperature of -1.3°C, and the hottest month is July, with an average temperature of 20.7°C. Access to the Project is possible throughout the year with no seasonal shutdowns of drilling required. Within the Bor region of Serbia several major mines are in operation, which are all able to operate all year round.

Annual precipitation is in the range of 500 mm to 1,130 mm, with the mean annual precipitation estimated to be 770 mm. The mean monthly precipitation is estimated to vary from about 47 mm in February to about 93 mm in both May and June. The mean annual potential evapotranspiration is estimated to be 554 mm, varying from about 8 mm in March to about 114 mm in July.

The Project is in a hilly area, mostly forested with steep-sided narrow valleys and broad interfluves. Figure 5.2 shows the typical landscape. The dominant habitat is beech woodland, interspersed with agricultural land comprising pasture and orchards with scattered homesteads (most seasonally occupied but now often abandoned). The majority of agricultural land was grazing pasture and is now disused, mainly reverting to meadow, and supports good species diversity. Much of the woodland present show signs of harvesting for timber production; some areas are composed of mature woodland and likely support high species diversity.

Several small streams drain into the northern and central parts of the Project, which are the tributaries to the main river Lipa, and is part of the catchment area of the river Pek and part of the Danube watershed. In the southern part of the Project, several streams drain into Crna Reka river, which flows into Tisnica and further into the Mlava basin. Many ephemeral riverbeds occur in valley floors around the site, likely seasonal watercourses fed by spring snow melt.

Figure 5.2 – Typical Landscape Directly Above the Čoka Rakita Project

Source: DPM, 2023

5.3 Local Resources and Infrastructure

Bor is a historical mining centre in eastern Serbia, which has been in near-continuous operation since 1902. Currently, the majority of the population is employed by the mining company Serbia Zijin Copper d.o.o, which in December 2018 became majority owner of the previously state-owned mining group, RTB Bor, which operates the Veliki Krivelj and Cerovo open pit copper mines and the underground Borska-Jama copper-gold operation, and the Bor smelter, all located near the town.

A considerable proportion of the population has experience in work activities associated with mining operations, and the local availability of technical staff for any future mining operations within the region is considered high.

While there is limited infrastructure within the Project area, there are existing power lines and networks of well-developed, gravel forestry roads. Aggregate for concrete can be supplied by an operating plant located some 30 km west of Bigar Hill, which is in good condition and currently supplies customers across the region. Water for drilling is sourced locally from permitted water sources. Preliminary engineering assessments by DPM indicate suitable locations for tailings storage and site infrastructure are present on the Čoka Rakita exploration license.

Habitation within the Project area is sparse and restricted to summer-months seasonal occupancy of rural farmsteads, although this practice is in decline. DPM has an operational base in the town of Bor (population approximately 40,000).

6 HISTORY

6.1 Prior and Current Ownership

Prior to DPM, only state-funded exploration is recorded on the Property. State-funded exploration efforts focused on the Dumitru Potok porphyry copper prospect, which is located approximately 1.5 km to the northeast of the Čoka Rakita prospect. Exploration efforts outlined weak porphyry copper mineralisation which was tested via means of underground drifting and a network of vertical surface drillholes. No historical records exist of the work undertaken.

No other private companies have historically explored on the Čoka Rakita exploration license. DPM has been active in minerals exploration in Serbia since 2004 and acquired several exploration licenses and concessions between 2004 and 2010 through its wholly owned subsidiary Avala.

In July 2010, Avala acquired Avala Resources d.o.o. (formerly named Dundee Plemeniti Metali d.o.o.) from DPM through a reverse takeover transaction, pursuant to which DPM retained an interest in the licenses, by acquiring a 51% share in Avala. In April 2016, DPM subsequently completed the acquisition of the 49% of Avala that it did not own, effectively re-acquiring full ownership of the Property. In November 2021 Avala Resources d.o.o. changed its name to DPM Avala d.o.o.

During 2022, the Potaj Čuka Tisnica license area was decreased, and a portion of the relinquished land was re-applied for by Crni Vrh Resources, a subsidiary of DPM, which was granted the Čoka Rakita license on 12 October 2022.

6.2 Regional Exploration History

The Timok region has a long history of exploration and mining, dating back to Roman times. Key periods include:

- Mining during Roman times, as demonstrated by the discovery of slag and mining tools.
- Geological mapping commenced in 1933 by Geozavod, Belgrade, and Geology Institute Bor.
- Geophysical exploration undertaken by French prospectors in the 1930s and during various periods until 1985 by the Institute for Geological and Geophysical Exploration, Belgrade.
- Several geochemical surveys, commencing in 1958, undertaken by Geozavod, Belgrade, and Geology Institute Bor.
- Small-scale adits developed prior to World War II.
- Limited exploration, including drilling, which commenced post-World War II, by RTB Bor (Mining and Smelting Combine Bor).
- Pits and adits of unknown age are scattered through the eastern and southern portions of the exploration licenses.

Historically, RTB Bor mined the adjacent Lipa high-sulphidation epithermal deposit with production occurring between 1958 and 1967 and producing about 1 Mt of material averaging 4 g/t Au and 1.1% Cu (Coffey, 2010). RTB Bor completed limited mining of the Valja Saka lead-zinc skarn, however, the extent and duration of this mining are not known. RTB Bor also established a small pit on the silica cap at the Kuruga high-sulphidation epithermal prospect where they undertook mining of silica flux for the Bor smelter. Minor historical mining in the form of disturbed ground or an old pit is present at Čoka Rakita but the age and history of this are unknown.

Exploration by RTB Bor on adjacent licenses commenced in the 1960s and continued intermittently until the 1980s. During this period, a total of 43 drillholes were drilled for 11,882 m ranging in depth from 90.0 m to 450.7 m. Drilling was based on a nominal grid spacing of 100 m x 300 m.

Extensive soil sampling and surface trenching programs were carried out during the 2007 to 2009 period by previous operator Avala Resources Ltd (Coffey, 2010). Four (581.7 m) diamond core drillholes and 152 trenches (28,014.6 m for 14,138 samples) were completed on adjacent licenses that are associated with the Timok Project to the north.

Limited historical gold exploration has occurred on the Čoka Rakita exploration licenses prior to DPM acquiring the Project.

6.3 Previous Mineral Resource Estimates

No previous MREs have been completed on the Čoka Rakita Project.

6.4 Historical Production

No production of any significance has been undertaken on the Property.

7 GEOLOGICAL SETTING AND MINERALISATION

7.1 Regional Geology

7.1.1 Regional Mineralisation

The Property is located within the north-western part of the Timok Magmatic Complex (TMC) in eastern Serbia. The TMC is part of the Western Tethyan Belt segment (Figure 7.1), which is part of the Tethyan (or Alpine-Himalayan) orogenic system that extends from Western Europe to Southeast Asia. The orogen resulted from the convergence and collision of the Indian, Arabian, and African plates with Eurasia, initially in the Cretaceous and continuing today.

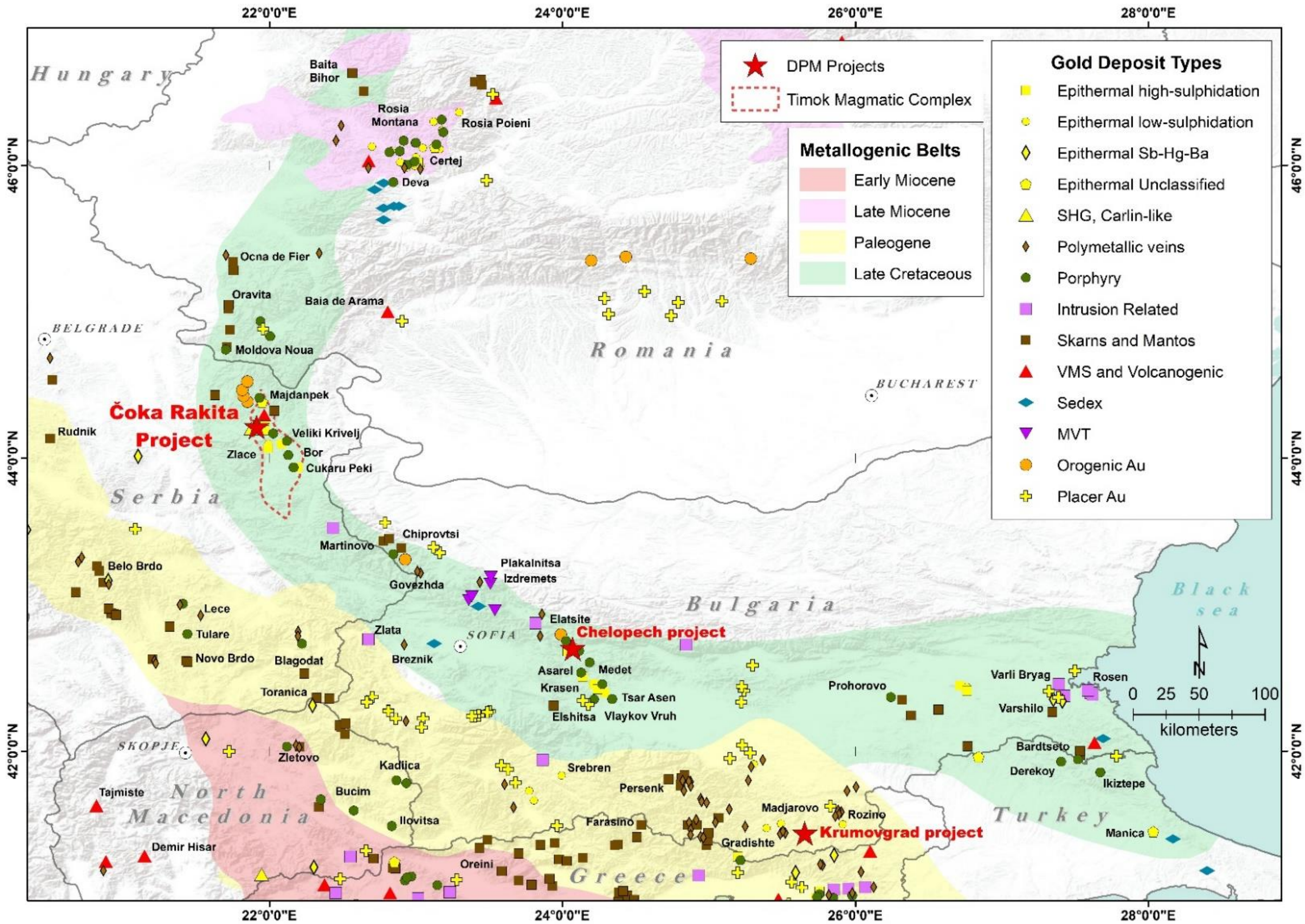
The complex arcuate geometry of the collision interface, and the presence of several micro-plates within the orogenic collage, resulted in a variety of collision products (Gallhofer et al., 2015). Some segments are characterised by extensive regional metamorphism, whereas others by calc-alkaline igneous activity. The structural complexity and present-day geometry of the region reflects large-scale oroclinal bending during post-collision tectonics throughout the Tertiary, including major transcurrent fault systems with overall dextral displacements exceeding 100 km (Knaak et al., 2016).

Orogenic segmentation resulted in a discontinuous distribution of mineral deposits within the Western segment of the Tethyan Belt and limited the lateral extents of the various metallogenic belts along the trace of the orogen. These Late Cretaceous to Miocene belts and adjacent segments host significant porphyry copper-gold deposits with related high sulphidation copper-gold mineralisation. The major deposits within the region are Skouries porphyry copper-gold in Greece, Chelopech high-sulphidation and porphyry in Bulgaria, Bor, Čukaru Peki, Veliki Krivelj, and Majdanpek high-sulphidation and porphyry in Timok, Serbia, as well as deposits skarns and porphyry copper deposits in Banat and Apuseni in Romania (e.g. Moldova Noua – Suvorov, Baita Bihor, Rosia Poieni, Deva, etc.).

Within the Western Tethyan, an economically significant segment comprises the Late Cretaceous subduction-related magmatic rocks and mineral deposits, referred as the Apuseni-Banat-Timok-Srednogorie Belt (abbreviated as ABTSB, Popov et al., 2000). This L-shaped belt extends from Romania, through Serbia, and into Bulgaria. Plate reconstructions show that the ABTSB originally had an east-west orientation in Late Cretaceous times (Gallhofer et al., 2015 and references therein).

The structural complexity, the present-day L-shape geometry of the region and clockwise rotation (~30°) of the TMC segment reflects large-scale oroclinal bending during post-collision escape tectonics throughout the Tertiary, including major transcurrent fault systems with an overall dextral displacement more than 100 km and associated alternating transpressive and transtensional episodes.

Figure 7.1 – Metallogenic Belts and Gold Deposit Types of the Western Segment of the Tethyan Belt



Source: DPM, 2023

Intrusive and extrusive rocks of the ABTSB were emplaced during a 30 million-year (Ma) period from ~90 Ma to 60 Ma and may have been associated with several different subduction zones of varying polarity (Gallhofer et al., 2015). The easternmost magmatic complex in Serbia, the TMC, bounds the Project area on the east.

7.2 Regional Structural Geology

Several fault populations of various inferred ages-of-formation have been identified in the TMC, characterised by relatively more intense development of strike length and density on the western margin of the TMC. From oldest to youngest, the populations constitute:

- Palaeozoic/Mesozoic faulting of metamorphic basement rocks. These faults were undoubtedly reactivated during syn-sedimentary TMC basin formation and subsequent emplacement of igneous intrusions.
- Early Cretaceous, currently northwest-striking, dislocations that appear to have controlled basin opening. These structures are interpreted as major accommodation-structures during Eocene-Oligocene deformation.
- Late Cretaceous strike-extensive reverse faults, trending north-south to northeast-southwest. These faults were reactivated by Alpine transpression that resulted in accommodation of dextral strike-slip motion. A discontinuous easterly-dipping subpopulation of these faults is developed through the sediment-hosted gold prospects and is interpreted as having been a single structure prior to disruption by subsequent deformation. This feature is defined as a domain-bounding structure and is discussed in Sections 7.4 and 7.5. Geology maps at 1:25,000 scale show north-trending, east dipping reverse faults as part of a larger north-trending reverse fault system at the north-western margin of the TMC.

Evidence for reverse movement is expressed as repetition/imbrication of stratigraphy and is also associated with local folding and variation in the dip of stratigraphic layering. Northeast-striking faults locally post-date sedimentary rock-hosted mineralisation, as evidenced by their intersection and offset of the margins of the Potaj Čuka monzonite, although the degree to which this can be attributed to fault reactivation is unknown.

Eocene to Oligocene northwest-striking, strike-slip faults that hosted sinistral movement as a result of oroclinal bending. These structures constrain numerous regionally pervasive, short strike-length northeast-trending faults that are typically expressed as topographic lows.

Late normal faults are responsible for the geometry of features such as the Miocene Žagubica Basin, which contains approximately 2,000 m of sedimentary infill. These structures extend eastward into Bigar Hill and offset the mineralised system. At Čoka Rakita, the presence of such structures has been interpreted in the north flank of the deposit, and locally displaces the stratigraphy and mineralisation by approximately 50 m.

Regionally developed east-west striking faults of variable strike length are expressed as discrete brittle structures at all scales and crosscut all other structural features.

Despite the age relationships indicated above, the assignment of individual faults to populations of particular ages is difficult. Surface expressions of faults are uncommon, and crosscutting relationships are rarely conclusive. Furthermore, a diversity of fault orientations is present, due to different ages of faulting, shifting far-field stress geometries over time, re-activation of older faults, and the role of pre-existing architecture during the formation of each successive stage of faulting. A critical element in the identification of faults has been the resolution of a consistent stratigraphic framework – the components of which can be identified regionally.

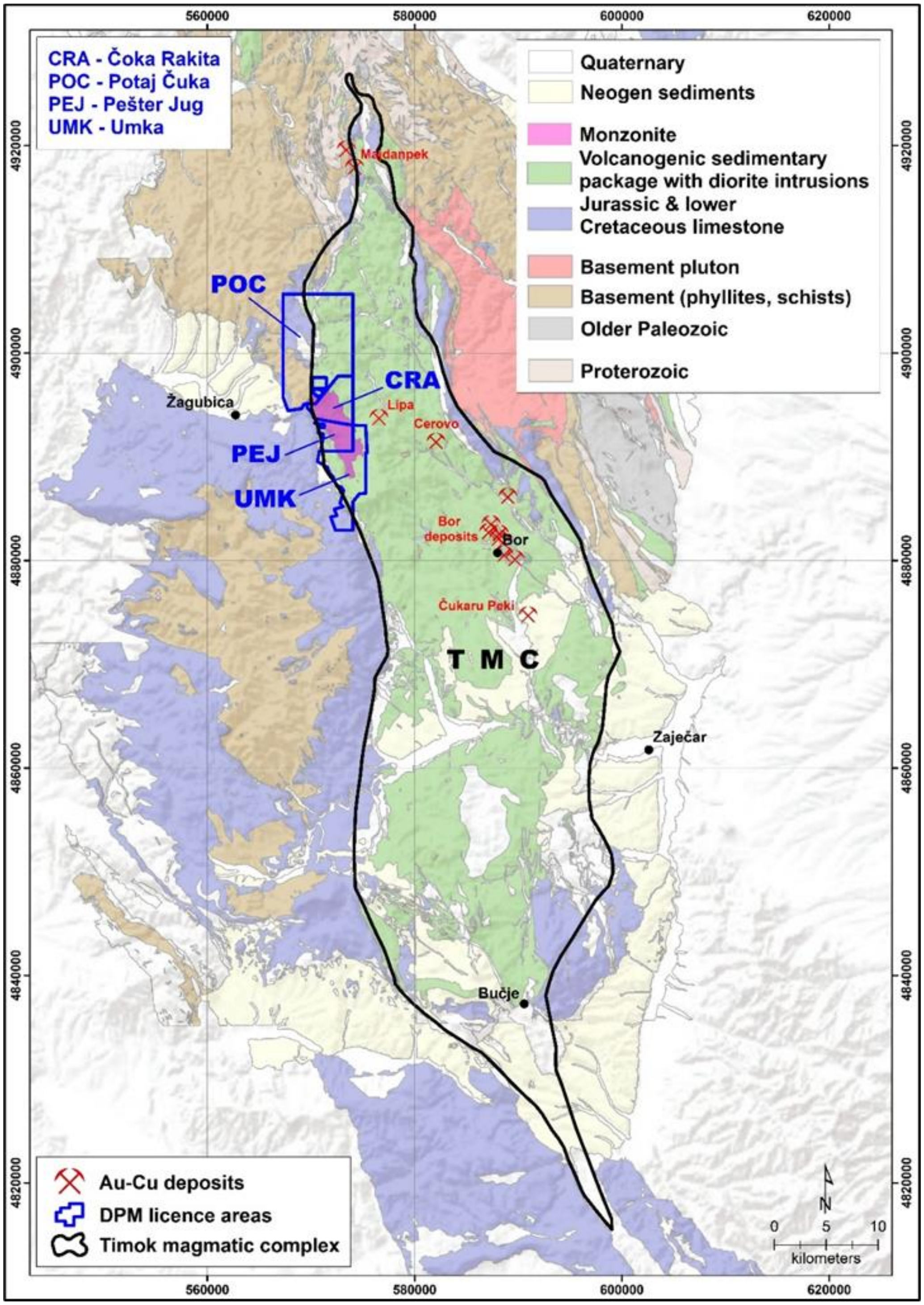
7.3 Local Geology

In eastern Serbia, magmatic activity of the Late Cretaceous ABTSB is developed along two subparallel north-trending branches: the narrow Ridanj-Krepoljin Belt to the west, and the wider TMC to the east. The latter branch contains several world-class Late Cretaceous copper-gold mineral deposits, including, Majdanpek, Veliki Krivelj, Bor, Čukaru Peki and Lipa, which are manifestations, at various levels, within porphyry to epithermal high-sulphidation metallogenetic environments. The TMC is approximately 85 km long and extends from the town of Majdanpek in the north to the village of Bučje in the south. The disposition of DPM's exploration licenses, and the local geology are shown in Figure 7.2.

The Late Cretaceous TMC developed in continental crust composed of different fault-bounded terranes composed of Proterozoic metamorphic to Lower Cretaceous rocks. The area is now incorporated in the Getic Nappe or the Kučaj Terrane, as part of the complex Carpathian Balkan Terrane in eastern Serbia. Upper Jurassic and Lower Cretaceous shallow marine sedimentary rocks, dominated by homogeneous, massive to bedded limestone and marl, unconformably overlie a metamorphic basement. Carbonate sedimentation terminated in the Early Cretaceous due to the impact of the Austrian deformational phase, which caused weak deformation, uplift, erosion, and subsequent paleokarst formation.

Clastic sedimentation commenced with an Albian transgression, unconformably burying the partially eroded and faulted carbonate platform rocks. These calcareous clastic rocks mark the start of the evolution of the TMC, beginning with Austrian deformation and followed by deformation in the Late Cretaceous (Albian). They outcrop along the eastern and western boundary of the TMC but rarely in the central part. Sedimentation continued through the Cenomanian, with an increasingly volcanic detrital component becoming important with decreasing age. During the Turonian, volcanism commenced, and progressed from east-to-west across the TMC. At this time, the TMC became a topographically positive volcanic area.

Figure 7.2 – TMC Geology Showing DPM Avala and Subsidiary Crni Vrh Exploration License Areas



Source: DPM, 2023

Contemporaneous sedimentation, magmatism, and hydrothermal activity were relatively continuous within the TMC throughout the entire Late Cretaceous, as illustrated in Figure 7.2. The sedimentation persisted from the Albian to the Maastrichtian. Late Cretaceous magmatic activity has been documented during a 10-millon-year period from ~89 Ma to 78 Ma and has been interpreted to generally progress from east to west, younging across strike towards the subduction zone. This process can be related to an arc under extension and gradual steepening and rollback of a northward subducting lithosphere slab, derived from the Vardar Ocean.

The TMC is dominated by alkaline to high-potassium calc-alkaline magmatic rocks, which are intercalated with Late Cretaceous volcaniclastic sedimentary rocks. Diorite dykes and sills are common, but locally difficult to distinguish from the volcanic supracrustal rocks.

A synthesis of previous studies by Banješević (2010) concluded that the TMC is interpreted as a succession of the following magmatic suites - Timok andesite, Metovnica epiclastite, Osnić basaltic andesite and Ježevica andesite, the Valja Strž plutonite and Boljevac latites.

The first phase of volcanism commenced during the Upper Turonian with mainly porphyritic, amphibole-andesitic magmatic rocks in the easternmost (present coordinates) parts of the TMC. This is typically referred to as Timok andesite or "Timocite". This is intercalated with Metovnica epiclastites which are composed of fragments derived from different volcanic facies of the Timok andesite suite.

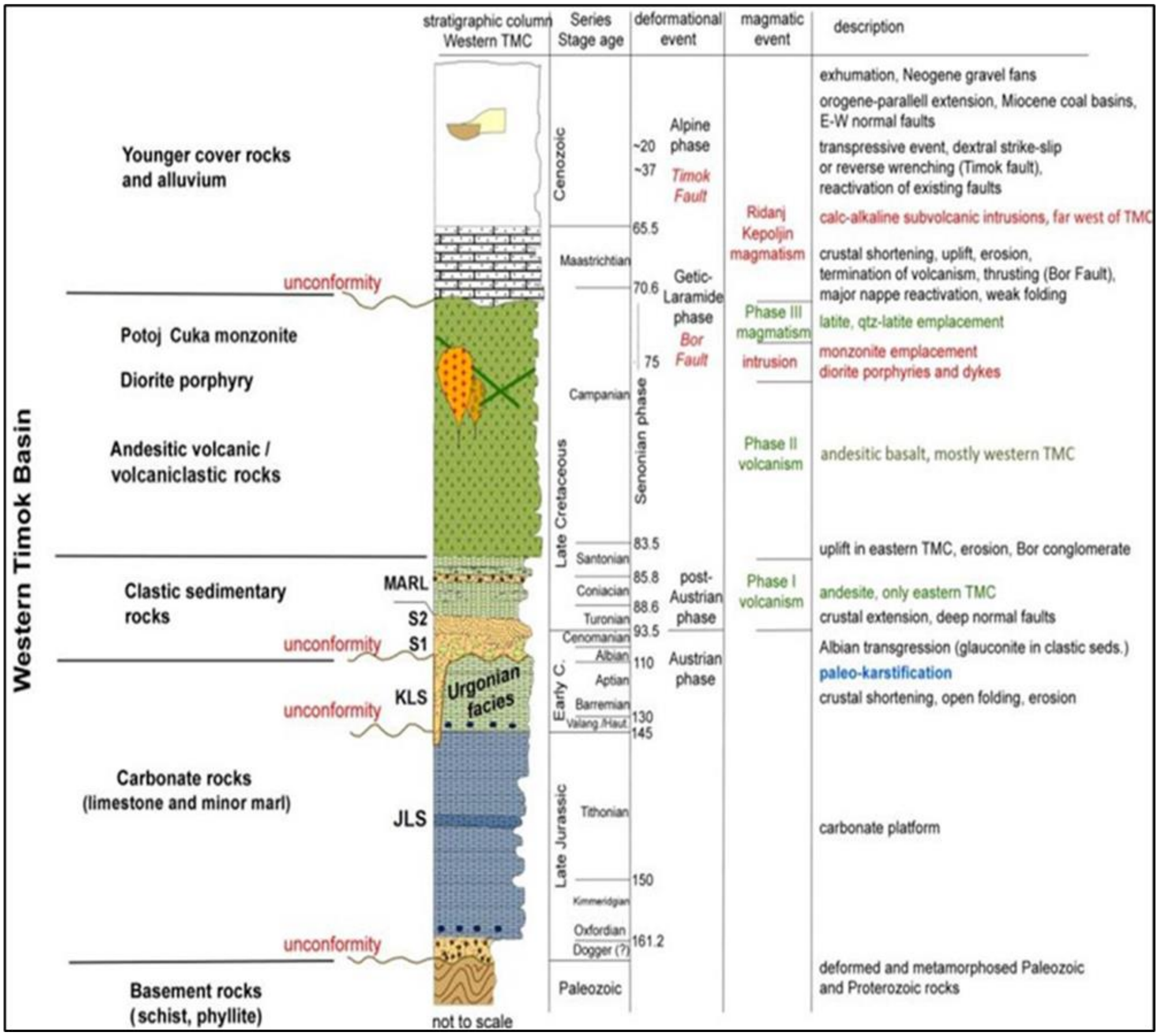
Subsequent phases of magmatism occurred from the Santonian to lower Campanian and comprised pyroxene basaltic andesite (Osnić basaltic andesite) and amphiboles andesite (Ježevica andesite). This suite is mostly found on the central and western portions of the TMC.

During Late Cretaceous (Campanian), diorite, quartz-diorite, and monzonite plutonic rocks were emplaced within the Valja Strž plutonite and the Boljevac latitic dykes. Such rocks from this phase are found in the northwest of the TMC.

The coarse-grained Bor conglomerate records exhumation of the basement within the eastern TMC. Calcareous rocks were deposited in the central part of the TMC at this time. The Upper Cretaceous rocks of the TMC are overlain by Paleogene to Neogene sedimentary rocks and deposits of quaternary sediments.

The structural complexity and present-day asymmetric lozenge-shaped geometry of the TMC area resulted from oroclinal bending during post-collision tectonics throughout the Tertiary. This has led to tectonic modifications of lithological contacts, including those that represent syn-depositional features, beds, or faults. The extent of deformation is commonly difficult to assess due to variable responses of different rock types to the same deformation event. Much of the deformation has been absorbed by argillaceous horizons due to their ability to accommodate shearing and shortening, whereas sandstone beds have resisted much of the deformation. Similarly, competent massive limestone units forming the base of the sequence exhibit minor deformation and much of this is expressed as fracturing near the contact with the overlying clastic sedimentary rocks.

Figure 7.3 – Schematic Stratigraphy of the Western TMC



Source: AMEC, 2014

7.4 Property Geology

Building upon public domain geologic maps and knowledge, mapping and an intensive drilling campaign have defined litho-stratigraphic interpretive units which are recognised as being important to the Čoka Rakita Project and surrounding areas and are summarised below.

7.4.1 Palaeozoic and Proterozoic Basement

Regionally, the oldest outcropping rocks are Palaeozoic phyllites, a meta-sedimentary sequence composed of sandstone, shale, and conglomerate protolith. These units, which have not been further differentiated, do not outcrop in the Project area, but have been encountered at the bottom of some exploration drillholes.

7.4.2 Carbonate Sequence, JLS and KLS

Within the wider Project area, two interpretative carbonate units were defined – the Upper Jurassic (JLS) and Lower Cretaceous (KLS). The older Jurassic age unit is characterised by massive limestone, most which is dominated by bedded and massive bioclastic and micritic, white, light-grey, and light brownish reef limestone of Tithonian age. The lower parts are commonly composed of micritic limestone with concretional chert nodules, and the contact with the underlying basement is commonly faulted. Unconformably overlying the Jurassic limestone (JLS) is Lower Cretaceous dark grey limestone with black concretional chert nodules, deposited during the Valanginian-Hauterivian (Vasić, 2012). This unit is overlined by well-bedded bioclastic, nodular, and stromatolitic, and locally sandy limestones deposited during the Barremian and Aptian; these are referred to as the Urgonian limestone.

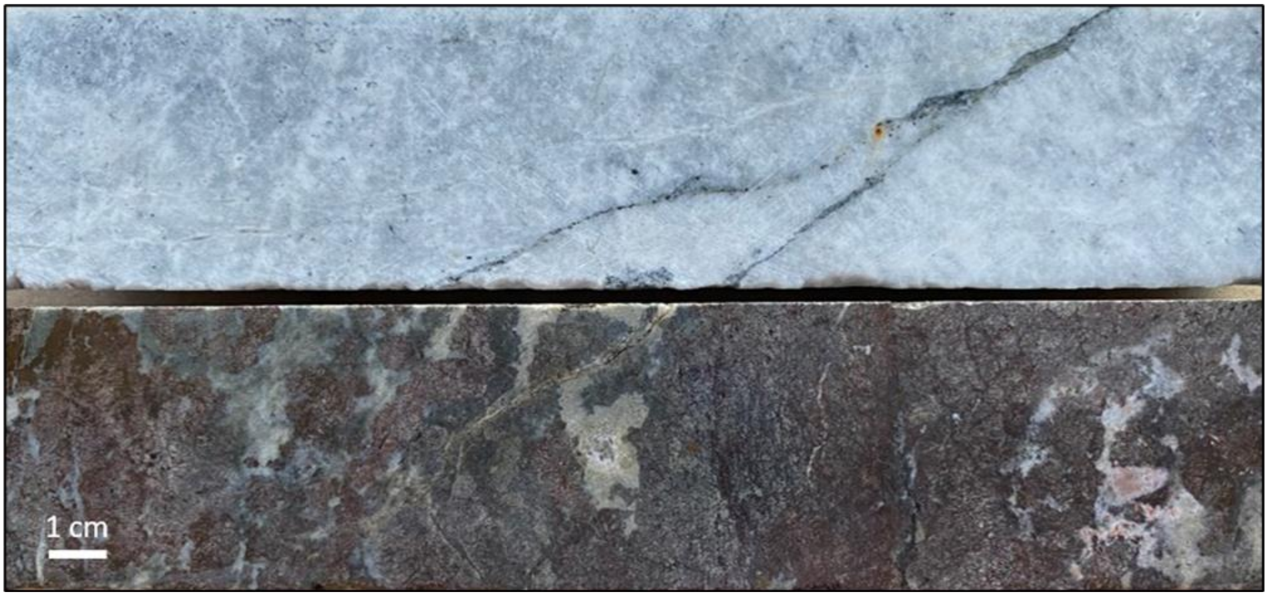
The limestone units are karsted, with the massive Jurassic limestone being more susceptible to karstification than the well-bedded Urgonian limestone. Some paleokarst formed prior to deposition of the younger and unconformably overlying clastic sedimentary rocks. These karst areas are partly filled by syn-karst fine-grained sedimentary rocks, as well as along the upper contact with finely laminated Lower Cretaceous (Albian) calcareous clastic sedimentary rocks. Locally, paleokarst collapse breccia is developed, the karsted zones might host gold mineralisation. Recent karst forms are also evident, including sinkholes and active caves.

Within the Čoka Rakita project area, limestones are exposed to contact metamorphism on the margins with the monzonite pluton, forming moderate to coarse-grained marble. Due to the metamorphism, the formation age of the protolith limestone was not possible to be determined. On the contact with the upper, clastic sequence and locally within the marble, skarn-altered, garnet-calcite-quartz-hematite-pyroxene domains are formed, potentially on the paleokarst protolith or in structurally predisposed areas. These skarn-altered zones frequently host copper-gold-polymetallic mineralisation. Marble and skarn-altered specimens are shown in Figure 7.4.

7.4.3 Calcareous Clastic Sedimentary Rocks, S1 and S2

Three (3) distinct units of calcareous clastic rocks unconformably overlie the carbonate sequence in the wider area of the Čoka Rakita Project. Various carbonate units lie beneath the unconformity, indicating exhumation and accompanying faulting during the depositional hiatus in the Early Cretaceous. Formation of the unconformity reflects the effect of the Cretaceous Austrian orogenic event. The clastic units, stratigraphically from lowest-to-highest, include calcareous sandstone and conglomerate with lesser siltstone-dominated sequence, overlain by reddish and iron-rich sandstone containing abundant andesitic volcanic detritus, capped by thinly bedded ferruginous marl. The stratigraphic sequence dips gently to east, at 20–30°, and its thickness extends to several hundreds of metres.

Figure 7.4 – Core Specimens of the Marble and Garnet-Hematite-Quartz Skarn

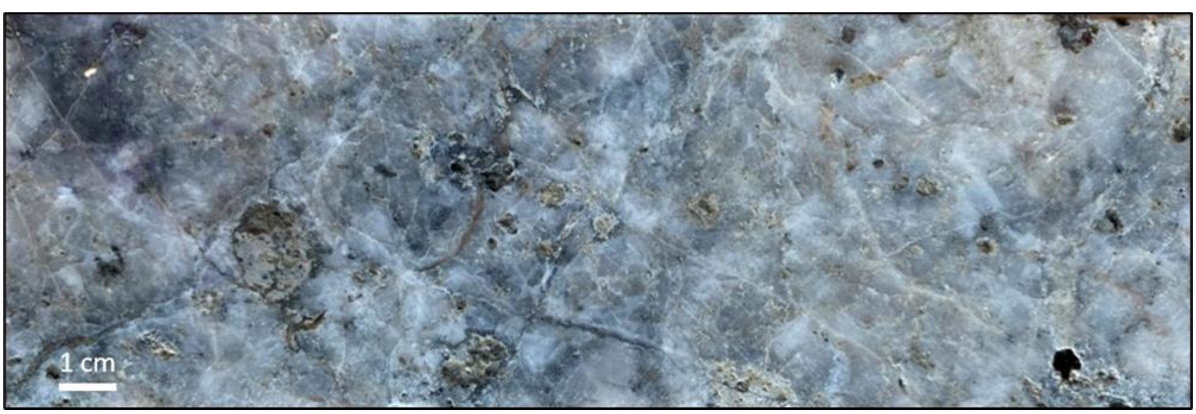


Source: DPM, 2023

Based on the composition and the alteration types exhibited at Čoka Rakita, the clastic sequence is divided into the S1Q, S1/S2, and marl units.

The S1Q unit is composed of recrystallised siliciclastic conglomerates and sandstones. Quartz is the dominant rock forming mineral, while calcite, kaolinite, chlorite, pyrite is present in subordinate amounts. Along the basal unconformable contact, the S1Q contains coarse blocks alongside smaller cobbles and pebbles of skarn-altered carbonate fragments, often hosting copper-gold-polymetallic mineralisation. The thickness of the unit varies between 50 m and 300 m. A typical core specimen from the S1Q unit is presented in Figure 7.5.

Figure 7.5 – Typical S1Q Unit: Recrystallised Quartz Conglomerate



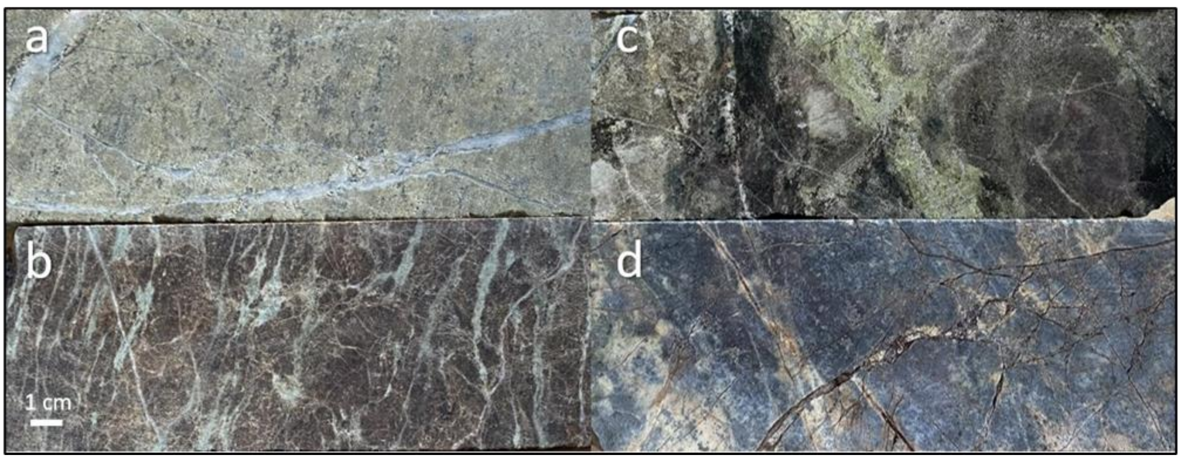
Source: DPM, 2023

The S2/S1 unit encompasses carbonate-rich sandstones, located on the hanging-wall of a fertile sill-like diorite body and abutting a monzonite pluton to the west. The proximity of the intrusive bodies, and associated contact-metasomatic processes led to the formation of skarns, and to a lesser extent of hornfels, on the account of the carbonaceous clastic protolith. Skarn-altered sandstones exhibit a large mineralogical and textural variability and are roughly divided into:

1. Prograde, garnet-dominated skarn, with subordinate pyroxene, quartz, wollastonite; and
2. Retrograde skarn, with epidote-chlorite-actinolite-secondary carbonate over prograde garnets, formed in proximity to syn-mineral diorite.

The S1/S2 unit is the main host rock for the gold-rich skarn mineralisation, with coarse gold mainly deposited within retrograde skarn. The thickness of the unit is between 80 m and 160 m. Examples of S1/S2 unit skarn-altered rocks are shown in Figure 7.6.

Figure 7.6 – Varieties of S2/S1 Skarn-Altered Sandstone



Yellowish, garnet-dominated prograde skarn. (b) Dark red/brown, garnet-dominated prograde skarn. (c) Garnet-epidote-actinolite retrograde skarn. (d) Pyrrhotite-magnetite skarn.
Source: DPM, 2023

The marl unit, overlying the S1/S2, was deposited during Santonian time and is typically finely laminated. Within the Čoka Rakita deposit area, the distinction between skarn-altered marl and S2/S1 sandstone is difficult, however to the north, south and west fringes of the mineralised zone, away from the thermal and metasomatic impact, fine-grained, bedded marls have been identified in core. A typical marl specimen is shown in Figure 7.7.

Figure 7.7 – Example of Marl Unit from Drillhole RADD022

Source: DPM, 2023

7.4.4 Epiclastics Unit

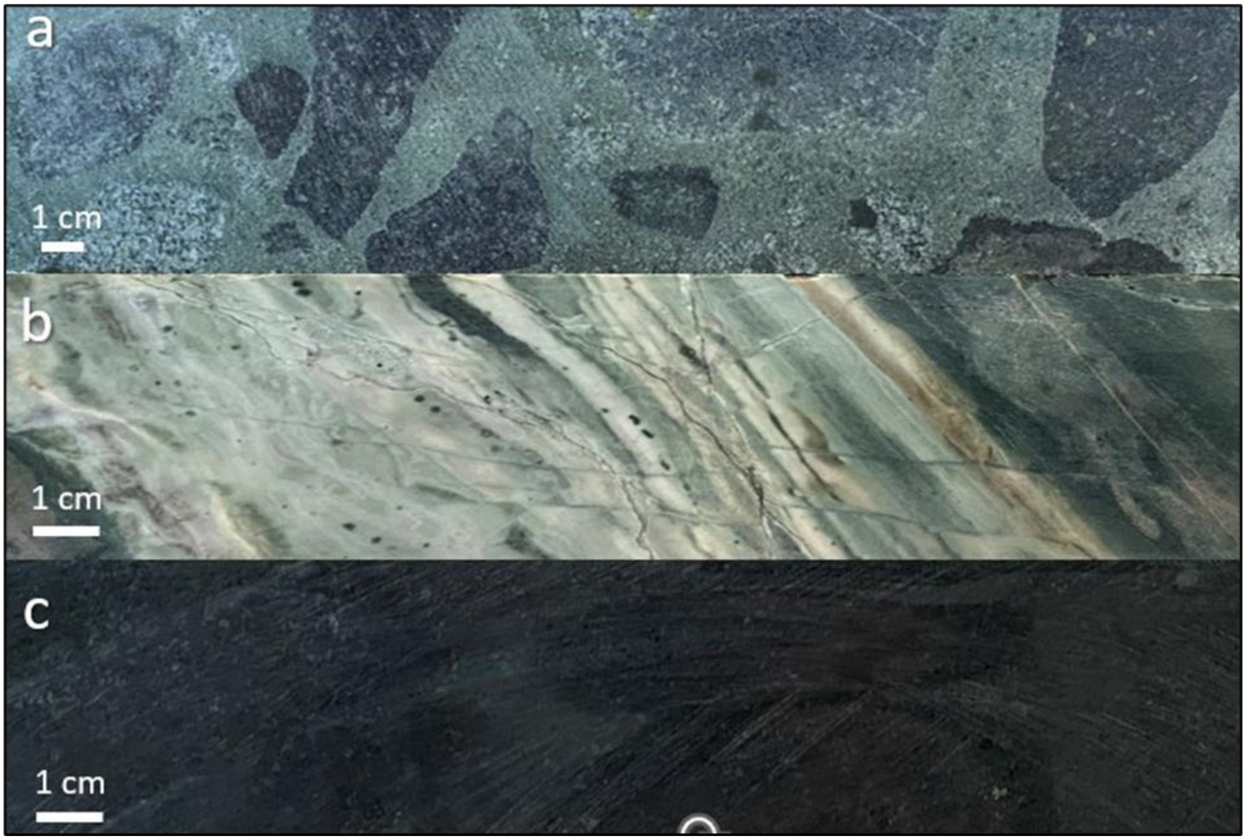
The Late Cretaceous epiclastic (SFD) unit is andesitic in composition and characterised by rapid facies changes throughout the sequence. The lower part of the epiclastic unit is characterised by polymictic basaltic andesite conglomerate and breccia, whereas the upper part is dominated by monomictic basaltic andesite breccia and conglomerate, which are interpreted being products of epiclastic debris flow deposits.

A finer grained sedimentary rock unit, consisting of well-bedded siltstone and sandstone locally forms a thin, but mappable sub-horizons within the debris flow deposits. Facies types within the epiclastic unit are presented in Figure 7.8.

7.4.5 Potaj Čuka Monzonite

The Monzonite unit comprises a coarse-grained equigranular intrusive with visible alkali feldspar phenocrysts, biotite, and minor magnetite and pyroxene. This monzonite is part of the Late Cretaceous Potaj Čuka pluton which, in the Campanian (79.8±0.6 Ma; uranium-lead in zircon), intruded the clastic sedimentary units in the region. The Potaj Čuka pluton is located immediately east of the western margin of the TMC and is elongated in a north-westerly orientation. Narrow, sub-vertical monzonite dykes, oriented sub-parallel with the main pluton, also intrude the clastic sequence.

Figure 7.8 – Examples of Facies Types from the Epiclastic Unit



(a) Polymictic breccia, with diorite, and skarn altered clasts. (b) Laminated siltstone with discrete skarn alteration along the bedding planes. (c) Dark, magnetic epiclastic sandstone with relic clast fragments.
Source: DPM, 2023

7.4.6 Diorite Intrusions

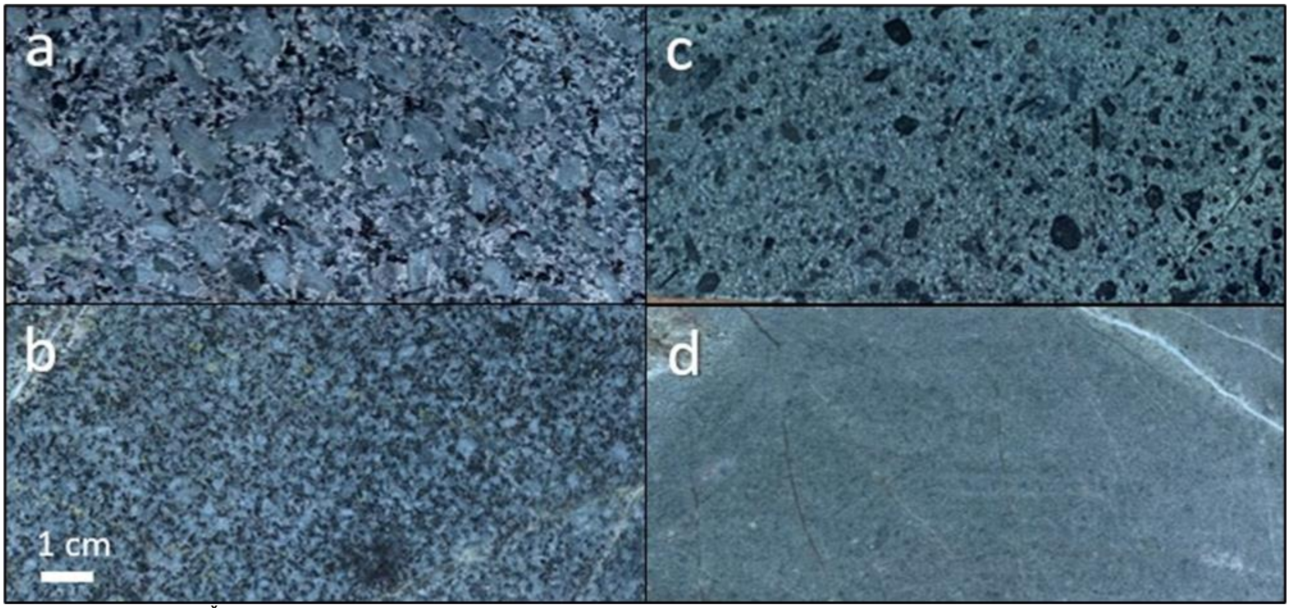
Numerous dioritic stocks, dykes, and sill-like intrusions are observed at Čoka Rakita. These bodies dominantly trend in a north-westerly orientation, which most likely represents a structural fabric in the subsurface that controlled their emplacement. The main intrusive phases identified on Čoka Rakita are early mineral porphyry (EMP), late mineral big hornblende porphyry (BHP), and post-mineral pyroxene porphyry (PXP).

- The pre-to-syn mineralisation EMP-type intrusive, is a sill-like plagioclase-hornblende phyric, micro-diorite. Locally the Diorite was exposed to potassic and phyllic alteration, and, in contact with S2/S1, to endoskarn alteration. EMP emplacement is dominantly controlled by S2/S1 and S1Q contact, hence it dips to the east at approximately 30°.
- BHP is more mafic diorite variety, with large hornblende phenocrysts. BHP is mainly found within the epiclastic unit and spatially most often in the northern part of the deposit.
- PXP-type intrusives are narrow (1–2 m thick), sill-like bodies comprising intensively altered mafic, pyroxene-rich porphyritic diorite. Such sills are mainly emplaced within S2/S1 unit and

appear to crosscut mineralisation. Within the PXP, potassium feldspar is plagioclase altered and primary texture is unobservable.

Based on crosscutting relationships, the Potaj Čuka Monzonite is the oldest intrusive body within the Čoka Rakita area, often cut by EMP. However, monzonite dykes have in turn been observed to crosscut EMP and sedimentary sequence, indicating a series of monzonitic intrusive events occurred. Typical specimens of the intrusive units are shown in Figure 7.9.

Figure 7.9 – Čoka Rakita Intrusive Units



(a) Potaj Čuka monzonite. (b) EMP – early mineral porphyry. (c) BHP – big hornblende porphyry. (d) PXP – post-mineral pyroxene porphyry.
Source: DPM, 2023

7.5 Structure

The structural regime of the Čoka Rakita deposit is represented by brittle type deformation and replicates the architecture of the western margin of the TMC on a local scale.

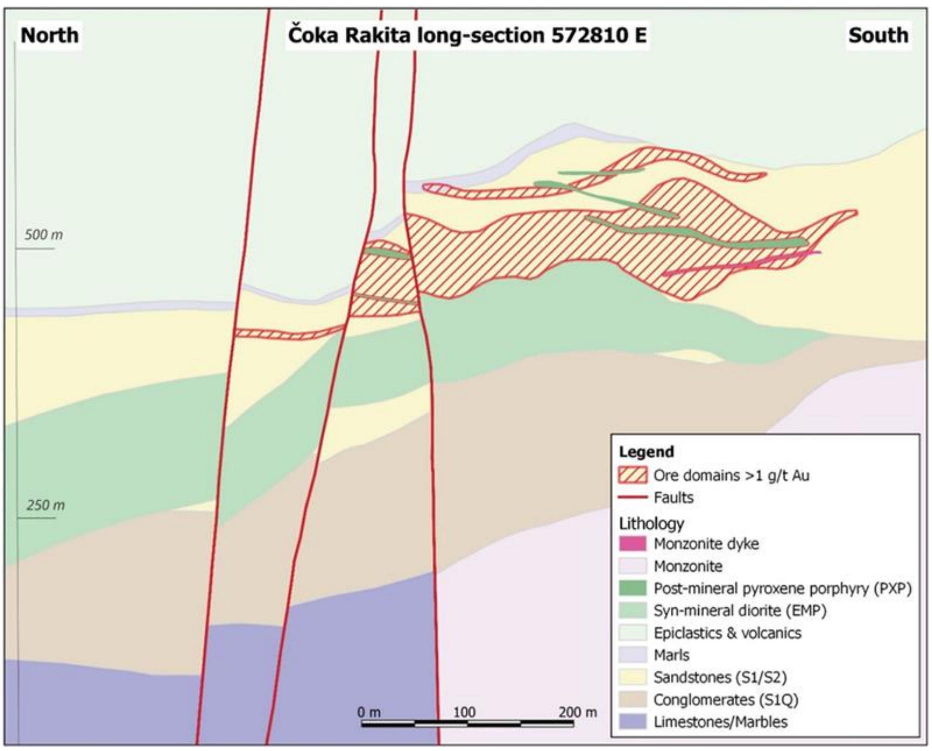
The Cretaceous clastic and epiclastic units are crosscut mainly by shallow angle faults, subparallel to the main stratigraphic bedding – dipping to the east, southeast and northeast at 10° to 40°. Such low angle faulting is localised to the contacts between the sedimentary units, such as epiclastics-marls, marls-sandstones and sandstones-S1Q. The S1Q unit exhibits more widespread evidence for low angle faulting, however, the sense of movement is yet to be defined.

Steep north-south striking structures are observed in the central and southern parts of the deposit, dipping to the west at 70° to 80°. They appear as brittle deformation zones and often occur as hydrothermal breccia bodies. On the eastern margin of the deposit, one of those structures is accommodated in the epiclastic and S1/S2 units and appears to limit the extent of the gold mineralisation and marks the boundary of the mineralisation.

North-south fabrics in the EMP diorite are found as magmatic breccia-like bodies, located mainly on the western flank, where brittle deformation is limited. At depth, within the S1Q and limestones, these fault structures are not observed. There is no clear evidence that indicates such structures persist through the lithology sequence.

Several late east-west faults are observed on the north flank of the Čoka Rakita deposit (Figure 7.10). They are steep dipping to sub-vertical and more commonly dipping to the north. Current working interpretations proposes these faults have both sinistral and normal kinematics, which explains the drop in the stratigraphy to the north of Čoka Rakita, as well as the sharp bends of the monzonite contact at surface. Vertical displacement within such fault structures is up to 50 m. In between the main east-west faults, conjugated east-southeast striking structures with the same kinematics are developed that further complicate the stratigraphy with displacement up to 25 m to 30 m.

Figure 7.10 – Schematic long-section, looking east, through Čoka Rakita displaying geological setting and different litho-tectonic blocks, divided by late east-west faults



Source: DPM, 2023

7.6 Mineralisation

The main mineralisation type found within the Čoka Rakita Project is the high-grade manto-like skarn gold-copper mineralisation, found as primarily stratigraphic controlled and to a lesser extent as structurally controlled massive stratabound lenses within calcareous S1 and S2 sandstones at the hanging-wall contact of the sill-like EMP intrusion.

The outlined high-grade gold-skarn mineralisation is intimately linked at deposit and project scale to other mineralisation types including:

- Porphyry gold-copper-molybdenum mineralisation at two stratigraphic levels, including:
    1. Stockwork quartz veinlets and disseminations related mineralisation in the potassic altered EMP, and
    2. Epiclastic-hosted gold mineralisation controlled by structural and lithology contacts.
- Stratabound copper-gold mineralisation at deeper stratigraphic settings, including:
    1. Conglomerate-hosted, copper-gold-polymetallic mineralisation, located on the footwall of the mineralised EMP intrusion, and
    2. Marble and skarn altered limestone-hosted copper-gold mineralisation with iron-hydroxides, pyrite, chalcopyrite, bornite and chalcocite.

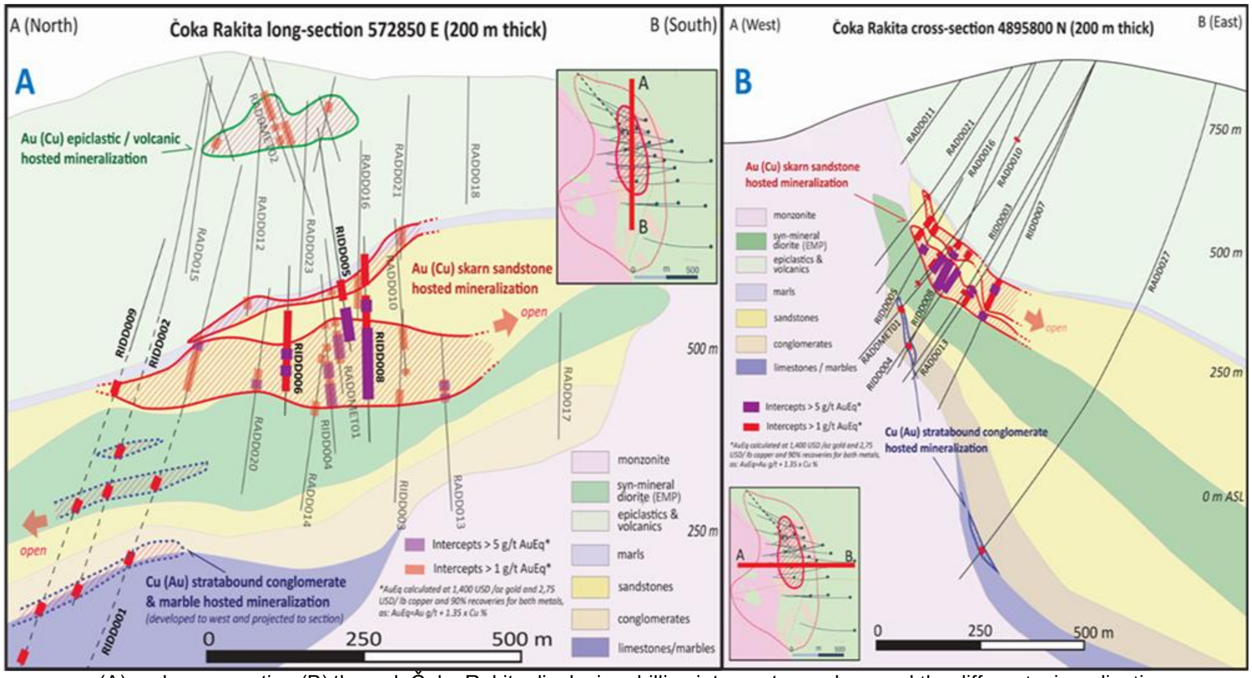
The location of the main mineralisation styles relative to the skarn-hosted gold mineralisation is shown in Figure 7.11.

7.6.1 High-Grade Manto-Like Gold-Copper Skarn Mineralisation

At Čoka Rakita, exoskarn formation in the S1/S2 calcareous clastic sedimentary rock sequence on the hanging-wall of the EMP intrusive is the principal mineralised horizon in terms of gold endowment.

The mineralisation is located between 250 m and 600 m below surface and has been traced over a footprint of 650 m x 350 m. It has variable thickness, from less than 20 m in the margins to more than 100 m in the core of the mineralised zone. The mineralisation forms a lens-like shape that dips between -40° to -50° to the east. Mineralisation is primarily stratigraphically controlled, with the lower boundary of mineralisation closely following the EMP sill contact. Endoskarn formation typically persists for a short distance within the EMP. As a second order control, steeper north-south striking structural trend can also be determined, which is evidenced by the north-south elongation of the high-grade mineralised zones and with the occurrence of mineralised subvertical phreatic breccia zones.

Figure 7.11 – Schematic Section



(A) and cross section (B) through Čoka Rakita displaying drilling intercepts, geology and the different mineralisation types identified to date
Source: DPM, 2023

Gold mineralisation is located within the andradite-grossular garnet skarn and is dominantly associated with a retrograde assemblage that comprises a quartz, K-feldspar, epidote, biotite, chlorite, albite, calcite, and apatite paragenesis. Gold is present in its native form and thought to have precipitated in a wide range of hydrothermal phases with the main ones being: (1) native gold and pyrite-dominant mineralisation, with minor chalcopyrite-bornite-chalcocite±molybdenite; and (2) native gold and pyrrhotite-magnetite mineralisation with minor chalcopyrite-sphalerite-pyrite-galena-bismuth sulfosalts-tellurides. A petrographic analysis of 48 samples from across the Čoka Rakita deposit (Pacevski, 2023) determined that although gold occurs in its native form, it almost always contains silver in different concentrations, preliminary SEM-EDS analysis indicating up to 10 wt% Ag content.

Gold appears as disseminations and often as visible aggregates that reach up to a few centimetres in size. These gold grain aggregates frequently occupy interstitial position between the garnet and pyroxene grains affected by the retrograde alteration. Gold grade continuity is variable; high levels of grade continuity are observed in the core of the system but this gradually decays moving outwards.

An example of coarse visible gold in retrograde skarn altered S2/S1 sandstone from drillhole RIDD025 is shown in Figure 7.12.

Figure 7.12 – Coarse Visible Gold from Drillhole RIDD025

(a) Core trays with HQ size half-core, with gold grades and marked visible gold aggregates (red circles, or red arrows indicating gold present on the other side of the core). (b) View at 10X magnification of an individual gold aggregate. (c) Half core photo from 478.4 m downhole with red marks highlighting visible gold. (d) Fine gold grains in association with pyrite from around 478.6 m.
Source: DPM, 2023

7.6.2 Porphyry Gold-Copper±Molybdenum Mineralisation

Porphyry gold-copper mineralisation occurs at two stratigraphic levels, including: 1) stockwork quartz veinlet related mineralisation in the potassic altered EMP, and 2) epiclastic-hosted gold mineralisation with a quartz-biotite-epidote-sericite-pyrite footprint controlled by structural and intrusion contacts.

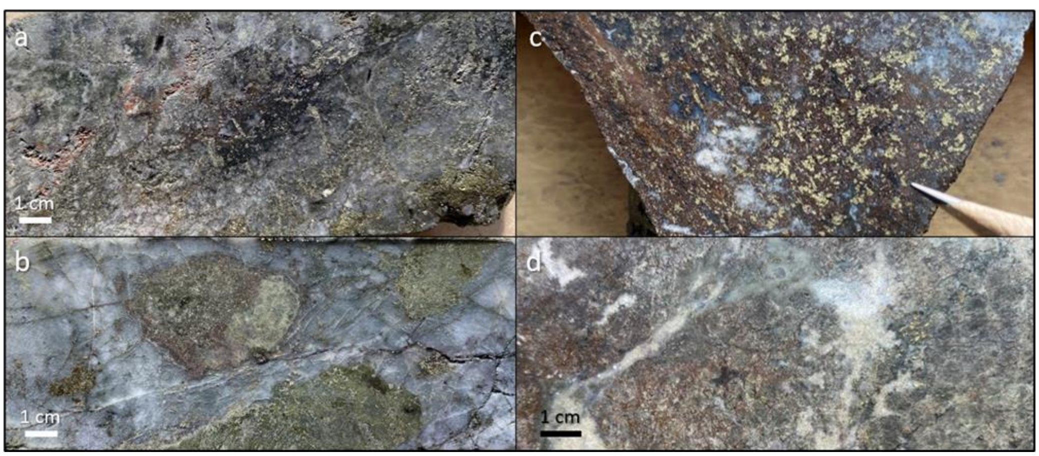
A pervasive secondary biotite±magnetite-K-feldspar assemblage formed in the EMP intrusion is associated with the low-grade disseminated and porphyry quartz-veinlet hosted copper±gold-molybdenum mineralisation. The mineralisation is represented by chalcopyrite, magnetite, pyrite and molybdenite. Although, the continuity of this mineralisation can be mapped and correlated across the entire EMP intrusion, the contained gold and copper grades are low and currently subeconomic.

Isolated occurrences of epiclastic-hosted gold mineralisation are found sporadically within the overlying epiclastic unit. Gold mineralisation is typically found with aggregates and disseminations of pyrite, often with minor sphalerite, galena, chalcopyrite and pyrrhotite and found with weak to moderate potassic alteration and overprinting phyllic alteration selvedges. Epiclastic mineralisation rarely forms contiguous zones. Its occurrence is thought to be controlled by a complex interaction of structures and lithology contacts. Current understanding is that the level of grade and geologic continuity between drillholes is poor. The most significant mineralisation of this type has been encountered directly above underlying skarn associated gold-(copper) mineralisation, indicating such manifestations maybe associated with leakage structures that have allowed hydrothermal fluid to escape upwards.

7.6.3 Stratabound Copper-(Gold) Mineralisation

Stratabound copper-gold-polymetallic mineralisation is located on the footwall of the EMP intrusive, approximately 550 m below surface, and is hosted by basal recrystallised siliciclastic conglomerates (S1Q unit) and sandstones, mainly in skarn altered carbonate fragments in the basal part of the unit, or in structurally predisposed zones. The mineralisation type is characterised by an assemblage dominated by pyrite and chalcopyrite, with molybdenite, sphalerite, galena, bornite, chalcocite, pyrrhotite in subordinate quantities. An example of pyrite and copper sulphide mineralisation in the S1Q unit is shown in Figure 7.13.

Figure 7.13 – Stratabound Copper-Gold Mineralisation



(a) Pyrite-chalcopyrite-bornite mineralisation in recrystallised quartz conglomerate, with sparse carbonate fragments, drillhole RIDD002 (b) Chalcopyrite – pyrite ± molybdenite partial replacement of carbonate fragments within recrystallised quartz conglomerates, drillhole RIDD009. (c) Chalcopyrite-pyrite-sphalerite massive sulphide mineralisation in skarn altered limestone, drillhole RIDD002. (d) Chalcopyrite-bornite disseminated mineralisation within skarn altered limestones, drillhole RADD044.
Source: DPM, 2023

At deeper levels, stratabound copper-gold-polymetallic mineralisation can also be found in marble-ised limestones and on the limestone-sandstone contact. It is mainly associated with skarn alteration and intensive iron-oxides replacements in paleokarst and structurally predisposed zones. The garnet-dominated skarn-altered limestone also contains pyroxene, actinolite, secondary calcite and silica. Chalcopyrite and pyrite are major sulphide minerals, with minor bornite, chalcocite, molybdenite, sphalerite and galena. Sulphide minerals occur in the form of disseminations, mottled aggregations, or veinlets. Sulphide mineralisation in limestone unit is shown in Figure 7.13.

Long intercepts of continuous copper-gold stratabound, limestone-hosted mineralisation had been encountered during initial scout drilling within the north of Čoka Rakita Project. So far this mineralisation type has been found between 650 m and 1,000 m below surface. Due to the depth of the prospective formation, systematic drilling has yet to be completed and the continuity of mineralisation yet to be established.

7.7 Alteration

7.7.1 Endoskarn

Endoskarn alteration is developed close to contacts with sedimentary units and typically extends 1–3 m internally within the EMP or monzonite intrusive bodies. Endoskarn mineral formation comprises epidote, actinolite, chlorite, K-feldspar, plagioclase, and biotite. Skarn alteration within intrusive bodies is significantly less prevalent compared to the corresponding exoskarn. Within epiclastic units, skarn alteration is structurally controlled, with the development of epidote, chlorite, garnets and magnetite. On rare occasions, endoskarn altered intrusives have been observed to host notable gold mineralisation, including visible gold aggregates, such is the case in drillholes RIDD005 (441–442 m, 354 g/t Au) or RIDT027 (481–483 m, 284 g/t Au and 234 g/t Au respectively).

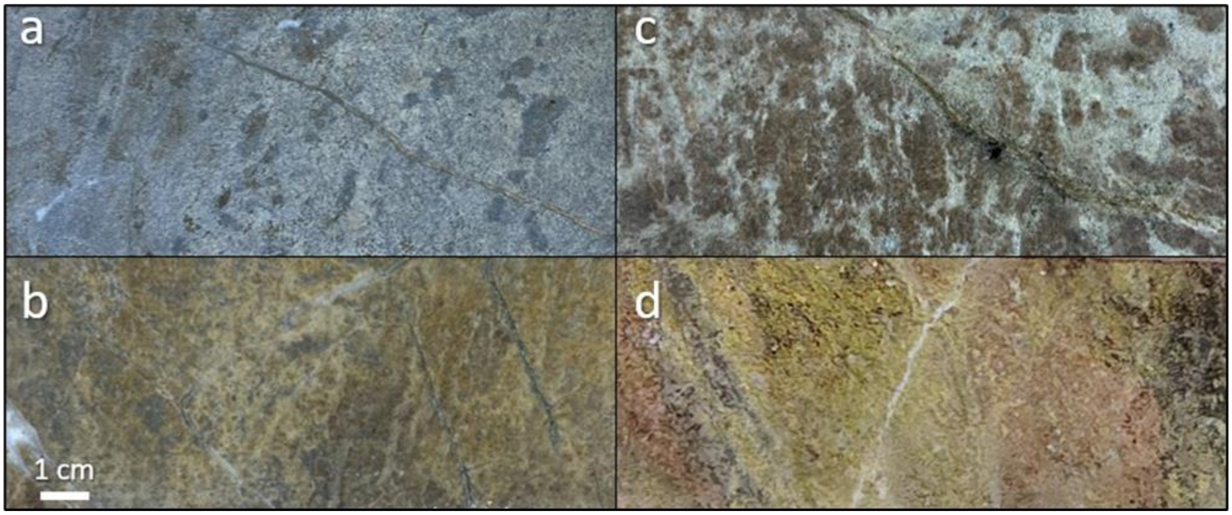
7.7.2 Exoskarn

The Čoka Rakita calcareous sedimentary rocks are exposed to intensive levels of skarnification. Both prograde and retrograde phases of alteration are developed. Mineral, chemical, and textural variability is common in both phases. Both the sulphide and gold mineralisation events are mainly associated with retrograde skarn.

Prograde skarn is dominated by garnets of andradite-grossular series, with lesser pyroxene, wollastonite and quartz. On the western part of the project area, the skarn formation is typically dark red, with massive, banded or “leopard” texture, while to the east, its colour changes to yellow ochre. Prograde alteration forms a broad alteration envelope that extends beyond the limits of known gold mineralisation.

The subsequent retrograde phase partially or fully overprints prograde skarn and is mainly developed near the contact with the early mineral diorite. The retrograde alteration assemblage consists of hydrous mineral assemblage of epidote, chlorite, actinolite, secondary calcite and K-feldspar. This skarn has a patchy appearance, with domains of various colours, and different mineral composition. Various exoskarn specimens from the project are shown in Figure 7.14.

Figure 7.14 – Exoskarn Alteration



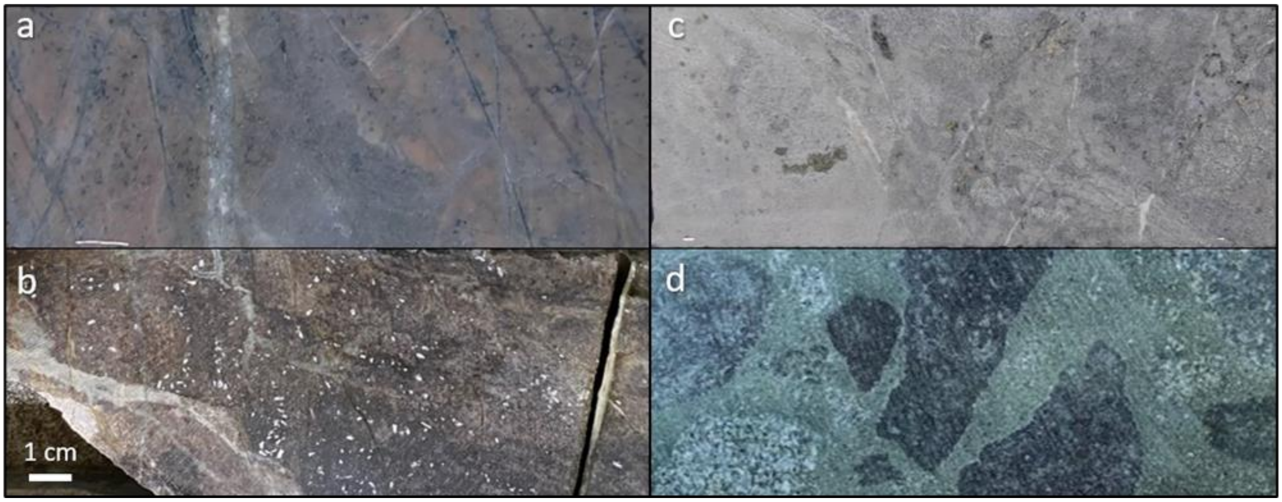
(a) Prograde, pale-yellow garnet-pyroxene skarn. (b) Prograde, yellow ochre garnet skarn. (c) Prograde, dark red garnet, wollastonite, pyroxene skarn with "leopard" texture. (d) Retrograde, epidote, actinolite, calcite skarn overprinting prograde skarn.
Source: DPM, 2023

7.7.3 Porphyry Mineralisation Related Alteration

The Čoka Rakita diorite and epiclastic sequence were exposed to porphyry mineralisation-related hydrothermal alterations, with the development of all three main generative types, potassic, phyllic, and propylitic. Potassic alteration is dominantly represented by secondary biotite, and subordinate magnetite, while K-feldspar is significantly less present, and the question remains if it is partly related with the skarn alteration. Pervasive secondary biotite±magnetite-K-feldspar assemblage formed on the EMP is associated with the low-grade disseminated copper±gold-molybdenum mineralisation.

Within the epiclastic unit, intensive potassic alteration is observed within structurally disturbed zones, while otherwise it appears as mottled, moderate to weak intensity feature, usually overprinted by phyllic association. Quartz, sericite, and pyrite are the main constituents of the phyllic alteration, which usually pervasively encompasses the host rock. Marginal, propylitic alteration is observed within the epiclastic unit, as a pervasive feature, frequently affecting the matrix material of epiclastic breccia. Main constituents of the propylitic alteration are epidote, chlorite, and carbonate. Typical examples of porphyry-related alterations are shown in Figure 7.15.

Figure 7.15 – Porphyry Mineralisation Related Alteration



(a) K-feldspar alteration in early mineral diorite, drillhole RIDT030B. (b) Biotite-actinolite alteration in epiclastic unit, drillhole RIDD007. (c) Quartz-sericite-pyrite alteration on epiclastic breccia, drillhole RIDD030. (d) Propylitic, epidote-chlorite alteration on epiclastic breccia matrix, drillhole RIDT009.
Source: DPM, 2023

8 DEPOSIT TYPES

8.1 Deposit Style

The Project conforms to an oxidised gold skarn type deposit. Such gold skarn deposits are exploited predominantly for gold, and exhibit calc-silicate alteration, usually dominated by garnet and pyroxene (Einaudi et al., 1981; Meinert et al., 2005). Most gold skarns form in orogenic belts at convergent plate margins. They tend to be associated with syn to late inter-oceanic island arc intrusions emplaced into calcareous sequences in arc or back-arc environments.

On the Project, skarn type mineralisation is primarily stratigraphically controlled and to a lesser extent structurally controlled and is found as massive, manto-like, stratabound lenses in Cretaceous calcareous clastic sedimentary rock sequence, intimately related with the proximity to fertile Late-Cretaceous dioritic intrusions.

The Čoka Rakita deposit exhibits characteristics that align with an oxidised gold skarn type. These qualities include:

- The dominance of garnet over pyroxene is typical characteristic of other oxidised gold skarns (Meinert, 1998). At Čoka Rakita, andradite-grossular garnet types are dominant with lesser pyroxene, wollastonite, and quartz in prograde alteration.
- In comparison to reduced gold skarns, oxidised gold skarns exhibit high garnet/pyroxene ratios, relatively poor iron garnet and pyroxene, and low total sulphide content (Brooks et al., 1991; Meinert, 2000). Gold at Čoka Rakita is present in sulphide-poor mineralisation assemblage and assumingly precipitated in a wide range of subsequent hydrothermal mineralising phases, the main ones being: (1) native gold and pyrite, with minor chalcopyrite-bornite-chalcocite-molybdenite; and (2) native gold, pyrrhotite and magnetite with minor chalcopyrite-sphalerite-pyrite-galena-bismuth sulfosalts-tellurides.
- Additionally, the highest gold grades are often associated with later retrograde alteration including abundant K-feldspar and quartz. The most significant gold mineralisation at Čoka Rakita, in terms of grade and continuity, is associated with retrograde alteration assemblages, which typically contain quartz, K feldspar, epidote, biotite, chlorite, albite, calcite and apatite. Native gold, in grains up to several millimetres in size, frequently occupies interstitial position between the garnet and pyroxene grains affected by this retrograde alteration.

In terms of deposit and mineralisation features, the Čoka Rakita deposit shares similarities with the documented Jurassic Nambija oxidised gold skarns in the sub-Andean zone of southeastern Ecuador (Fontboté et al., 2004; Vallance et al., 2009). In the Nambija district, which has been exploited since the 16th century, the gold deposits occur mainly in skarn bodies developed in a Triassic volcano-sedimentary rocks of the Triassic Piuntza unit. Total exploited and remaining resources at Nambija were estimated in range of 15 to 20 M oz gold.

Gold grades are typically high (average 10 to 30 g/t and up to 1,000 g/t), whereas the contents of copper, zinc, lead, and other metals are very low in most mines (Prodeminca, 2000). The Nambija skarns also consist dominantly of granditic garnet with subordinate pyroxene (diopside–hedenbergite) and epidote and are spatially associated with porphyritic quartz-diorite to granodiorite intrusions.

Endoskarn is developed at the intrusion margins and grades inwards into a potassic alteration zone. Exoskarn has an outer potassium- and sodium-enriched zone in the volcano-sedimentary unit. Gold mineralisation is associated with the weakly developed retrograde alteration of the exoskarn and occurs mainly in sulphide-poor vugs and milky quartz veins and veinlets in association with hematite (Vallance et al., 2009).

8.2 Concepts Underpinning Exploration

Exploration for skarn-type deposits such as Čoka Rakita require careful analysis of spatial data and temporal relationships. Detailed interpretation of the spatial distribution of the calcareous clastic sedimentary host-stratigraphy, the fertile intrusions, and the overprinting skarn and potassic alteration assemblages is critically important when exploring for skarn mineralisation on the Project.

Zonation in alteration and mineralisation is a common facet of skarn deposits and this is clearly evident at Čoka Rakita. To that end, the interpretation of surface geochemical footprints (from rock and soil sampling surveys) is a key targeting vector, with special emphasis on the analysis and distribution of various magmatic-hydrothermal related chalcophile elements components (gold-copper-molybdenum-bismuth, gold-silver-lead-zinc, gold-arsenic-antimony-thallium).

Geophysical techniques have been extensively utilised to help evaluate the underlying subsurface architecture and identify potential targets within the DPM licenses. Understanding the various components of magmatic-hydrothermal systems associated with skarn deposits has been guided by the acquisition and interpretation of electrical geophysical data (magnetotellurics and induced polarisation surveys). The interpretation of magnetic geophysical surveys data has helped to outline the extent of various magmatic intrusions and to vector toward mineralisation-related, magnetic mineral assemblages (pyrrhotite, magnetite). Furthermore, gravity surveys have been used to improve the modelling of magmatic intrusions and vector toward denser, garnet-pyroxene skarn assemblages.

9 EXPLORATION

9.1 Introduction

Following the granting of the Čoka Rakita exploration license, DPM completed extensive soil sampling on the Project between 2007 and 2009 and identified a series of gold-in-soil anomalies. Wide-spaced follow-up drilling intercepted shallow, structurally controlled, epiclastic breccia hosted gold mineralisation. The mineralisation was highly complex and was evaluated as possessing poor metallurgical characteristics, and as such, DPM deemed that the prospect had limited resource potential. Although the drilling during this phase adequately evaluated the near-surface mineralisation, it failed to reach the target skarn stratigraphy and consequently, the Čoka Rakita gold skarn deposit was not detected.

A hiatus in exploration at Čoka Rakita occurred during this time while focus was on developing sediment-hosted gold mineralisation, found on the adjacent Potaj Čuka Tisnica exploration license. This phase of work culminated in a Preliminary Economic Assessment (PEA) for the Timok Gold Project being disclosed in May 2014, based on MREs on the Bigar Hill, Korkan and Kraku Pester prospects and subsequently updated in 2019.

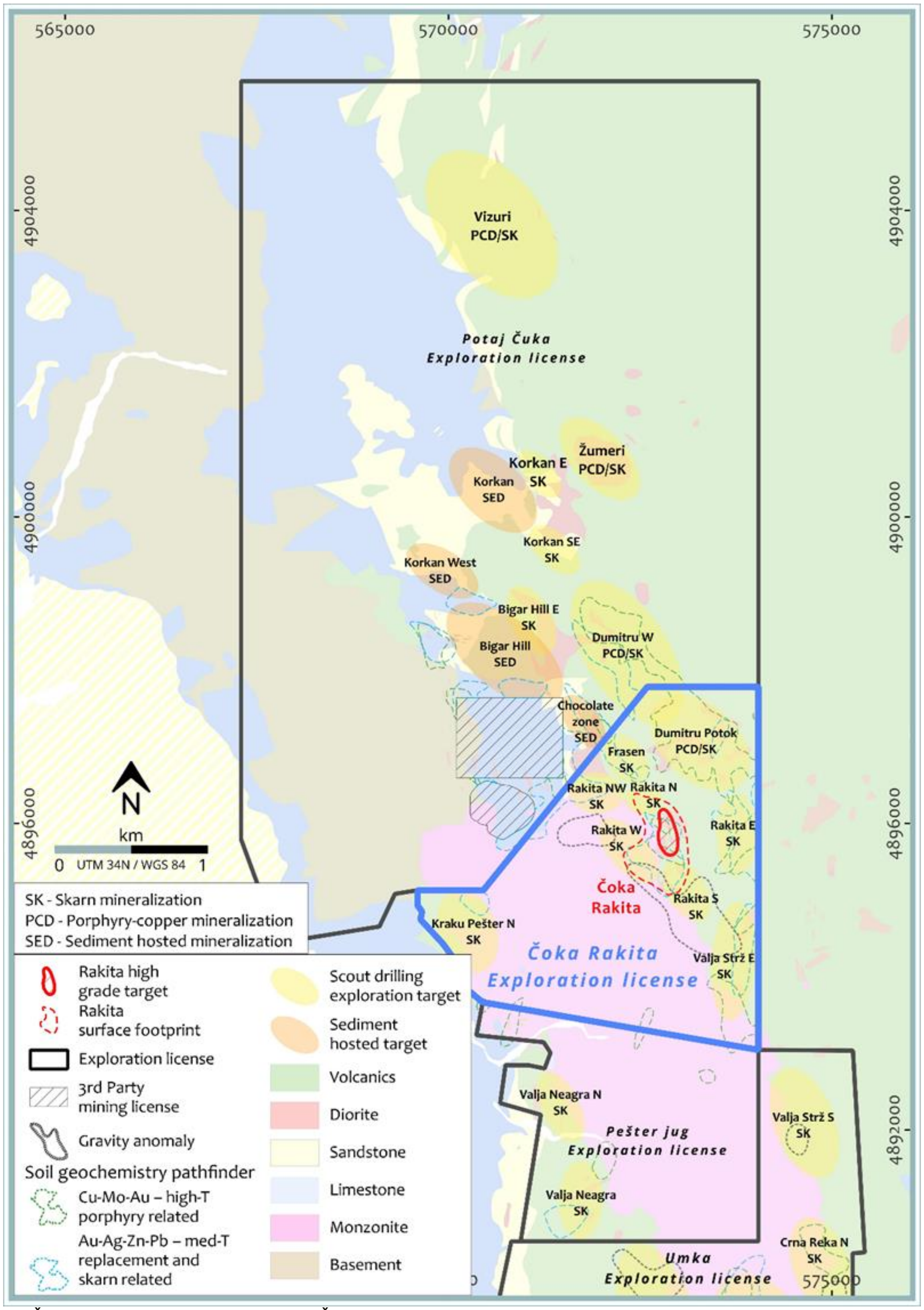
When exploration efforts were resumed, a new phase of drilling in 2016 was completed. A single deeper drillhole, that aimed to understand the potential for porphyry copper-gold mineralisation and search for potential skarn formation beneath the known extents of epiclastic-hosted gold mineralisation was undertaken as part of this program. Drillhole RADD010 intercepted intensely skarn-altered sandstones and returned assay results with an interval of 21 m at 2.61 g/t Au from 514 m downhole. Although this drillhole returned encouraging results, the potential for skarn-hosted gold within the TMC remained poorly understood and no follow up drilling was performed.

In 2020, a camp scale re-evaluation of the exploration potential lead to the resumption of drilling at Čoka Rakita to better understand and evaluate the potential for deeper, skarn-hosted gold mineralisation and follow up on the earlier RADD010 results. DPM intercepted gold-rich skarn in 2020 within drillhole RADD013 which intercepted 36 m at 4.41 g/t Au and confirmed the potential for a sizable deposit of skarn-hosted gold mineralisation at Čoka Rakita.

In late 2022, DPM embarked on an intensive drilling program to evaluate the mineral resource potential. The formal gold discovery at Čoka Rakita was announced by DPM in a news release dated 16 January 2023.

There are numerous exploration targets located on the Čoka Rakita exploration license. The location of the exploration targets is shown in Figure 9.1.

Figure 9.1 – Overview Map of the Čoka Rakita Exploration License



Čoka Rakita license in blue, Potaj Čuka and Pešter Jug Licenses outlined in black with exploration targets shown on surface geology.
Source: DPM, 2024

9.2 Geological Mapping

Outcrop exposure over the exploration licenses is generally poor. However, in areas with outcrop, ground geological mapping together with rock sampling was undertaken. All existing surface outcrops have been mapped, including those created by earthworks activities associated with drill pad construction and cuttings for access roads. Geological maps were created using available observed lithology, alteration, and structure data, followed by interpretation.

9.3 Soil Geochemistry

Soil sampling has proven to be a very effective exploration method for localising potential epithermal, skarn and porphyry type mineralisation. Gold, as well as low-temperature pathfinder elements such as arsenic, mercury, and thallium, have been found to be important elements in soil geochemistry surveys. An overview map of the gold-in-soils results is shown in Figure 9.2.

Follow-up or detailed sample grids were configured at a line spacing of 100 m, with 50 m samples collected along each line. The sampling approach was based on orientation surveys completed by the Issuer in a similar environment from the Eastern Rhodope Mountains of Bulgaria. Soil field duplicates were collected at frequency of one in 20. Soil samples were collected by field staff and transported to the core storage facility in Bor on the same day they were sampled.

As of November 2023, 2,592 soil samples have been collected over the current Čoka Rakita license.

The results of all soil sampling to date have highlighted a near-continuous 20 km-long combined gold-arsenic-antimony-mercury-thallium anomaly.

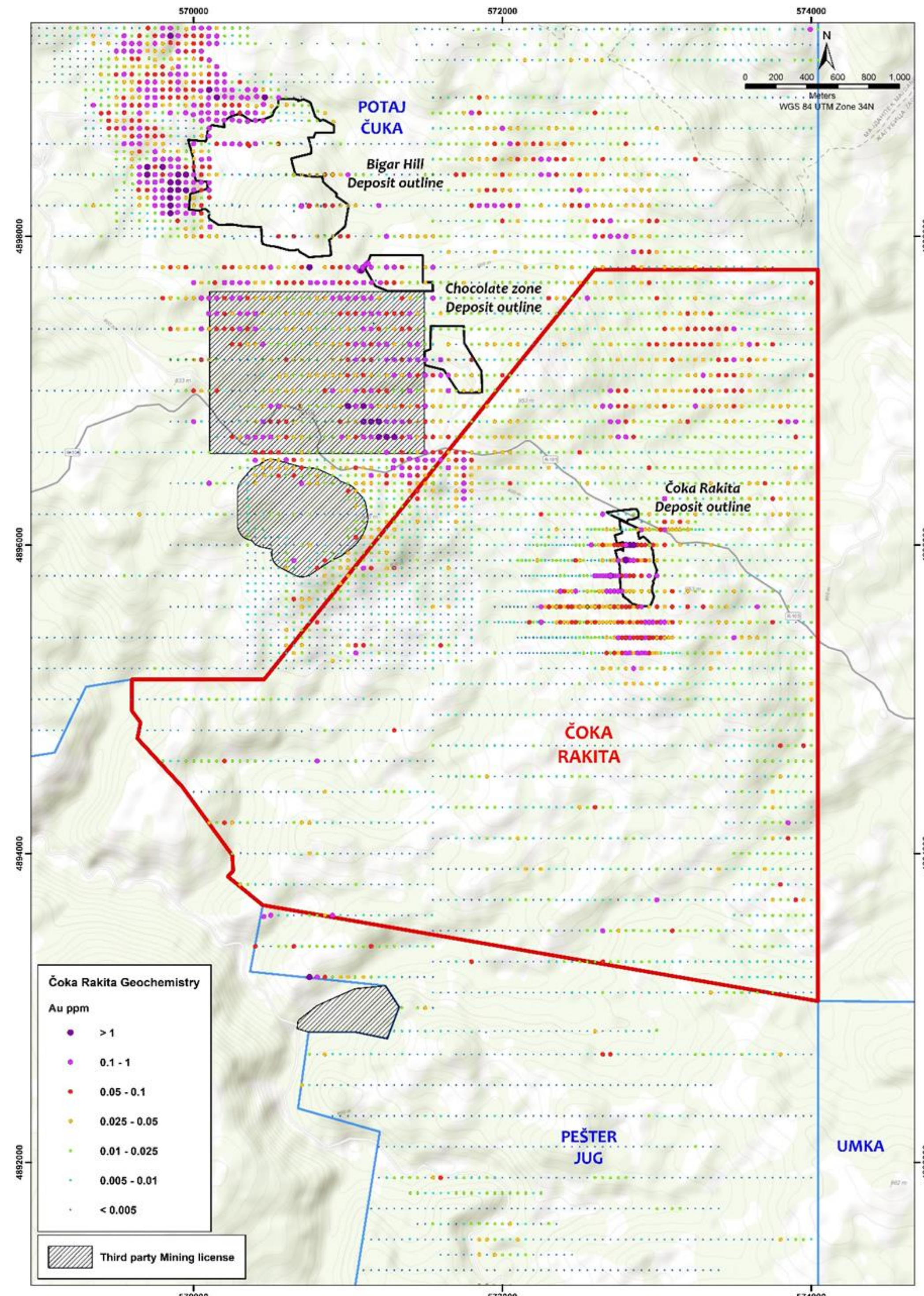
Within the Čoka Rakita exploration license, gold-in-soil anomalies to the northeast of the license are associated with porphyry mineralisation that broadly follows the trace of the Dimitru Potok porphyry system. Anomalous gold in soil results located above the Čoka Rakita deposit are related to epiclastic breccia hosted vein type mineralisation that is located above the skarn mineralisation.

9.4 Trenching and Channel Sampling

Trenching was used as early as 2007 as a follow-up strategy to explore areas with anomalous soil geochemistry and to assist in defining key geological relationships due to the limited outcrop in the Project area. There was a high success rate in intersecting gold mineralisation by drilling near extensive and well-mineralised trench intercepts.

Channel samples were routinely taken on road cuttings or where outcrop existed. Channels were typically cut using a hammer and chisel, which allows sufficient penetration to excavate a channel approximately 100 mm high and 30 mm deep. Samples were caught into a chip tray which is cleaned at the end of every interval.

Figure 9.2 – Gold Assay Results from Soil Sampling Activities on the Čoka Rakita and Surrounding Licenses



Čoka Rakita exploration license shown in bold.
Source: DPM, 2024

Trenches were completed under the supervision of DPM exploration geologists. The dimensions of the trench are set out according to safety regulations, with a maximum depth of 1.5 m and a minimum width of 0.8 m. During excavation, the upper humus layer is separated from the underlying soil material so that it can be replaced and revegetated during rehabilitation.

Trenches were sampled as channels, with channel samples collected just above the trench floor at either 1 m or 2 m intervals. Except where extensive soil cover is encountered, trenches are sampled in their entirety. The samples were routinely weighed prior to final bagging to maintain an even sample size and to avoid sampling bias in harder rock types. An average channel sample weight of 3 kg/m was maintained. Field duplicate samples and certified standards were taken at a frequency of 1:20. All data collected in the field was routinely entered into geology and structural geology spreadsheets using Field Marshal software and later exported to an acQuire database.

Both channel and trench samples were collected by DPM field staff and transported to the core storage facility in Bor on the same day they were sampled.

As of November 2023, 5,163 m of surface trenching and an additional 622 m of surface channel sampling has been undertaken on the Project.

9.5 Geophysics

In 2006, DPM initiated a heliborne VTEM geophysical survey on the Čoka Rakita license as part of larger survey over all the licenses held by DPM on the Timok Belt. The electromagnetic response and magnetic signal (Total Magnetic Intensity) were recorded during the survey. The airborne survey was flown along traverses oriented at an azimuth of 080° and a nominal line spacing of 100 m with significant portion of infill at a 50 m line spacing. The objective of VTEM survey was to identify conductive targets in the first couple of hundred metres below surface which could be caused by high sulphidation and possible porphyry styles of mineralisation.

The outputs of the survey have been used extensivity to determine the lithological and structural architecture of the license area. The results of this study identified that unaltered intrusives (monzonite batholith), dykes and volcanic epiclastic units appear as the most magnetic units in the Project area. The basal breccias, which are an erosional product that lies between the Jurassic limestones and S1 sandstone can be magnetic, particularly if volcanic clasts are present. The core of the Dimitru Potok porphyry systems, located to the east of the Čoka Rakita skarn deposit, appears as cluster of small bodies with moderate magnetic intensity.

Induced polarisation surveys have been used since the commencement of exploration works at Timok using profiling arrays (dipole-dipole) with variable dipole spacing, depending on the target in question. A significant portion of the area of the license was surveyed in 2007 with a large dipole (200 m), to target blind porphyry systems. Subsequently during 2018, smaller (50 m or 25 m) dipoles were employed with the aim to achieve better resolution at shallow levels, targeted around areas related to sediment hosted gold mineralisation. Based on this survey, the near surface lithologies, that sit stratigraphically above the Čoka Rakita Skarn deposit, appear as a positive anomaly of chargeability and moderate to low resistivity.

In 2022, detailed gravity surveys, using an approximately 250 m spaced random grid, identified a distinct gravity low over the Dimitru Potok porphyry system. The Čoka Rakita deposit is located in between the gravity low and on the eastern margin of positive gravity anomaly. Second rank positive gravity anomalies occur between the major minimum-maximum domains, some spatially very close to the deposit.

A ground radiometric survey identified a distinct K/Th anomaly at surface along northeast elongated lineament bounding Čoka Rakita mineralisation towards the south. A few anomalous domains corresponding to elevated positive geochemical signals at surface were selected for further follow-up.

In 2023, a magnetotelluric survey was undertaken over the area between Čoka Rakita and the Dimitru Potok porphyry. Numerous conductive targets were identified and selected anomalies that may represent deep manto or skarn type mineralisation, will be tested during later drilling campaigns.

A map of the coverage of all geophysical works completed on the Čoka Rakita license areas is shown in Figure 9.3.

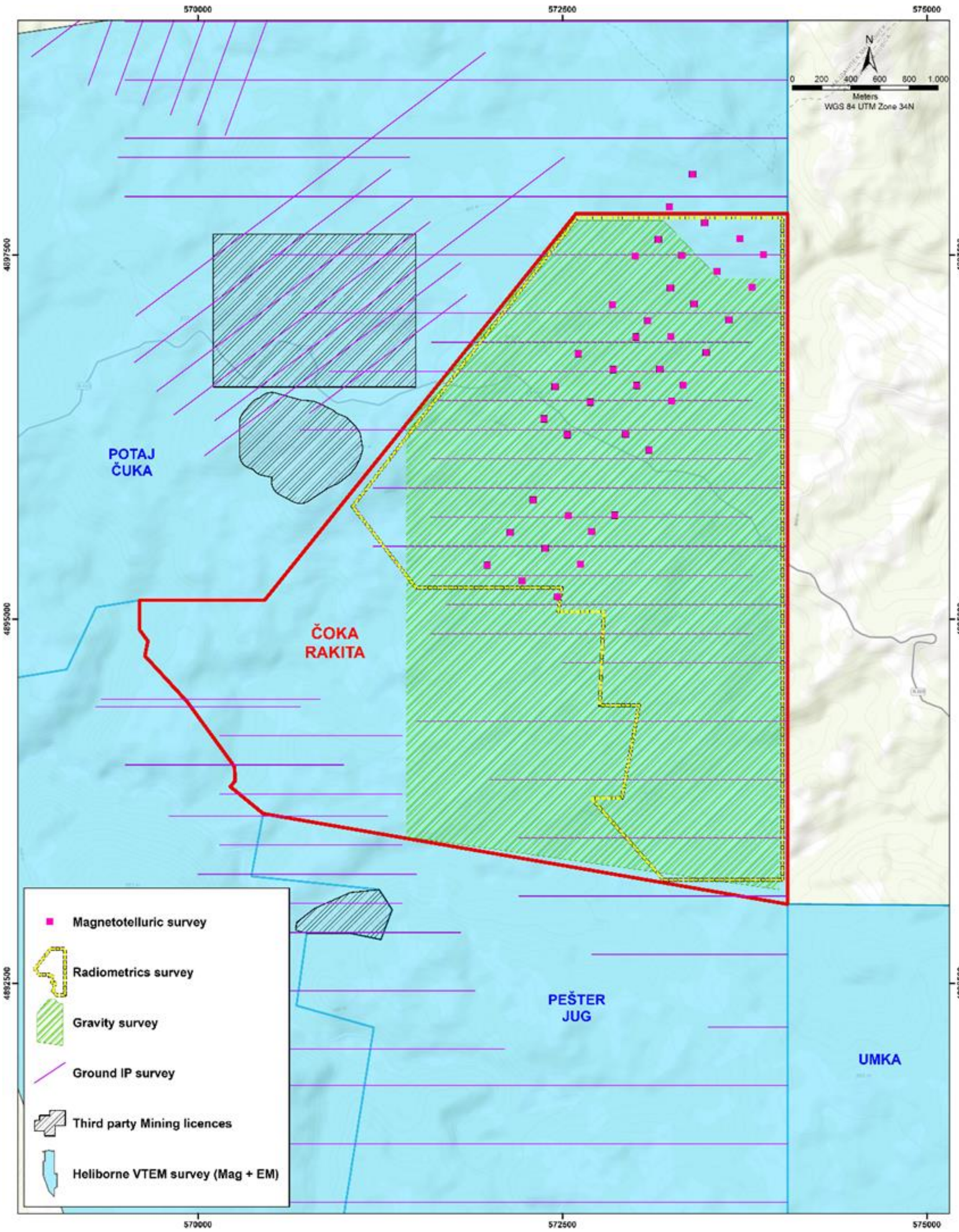
9.6 Topographic Surveys

All survey activities have been conducted using a licensed third-party surveyor. A base geodesic operational network within the Project area has been established that covers the entire exploration areas. This primary survey control network was implemented using AUSPOS, an online global positioning system (GPS) processing service provided by Geoscience Australia.

A high-resolution topographic survey, which covered two DPM exploration licenses (Umka and Čoka Rakita), was finalised in April 2022. However, the survey did not encompass the full extent of the Čoka Rakita license. The total area surveyed was approximately 51.53 km². The Universal Transverse Mercator (UTM) coordinate system was used for recording all coordinates, specifically Zone 34 North in World Geodetic System (WGS) 84 datum.

Drone topographic mapping was carried out by a licensed third-party surveyor, but all data processing was handled internally by DPM staff. The survey was conducted using a Wingtra unmanned aerial vehicle. Detailed Orthophoto mosaic was not created, but a Digital Terrain Model (DTM) with a resolution of 80*160 cm was generated for the entire area. This survey has been used to provide better resolution for more precise terrain corrections for gravity survey.

Figure 9.3 – Plan View of Geophysical Works Completed on the Project



Source: DPM, 2024

A detailed Digital Elevation Model (DEM) has been calculated in-house by DPM's engineers using Agisoft Metashape Professional v1.6.3. Filtering was applied with the aim of removing vegetation using "Cloth Simulation Filter" (Cloud Compare v2.11.3). Final resolution after filtering is at 2.0 m grid cell size.

10 DRILLING

10.1 Drilling Summary

DPM has employed a combination of diamond drilling and reverse circulation (RC) drilling approaches at Čoka Rakita. Drilling was carried out by various Serbian drilling contractors using Atlas Copco CS-14 and Atlas Copco Mustang 9/13/18, Alton HD, Coretech YDX 1300G / YDX-3L, Epicor CT20, Sandvik DE710 / DE712, HANJIN HYDX, UDR 200D and Gemex MP 1200 rigs for diamond drilling, and GEMSA 500RC rigs for RC drilling. Examples of drilling activities are shown in Figure 10.1.

Figure 10.1 – Diamond Drill Rig (Left) and a RC Drill Rig (Right) at Čoka Rakita

Source: DPM, 2023

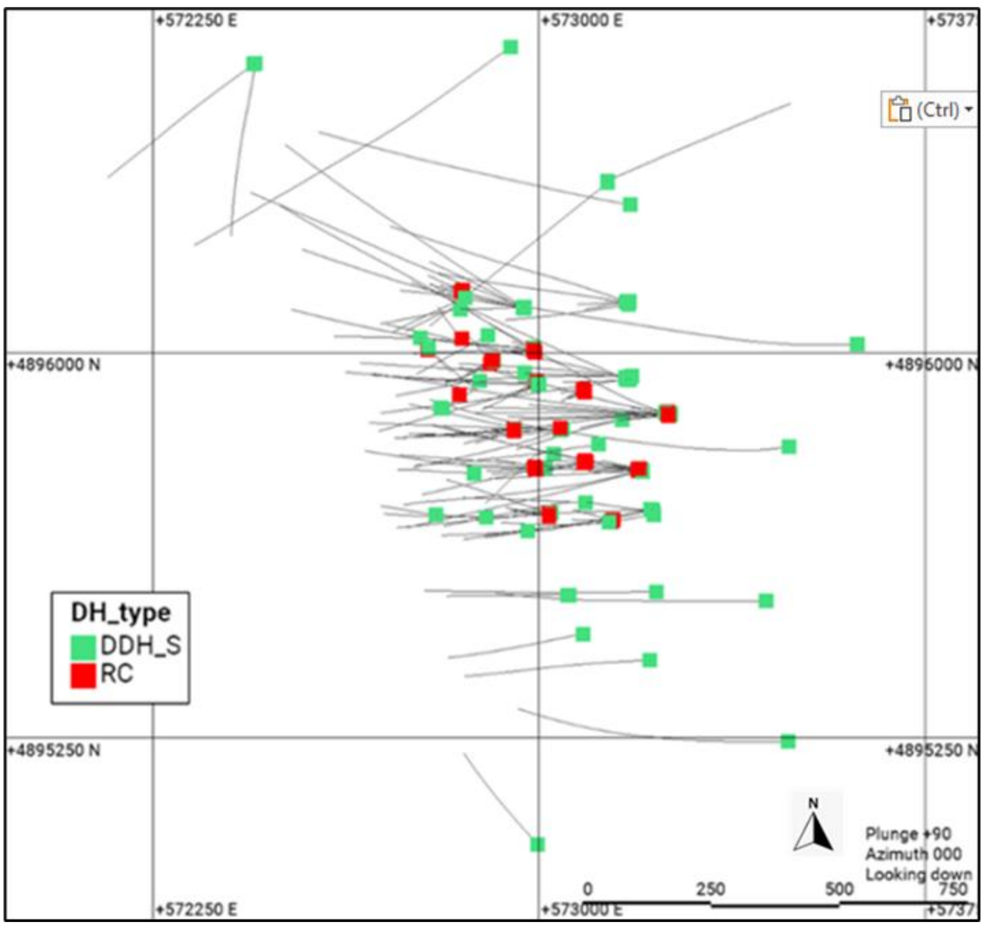
Drilling operations are summarised by area and year completed in Table 10.1. All the drilling activities on the Project have been completed during the tenure of DPM. Figures 10.2 and 10.3 present the drillholes completed at each deposit.

Table 10.1 – Summary of Drilling by Type and Year at Čoka Rakita, up to November 2023

Company	Year	Diamond		RC		Diamond Tail		Metallurgical	
		Number	Metres	Number	Metres	Number	Metres	Number	Metres
DPM	2008	-	-	4	474	-	-	-	-
	2009	9	1,319	-	-	-	-	-	-
	2016	1	588	-	-	-	-	-	-
	2017	1	225	-	-	-	-	-	-
	2020	3	2,298	-	-	-	-	-	-
	2021	25	16,030	-	-	-	-	2	1,160
	2023	60	37,678	44	7,483	24	13,469	-	-
Total		**99**	**58,138**	**48**	**7,957**	**24**	**13,469**	**2**	**1,160**

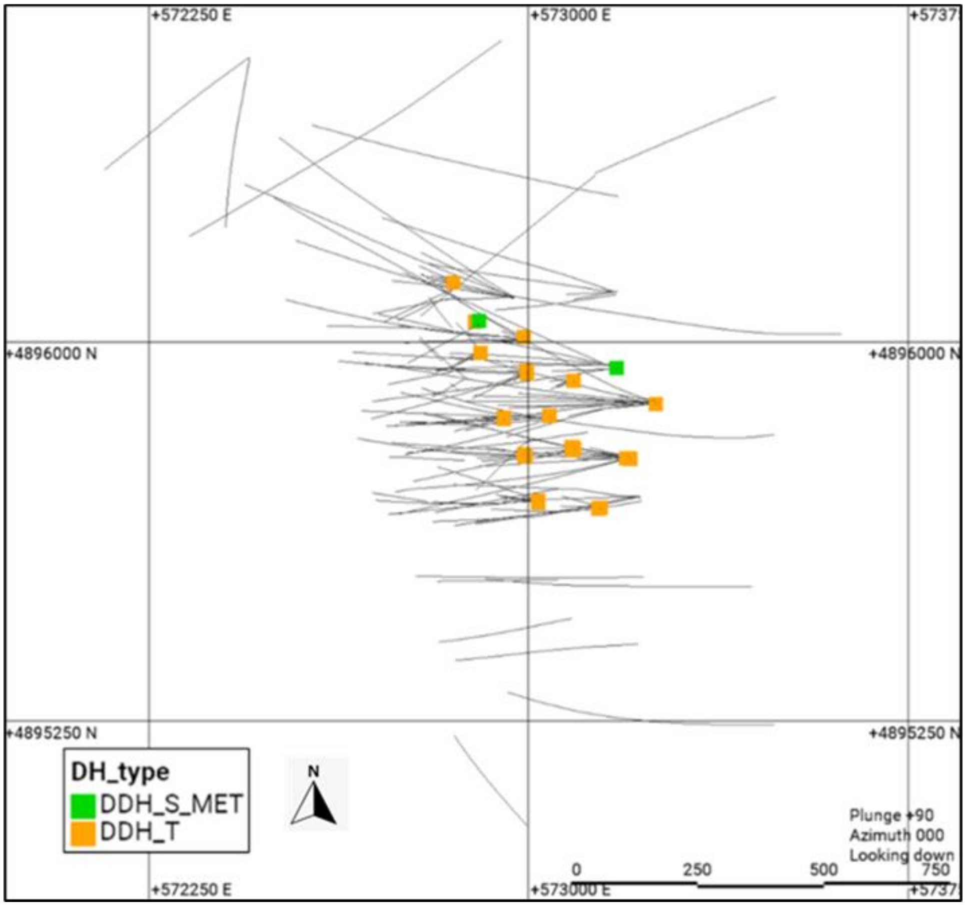
Source: DPM, 2023

Figure 10.2 – Plan Map of Diamond and RC Drillholes Completed on the Čoka Rakita Project



Source: DPM, 2023

Figure 10.3 – Plan Map of Diamond Metallurgical and Diamond Tail Drillholes Completed on the Čoka Rakita Project



Source: DPM, 2023

10.2 Collar Surveying

Drillhole collar surveying was undertaken using either Total Station (when drilling in forested area) or differential GPS by a contracted surveyor. Once approved by the Project Geologist, the collar data was then imported into the DPM acQuire database.

10.3 Downhole Surveying

Up until 2017, a Devi Tool digital multi-shot camera was used for downhole surveying of diamond holes and RC holes. Diamond drillhole downhole surveys were carried out by drilling contractors at 30 m intervals.

Drilling from 2020 onward has a Devico gyro tool to provide both single and multi-shot surveys of both RC and diamond drillholes. Multi-shot surveys provide measurements every 3 m downhole. The gyro tool is re calibrated weekly and serviced every six months by the vendor.

For diamond drillholes, the procedure requires the final survey to take priority over previous surveys once the drillhole is complete.

For RC drilling, survey readings were found to encompass excess levels of variability during early stages of RC pre-collar drilling. This was thought to be because of the larger hole diameter effecting the way the tool travels downhole. Due to these initial observations, the current procedure for RC pre-collar holes is to use a combined average of all measurements to generate final downhole survey volumes.

10.4 Drilling Orientation

The majority of the drillholes are designed to test for stratigraphic-hosted mineralisation and are designed with a roughly westerly azimuth and a -60° dip. Drilling is perpendicular to the orientation of target lithologies, to best intercept the true thickness of the skarn mineralisation.

During the 2023 drilling campaign, a small amount of navigational drilling was conducted to correct the azimuth of a series of drillholes. The use of navigational drilling enabled multiple target intersections by drilling branch holes off a “parent hole” and then navigating the hole to reach a target in three-dimensional (3D) space.

10.5 Diamond Drilling

10.5.1 Drilling Procedures

DPM staff and drilling contractors followed a comprehensive set of drilling QA/QC and safety procedures for all diamond core drilling programs. Diamond drilling begins with the use of a PQ diameter core barrel (85 mm core diameter) and then reduced to HQ triple tube (HQ3) core barrel (61.1 mm core diameter) once competent rock is intersected. The diamond drill core size was maintained at HQ3 for as long as possible. NQ2 core barrel (50.6 mm core diameter) was used to extend diamond holes to reach deeper targets.

Core was transferred directly from the core barrel into appropriately labelled aluminium core boxes to ensure that core was correctly placed, and no core was lost. Wooden core blocks were placed between runs, recording the length of the run and any core loss. Forced breaks made by the drillers were marked on the core with a red cross on both sides of the breaks. At the drill site, core was washed clean of surface mud or other drilling fluids. All core boxes were labelled with the drillhole number, starting and ending depths for the core box, and box number.

Drill core orientation procedures were carried out at approximately 3 m intervals, and less in mineralised zones or areas of poor ground conditions. EzyMark, or occasionally spear-orientation equipment was used to mark the orientation of drill core.

Core boxes were collected by DPM staff at least once a day from the drilling rigs and transported to the DPM core storage facility in Bor on the same day. For transportation, core box lids were fitted by adhesive-coated fastening tape, and boxes were firmly secured with strapping in the transport vehicle.

10.5.2 Recovery

Diamond drilling core recovery averages, excluding those intervals where navigational drilling was undertaken, is 98.45% for all rock types. The majority of drill core was HQ3 size, followed by PQ3 and a small proportion of NQ. Specialised drilling muds and polymers were used throughout the program to maximise core recovery, and in areas of poor core recovery, drill runs were reduced to less than 0.5 m.

Where navigational drilling was employed to steer a drillhole toward a target, no core was recovered. These intervals were completed within un-mineralised intervals of overlying epiclastic breccias. No navigational drilling was conducted in lithologies where gold mineralisation was expected.

10.6 Logging and Sampling

At the DPM core facility, all core is photographed dry and wet using a digital camera before logging commences. Core photos record the drillhole number, box number, starting and ending depths, and date. Photo sets are integrated with the acQuire drillhole database.

Logging procedures are initiated with geotechnical logging, during which rock quality designation (RQD), joint strength and roughness, rock strength classification, and detailed core recovery are recorded. Core with drilling orientation marks is aligned with adjacent core intervals so that an orientation line can be drawn consistently over most of the drill core.

Geological structures are measured based on alpha, beta, and gamma angles relative to the orientation line. True orientations of features are determined using either a jig or by calculation. Geological logging is recorded using a digital logging form that provides an extensive geological description through a system of codes for lithology, alteration, veins, mineralisation, weathering, and vein descriptors.

After core logging has been completed, core is marked up for sampling at regular 1.0 m intervals corresponding to drilled depths. The 1.0 m sample intervals may be adjusted at key geological contacts or in sample intervals with significant core loss. These intervals must be less than 1.5 m and greater than 0.5 m long. Core is split approximately 1 cm from the orientation lines using a diamond saw. Half the core is placed in a heavy cotton sample bag, together with a sample tag. Core samples weigh (on average) 3 to 4 kg. The remaining split core is replaced in the core box and retained at DPM's core shed facilities in Bor.

10.7 Reverse Circulation Drilling

10.7.1 Drilling Procedures

DPM staff and drilling contractors followed a comprehensive set of drilling quality control and safety procedures for all RC drilling programs. RC drilling was conducted under constant on-site supervision by the rig geologist.

RC drilling was completed using downhole hammers with face sampling drill bits. All drilling and sampling were confined to dry downhole conditions. All collars were lined with a 6 m casing of polyvinyl chloride (PVC) pipe.

To ensure sampling was under dry conditions, and to enhance sample recovery, two 35 m^3 per minute compressors and booster were used at each drill site. Pressurised air blowbacks were routinely used after every metre of advance so that all the material within the drill stem was displaced into the sample bag prior to advancing to the next metre. At every rod change, compressed air blowdowns were used for cleaning the air system and for conditioning the hole before drilling resumed.

If drilling could not be continued under dry conditions, the RC drillhole was abandoned and re-entered using a diamond core drill to advance the hole.

A dedicated compressed airline from the rig compressor was always available for cleaning the cyclone and the sample splitter. All RC sample splits were collected daily by DPM staff from the drill rigs and transported to a secure core shed facility in Bor where they were maintained under 24-hour security. Upon arrival at the core shed, all RC samples were measured for magnetic susceptibility, using a handheld meter. A small sample split was washed, and the chips kept in a chip tray for logging and reference.

The RC drilling at Čoka Rakita completed during 2008, did not reach the required depth to intercept gold-bearing skarn mineralisation and as such, has not been used for grade and resource estimation purposes, however logging data has been used to inform the geological model.

10.7.2 Logging and Sampling

RC drilling samples have been routinely collected at 1 m intervals. Drill cuttings for each drilled metre are collected in a new plastic bag and marked with the drillhole number and interval sampled. Each bag of cuttings is weighed at the drill site using electronic scales. Cutting weights are recorded using handheld data loggers for input into the acQuire database and are monitored in real time during drilling for consistency using expected weights based on drill rods, bit sizes and shroud sizes being used and rock types. Changes in the weight of cuttings are also monitored by evaluating the statistical variations of cutting weights for each drillhole.

Routine sampling procedures require that the cyclone be cleaned at each rod change and after a wet sample. Drill cuttings are split using a Jones three-tier riffle splitter to provide a sample that will be submitted to a laboratory for analysis. The riffle splitter is cleaned with compressed air and bottle brushes after each sample is split.

10.7.3 RC Pre-Collar Drilling

During 2023, approximately 44 RC pre-collar drillholes were undertaken within the first 200 m of the overlying cover sequence that lies on top of the Čoka Rakita, gold-rich skarn prospect, and subsequently completed using a diamond tail to the reach the target depth. The length of RC pre-collar ranged from 85 m to 241 m but averaged 170 m in length.

Not all RC pre-collars were continued using diamond drilling. Such holes were abandoned mostly due to excessive hole deviation or occasionally due to poor ground conditions downhole.

No samples were collected from RC pre-collar drilling. RC pre-collaring was undertaken in lithologies where no mineralisation could be expected, and no analytical results were deemed necessary.

10.7.4 Recovery

RC recovery was calculated by dividing actual sample weight (split + reject) by the theoretical sample weight. The theoretical average sample recovery from RC drilling during the 2008 campaign was 83.3%, for all rock types, based on an estimated theoretical weight of 32.7 kg. No recovery value has been determined for the pre-collar holes completed in 2023.

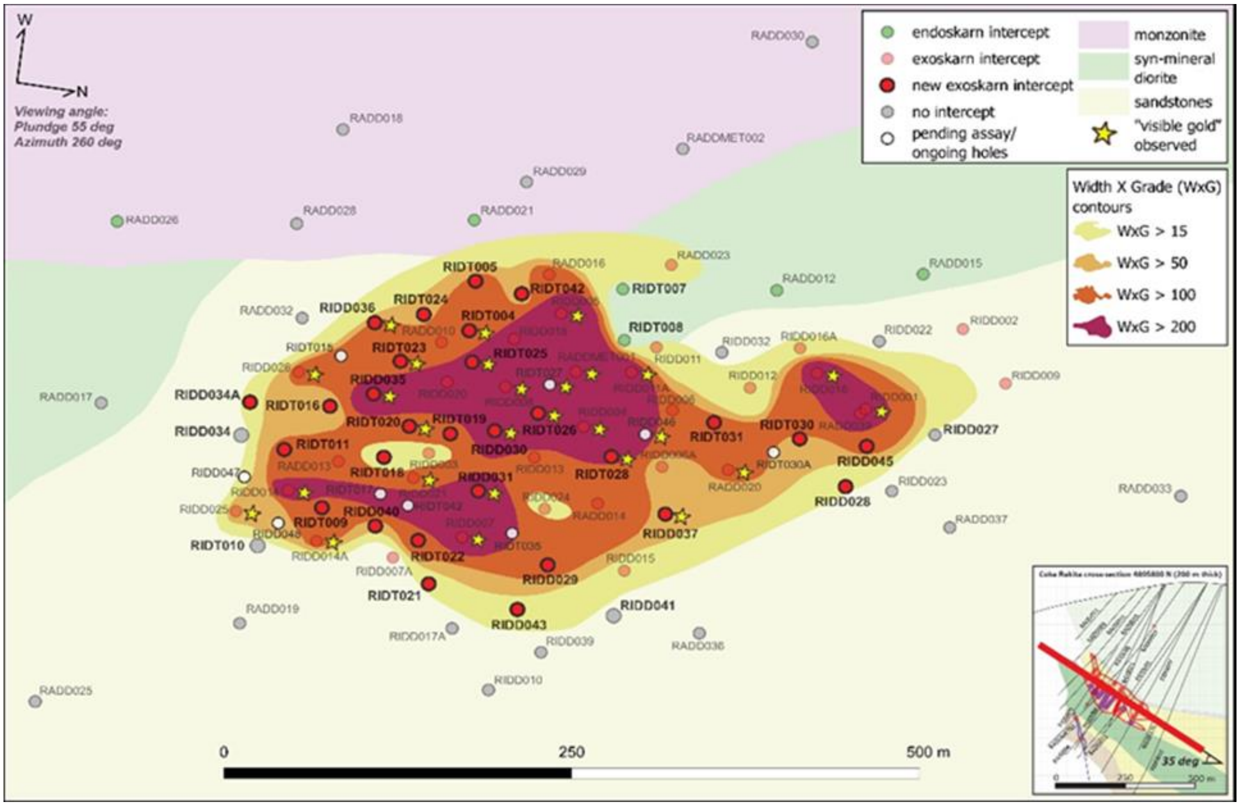
10.8 Metallurgical Drillholes

Since 2020, two (2) diamond drillholes (HQ core diameter) have been completed for the purposes of metallurgical sample collection. Both holes are twins of original diamond holes, with a step out distance of between 15 m and 30 m.

10.9 Drilling Results

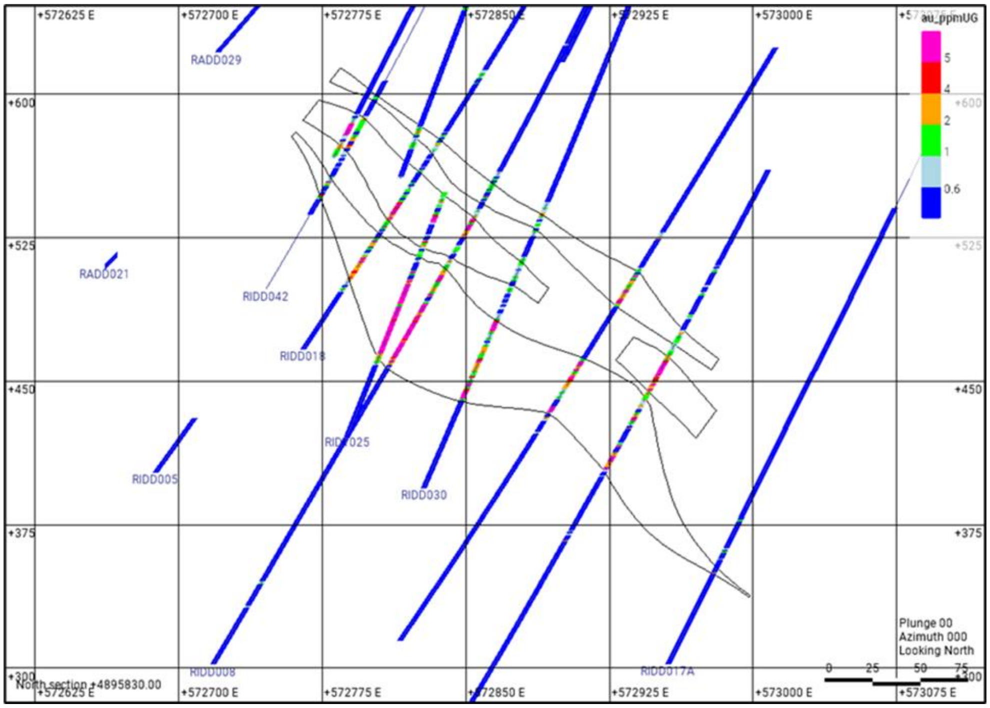
Drilling results received as of 16 November 2023, defined a wide zone of skarn-hosted gold mineralisation over a footprint of 650 m long, up to 350 m wide, and with variable thickness from less than 20 m in the margins to more than 100 m in the core of the mineralised zone. A tilted slice through the deposit is shown in Figure 10.4, which shows a high-grade core of mineralisation that can be delineated by the width multiplied by grade contour greater than 200. Representative drill sections through the deposit are shown in Figure 10.5 and Figure 10.6 .

Figure 10.4 – Tilted Slice Along High-Grade Skarn Mineralisation Displaying Drilling Intercepts and Ongoing Infill Drilling at Čoka Rakita



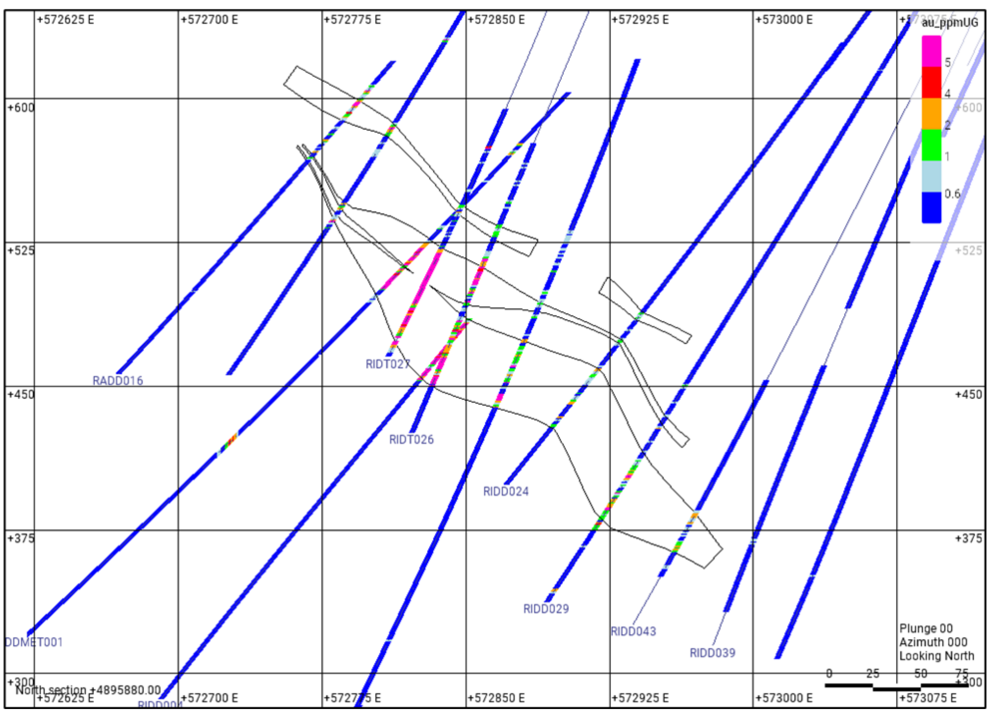
Source: DPM, 2023

Figure 10.5 – Cross Section 4895830 mE Showing Drilling and Interpreted Mineralisation



Source: DPM, 2023

Figure 10.6 – Cross Section 4895880 mE Showing Drilling and Interpreted Mineralisation



Source: DPM, 2023

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Introduction

The QP reviewed the policies and procedures for sample methods, analyses, and transportation, as supplied by DPM and they were found to be in line with the latest CIM exploration best practice guidelines and industry best practice.

The QP is satisfied that the relevant procedures have been followed consistently, all laboratories used for analyses are adequately certified, and do not have any undue relationships with DPM, and that the standards used as part of the QA/QC routine adequately reflect the characteristics of the mineralisation.

The QP also supervised the production and review of the QA/QC reports to verify the accuracy and precision of the assayed QA/QC material and samples.

11.2 Sampling Techniques

DPM has collected different types of samples including density, soil and trench samples and samples from RC and diamond core drilling. Sampling techniques appear to have been consistent throughout the Project's exploration history.

11.2.1 Dry Bulk Density Measurements

Bulk density measurements were restricted to diamond core only. Half-core samples of 20 to 30 cm were collected approximately every 3 m.

11.2.2 Soil, Trench and Channel Samples

Soil field duplicates were collected at a frequency of 1 in 20. Blanks and low-level gold CRMs were inserted at the same frequency.

Trench and channel samples were routinely weighed prior to final bagging to maintain an even sample size and to avoid sampling bias in harder rock types. An average channel sample weight of 3 kg/m was maintained. Field duplicate rock samples were taken as a second sample (normally 5 to 10 cm below) during trenching and channelling on a 1:20 basis. CRMs were inserted at a frequency of 1 in 10. Since the first quarter 2017, blanks were similarly inserted at a frequency of 1 in 20 samples.

11.2.3 RC Hole Samples

RC field duplicates, pulp duplicates, and certified standard reference material are submitted to the laboratory at a frequency of 1 in 20 samples. DPM used a non-certified coarse blank (BLANK_BOR), composed of unmineralised quartz from a local quarry.

11.2.4 Diamond Drill Core Hole Samples

Core field duplicates are prepared by producing split samples after the jaw crushing stage of sample preparation, with each split being assigned a unique sample number. Pulp duplicates and certified standard reference material are submitted into the assay sequence at a frequency of 1 in 20 samples. Blank samples of unmineralised quartz sand were submitted at one in every batch submitted to the analytical laboratory at the beginning of the batch sample sequence. The procedure was updated in 2017, wherein coarse blanks (rocks) are now used instead of sand, and blanks are now inserted at a 1 in 20 frequency.

11.3 Laboratory Sample Preparation and Analyses

Table 11.1 lists several independent laboratories that were contracted by DPM (and Avala prior) to complete analytical tests on rock, chip and core samples collected during exploration and drilling programs at Čoka Rakita. All of these analytical laboratories are ISO-certified and fully independent of DPM.

Table 11.1 – Laboratories Used to Complete Analytical Works on Samples Taken from the Čoka Rakita License

Name and Location	Dates (Primary Assaying)	ISO Certification	Testwork Performed
SGS, Chelopech, Bulgaria	2008 to 2010	ISO9001:2015	Gold, silver, sulphur and base metal analysis of trench, channel, RC and diamond core samples.
Genalysis/Intertek, Perth, Australia	2007 to 2008	ISO17025	Gold, silver, sulphur and base metal analysis of trench, channel, RC and diamond core samples.
SGS, Bor, Serbia	2010 until present	None	Crushing and pulverising of soil, trench, channel, RC and diamond core samples. Density determination. Gold, silver, sulphur and multi-element analysis of trench, channel, RC and diamond core samples.
ALS Rosia, Montana, Romania	2007, 2020 to 2023	ISO9001:2008 and ISO/IEC 17025:2017	Gold, silver and sulphur analysis. Gold and multi-element analysis of soil and stream sediment samples in 2007 and 2019. Metallic screen fire assaying of gold.
ALS Chemex, Bor, Serbia	2020 to 2023	ISO9001:2008 and ISO/IEC 17025:2005	Crushing and pulverising of soil, trench, channel, RC and diamond core samples.
ALS Chemex, Vancouver, Canada	2007 to 2009	ISO9001:2000 and ISO:17025	Gold, silver, sulphur and base metal analysis of trench, channel, RC and diamond core samples.

Source: DPM 2024

11.3.1 Laboratory Sample Preparation

All submissions to the sample preparation facility are accompanied by sample submission forms with instructions for preparation methods, insertion-of-standards protocols, and analytical process codes. Once the samples are delivered to the SGS sample preparation facility, chain of custody records are maintained until reject sample pulps are returned to DPM’s jurisdiction. The SGS Bor preparation facility is owned by Avala and independently managed by SGS with the chain of custody transferred from Avala/DPM at the laboratory door.

All samples submitted to the facility are initially dried at 105°C for a minimum of 12 hours. Core, trench, and rock samples are then crushed to 4 mm, using jaw crushers. Crushing is checked by confirming that 85% of the crushed material can pass through a 4 mm sieve. Core field duplicates are produced by riffle splitting crushed samples on a 1 in 20 basis. Each field duplicate is assigned its own identification number for the remainder of the assay procedure. All crushed sample material is then pulverised using LM5 pulverising mills (of which there is currently a bank of eight).

RC drilling samples are pulverised in their entirety using the LM5 pulverising mills. A standard part of the SGS operating procedures is for 1 in 10 pulps to be wet sieved using a motorised sieve bank to confirm that the sample passes a P90 of 75 µm. If a sample fails the test, the previous 10 samples are re-pulverised.

Pulverised material from all sample types is split into 250 g and 600 g pulps, where the former is used for assay determination, and the latter is stored as part of the reference pulp library. An additional 250 g pulp duplicate is split from the pulverised material at a frequency of 1 in 13.

11.3.2 Laboratory Analyses

Routine analysis of samples is currently performed at the SGS analytical laboratory in Bor, or during earlier phases of exploration at the SGS analytical laboratory in Chelopech. All laboratory methods, procedures, and QA/QC protocols are consistent with standards adopted by SGS worldwide standards and are ISO certified.

Gold analysis methodology is conventional 50 g fire assay (FA), with an atomic absorption finish. Silver and base metal analyses (copper, molybdenum, arsenic, bismuth, lead, antimony, and zinc) are performed using a 0.3 g charge, aqua regia digestion, and atomic absorption analysis. Sulphur samples are analysed by combustion with an infrared finish.

The procedures routinely used at both the SGS laboratories include the following established and standard specifications used at all SGS laboratories worldwide:

- Cross-referencing of sample identifiers.
- Use of compressed air gun and vacuum gun, along with routine barren quartz “washes”, for cleaning of crushing and pulverising equipment.
- Routine assaying of quartz washes.
- Assaying of SGS-submitted certified standards at a rate of two per batch of 40 original samples.
- A minimum of 10% of submitted samples are subject to repeat analysis.

Second splits generated by the SGS CCLAS (Comlabs Computerised Laboratory Automation) system are produced at a rate of 1 in 13 and represent a second subsample taken from the LM5 pulverised pulp.

Soil samples were assayed by ALS Chemex Perth, using methods Au-TL43 (gold by aqua regia digestion with inductively coupled plasma-mass spectrometry – ICP-MS) and ME-MS41 (combined ICP-MS and inductively coupled plasma-atomic emission spectrometry (ICP-AES) dependent on concentration) for multi-elements. More recently, the same analytical methods have been undertaken at ALS Rosia Montana. Elements assayed for are silver, aluminium, arsenic, boron, barium, beryllium, bismuth, calcium, cadmium, cerium, cobalt, chromium, caesium, copper, iron, gallium, germanium, hafnium, mercury, indium, potassium, lanthanum, lithium, magnesium, manganese, molybdenum, sodium, niobium, nickel, phosphorous, lead, rubidium, rhenium, sulphur, antimony, scandium, selenium, tin, strontium, tantalum, terbium, thorium, titanium, thallium, uranium, vanadium, tungsten, yttrium, zinc and zirconium.

An ICP-MS machine has been in use at the SGS Bor laboratory since 2012, where core and RC samples are analysed for 49 elements. In 2021, the existing ICP-MS machine was upgraded with a newer version.

Pulp aliquots for dispatch to other laboratories (abroad) were packed in boxes which were plastic-wrapped or taped-shut for transport in sealed containers. The sealed sample boxes, accompanied by chain-of-custody documents, were transported door-to-door by an international courier delivery company. Returned reject pulps are stored in the pulp library.

11.3.3 Screen Fire Analysis

Mineralogical studies and systematic core logging indicate that gold is present in its native form and its grains can measure a few tens to more than 100 µm, often grouped as “visible gold” aggregates, with uneven, nuggety distribution at a deposit and sample scale. Samples are submitted for screen fire assay (SFA) analysis where coarse gold is suspected at Čoka Rakita. Sample preparation work is completed at ALS Bor, and samples are analysed in ALS Rosia, Montana.

To better quantify the variability of gold, all FA results received from SGS Bor, with results over 1 g/t Au, are re-assayed by means of SFA (Au_SCR24) at ALS laboratories. The selection of samples for re-assaying aims to capture broad and continuous intervals. It is triggered at the first and last interval above a 1 g/t Au grade threshold downhole once the drillhole enters prospective, skarn-type mineralisation. The selection includes internal intervals beneath the 1 g/t threshold, as well as an additional five (5) samples either side of the initial selection.

Each sample consists of approximately 1 kg of crush duplicate–coarse reject material (4 mm), provided by SGS Bor after completing the standard gold FA procedure. Preparation in ALS includes additional crushing and pulverising, then screening at 106 µm, to separate the sample into a coarse fraction (>106 µm), and a fine fraction (<106 µm). Fractions weights are reported in grams. After screening, two 50 g aliquots of fine fraction are analysed using the traditional FA method and atomic absorption spectrometry (AAS) finish. High-grade assays are repeated using gravimetric finish. ALS reports each assay and their calculated means.

(Au (-) F_SCR24_ppm). The entire coarse fraction is assayed to determine the contribution of the coarse gold using FA and gravimetric finish. Gold is reported in milligrams and back calculated in parts per million (ppm) (Au (+) F_SCR24_ppm). A total gold calculation for the entire sample is based on the weighted average of the coarse and fine fractions (Au Total_SCR24_ppm). The SFA results (Au Total_SCR24_ppm) are used as priority gold assay data over the initial SGS FA results.

11.3.4 Dry Bulk Density Measurements

Half-core billets are submitted to the SGS sample preparation facility at Bor for determination using a wax-sealed core water immersion method – PHY04V. After measurements have been completed, the core is returned to the core boxes.

11.3.5 Spectral Measurement

As of 2020, DPM has undertaken TerraSpec™ shortwave infrared spectral measurements at an onsite facility and results sent offsite for interpretation. Coarse sample reject material from every processed sample has been systematically measured during the Čoka Rakita drilling program.

11.4 Quality Assurance and Quality Control

11.4.1 Assay QA/QC Database Checks

DPM has performed routine checks on every laboratory submission upon import to the drillhole database, using acQuire QA/QC tools. These checks were initially undertaken on receipt of the assay results to determine if the submission had passed the control test. If the submission failed, it was re-assayed. On a monthly basis, the QA/QC data was assessed using custom acQuire tools to identify any quality control issues or trends. Failures in quality control samples were immediately discussed with the analytical laboratory and, if needed, batches were rapidly re-submitted.

11.4.2 Certified Reference Materials

All sample dispatches include routine insertion of CRMs to monitor accuracy, which were certified for gold, silver and sulphur and covered a wide grade range into the sample submission stream. A small number of CRMs were additionally certified for arsenic and copper. The CRMs used were a mixture of commercially available CRMs (supplied by Geostats) as well as project-specific standards (certified by Geostats). The samples were in standard pulp packets, but the recommended values of the samples were unknown to SGS laboratories. Previously, RC field duplicates were also inserted into the sample sequence. Coarse crush duplicates were produced from diamond core samples by the SGS sample preparation laboratory and included for analysis.

A CRM that assayed 10% outside the expected value for gold, silver, and sulphur, or 15% outside the base metal expected values was considered a failure that required the laboratory to re-assay 10 samples prior to, and 10 samples following the failed quality control assay. This instruction included the submission of standard reference material control samples. If more than two standards failed in a submission, the entire submission was re-assayed. If a failed standard was amid a sequence of results below the detection limit, it was up to the geologist assessing the data to determine if re-assay was required.

In 2021, DPM changed the failure limits for CRMs from percentage tolerance limits to standard deviations. Any CRM result that varies from the expected value by more than three (3) standard deviations, or any two (2) consecutive standards differing more than two (2) standard deviations constitutes a failure and the project geologist is required to submit the affected batches for re-assaying.

11.4.3 Blank Samples

Blanks samples were inserted into the sample stream to monitor for sample contamination and go through the same sample preparation and analytical procedure as all samples sent within the same dispatch to the laboratory. The results were monitored using warning and failure limits of three and 10 times the lower detection limit respectively for the analytical method used. If two or more batches of samples in sequence contained blank assay values above the warning threshold, the batches were re-assayed. If a blank sample returned an assay value above the failure limit, the entire batch was re-assayed. DPM internal controls identified a limited number of warnings; however, no failures were noted with the blank materials.

11.4.4 Duplicates

Diamond core field duplicates, which are duplicate samples taken at the jaw crushing stage, were inserted into the sample stream on a 1:20 frequency to assess precision. Results were monitored by DPM staff by comparing results on scatterplots as well as by means of statistical review. The results indicate no bias as well as good levels of repeatability.

11.4.5 Screen Fire Assaying – Blank Samples

The QA/QC procedure to monitor possible contamination during the ALS SFA procedure (crushing, milling, screening) includes insertion of known unmineralised, barren reject material at every 20th sample.

During March 2023, DPM noted sporadic blank failures within the ALS sample stream that indicated cross contamination was occurring within the screening process. Since the standard ALS SFA procedure includes washing of crushers and mills only at the beginning and the end of the job, extra washes with a barren flush have been specified by DPM after each sample that has over 5 g/t Au grade based on initial FA results. In addition to the controls assessed by DPM, ALS provides laboratory CRMs and duplicate data for DPM's analysis and review. After these additional controls being in place, blank performance has improved significantly.

11.4.6 QP QA/QC Report: 1 March 2007 to 31 October 2023

DPM supplied an export of the data from the acQuire database to the QP. The data were investigated per laboratory for diamond drilling samples. QA/QC data for the following laboratories were reviewed: Genalysis Perth (GEN_PE), ALS Vancouver (ALS_VA), SGS Bor (SGS_BO), SGS Chelopech (SGS_CH), and ALS Bor (ALS_BO).

The quality control data for all elements did not show any fatal flaws. There are CRMs that failed. However, these seem to be due to sample swaps or incorrect CRM names. The CRM results display bias, but not systemic bias as the bias is not consistently negative or positive.

All incoming assay results are emailed as digital files from the analytical laboratory. The database export does not contain expected reference values for specific assay methods for all the relevant CRMs used during the period under review. This does not conform to industry best practice and CRM samples are associated with more than one analytical method (analysis suite).

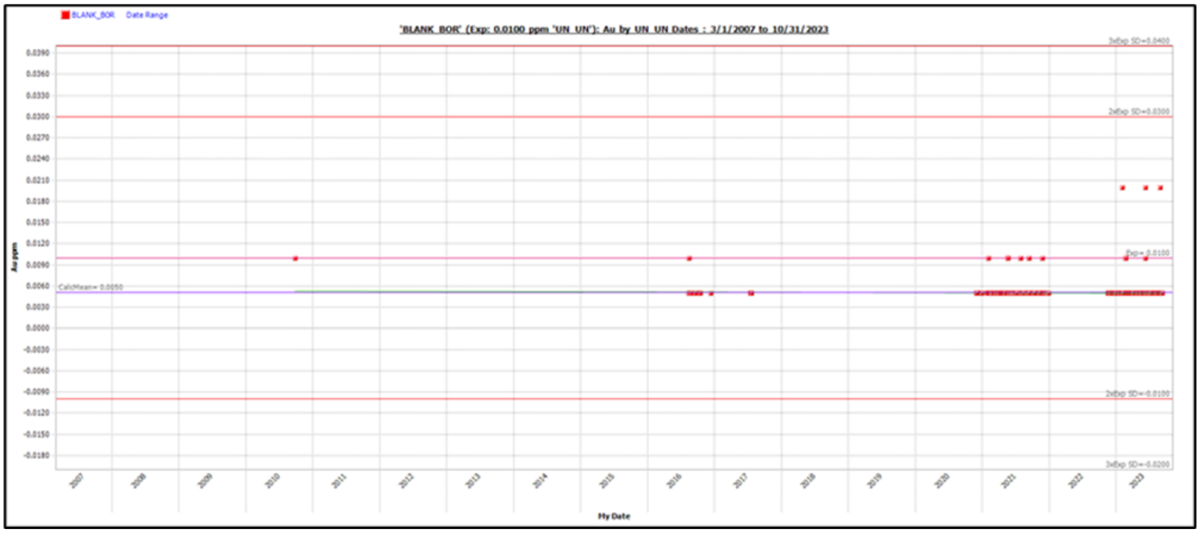
The CRMs used as part of the quality control program were not captured in the database. The expected values and allowed standard deviations per element, per analysis method were not captured in the database. The implication of this is that a CRM cannot fail analysis, as there are no results associated with an element and analysis method. Under the supervision of the QP the expected results were captured based on the certificates supplied by DPM. The results were recorded as analysis method "Unknown". In many cases the analysis method result differences are not statistically significant. However, according to industry best practice this way of setting up the database is not the best way.

The ideal database setup for analysis method for CRMs should include the CRM ID, the element, the laboratory analysis method, the expected value, and the standard deviation expected. This will also ensure that material bias may be detected when considering CRM results.

11.4.6.1 Contamination

The data for the blank, BLANK_BOR, did not show any fatal flaws. The results were within the expected range (Figure 11.1).

Figure 11.1 – Blank Performance for SGS Bor

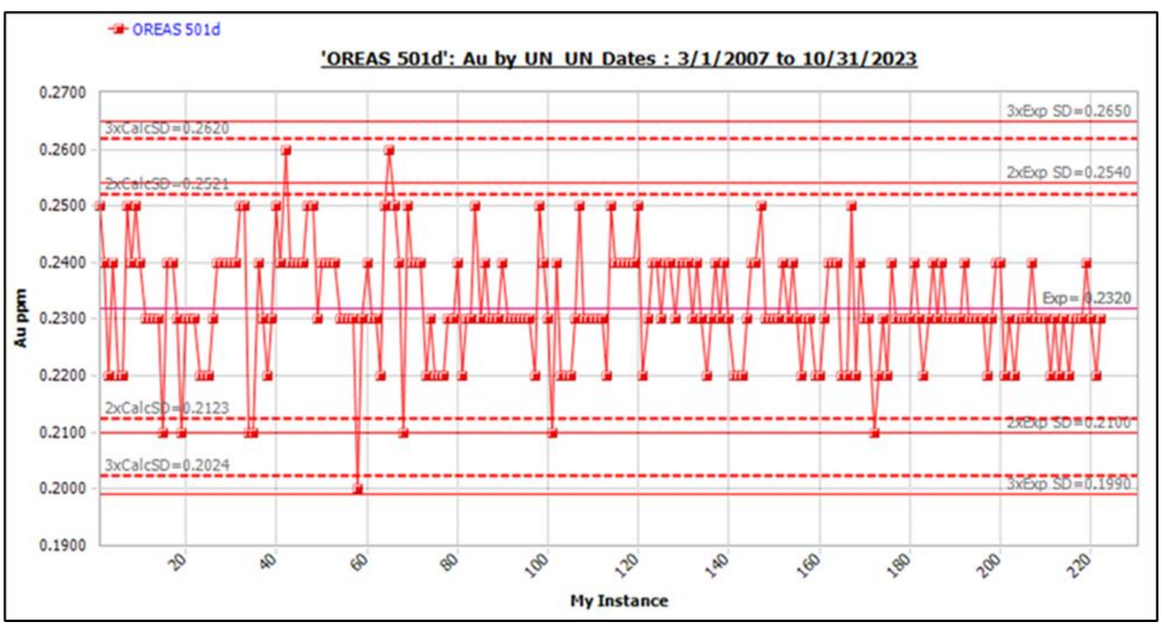


Source: CSA Global, 2023

11.4.6.2 Accuracy

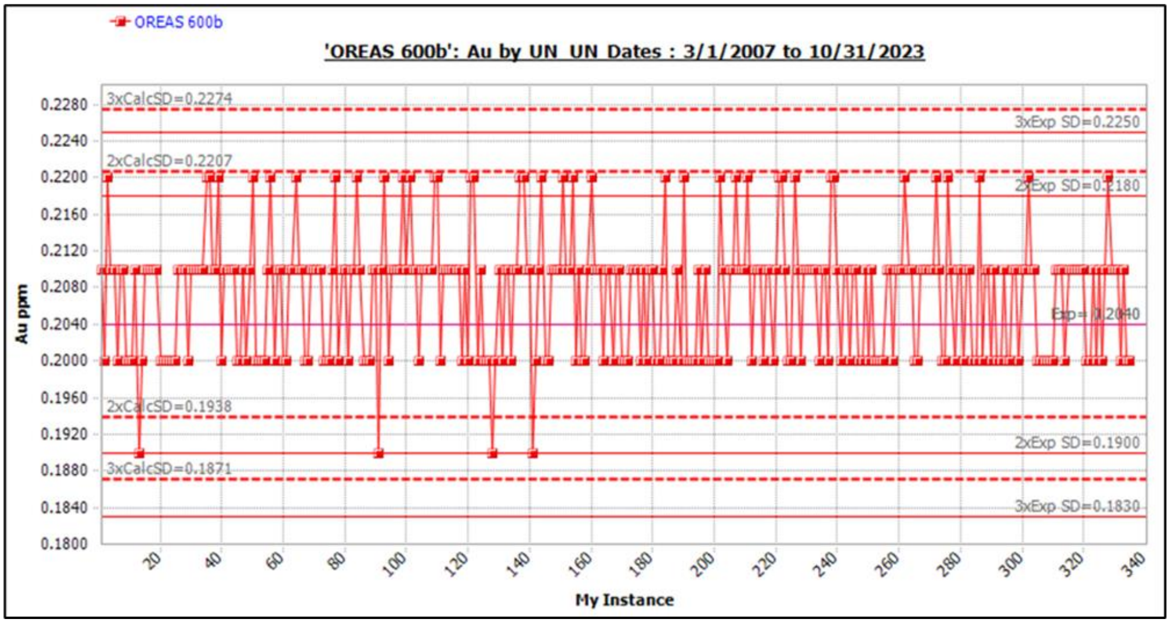
The CRM results were investigated per individual CRM for all laboratories combined, as well as per CRM per laboratory. There were some biases noted but are not indicative of any systemic biases. The results do not highlight any fatal flaws, and samples that failed can be attributed to sample swaps or incorrect labelling of CRMs. The following graphs illustrate the performance per CRM, laboratories combined (Figure 11.2 to Figure 11.5).

Figure 11.2 – OREAS 501d Performances for Au for All Laboratories



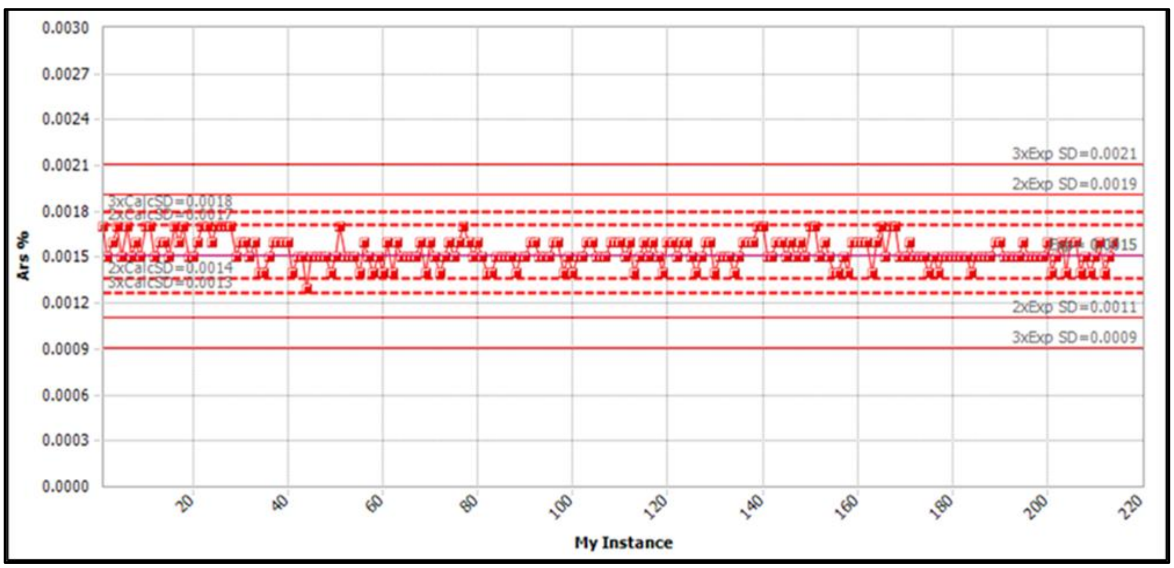
Source: CSA Global, 2023

Figure 11.3 – OREAS 600b Performance for Au for All Laboratories



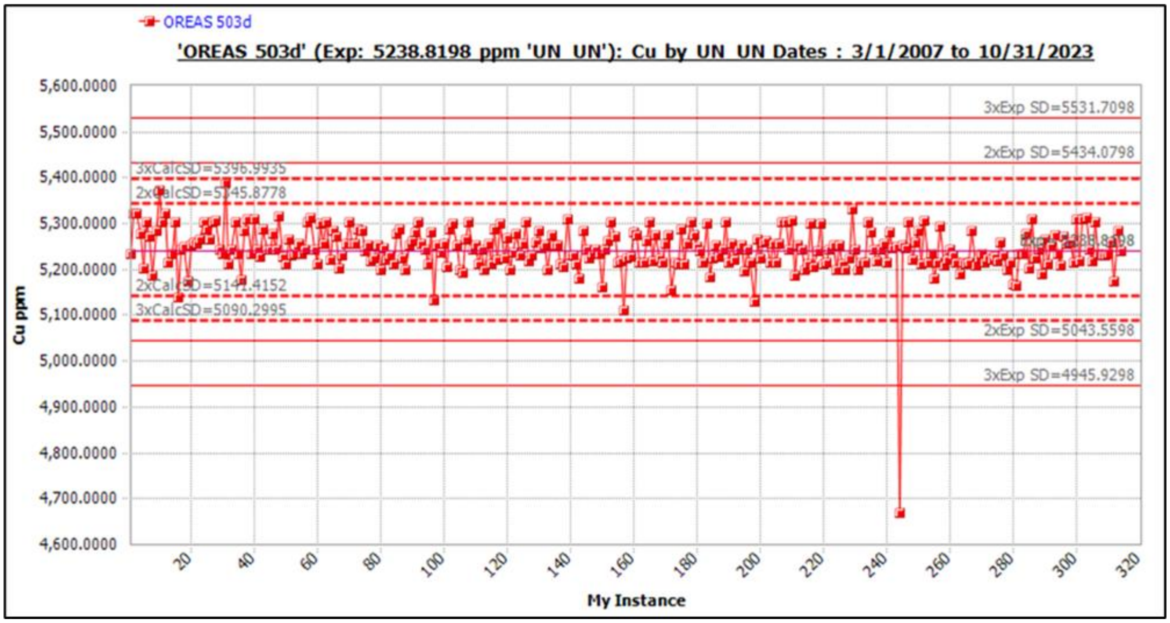
Source: CSA Global, 2023

Figure 11.4 – OREAS 501d Performance for As



Source: CSA Global, 2023

Figure 11.5 – OREAS 503d Performance for Cu, All Laboratories Combined



Source: CSA Global, 2023

The failed sample may indicate a swapped sample or a mislabelled CRM.

11.4.6.3 Precision

Preparation duplicates as well as external check (SFAs as Umpire) results were compared for diamond drill samples. The duplicate results were compared for primary samples submitted to SGS Bor, SGS Chelopech, and ALS Bor. The precision data for gold for sample type half diamond core fall within acceptable practice limits (Table 11.2). The precision for the trench samples were not within acceptable practice limits, though this is not material to underground Mineral Resources which is the subject of this Report.

11.5 Sample Security

Samples collected from field operations are transported to the DPM core shed based in Bor where the samples are geologically logged and prepared for chemical analysis by DPM staff. The sampling procedures are appropriate and adequate security and supervision exists on the site to minimise any risk of contamination or inappropriate mixing of samples. A pulp library is maintained of all samples prepared by SGS Bor, which are stored in a locked warehouse onsite.

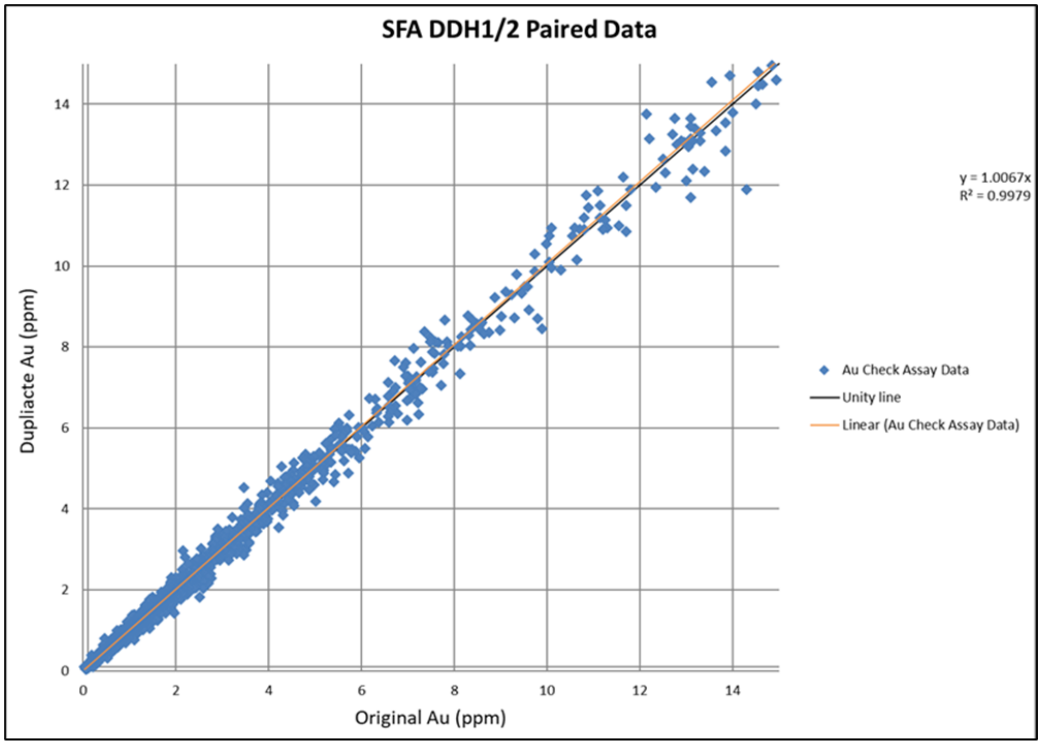
The core shed, sample preparation laboratory and pulp library facilities are located within a gated compound in Bor that requires a secure key card to access. The facility has an alarm system and closed-circuit television (CCTV) cameras distributed across the site.

Table 11.2 – Summary of Duplicates and SFA Data, Including Acceptable and Best Practice Limits Where Applicable

Type	**Sample Type**	**Best Practice Limits**	**Acceptable Practice Limits**	**Pairs** (total)	**Count of Pairs** (>10 x DL)	**CV(AVR)** %	**Mean Au Original** (ppm)	**Mean Au Duplicate** (ppm)	**Bias**
FDUP	DDH ½	20	30	2,914	1,141	16	0.91	0.94	3%
SFA UMPIRE	DDH ½	10	20	2,883	2,821	7	2.78	2.80	1%

Source: CSA Global, 2023

Figure 11.6 – SFA vs FA Data Displaying Excellent Precision



Source: CSA Global, 2023

11.6 Summary Opinion of Qualified Person

The QP concludes that the sample preparation, security, and analytical procedures are robust and follow CIM Exploration Best Practice Guidelines and industry best practice. The QA/QC procedures are comprehensive and are suitable to monitor assay contamination, accuracy, and precision. The QP makes the following conclusions and recommendations.

- Approximately 87% of samples (+3,000 samples) are analysed by SFA (samples >1 g/t Au), which is a high percentage of samples and encouraging. All SFA samples are analysed by ALS Romania. The QP recommends sending a selection (5 to 10%) of coarse reject samples to another laboratory for SFA. This provides a second laboratory check and confirmation that essentially similar SFA procedures are providing similar results (and if not, another investigation would be required). As a first step, compare the analytical SFA techniques with a second laboratory to ensure a like-for-like process (which is preferable) so that only assay results need to be compared.
- Consider having blanks and CRMs strategically placed in and around areas of high-grade material. Blanks are particularly critical. This is where contamination may be expected to occur and provides a good data set within the resource model, which can be very useful when reviewing reconciliation at the mining stage.
- Keeping the systematic placement of CRMs and blanks as per the DPM manual is recommended for general lab performance, but when mineralisation represents a relatively small part of the samples assayed it is important to have QA/QC materials inserted in and around the mineralisation.
- No analytical umpire samples are available for the mineral resource drilling programs at Čoka Rakita. DPM procedure is that approximately 5% of all samples exhibiting a gold grade greater than 0.1 g/t Au are sent for umpire analysis to a third-party laboratory to assess the reliability of primary analytical data. The QP understands that umpire samples have been selected and will be assayed during 2024. however, the results were not available at the time of reporting.
- DPM should strive to ensure a suite of CRMs are available that match the grade tenor of the Čoka Rakita deposit. The current suite of CRMs is generally suitable for lower-grade porphyry and sediment-hosted gold type grades; however higher grades, as those seen within gold-rich skarn deposits, are underrepresented. The QP understands that appropriate gold grade CRMs have been ordered and will be inserted in the 2024 sampling programs.
- Continual vigilance is required considering the extremely high gold grade values that have been encountered while drilling.
- The QP finds that the QA/QC procedures are adequate and that there are no evident fatal flaws.

12 DATA VERIFICATION

DPM has implemented an acQuire GIMS to manage drillhole data. All data, such as collar, survey, geological, geotechnical, structural, assay, etc. are imported daily into acQuire from the server or via email. After validation, data is one-way synchronised Datamine™ for Mineral Resource estimation purposes. The acQuire GIMS was also used to generate monthly, quarterly, and yearly QA/QC reports.

Data used to support the MRE have been subjected to validation, using inbuilt and modified acQuire GIMS triggers that automatically check data for a range of data entry errors. Verification checks on surveys, collar coordinates, lithology, and assay data have also been conducted.

Data underwent further validation by the QP (Maria O'Connor, MAIG) through a series of spot checks for factual errors and further observations from the QP's verification program include:

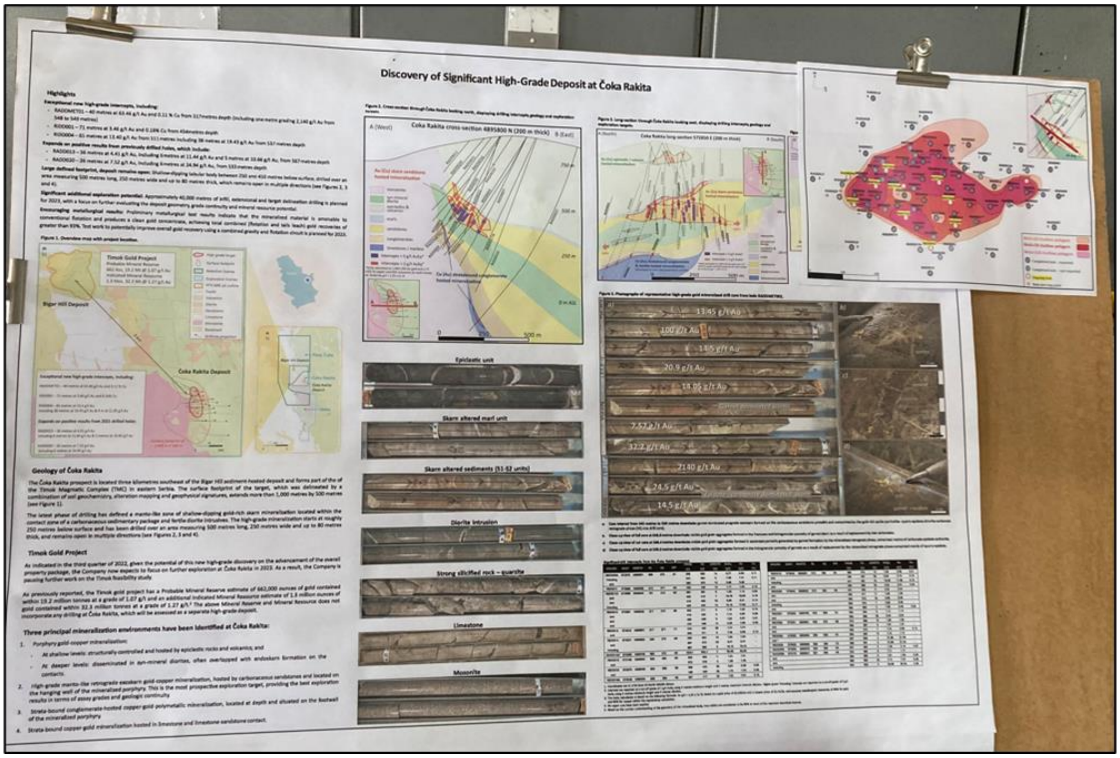
- Assay certificates for eight (8) drillholes – RADD013, RADD020, RIDD004, RIDD007, RIDD008, RIDD017A, RIDD020, RIDT004 – were requested by the QP due to their materiality to the MRE. These were reviewed for both FA (SGS) and SFA (ALS) results to enable spot checks to be completed against the database. No errors were found in the spot checks made. SGS assay certificates were signed off by George Daher, Laboratory Manager SGS Bor while the ALS assay certificates were signed off by Adrian Bogdan, General Director Romania.
- A series of discussions were held with DPM geologists, geochemists and database personnel continuously throughout the estimation process, both on and off-site. This enabled a fuller understanding of the data and the interpretation of mineralisation controls by the QP.
- Sample submission forms were viewed to understand the process. Drilling records and hard copy assay certificates were found to be filed in a neat and orderly manner (Figure 12.1). Housekeeping around drill rigs, sampling and core logging was found to be good.
- The SGS laboratory in Bor was toured by the QP and equipment was found to be clean and well managed by an experienced laboratory manager George Daher. No issues were identified.
- The QP observed from discussions on site that the geologists who log the core have a clear understanding of the geology and efforts are made to ensure consistency with rock boards being used to show representative sections of core. Logging is revisited as assays are returned and in conjunction with multi-element geochemistry to fully understand the stratigraphy and alteration and there is a good level of communication across the team as the interpretation develops (Figure 12.2).
- Drilling activities were viewed on site to assess how core was recovered and treated at the drill site. Drillhole deviation is carefully monitored on an ongoing basis.
- Drilled collars were visited and collar coordinates were identified in the field (Figure 12.3) and picked up using a handheld GPS to confirm collar coordinates held in the database. These reconciled well and no issues were identified. Drill core was viewed at drill rigs (Figure 12.5).

- Representative core from five (5) drillholes was viewed, with significant intercepts being inspected and verified visually and in conjunction with assay results by the QP (Figure 12.4).
- Core recovery was reviewed and found to be >98% for all rock types. There is no apparent relationship between recovery and grade.
- QA/QC was reviewed as described in Section 11.4.6 and no significant issues were found.
- The SFA process was reviewed in detail given the presence of coarse gold and the predominance of this method in the assays used to underpin the MRE.
- Overall, the QA/QC procedures are appropriate, and the high amount of SFAs is encouraging. With the resource at the current inferred stage, a review of the QA/QC in the context of domaining, focusing on different mineralised grade zones is recommended.

Figure 12.1 – Example Hard Copy Sample

Submission Form Left and Folders of Assay Results, Right
Source: CSA Global, 2023

Figure 12.2 – Current Interpretation on Display in Core Yard

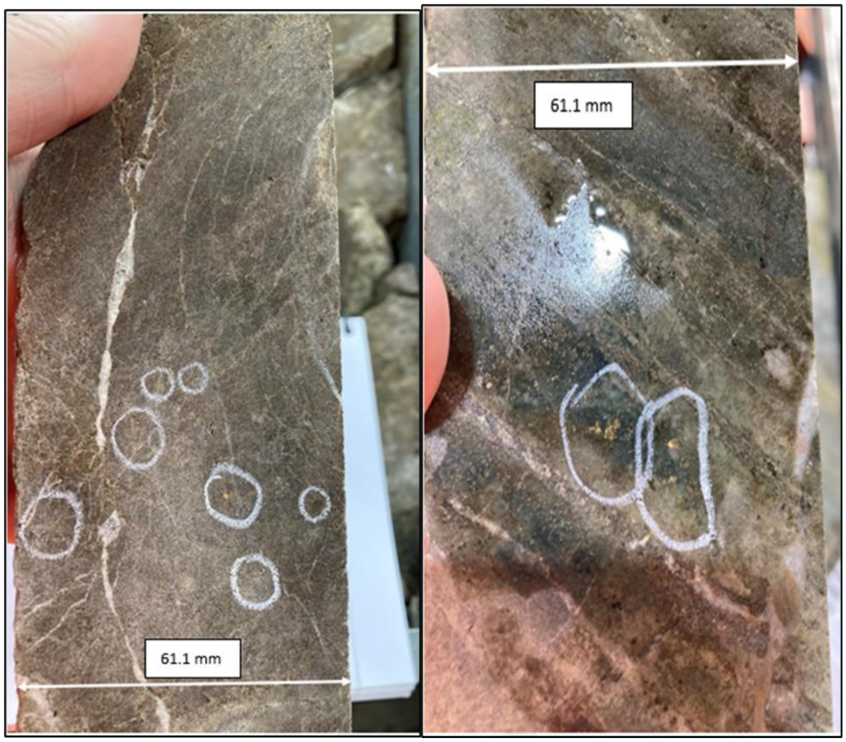


Source: CSA Global, 2023

Figure 12.3 – Examples of Drillhole Collars Identified in the Field

Source: CSA Global, 2023

Figure 12.4 – Visible Gold Seen in Core in Drillhole RIDD008



Source: CSA Global, 2023

Figure 12.5 – Drill Core at Drill Rigs while Drilling Takes Place

RIDD044 Left, RIDT031 Right
Source: CSA Global, 2023

Database validation completed by DPM geologists and verified by the QP is summarised below:

- Coordinates are captured at various stages using different methodologies which are ranked accordingly and those with the highest (best) ranking are captured in the "Best" field in the database. These coordinates were used in the MRE. Highest to lowest ranked methods are as follows – differential GPS->Total station->Digitised->Transformed Historic->Planned.
- Collar information was received via email from the Survey Department in pre-specified templates and imported into the acQuire database.
- There were no issues identified with the data in the collar table.
- No issues were identified with downhole surveys.
- There are unlogged intervals for geology and alteration. Industry best practice would usually have relational validation in place that would disallow this from happening, so a database audit is recommended to ensure best practice in database procedures. Since the specified intervals relate to drillholes not yet logged, this is not material to the MRE under discussion here.

Queries identified during the database load-up and validation process for relevant data were discussed with DPM. Outcomes are summarised as follows:

- Four (4) drillhole had zero depth recorded in the database – RIDD007B, RIDT012, RIDT014, RIDT039. These were terminated due to technical drilling reasons.
- Seven (7) drillholes were in progress – RIDD047, RIDD048, RIDD049, RIDT029, RIDT030A, RIDT035, RIDT042.
- 37 holes had no samples/assays recorded:
  1. These were holes that were terminated early (RIDD007B, RIDT012, RIDT014, RIDT039);
  2. In progress (RIDT030A); or
  3. Waiting to be sampled (RIDD021A, RIRC013, RIRC014, RIRC016, RIRC017, RIRC018, RIRC019, RIRC020, RIRC021, RIRC022, RIRC023, RIRC024, RIRC025, RIRC026, RIRC027, RIRC028, RIRC029, RIRC030, RIRC031, RIRC032, RIRC033, RIRC034, RIRC035, RIRC036, RIRC037, RIRC038, RIRC039, RIRC040, RIRC041, RIRC042, RIRC043, RIRC044).
- Samples recorded in the assay file with no assay results – these were holes that had been sampled but were awaiting assay results, where there were voids (560 m), or where the interpreted unit was SFD. Notably, there were no missing assays within the skarn mineralisation which is the focus of this MRE.
- All drillholes had downhole survey records and geotechnical logging.
- All completed drillholes had logged geology and alteration with the exception of 44 drillholes. Many (31) of those holes with unlogged geology are RC pre-collars, meaning only 13 diamond drillholes do not have logged geology/alteration.

- There are 53 diamond drillholes with no density measurements; however, there is a large number of density measurements that are spatially and materially representative of the waste and mineralisation, therefore the QP does not consider this an issue.

Overall, the drillhole database is clean and as complete as possible given that drilling continues at the Project and the database received is a snapshot in time. It is therefore reasonable that some drillholes contain missing data (e.g. waiting to be logged for geology etc.) as they wait to get processed.

Based on the checks completed, the QP is comfortable that the available information and sample density allow preparation of a reasonable estimate of the geometries, tonnage and grade continuity of the mineralisation in accordance with the level of confidence established by the MRE categories in the CIM Definition Standards.

The QP is of the opinion that the data used and described in this Report is adequate for the purposes used in this Report.

13 MINERAL PROCESSING AND METALLURGICAL TESTWORK

13.1 Introduction

A review of the available metallurgical data was carried out by DRA, with a focus on obtaining pertinent results affecting the flowsheet development and process design criteria. The findings of the review are presented in this Section.

Wardell Armstrong International performed testwork in 2021 on five (5) samples provided by DPM. The objective of the program was to perform exploratory testing to investigate the samples' amenability to gravity concentration, cyanidation and flotation.

Table 13.1 summarises the results of this program and shows that recoveries exceeding 90% can be achieved with a flowsheet comprising flotation followed by leaching of the float tails.

Table 13.1 – Wardell Armstrong Test Program Summary

Lithology	Gold Recovery (%)							
	Sample ID	Head Au Assay (g/t)	Gravity Conc.	Whole Material Leach	Rougher Flotation	Cleaner Flotation	Flotation Tails Leach	Flotation Tails Leach
Retrograde Exoskarn	Met_Ra_P01	2.68	50.7	88.3	75.1	73.7	73.2	92.9
Phyllic Exoskarn	Met_Ra_P03	3.91	57.2	88.7	75.8	73.5	74.8	93.3
Retrograde Skarn	Met_Ra-P05	18.54	63.6	93.0	82.3	81.3	82.3	96.7
Retrograde Endoskarn/ Potassic Porphyry	Met_Ra_P02	0.55	40.5	88.0	79.5	77.2	76.7	94.7
Epiclastic-Volcaniclastic	Met_Ra_P04	2.36	35.6	81.7	55.2	51.3	79.7	90.1

Source: Wardell Armstrong, 2021

The 2023 BaseMet Laboratories (BaseMet Labs) testwork program was performed on three (3) composite samples, representing low, medium, and high gold grades. The BaseMet Labs test program was more comprehensive than the exploratory Wardell Armstrong program and covered many processing aspects including sample mineralogy, gravity concentration, flotation, cyanidation, sedimentation as well as filtration. Extensive testing was conducted to explore and optimise test conditions, which are all reported in the sections below. The testing culminated in a set of locked cycle tests (LCTs) which essentially simulate the selected flowsheet and employ optimised test conditions.

Table 13.2 below summarises the gold recoveries achieved during these LCTs and compares it with a scenario where the tails from this LCT was subsequently leached using cyanide and, finally, against a control test which simply comprised direct cyanidation of the original sample. The result shows that cyanidation of the flotation tailings consistently produced the best gold recoveries. However, DPM did not consider the use of cyanide appropriate for this project, hence the flowsheet selected uses gravity concentration followed by flotation only.

Table 13.2 – BaseMet Labs Test Program (2023) Results Summary

Sample ID	Test Procedure	Head Au Grade (g/t)	Au Recovery (%)	Tails Au Grade (g/t)
MetCRA23-01	Gravity -> Flotation Locked Cycle Test (LCT)	3.33	88.1	0.41
	Gravity-> LCT->Tails Cyanidation	3.33	92.4	0.12
	Direct Cyanidation	3.44	88.4	0.40
MetCRA23-02	Gravity -> Flotation Locked Cycle Test (LCT)	5.93	84.2	0.94
	Gravity-> LCT->Tails Cyanidation	5.93	92.4	0.19
	Direct Cyanidation	6.36	92.1	0.51
MetCRA23-03	Gravity -> Flotation Locked Cycle Test (LCT)	11.0	90.9	1.04
	Gravity-> LCT->Tails Cyanidation	11.0	96.2	0.18
	Direct Cyanidation	10.5	95.4	0.48

Source: BaseMet, Labs 2023

13.2 Historical Metallurgical Testwork

To the best of DRA's knowledge there is no historical testwork for this deposit other than the testwork described in this document.

13.3 Wardell Armstrong Testwork Campaign (2021)

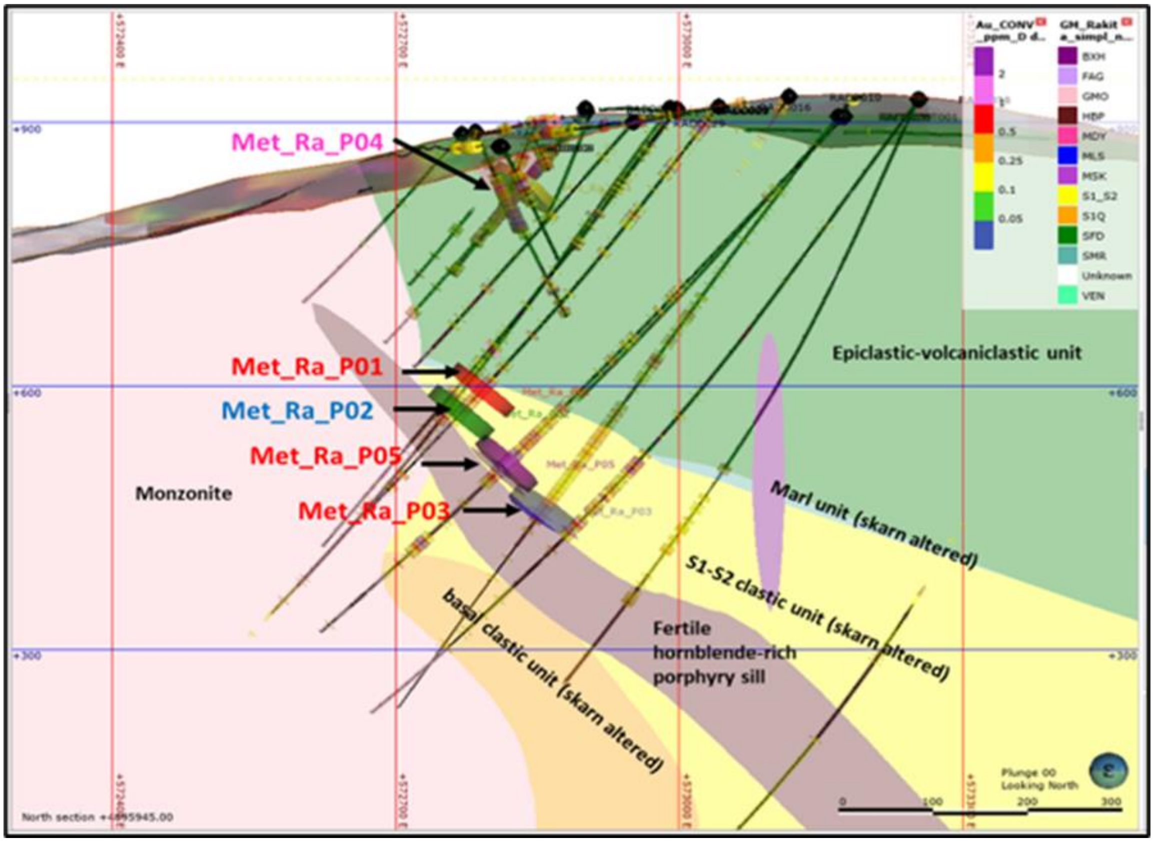
13.3.1 Sample Selection

The aim of this initial metallurgical testwork program was to understand variability related to various mineralised zones and paragenesis. The five (5) metallurgical test samples included three (3) samples from the deeper high-Au grade manto/skarn mineralisation, one (1) sample from the low-grade Au-Cu porphyry mineralisation and one (1) sample from the shallow moderate Au-grade epiclastic hosted mineralisation. All samples consist of a single uninterrupted interval which yielded approximately 30 kg of sample mass.

Figure 13.1 presents a schematic E-W cross section view (N4895945, +/-100 m, looking North) showing the distribution of the selected metallurgical samples relative to geology and mineralisation types. This figure depicts the drill holes and sample locations for the five (5) samples used in the

Wardell Armstrong testwork campaign. The drill cores were selected to represent all five (5) major lithologies and to provide a range of gold grades.

Figure 13.1 – Sample Locations for the Five Wardell Armstrong Samples



Source: Wardell Armstrong, 2021

13.3.2 Head Assays

Table 13.3 summarises the head assays determined for these samples.

Table 13.3 – Head Assay for Wardell Armstrong Test Program

Element	Units	Met_Ra_P01	Met_Ra_P02	Met_Ra_P03	Met_Ra_P04	Met_Ra_P05
Au (FA)	ppm	2.69	0.53	2.69	2.23	16.45
Au (AR)	ppm	2.68	0.55	3.91	2.36	18.54
As	%	0.006	0.007	0.010	0.032	0.005
S(tot)	%	2.94	2.68	2.25	2.67	1.89
S(sulph)	%	2.88	2.66	2.23	2.65	1.79

Source: Wardell Armstrong, 2021

The gold grades of three (3) of these samples are below the cut-off grade for the mine. Sample Met_Ra_P02 has a grade of 0.5 g/t only and consequently represents waste. Sample Met_Ra_P05 is representative of a high-grade pocket. The low grades implies that recoveries recorded during this testwork program would likely be more conservative than what can be expected from the mine during regular operations.

Arsenic levels were low throughout. Other elements analysed include carbon and base metals; these were all insignificant.

13.3.3 Mineralogy

Highlights from the mineralogical examination are as follows:

- Garnet and quartz are the most abundant minerals in these samples.
- Sulphur minerals are mostly pyrite and chalcopyrite.

13.3.4 Gravity Concentration

Wardell Armstrong subjected the five (5) samples to a two-stage gravity concentration procedure involving a centrifugal concentration step followed by re-concentration of the initial concentrate on a Mozley shaking table. The tailings from both gravity concentration steps were combined to produce a final tails sample. Table 13.4 below summarises the results achieved.

Table 13.4 – Two-Stage Gravity Test Results

Sample ID	Head Grade	Gravity Concentration		
	Au (g/t)	Grade (g/t)	Recovery (%)	Mass Pull (%)
Met-RA-P01	2.68	66.8	50.7	2.0
Met-RA-P02	0.55	11.6	40.5	1.9
Met-RA-P03	3.91	105.3	57.5	2.1
Met-RA-P04	2.36	47.9	35.6	1.8
Met-RA-P05	18.54	496.3	63.6	2.4
Average		145.6	49.6	2.0

Source: Wardell Armstrong, 2021

The results show that approximately half of the gold present in these samples can be recovered using gravity concentration. Given that the grades of the samples are low compared to what the eventual plant will process it is evident that a gravity concentration step would be beneficial.

There is significant variation in gravity recovery with results ranging from 35.6% to 60.5%. This is not unusual and is indicative of the presence of coarse gold nuggets that are unevenly distributed within the samples.

13.3.5 Whole-Ore Leaching

Wardell Armstrong subjected their five (5) samples to whole-ore cyanidation to test the samples' amenability to leaching. The tests were conducted at grinds of 80% passing 106 µm and 80% passing 75 µm. A lengthy duration of 48 hours and a cyanide concentration maintained at 1 g/L ensured complete dissolution of all leachable gold.

Gold recoveries from these whole-ore cyanidations ranged from 82 to 93% with the average around 87%. Grind size did not affect the results appreciably. Cyanide consumption averaged around 2.3 kg/t which is higher than for a typical gold plant.

13.3.6 Flotation

Wardell Armstrong performed scouting rougher flotation tests on the five (5) samples tested in 2021. This was followed by assessing whether cleaner flotation of the rougher concentrate could improve the quality of the concentrate significantly. Lastly, the first cleaner tails samples were leached to assess whether overall gold recovery could be improved sufficiently to justify the introduction of cyanide to the flowsheet.

Table 13.5 summarises the results of the scouting rougher flotation tests performed in 2021. Rougher flotation tests were performed after grinding the samples to 80% passing 105, 75, 45 and 30 µm respectively. The slurry samples were conditioned with $CuSO_4$ prior to adding PAX and MIBC at the start of the flotation stage. It is noted that the results for sample P02 are omitted below for the sake of brevity because it is far below the cut-off grade.

Table 13.5 – Gold Recoveries and Grades at Various Grind Sizes After Rougher Flotation

Grind P_{80} (mm)	**Gold Recovery** (%)				**Rougher Concentrate Gold Grade** (g/t)			
	P01	P03	P04	P05	P01	P03	P04	P05
105	51.7	74.5	59.5	87.9	27.9	35.3	13.3	194.0
75	79.0	76.9	57.3	82.9	39.8	41.3	13.0	251.0
45	80.3	59.2	62.6	85.3	32.6	46.9	14.0	246.9
30	85.8	64.2	64.4	82.2	36.4	44.4	11.4	251.0

Source: Wardell Armstrong, 2021

Only sample P01 showed a consistently improving trend in gold recovery as the grind became finer, with the finest grind yielding 85.8% recovery, while the coarsest grind only achieved 51.7%. There is no clear trend for any of the other samples tested, which is possibly due to the presence of gravity gold in the samples that results in significant variations in recovery at all grinds.

A further set of tests were performed at grinds of 75 µm and 45 µm, where a cleaner step was added after rougher flotation to assess how much the grades could be improved through cleaning. Table

13.6 summarises the results for the 75 µm set (45 µm results omitted for the sake of brevity as the trends are similar).

Table 13.6 – Summary of Rougher-Cleaner Flotation Tests at a P_{80} of 75 µm

Sample ID	Rougher Concentrate		Cleaner Concentrate	
	Au (g/t)	Recovery (%)	Au (g/t)	Recovery (%)
Met_Ra_P01	38.5	75.1	51.2	73.7
Met_Ra_P02	4.2	79.5	5.8	73.2
Met_Ra_P03	35.8	75.8	51.0	73.5
Met_Ra_P04	15.9	55.2	22.0	51.3
Met_Ra_P05	322.5	82.3	455.9	81.3

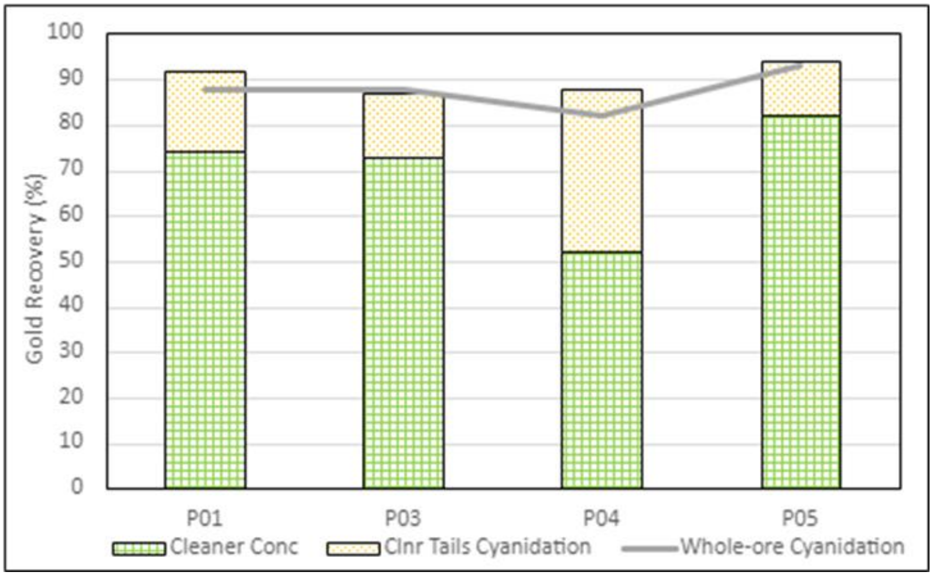
Source: Wardell Armstrong, 2021

The results show that the addition of a cleaner step will improve final concentrate grade substantially (approximately 50% increase) at the expense of only a few percentage points of lower recovery. Given these results a cleaner step is recommended as it will reduce the mass of concentrate to be transported significantly for only a modest decrease in total gold production.

Finally, Wardell Armstrong performed cyanidation of the cleaner tails to see if the additional gold extracted in this way could improve recoveries. Figure 13.2 shows the contributions of flotation and cyanidation as a stacked column and compares that to the recoveries achieved through whole-ore cyanidation (the line). The graph shows that the additional gold extracted into the leach solution resulted in combined gold recoveries comparable to what had been achieved through whole-ore cyanidation. For samples P01 and P04, this route even exceeded whole-ore recoveries marginally.

While the additional recovery is substantial the downside is that this introduces cyanide to the site which has implications for permitting and how the tailings need to be managed. A decision has since been taken by DPM to avoid cyanide on site in favour of a flotation-only flowsheet.

Figure 13.2 – Comparison of Gold Recovery-Whole-Ore Leach versus Flotation-Cyanidation of Cleaner Tails



13.4 BaseMet Labs Testwork Program 2023

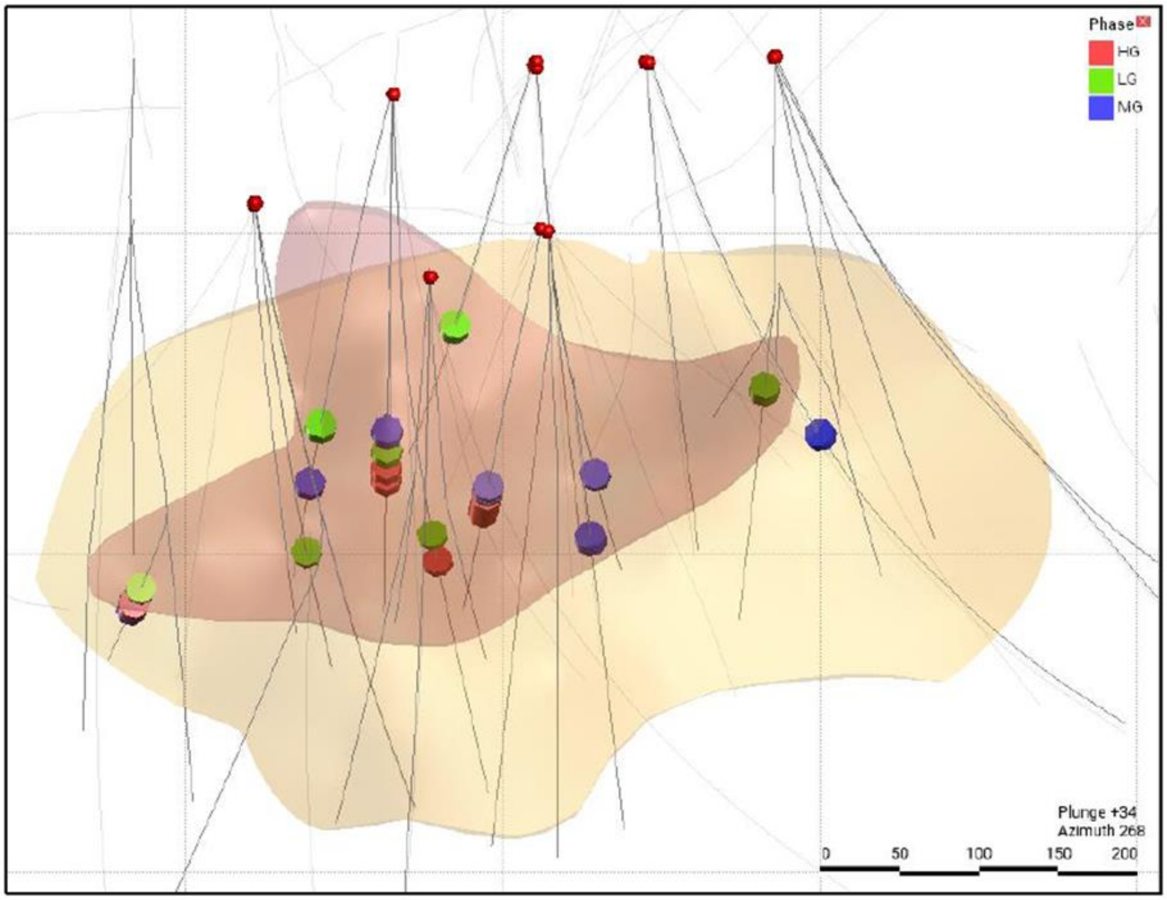
13.4.1 Sample Selection

Figure 13.3 shows the locations of the drill core samples selected to produce the three (3) composite samples used in the BaseMet Labs testwork program. The primary objective with the three (3) composites was to produce low, medium and high gold grade samples.

Cancha software was used to select the intervals for the three (3) composites. Quarter cores were used. Older holes were not considered as these may be oxidised and were hence deemed unsuitable. A minimum cut-off grade of 3.0 ppm Au was specified at the time of sample selection. The aim with the sample selection was to include as many as possible drillholes to ensure good representativeness of the entire mineral body. An equal weighting of lithology types was also specified.

The low-grade composite sample (MetCRA23-01) consists of ten (10) intervals selected from ten (10) drillholes for a weighted average gold grade of 3.16 g/t. The medium-grade composite sample (MetCRA23-02) contains nine (9) intervals selected from eight (8) drillholes for a weighted average gold grade of 6.36 g/t. The high-grade composite sample (MetCRA23-03) contains nine (9) intervals selected from seven (7) drillholes for a weighted average gold grade of 12.52 g/t.

Figure 13.3 – Sample Locations for the Drill Cores Selected for the Three 2023 Composite Samples



Source: BaseMet Labs 2023

13.4.2 HEAD ANALYSIS

Table 13.7 summarises the head analysis performed on the three composites tested in 2023. This table shows acceptable agreement between the gold analysis through fire assay (FA) and through the screened metallics (SM) procedure. Sample MetCRA23-02 has a similar head gold grade to the LOM grade of around 5.5 g/t.

Table 13.7 – Head Assay for BaseMet Labs Test Program (2023)

Sample ID	Au by SM (g/t)	Au by FA (g/t)	S by LECO (%)
MetCRA23-01	3.82	3.35	3.03
MetCRA23-02	5.91	4.92	2.52
MetCRA23-03	11.70	11.10	2.28

Source: BaseMet,Labs 2023

Notable elements are silver ranging from 0.7 to 1.1 ppm and arsenic ranging from 48 to 115 ppm. Both are low and should not affect the process design appreciably.

13.4.3 Mineralogy

AGAT Geology Dept (directed by BaseMet Labs) performed XRD analysis on powdered samples of the three metallurgical samples tested by BaseMet Labs in 2023. Highlights from the mineralogical examination are as follows:

- Garnet was the most abundant mineral at around 47% of the total mass.
- Calcite (12%), quartz (9%) and potassium feldspar (8%) are also significant.
- Pyrite was the only significant sulphide mineral at around 4%.
- Clay minerals were observed but were insignificant at <1%.

13.4.4 Laser Ablation Analysis

Table 13.8 presents the results of the laser ablation analysis of the pyrite showing the distribution of metals.

Table 13.8 – Metal Concentrations within Pyrite

Sample ID	Average Elemental Concentration within Pyrite (ppm)											
	Mn	Co	Ni	Cu	Zn	As	Ag	Cd	Sb	Au	Hg	Pb
Met_CRA23_01	92.7	278.6	363.2	102.6	2.51	562.5	0.99	0.08	2.80	**0.79**	0.93	n/a
Met_CRA23_02	26.8	209.4	238.4	258.2	58.12	559.2	6.19	1.61	52.90	**0.61**	1.47	216.2
Met_CRA23_03	13.8	192.5	265.9	192.3	17.27	259.1	1.00	0.48	0.55	**0.22**	0.77	37.40

Source : BaseMet Labs 2023

The low gold concentrations indicate that there is very little refractory gold locked in pyrite.

13.4.5 Comminution

Table 13.9 summarises the comminution results produced for the PEA phase of the Project. Three (3) metallurgical composite samples were tested individually. There was no testing of comminution parameters during the 2021 program performed by Wardell Armstrong.

Table 13.9 – Summary of Comminution Test Results (2023 Test Program)

Sample ID	DWi	Axb	SG	SCSE	Ai	BBWi
	(kW/m^3)			(kW/t)		(kW/t)
Met_CRA23_01	5.57	57.5	3.21	9.12	0.123	13.4
Met_CRA23_02	5.61	55.1	3.11	9.21	0.138	13.2
Met_CRA23_03	6.32	49.8	3.14	9.70	0.154	13.3

Sample ID	DWi	Axb	SG	SCSE	Ai	BBWi
	(kW/m^3)			(kW/t)		(kW/t)
Average	5.83	54.2	3.15	9.34	0.138	13.3
75th or 25th percentile	5.97	52.5	3.18	9.46	0.146	13.4

Source : BaseMet Labs 2023

For the Axb parameter a smaller number generally indicates a more competent material hence the 25th percentile is used as the most onerous result for design purposes. Results show only minor variations in hardness and other characteristics between the samples tested. However, these composite samples are blends of samples from many areas within the mineral body and may hence have yielded largely averaged values for these parameters. Further testing is recommended during future phases using more distinct variability samples to assess the true spread of characteristics within the mineral body.

The hardness parameters measured for the samples place the Čoka Rakita material in the average hardness category. Similarly, at an abrasion index value of around 0.14 the samples can be described as only mildly abrasive.

13.4.6 Gravity Concentration

BaseMet Labs subjected the three (3) composite samples to an Extended Gravity Recoverable Gold (EGRG) procedure comprising staged recovery of gravity gold at increasingly finer grinds. Table 13.10 summarises the results for the concentrate collected at all grinds.

During sample preparation, a 20 kg portion of each master composite was split and stage crushed to 100% passing 1.7 mm to produce a target K_{80} of ~1,200 µm. This material was utilised for EGRG testing. The EGRG test is conducted by passing the entire sample of crushed material through a Knelson MD-3 concentrator at a force of 60 Gs. The concentrate is retained and sized for assay; the tailings are sub-sampled for sizing. This procedure is then repeated using the tailings sample from the preceding pass for another 3 passes through the gravity concentration step.

Table 13.10 – EGRG Results Summary

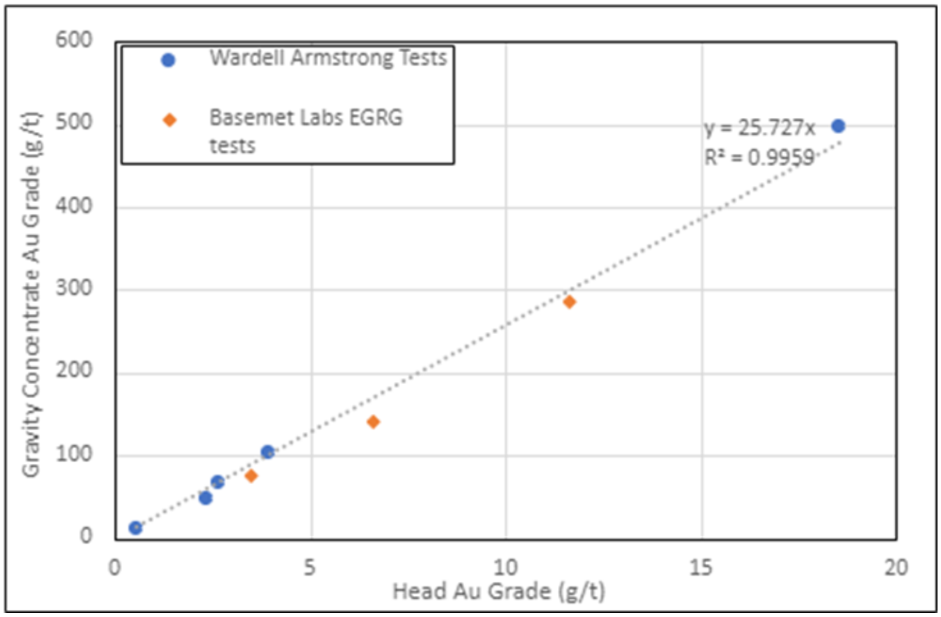
Composite (Test)	Product	Feed Size K_{80} (µm) per Stage	Mass (%)	Assay Au (g/t)	% Au Distribution
Met_CRA23_01 GRG (T-05)	Stage 1 Conc.	1281	0.56	90.0	14.4
	Stage 2 Conc.	228	0.56	127	20.3
	Stage 3 Conc.	96	0.74	70.1	14.9
	Stage 4 Conc.	54	0.65	32.2	6.0
	Tailing		94.3	1.65	44.4

Composite (Test)	Product	Feed Size K_{80} (µm) per Stage	Mass (%)	Assay Au (g/t)	% Au Distribution
	Comb Conc. 1-4		2.52	77.3	55.6
	Feed (calc.)			3.50	
Met_CRA23_02 GRG (T-05)	Stage 1 Conc.	1040	0.51	236	18.3
	Stage 2 Conc.	110	0.54	179	14.9
	Stage 3 Conc.	71	0.61	112	10.4
	Stage 4 Conc.	58	0.58	61.7	5.4
	Tailing		94.6	3.55	51.0
	Comb Conc. 1-4		2.25	143	49.0
	Feed (calc.)			6.58	
Met_CRA23_03 GRG (T-05)	Stage 1 Conc.	1241	0.53	441	19.9
	Stage 2 Conc.	258	0.56	532	25.7
	Stage 3 Conc.	106	0.65	208	11.6
	Stage 4 Conc.	57	0.76	69.2	4.5
	Tailing		94.5	4.71	38.2
	Comb Conc. 1-4		2.50	288	61.8
	Feed (calc.)			11.6	

Source: BaseMet, Labs 2023

Figure 13.4 shows a good correlation between the gravity concentrate gold grade and the head grades for both the Wardell Armstrong and the BaseMet Lab tests. The good correlation coefficient of 0.996 over a wide range of head grades suggests that a constant upgrade ratio can be used with confidence to model the gravity circuit at different gold head grades. For these tests the ratio was 26 but this could vary depending on how the gravity circuit is configured.

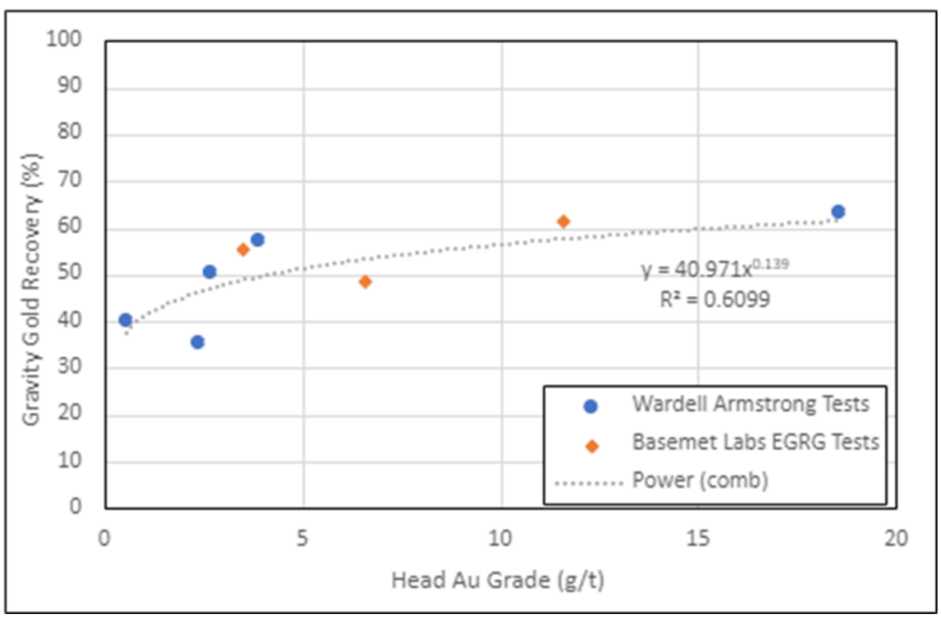
Figure 13.4 – Gravity Concentrate Gold Grade versus Head Grade



Source: DRA 2023

Figure 13.5 shows a similar graph for gold recovery into the gravity concentrate versus gold head grade again combining the results of the two testwork programs into one graph.

Figure 13.5 – Gravity Gold Recovery versus Head Grade



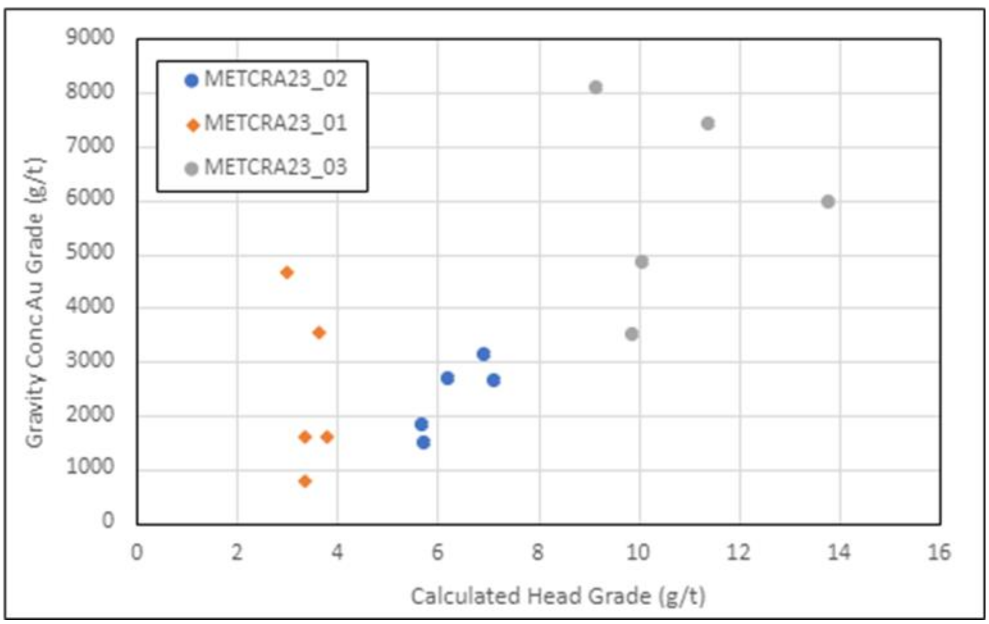
Source: DRA 2023

The correlation is not as strong as for the concentrate grade. This is possibly again due to the presence of nuggets of coarser gold that introduces variability around the mean. Despite the variability the correlation is sufficient to allow it to be used to estimate gravity gold recovery at any head grade within the wide range tested. It yields a small increase in recovery with an increase in head grade for grades above the cut-off of around 3 g/t, which is probably more realistic than simply assuming a constant recovery.

BaseMet Labs also conducted gravity concentration tests prior to all the individual rougher flotation tests. The procedure was also two stages of an initial Knelson (centrifugal) test followed by cleaning of the initial concentrate on a shaking table. These tests yielded very low gravity concentrate mass pulls (around 0.1 % as opposed to the approximately 2% recorded by Wardell Armstrong and BaseMet Lab's own EGRG test). It appears the settings on the shaking table and/or Knelson must have been modified to produce a higher grade gravity concentrate. Because of this change in test procedure, it is not possible to compare these different sets of results directly.

Figure 13.6 shows the graph of gravity concentrate grade versus head grade for this set of tests (at all five (5) grinds per sample). The fit is not as smooth as it was for the data shown in Figure 13.3 (Wardell Armstrong and EGRG tests). Despite the increase in variability, it is still evident that there is a proportional correlation between the two grades. The significant increase in variability is partly due to different grinds, but likely mostly due to the change in methodology which targeted a much smaller mass pull. Concentrating a similar mass of high-density material into a much smaller concentrate mass amplifies the nugget effect which then manifests in amplified variances around the average trendline.

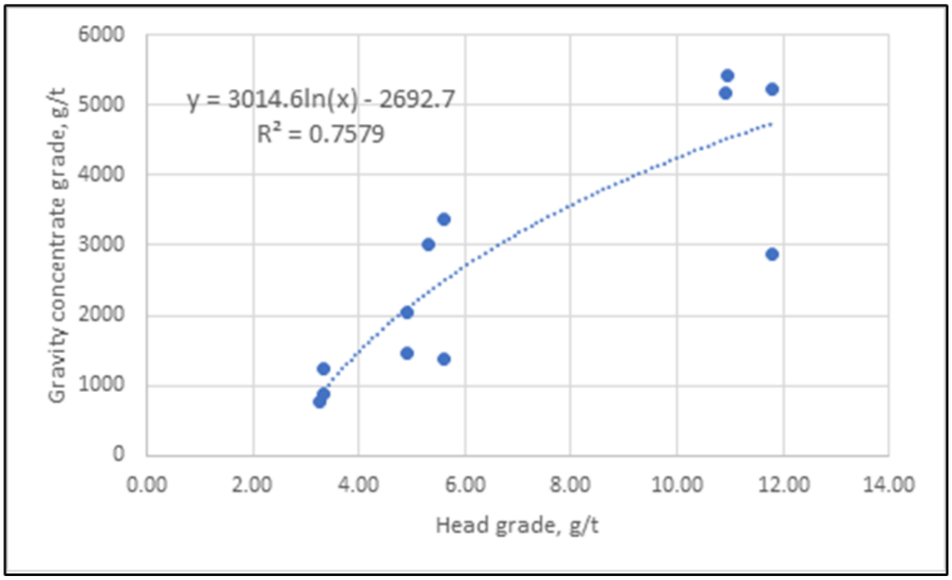
Figure 13.6 – Gravity Concentrate Gold Grade versus Sample Head Grade



Source: DRA 2023

Figure 13.7 shows the gravity concentrate grade versus head grade for all batch flotation tests conducted by BaseMet Labs at a grind of 53 µm for all three (3) samples combined. It is evident that there is still a good correlation between the grades as was reported before.

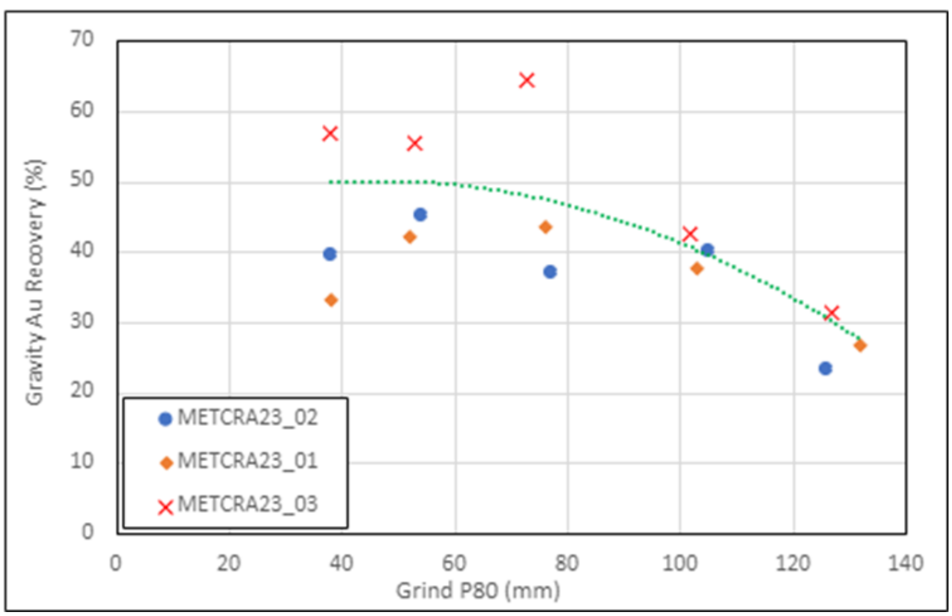
Figure 13.7 – Gravity Concentrate Grade versus Head Grade (53 µm grind)



Source: DRA 2023

Figure 13.8 presents the gravity gold recoveries recorded at different grind sizes for this series of gravity-rougher flotation tests conducted at BaseMet Labs.

Figure 13.8 – Gravity Gold Recovery versus Grind Size



Source: DRA 2023

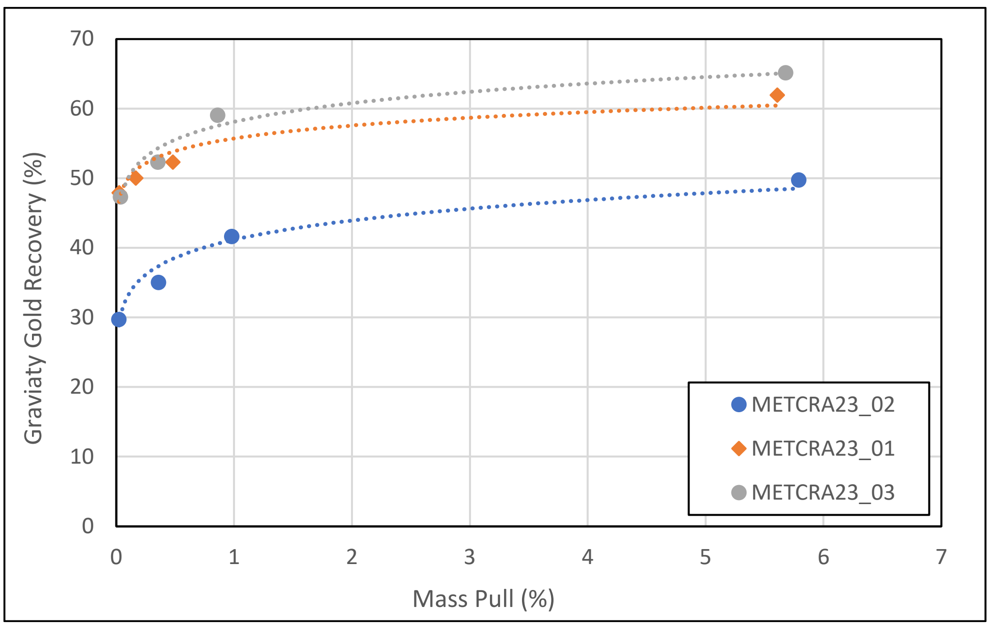
From Figure 13.8, it can be concluded that gravity gold recovery decreases as the grind coarsens above 53 µm. Below 53 µm the recovery of coarse gold appears to be unaffected by the grind size. It appears all gravity recoverable gold is fully liberated at a grind of 80% passing 53 µm.

The highest-grade sample yielded higher recoveries, but recovery appears to be independent of grade for the lower grade samples. This is similar to what was observed during the EGRG and Wardell Armstrong testing, which showed that gravity recovery was mostly independent of grade.

BaseMet Labs performed another set of three (3) gravity gold recovery tests where the initial Knelson concentrate was passed through a Mozley table. The Mozley table was set-up to produce a final concentrate and two (2) middlings products. The results from these tests are presented in Figure 13.9 as a graph of cumulative gold recovery versus mass pull.

The result shows that most of the gravity recoverable gold can be recovered in a small amount of concentrate. Increasing the mass pull beyond 1% leads to diminishing increases in gold recovery. There is no clear correlation between gravity recovery and gold grade as the curves do not show a clear trend from the lowest to highest grades.

Figure 13.9 – Gravity Gold Recovery versus Mass Pull



Source: DRA 2023

13.4.7 Whole-Ore Leaching

BaseMet Labs subjected their three (3) samples to whole-ore cyanidation to test sample amenability to direct cyanidation. The tests were conducted over 48 hours and at a cyanide concentration maintained at 1 g/L to ensure complete dissolution of all leachable gold. All three samples were ground to 80% passing 53 µm.

For these samples the gold recovered through whole-ore cyanidation was better than that of the previous Wardell Armstrong samples with the average grade yielding a recovery just above 92%. The cyanide consumption was also marginally lower but still high by industry standards. The kinetics curves produced (per the BaseMet Labs report) shows that a leach duration of 24 hours is sufficient.

13.4.8 Gravity and Rougher Flotation Test Program

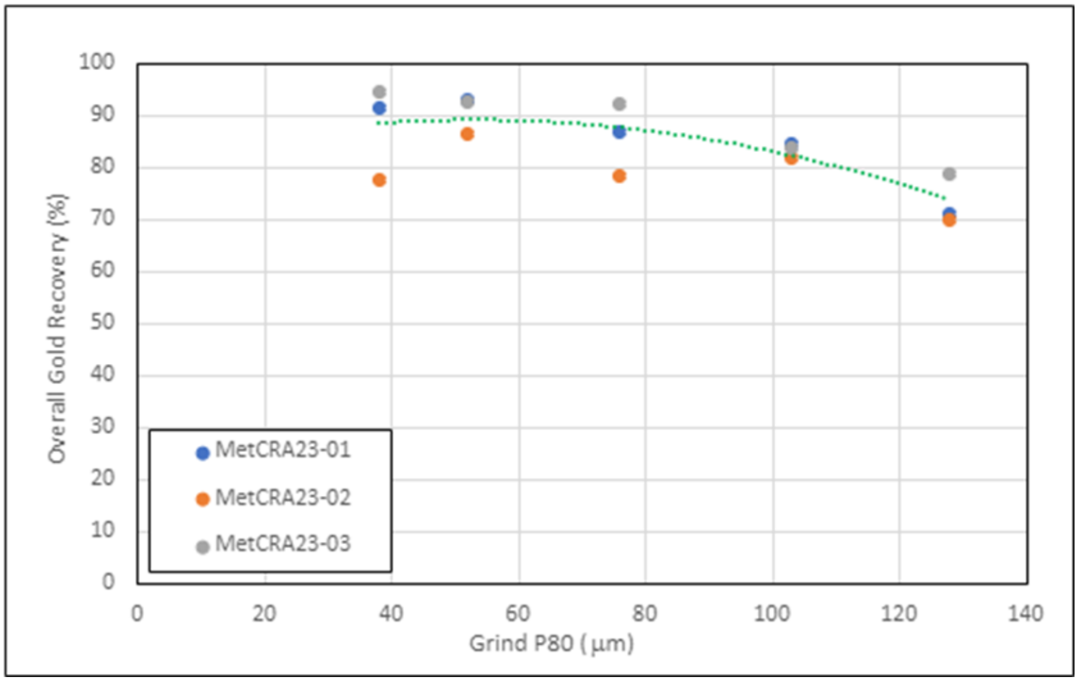
BaseMet Labs subjected their three (3) samples to a series of tests comprising gravity separation followed by rougher flotation. The program investigated, primarily, the role of grinding but also explored extended flotation times and adding more reagent.

The gravity step comprised a centrifugal step (Knelson concentrator) followed by re-concentration of the Knelson concentrate on a shaking table (Mozley table). The Mozley concentrate represents the final concentrate while both the Knelson and Mozley tails were combined to form the gravity tails sample used in downstream rougher flotation tests.

All rougher flotation tests were conducted by first conditioning the sample for 5 minutes with 250 g/t $CuSO_4$. The float test was initiated by adding 100 g/t PAX and some MIBC, which was added throughout the tests as required and was hence not exactly the same for each test. Five rougher concentrate samples were collected after incremental durations of 1, 2, 3, 4 and 5 minutes respectively for a total duration of 15 minutes. Each concentrate was analysed separately.

Figure 13.10 summarises the results by showing the overall gold recovery from both gravity concentration and rougher flotation versus the grind P_{80} size.

Figure 13.10 – Overall (Gravity-Rougher Float) Gold Recovery versus Grind P_{80}



Source: DRA

The green dotted line is the best fit to all data points; it shows an optimum grind around P_{80} of 53 µm. The curve flattens out below 53 µm with no significant further increase in overall gold recovery.

The individual datapoints exhibited significant variation (noise) in overall recovery. Given the high gravity gold recoveries it is suspected that the significant variability is due to the presence of coarse nuggets of gold unevenly distributed throughout the samples. The large variability makes it difficult to see trends clearly (hence the need to fit a curve to the averages of all the data). It is suggested that the nugget effect need to be addressed during future testwork programs by first removing the very coarsest gold in a pre-treatment gravity concentration step from the bulk sample before it is split into smaller sub-samples for downstream testing. The gold removed during this step should then be added to any subsequent gold recoveries to arrive at a total overall gold recovery.

BaseMet Labs also explored the effects of adding additional PAX, extending flotation duration and adding collector A3477 on flotation performance. Table 13.11 summarises the overall (gravity +rougher float) gold recoveries recorded with these changes from the base case test procedure.

Table 13.11 – Summary of Gravity Plus Rougher Flotation Recovery at P_{80} of 53 µm

Test Conditions	Met_CRA23_01	Met_CRA23_02	Met_CRA23_03
Base Case	93.0	86.3	92.6
2x PAX addition	92.5	77.8	93.6
Extended duration	93.0	78.1	94.7
Adding 3477 and Extended Time	96.2	88.8	94.2

Source: BaseMet, Labs 2023

There is no clear benefit with any of the changes made from the base case procedure. However, this may be entirely due to how the nugget effect masks other effects.

13.4.9 CLEANER FLOTATION TEST PROGRAM

BaseMet Labs subjected the three (3) 2023 samples to cleaner flotation testwork to investigate whether this could improve concentrate quality. The effects of regrinding and primary grind size were also explored during these tests. Table 13.12 summarises the results of this set of tests.

Table 13.12 – Gravity-Rougher-Cleaner Tests (2023)

Sample ID	Grind P_{80} (µm)	Regrind	Rougher Concentrate		Cleaner Concentrate	
			Au (g/t)	Recovery (%)	Au (g/t)	Recovery (%)
Met_CRA23_01	75	-	13.7	89.5	24.2	85.3
	53	-	18.1	92.6	31.0	87.5
	53	Yes	15.9	89.9	24.2	84.0
Met_CRA23_01	75	-	28.2	81.6	47.5	78.6
	53	-	24.0	92.0	52.8	88.5
	53	Yes	15.8	82.1	46.9	74.0
Met_CRA23_01	75	-	45.7	91.6	81.3	89.3
	53	-	40.5	93.3	78.0	91.2
	53	Yes	37.8	92.1	71.9	88.5

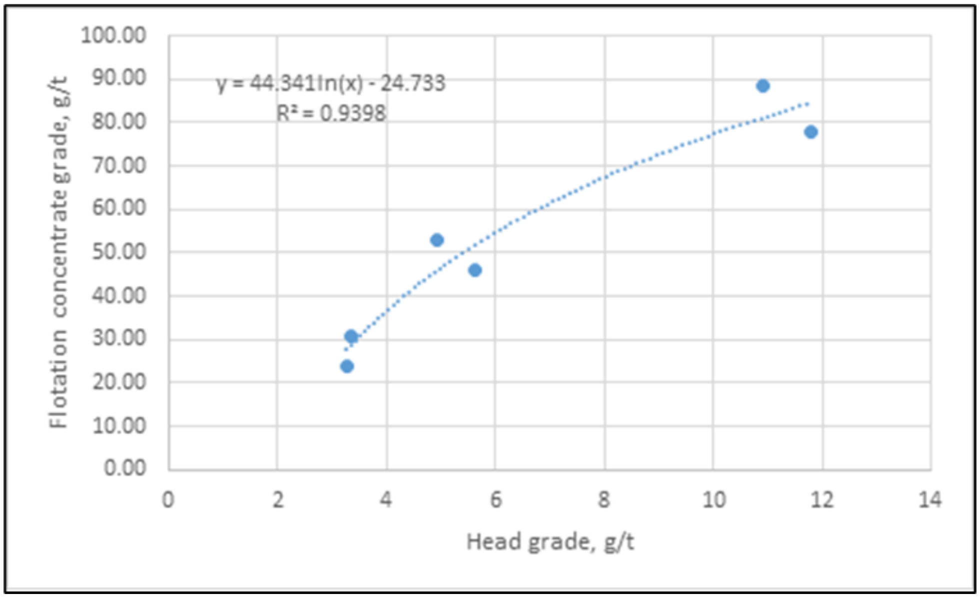
Source: BaseMet, Labs 2023

All the tests showed that cleaning can be used to improve concentrate grade but at the expense of some recovery. The cleaner concentrate grades are about double the rougher concentrate grades while recoveries decreased by approximately 4% points. A trade-off calculation is required to determine whether the improvement is worth the revenue sacrificed.

The results with regrinding were generally worse than with no regrinding which is contrary to expectations. Decreasing the primary grind from 75 µm to 53 µm was beneficial, particularly for sample Met_CRA23_02 for which the recoveries improved by approximately 10%. Further testwork is required to validate the optimum grind.

Figure 13.11 shows the cleaner concentrate grade profile versus the sample head grade. With a good proportional correlation between these and the mass pull largely independent of grade a constant recovery can be assumed that is independent of grade.

Figure 13.11 – Flotation Cleaner Concentrate Grade versus Head Grade

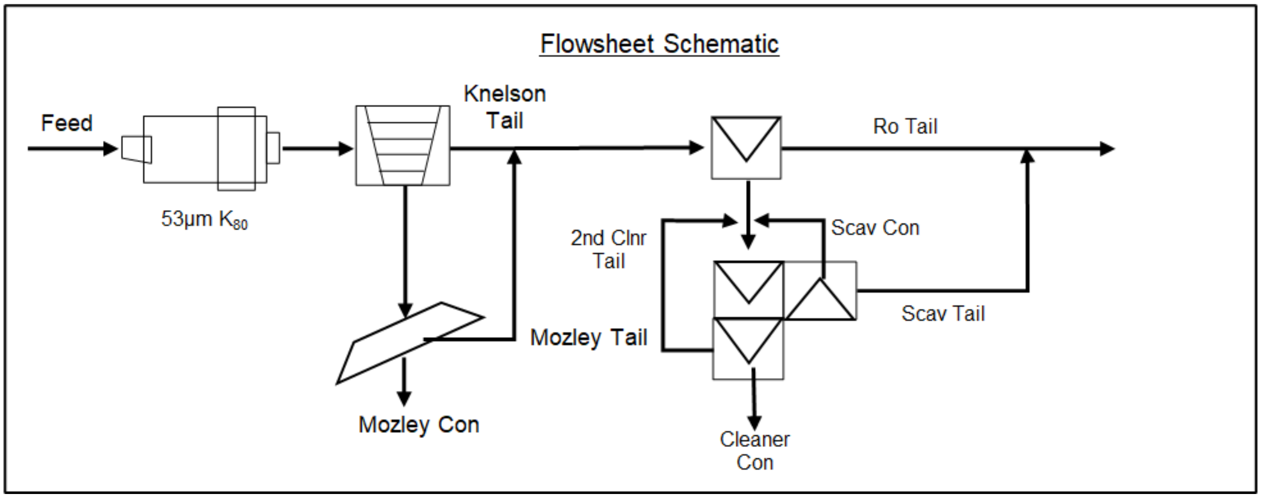


Source: DRA 2023

13.4.10 Flotation Locked Cycle Tests

BaseMet Labs subjected the three (3) 2023 samples to standard flotation locked-cycle testwork to investigate whether the closed cleaner circuit would improve the final concentrate grade. Figure 13.12 shows a schematic of the procedure employed (courtesy of BaseMet Labs).

Figure 13.12 – Schematic of Locked-Cycle Test



Source: BaseMet Labs 2023

Three (3) LCTs were performed, one for each of the 2023 samples. The sample was first ground to 80% passing 53 µm in a lab grinding mill. The ground sample was then passed through two stages of gravity concentration with the Mozley table concentrate collected and analysed as the final gravity concentrate product. Tails from both gravity steps were combined and used as feed for the rougher flotation step. Test conditions were as follows:

- 5 minutes conditioning of rougher feed with 250 g/t $CuSO_4$;
- 125 g/t PAX and 125 g/t A3477 collector addition to roughers;
- Total rougher float time = 30 minutes;
- Cleaner 1 duration = 15 minutes, and cleaner 2 duration = 10 minutes;
- Cleaner scavenger = 6 minutes;
- Add 20 g/t PAX and 3477 to cleaner 1;
- Add 10 g/t PAX and 3477 to cleaner 2 and cleaner scavengers; and
- MIBC addition varied depending on visual assessment of the froth.

Table 13.13 summarises the mass and pertinent gold data obtained from the LCTs.

Table 13.13 – Locked Cycle Test Results

Sample ID	BL1291-52 Met_CRA23_01			BL1291-53 Met_CRA23_02			BL1291-54 Met_CRA23_03		
	wt%	Au (g/t)	Au Distribution %	wt%	Au (g/t)	Au Distribution %	wt%	Au (g/t)	Au Distribution %
Mozley Conc	0.21	718	48.4	0.17	1318	41.7	0.14	3867	53.3
2nd Clnr Conc	5.3	23.2	39.7	4.7	46.4	41.7	3.8	100.5	36.8

Sample ID	BL1291-52 Met_CRA23_01			BL1291-53 Met_CRA23_02			BL1291-54 Met_CRA23_03		
	wt%	Au (g/t)	Au Distribution %	wt%	Au (g/t)	Au Distribution %	wt%	Au (g/t)	Au Distribution %
Clnr Scav Tail	6.1	2.03	4	5.9	3.8	4.3	4.8	7.7	3.5
Ro Conc	11.4	11.9	43.6	10.6	22.8	45.9	8.6	49.0	40.3
Ro Tail	88.3	0.28	7.9	89.3	0.7	12.4	91.3	0.7	6.4
Flotation Feed	100	3.12	100	100	5.3	100	100	10.4	100
Combined Concs	5.5	49.7	88.1	4.9	89.7	83.3	4.0	237.2	90.1
Combined Tails	94.4	0.35	11.9	95.1	0.8	16.7	96.0	0.82	9.9

Source: BaseMet Labs 2023

The combined overall gold recoveries were 88.1%, 83.3% and 90.1% respectively for the three (3) samples. The average overall gold recovery is close to 87% which is consistent with previous test results. The lower recovery recorded for the intermediate grade sample (Met_CRA23_02) is probably due to the lower amount of gravity recoverable gold present in this sample. Only 41.7% of the gold in this sample reported to the gravity concentrate compared to 48.4% and 53.3% for the other two (2) samples.

Rougher tails contained 12.4% of the 16.7% gold reporting into the combined tails while cleaner scavenger tails contributed only 4.3%. It is recommended that future testwork should explore why this gold was not recovered by rougher flotation. This could be done by subjecting the tails sample to a diagnostic leach procedure which can be used to identify whether the gold lost to tails was locked within non-sulphide minerals and/or whether this could still be recovered through cyanidation.

The final flotation concentrate mass pull was around 4% to 5% and the grades are not as high as was hoped for. The implication of this is that transport costs could be high. Future testwork programs should explore ways of upgrading the final product on site. Improvements to the flotation circuit should also be explored, e.g., introducing a regrind step to improve liberation and/or more efficient flotation devices to improve selectivity.

13.4.10.1 Analysis of Concentrate Samples

The final concentrates from the LCTs as well as the gravity concentrate were analysed for elemental constituents. Table 13.14 summarises the most significant elements:

Table 13.14 – Analysis of Concentrate Samples

Element	Units	Met_CRA23_01	Met_CRA23_02	Met_CRA23_03	Gravity Conc
Ag	ppm	9.2	15.3	10.4	42.2
As	ppm	1,522	1,447	983	778
Cu	ppm	17,821	20,461	21,051	2774
Fe	%	42.8	40.56	41.23	47.1

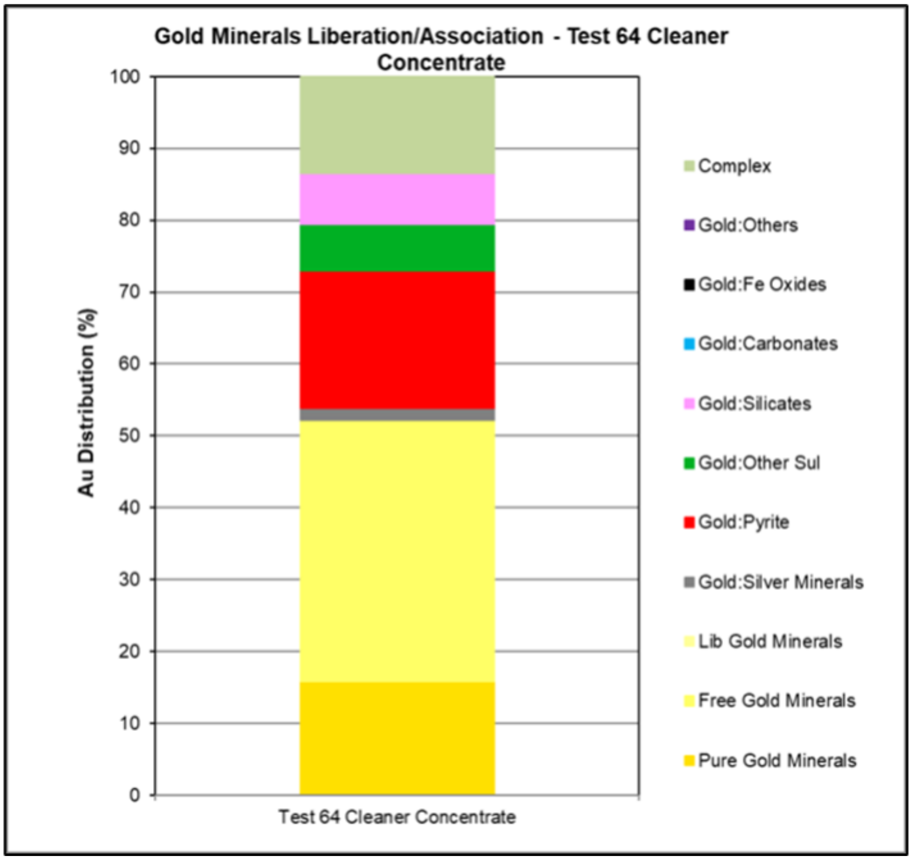
Source: BaseMet, Labs 2023

The silver contents of the three (3) flotation concentrates are too low for a revenue credit as it is only payable above 30 g/t. The gravity concentrate may qualify. Arsenic is present but below the penalty threshold of 2,000 ppm. The high copper and iron contents are as expected and due to their association with sulphur.

X-ray diffraction analysis of the primary concentrate sample from test BL1291 yielded a 97.6% pyrite content with only traces of other minerals (magnetite, quartz, garnet, and galena). X-ray diffraction analysis of the sample labelled BL1291-64 Cl Con (the final concentrate) yielded 81% pyrite and 11% chalcopyrite as the dominant minerals with all other minerals contributing less than 3%.

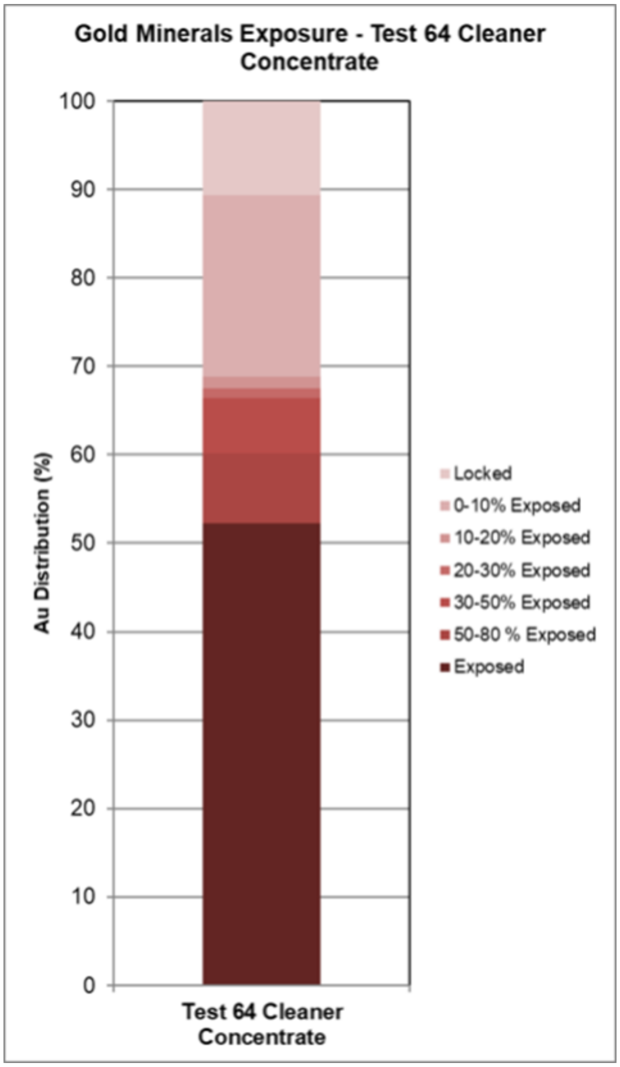
The sample of cleaner concentrate from test 64 (BL1291-64) was also subjected to a QEMscan analysis to determine liberation, exposure and gold association. Figure 13.13 and Figure 13.14 are reproduced here from the BaseMet Labs report. From Figure 13.14, only 10% of the gold is locked which explains why the sample was amenable to cyanidation. More than half of the gold in the sample was fully exposed.

Figure 13.13 – Gold Minerals/Association for a Sample of Cleaner Concentrate



Source: BaseMet Labs 2023

Figure 13.14 – Gold Exposure in a Sample of Cleaner Concentrate

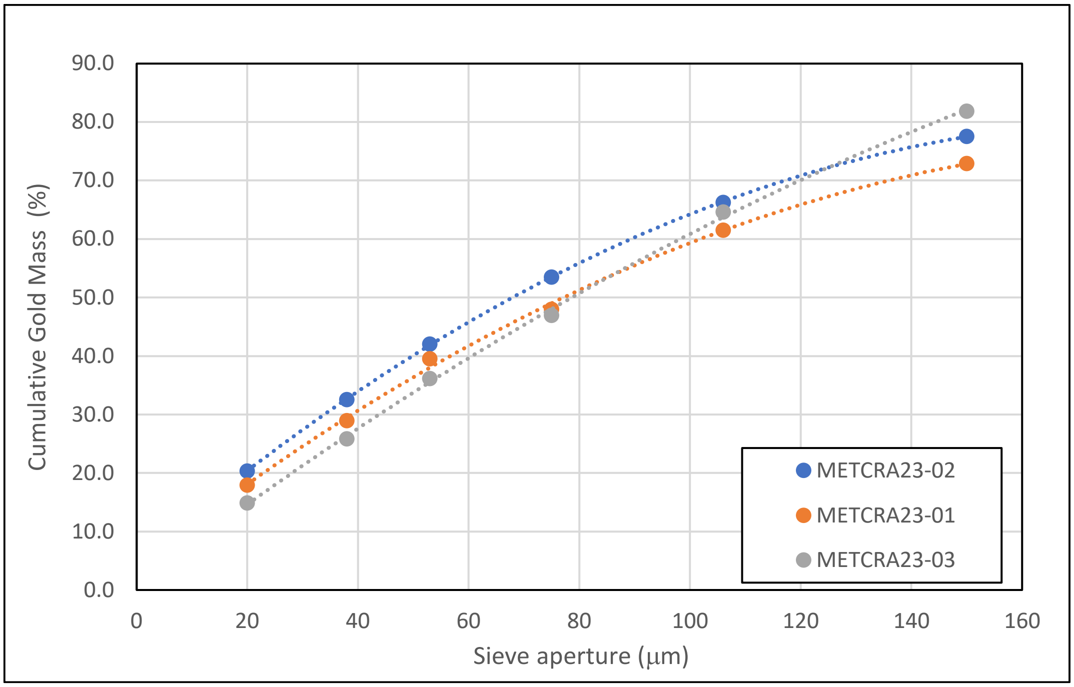


Source: BaseMet Labs 2023

13.4.10.2 Gold Distribution by Size in Feed and Tails Samples

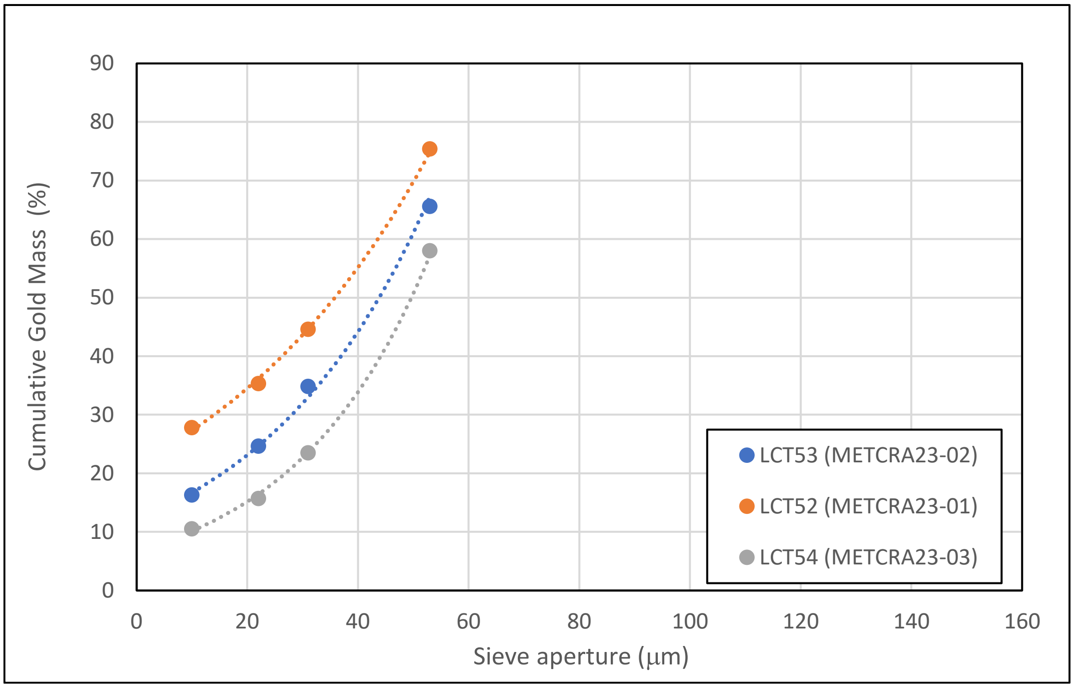
The three (3) feed samples and three (3) LCT tails samples were screened into size fractions and then each size was analysed for gold grade. Figure 13.15 and Figure 13.16 illustrate the cumulative gold mass versus size curves for these samples.

Figure 13.15 – Cumulative Gold Distribution by Size Class for the LCT Tests Feed Samples



Source: DRA 2023

Figure 13.16 – Cumulative Gold Distribution by Size Class for the LCT Tests Tails Samples



Source: DRA 2023

The analysis shows that gold is evenly distributed in the feed sample across the range of screen sizes used in this analysis. It also shows an almost identical curve for all three (3) feed samples despite the significant difference in head grade. The implication is that the increase in grade is not simply due to more coarse gold for the higher grade but an even increase across all size ranges.

The tails graphs were plotted on the same scale as the feed to clearly show how the curves for the tails samples have shifted to the left (finer). The shift indicates that the gold lost to tailings are mostly in the fine sizes and hence that the process was more efficient at recovering coarser gold (as expected). This result also suggests that flotation technologies such as the Jameson cell would be beneficial as these more modern cells are known to be more efficient at floating finer particles.

13.4.11 CYANIDATION OF LOCKED CYCLE TAILS

The two (2) rejects products from each of the three LCTs were combined and subjected to a 48 hour cyanidation. Table 13.15 summarises the results.

Table 13.15 – Cyanidation of Flotation Tails (Combined Rougher Tails and Cleaner Scavenger Tails)

Test Condition	Met_CRA23_01	Met_CRA23_02	Met_CRA23_03
Calc. Combined Tails Grade	0.19	0.39	0.42
Leach/Final Residue Grade	0.12	0.19	0.18
Leach Stage Extraction	36.8	51.3	57.1
ROM Feed Grade (from LCT)	3.1	5.3	10.4
Gravity+Float+Leach Recovery %	96.2	96.4	98.3

Source: BaseMet,Labs 2023

A significant amount of gold was leached into solution from the flotation tails samples. Calculating the overall recovery for the entire flowsheet using the final residue grades and head grade for the LCT tests yields overall recoveries exceeding 96% for all three samples. While this step would increase gold recovery significantly it would also introduce cyanide to the site which is currently not considered. A decision has been made to avoid the use of cyanide on site.

The average cyanide and lime consumptions recorded during these tests were 0.4 kg/t NaCN and 1.9 kg/t lime. Based on the kinetic curves and the reported solution analyses a total leach duration of 24 hours is sufficient.

Comparing the 96% recovery against the 88% recovered through whole material leaching it appears that the gold recovered into the flotation concentrate was locked in sulphides and not leachable. This is deduced from the fact that the flotation step is the key difference between the two flowsheets as it can be assumed all gravity gold will also have leached in the 48-hour leach.

13.4.12 CYANIDATION OF GRAVITY CONCENTRATES

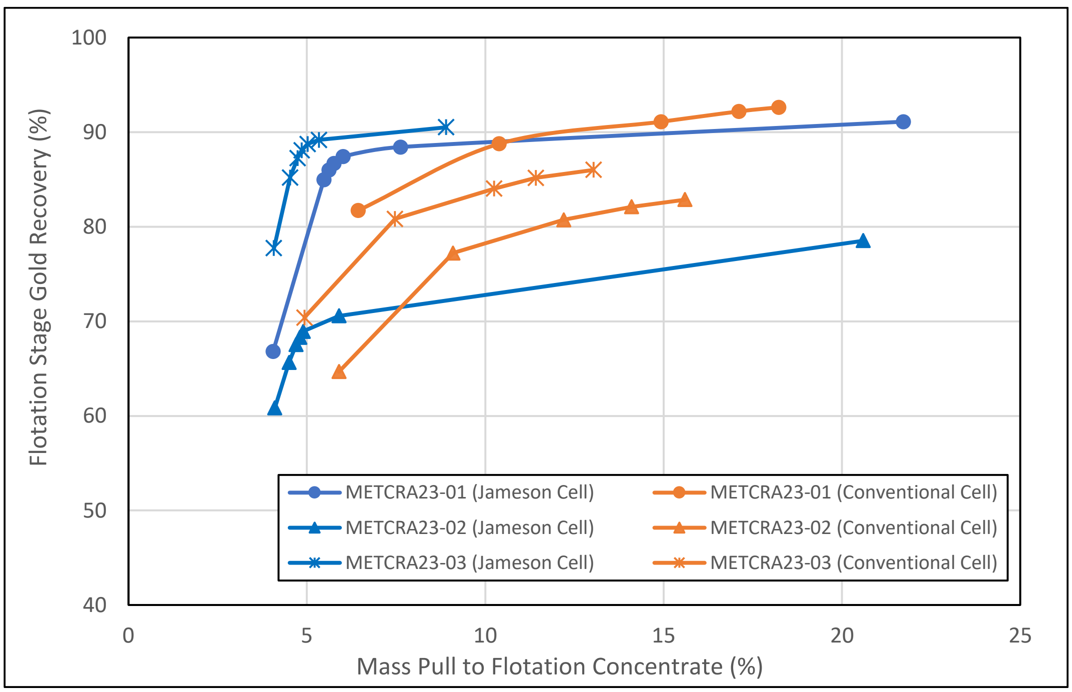
Bulk 20 kg samples of the three (3) metallurgical samples were processed using the locked cycle procedure to generate bulk concentrate samples for subsequent cyanidation testing. Cyanidation of the bulk gravity concentrates were performed at 4:1 liquid to solid ratios, a cyanide addition of 5,000 ppm NaCN and with oxygen sparging.

The results showed that near complete leaching is achievable (99.8% to 99.9%) after 72 hours of cyanidation. Leach kinetics were slow with about 7% of the leaching occurring in the final interval from 48 hours to 72 hours. Cyanide consumptions ranged from 30 kg/t to 80 kg/t but this is possibly due to the oxidation to cyanate given that oxygen was sparged through the sample throughout. The result shows that production of doré bars would be feasible if cyanide is allowed on site.

13.4.13 JAMESON FLOTATION CELL TESTS

BaseMet Labs performed three (3) tests using a procedure designed to mimic the operation of Jameson flotation cells. The tests were preceded by the normal gravity step to remove coarse gold ahead of the flotation procedure. Figure 13.17 presents the stage recovery of gold from the gravity tails to the concentrate versus mass pull to the flotation concentrate for the three samples tested.

Figure 13.17 – Gold Stage Recovery to Flotation Concentrate versus Mass Pull



Source: DRA 2023

It is evident from the above that the Jameson cells can be expected to yield superior recoveries at lower mass pulls i.e. it will produce a higher-grade concentrate product.

Unfortunately, the gravity recoveries preceding the flotation steps were vastly different for the individual tests. For Met_CRA23_01 the gravity recovery preceding the conventional procedure was 47.9% whereas the gravity step preceding the Jameson test removed only 28.4% of the gold from the sample tested. The gravity recoveries for the Met_CRA23_02 samples were similarly different. The consequence of this is that the overall recovery from gravity plus flotation yielded better results for the conventional cell tests. However, this is solely due to the abundance of coarse gold in the two sub-samples used in the control tests. This large difference in gravity recovery makes it difficult to compare the performance of the subsequent flotation steps and to arrive at a reliable recovery benefit for the Jameson cells; more testing is recommended.

13.4.14 Sedimentation Testing

BaseMet Labs conducted scoping static settling tests as well as dynamic tests using both tailings and concentrate samples. A description of the additional testing, performed specifically to support the design of the paste plant, is presented in Section 18.

The main objective with the scoping tests was to select the best flocculant for the material. Static tests were conducted at pH 8 and using 5 different flocculants and showed that the AN913 flocculant outperformed the other flocculants, especially for the tails sample.

13.4.14.1 Static Testing

Static testing was performed for both the tailings and concentrate samples using different dosages of the selected flocculant. For the tailings sample, static testing flocculant dosage rates of 30 and 60 g/t, pH modification and a final test using coagulant were explored. Two (2) flocculant dosage rates were explored with the concentrate sample.

With respect to the tailings sample the following outcomes were observed:

- Increasing flocculant dosage from 30 to 60 g/t increased underflow density from 55 to 58% w/w solids;
- Increasing pH had minimal impact on underflow density but improved solution clarity; and
- Adding coagulant significantly improved the clarity of the supernatant solution.

With respect to the concentrate sample static settling tests, it was observed that a lower flocculant dosage of 10 g/t produced a higher density of 71% compared to 65% w/w solids for 20 g/t flocculant.

13.4.14.2 Dynamic Testing

The results of dynamic settling tests are summarised in Table 13.16; due to the small sample size only one test was performed using the concentrate sample. The tests were all performed at pH 8, a grind of 80% passing 53 µm, feed density of 15% w/w solids and with coagulant added.

Turbidity was low for all tests due to the use of the coagulant. Over the range of loading rates tested underflow densities exceeded 60% w/w solids, which is an acceptable density for a filter feed stream. The underflow density achieved with the concentrate sample was exceptionally high at 76.4% w/w solids. A solid loading rate of 1 tph/m^2 can be used for the sizing of both thickeners as this rate allowed underflow densities exceeding 60% w/w solids, while also producing acceptably clear overflows. The only issue of mild concern is the underflow sample viscosities, which ranged from 67 Pa to 153 Pa, and may be indicative of a slurry that is difficult to pump. However, it should be noted that these were recorded at high underflow densities, and it is expected that a decrease to say 55% w/w solids may result in much less viscous slurries. Shear thinning can also be employed to remedy this situation if required.

Table 13.16 – Dynamic Settling Tests

Sample	**Loading Rate** (tph/m^2)	**Flocculant Addition** (g/t)	**Rise Rate** (m/h)	**Turbidity** (ppm)	**Underflow %solids**	**Viscosity** (Pa·s)
Tails	0.5	50	3.11	43	61.8	67
Tails	0.7	50	4.35	41	61.0	153
Tails	1.0	50	6.21	52	60.6	98
Tails	0.7	30	4.28	97	61.6	129
Tails	0.7	30	4.28	32	59.8	119
Conc	1.0	20	5.97	123	76.4	88

Source: Basemet Labs 2023

13.4.15 Filtration Testing

BaseMet Labs performed filtration tests on samples of the combined flotation tailings and a final concentrate sample to generate information for the sizing of filtration equipment. The tests involved filling chambers with slurry (forming phase) followed by pressing the chambers together, followed by blowing air through the cake to remove additional filtrate. For each phase the amount of filtrate recovered was measured allowing the calculating of cake moisture content and filtration rate data as summarised in the Table 13.17. The starting densities of the slurries were around 60 to 70% w/w solids in both cases. The second air blow duration applied for the concentrate was three (3) minutes for the tails test and six (6) minutes for the concentrate test.

Table 13.17 – Filtration Tests Summary

Sample	**Parameter**	**Formed**	**Pressed**	**Air Blow (1 min)**	**Air Blow (3 to 6 min)**
Tails	Filtration rate (kg/h/m^2)	1,649	1,064	449	349

	Moisture Content (%)	28.8	22.7	18.5	16.3
Conc	Filtration rate (kg/h/m^2)	723	481	361	110
	Moisture Content (%)	16.0	13.0	4.1	2.9

Source: BaseMet Labs 2023

The results showed that a much smaller press can get the tails dewatered to less than 23% moisture which may be adequate for conveying to a dump and stacking (to be determined by slope stability testing). If this is not sufficient then air blowing can be used to reduce the cake moisture content to around 16% if required.

The concentrate dewatered to a low moisture content of 13% by pressing only. Air blowing was very efficient reducing the cake moisture content to below 5%.

13.4.16 Environmental Testing

BaseMet Labs tested the acid generating potential of the combined final tailings sample from test BL1291 and also analysed it for acid-base accounting.

The analysis yielded an acid potential (AP) value of 2.5 in equivalent kg $CaCO_3$/t and a neutralising potential (NP) of 107.3 kg $CaCO_3$/t. This yields a net neutralising potential (NNP) of 104.8 equivalent kg $CaCO_3$/t. The ratio of neutralising to acid generating potentials is substantially in favour of neutralisation.

A single addition (EGi Method) acid generation test was conducted on the sample. This involves contacting the sample with a strong oxidant (H_2O_2) and measuring the acid generated. The pH of the solution at the conclusion of the test was measured as 7.95 units which shows that there was no acid generated. The net acid generating potential at a pH of 4.5 was recorded as 0.0 equivalent kg H_2SO_4/t.

13.5 Process Design Criteria Derived from Test Results

- Axb = 54.2 (ave) and 52.5 (design);
- Ai = 0.138 (average for Opex);
- Bond Ball Mill Work Index = 13.3 kWh/t (ave) and 13.4 kWh/t (design);
- Optimum primary grind for gravity plus rougher flotation route is 53 µm – more testing may change this value;
- The following LCT test conditions were adopted as design criteria for the flotation circuits:
    - 5 minutes conditioning with 250 g/t $CuSO_4$;
    - 125 g/t PAX and A3477 collector addition to roughers;

- Total rougher float time = 30 minutes;
- Cleaner 1 duration = 15 minutes;
- Cleaner scavenger = 6 minutes;
- Cleaner 2 duration = 10 minutes;
- Add 20 g/t PAX and 3477 to cleaner 1; and
- Add 10 g/t PAX and 3477 to cleaner 2 and cleaner scavengers;

- Thickener solids loading rate of 1 tph/m^2 can be used with underflow densities of 60% solids;
- Flocculant addition rates of 50 g/t for tailings thickener and 20 g/t for concentrate thickener.

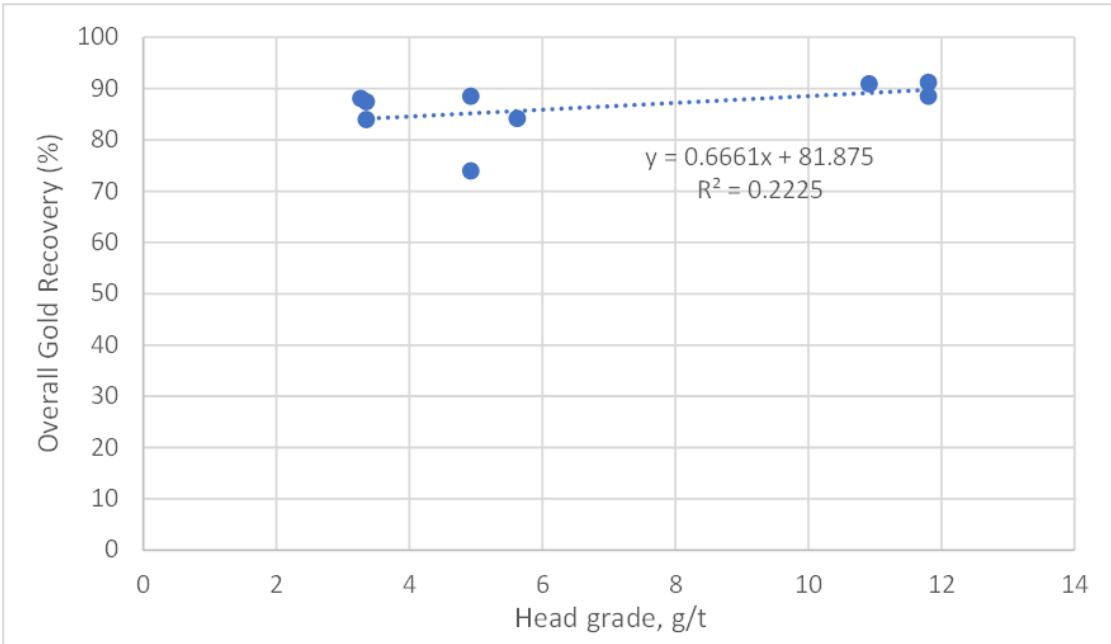
13.6 Recovery Model

With respect to gravity concentration two (2) observations were made from the test results that have been adopted into the design criteria to predict the gravity gold recovery and concentrate grade:

- The different sets of tests all yielded good correlations for a constant upgrade ratio from head gold grade to gravity concentrate gold grade. The actual value of this upgrade ratio would depend on how the gravity concentration step was performed and varied considerably from set to set of tests. For the process design criteria an upgrade ratio was selected that closely mimics the locked-cycle test results as these tests were also used as design criteria for the downstream flotation section.
- Gravity Recovery = 41 x [Head Au]^0.14 with [Head Au] in g/t and Recovery as %. With both the upgrade ratio and gravity recovery defined the concentrate mass pull can be calculated at different head grades (it should be in the vicinity of around 2%)

With respect to overall gold recovery there is no clear correlation that could be established. Figure 13.18 shows the results for all tests performed by BaseMet Labs for the set of cleaner batch tests and LCTs that involved a grind of 53 µm. It is evident that the correlation is weak and that the most appropriate model for overall gold recovery is to adopt a simple constant recovery of approximately 86 to 90%.

Figure 13.18– Overall Gold Recovery versus Head Grade



Source: DRA 2024

13.7 Deleterious Elements

Arsenic was present in the final product (concentrates) samples. For the three (3) samples tested by BaseMet Labs the arsenic concentration in the final flotation concentrate ranged from 983 ppm to 1,522 ppm. The gravity concentrate contained 778 ppm arsenic. These are all below the 2,000 ppm threshold for attracting penalties; nonetheless arsenic levels will continue to be monitored in future testwork programs.

All other deleterious elements were insignificant.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

The maiden MRE for Čoka Rakita was estimated in November 2023 using drilling data collected since 2009 but dominated by data collected in 2021 and 2023 when the mineralised skarn, that is the focus of this Report, was identified as a target with gold potential. The MRE was estimated by QP, Maria O'Connor with support from other CSA Global geologists. The model has gold, silver, copper, arsenic and sulphur estimated in both mineralised and non-mineralised (waste) domains. This section focuses on the estimation of gold and silver as reported in the MRE.

14.2 Database Cut-Off

The database cut-off is 16 November 2023, which is also the effective date of the MRE. The QP started work on a preliminary dataset in September 2023 to explore the data, lithogeochemistry, geological and mineralisation interpretations, and establish estimation methodology, with the assumptions checked and updated on receipt of the final database.

The following data was provided as a set of comma-separated values (csv) files, exported from the acQuire database managed on-site by DPM geologists:

- RA_Alteration.csv
- RA_Assay_BM.csv
- RA_Assay_ME.csv – for gold, SFA used in preference to FA where available
- RA_BulkDens.csv
- RA_Collar.csv
- RA_Geotech.csv
- RA_Lithology.csv
- RA_MagSusc.csv
- RA_Screen Fire Assay.csv
- RA_Spectral.csv
- RA_Structure.csv
- RA_Sulphides.csv
- RA_Survey.csv
- RA_Vein.csv

In addition, three (3) files were provided as exports from Leapfrog (where the interpretation was undertaken):

- collar.csv
- survey.csv
- assay.csv

Files were loaded into Datamine and subjected to a series of validation checks. The collar files from the database were compared against those exported from Leapfrog and differences were compared. Differences in collar records for 16 holes were identified. DPM confirmed that the collar (pick-up) surveys had been updated for the data in Leapfrog, and therefore the coordinates from Leapfrog were used in preference to the csv exports.

Table 14.1 – Summary of Collar Data Imported

Hole Type	Year	Number of Holes	Total Metres
Diamond	2009	9	1,319
	2016	1	588
	2017	1	225
	2020	3	2,298
	2021	27	17,190
	2023	60	37,678
Diamond – subtotal		*101*	*59,298*
Diamond tail	2023	24	13,469
Diamond tail – subtotal		*24*	*13,469*
RC	2008	4	474
	2023	44	7,483
RC – subtotal		*48*	*7,957*
TOTAL		**173**	**80,723**

Queries identified during the load-up and validation process for relevant data were discussed with DPM. Outcomes are summarised as follows:

- Four (4) drillholes had zero depth recorded in the database – RIDD007B, RIDT012, RIDT014, RIDT039. These were terminated due to technical drilling reasons.
- Seven (7) drillholes were in progress – RIDD047, RIDD048, RIDD049, RIDT029, RIDT030A, RIDT035, RIDT042.
- Thirty-Seven (37) holes had no samples/assays recorded:
    1. These were holes that were terminated early (RIDD007B, RIDT012, RIDT014, RIDT039);
    2. In progress (RIDT030A); or

3. Waiting to be sampled (RIDD021A, RIRC013, RIRC014, RIRC016, RIRC017, RIRC018, RIRC019, RIRC020, RIRC021, RIRC022, RIRC023, RIRC024, RIRC025, RIRC026, RIRC027, RIRC028, RIRC029, RIRC030, RIRC031, RIRC032, RIRC033, RIRC034, RIRC035, RIRC036, RIRC037, RIRC038, RIRC039, RIRC040, RIRC041, RIRC042, RIRC043, RIRC044).

- All drillholes had downhole survey records and geotechnical logging.
- All completed drillholes had logged geology and alteration except for 44. Many (31) of those holes with unlogged geology are RC pre-collars, meaning only 13 diamond drillholes do not have logged geology/alteration.
- There are 53 diamond drillholes with no density measurements; however, there is a large number of density measurements that are spatially and materially representative of the waste and mineralisation, therefore the QP author does not consider this an issue.

Overall, the drillhole database is clean and as complete as possible given drilling continues on the Project and the database received is a snapshot in time. It is therefore reasonable that some drillholes contain missing logging data as they wait to get processed.

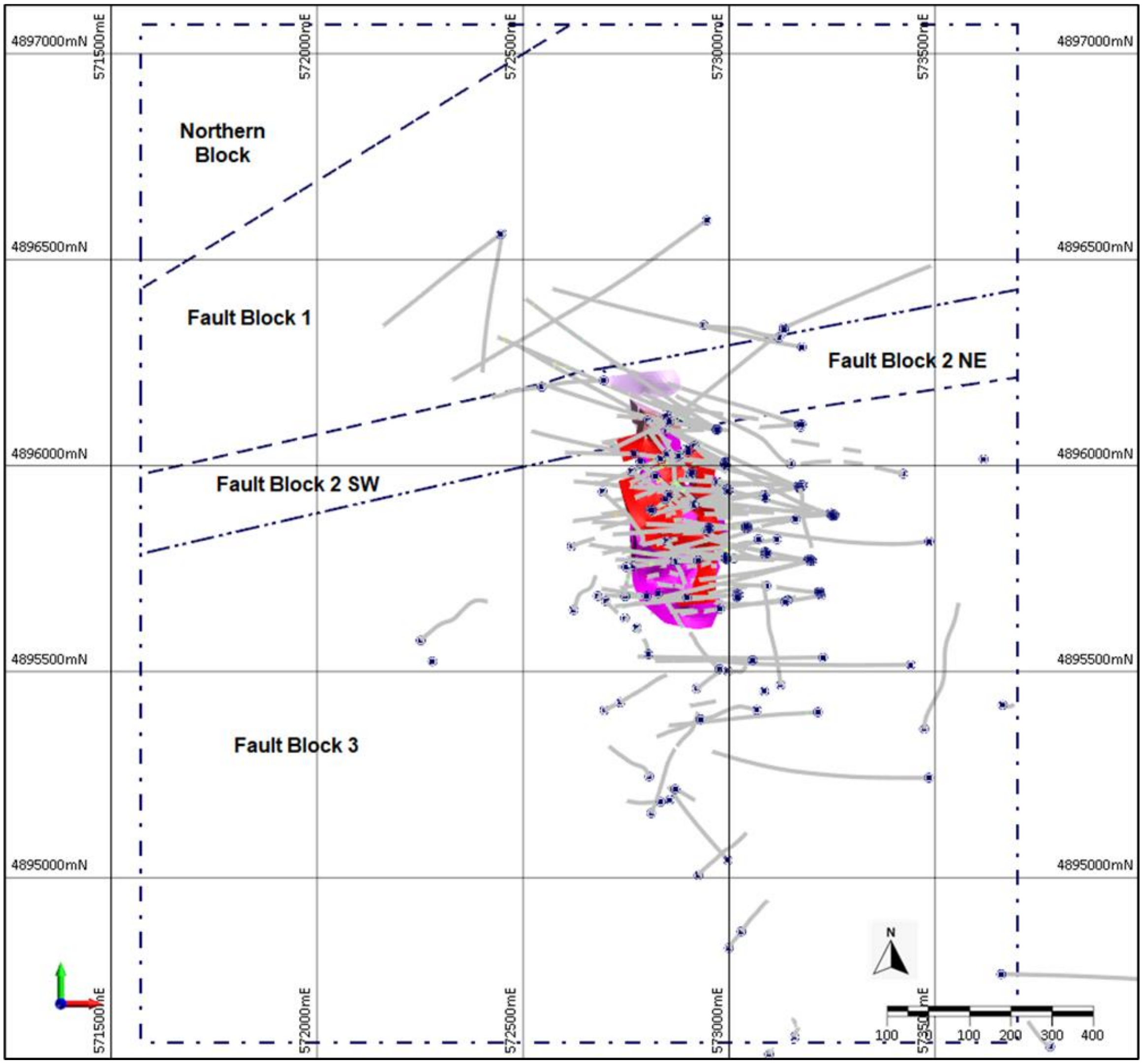
14.2.1 Data Excluded

Trenches were excluded both because of limited quality control but more significantly because they are surface samples and are therefore not relevant to the skarn mineralisation modelled for this MRE. No other data was excluded.

14.3 Preparation of Wireframes

Geological interpretation by DPM geologists takes place in Leapfrog on an ongoing basis as new drilling becomes available. The model was constructed within a group of faulted blocks (Figure 14.1). Faults were interpreted by the DPM geological team through a combination of mapping, drillhole logging and geological inference. Major offsetting faults that crosscut the area were used to define these blocks.

Figure 14.1 – Plan View Showing Location of Fault Blocks in Relation to Mineralisation Wireframes



Source: CSA Global, 2023

The Leapfrog Geo project containing the lithology, structure and oxidation model for the area was provided by DPM. The QP author reviewed and validated the models and made small adjustments to them to either improve their construction locally or to facilitate their use in statistical analysis of data and mineral resource estimation.

The following minor issues were noted and corrected (unless otherwise stated) under the supervision of the QP prior to using the models in the MRE:

- Inconsistent snapping to drillholes – the majority of drillhole intervals were correctly snapped to the drillhole intersections, but a small number were not. This was adjusted to ensure drillholes were snapped to for all modelled units. When checked, this was due to inconsistent snapping being used for each lithological unit being built (using custom snapping).

- Cross cutting lithologies – where no contact point is created, at the end of the hole or as a unit pinches out in drilling, control points were included to control the 3D volumes, so they do not crosscut drillholes containing other lithologies. This only occurred away from the MRE area of interest, so no corrective action was taken.

The following changes were made to enable the statistical evaluation of data within the exported wireframes:

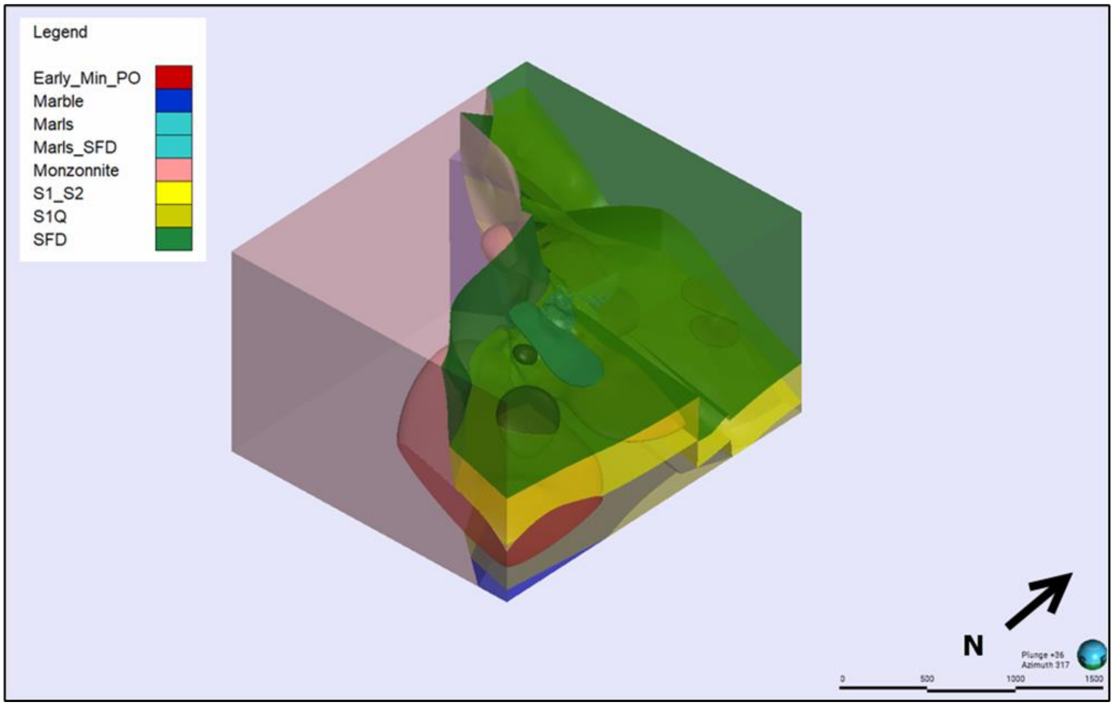
- The lithology model was exported individually so that the mineralisation wireframes could be reviewed by lithology, which was not possible to do when mineralisation was integrated into the lithology model.
- Removal of the topography used to constrain the upper limit of the model to remove any misalignment in the resolution of the topography and model.
- Models were built within each of these faulted blocks and the output wireframes were combined into a single wireframe for easier use in later software packages.

14.3.1 Lithology and Structure

The lithology model is shown in Figures 14.2 and 14.3 and comprised the following units, listed from youngest to oldest:

- Early mineralised porphyry (Early_Min_PO) – modelled as an intrusive body with a moderate dipping trend towards the east-northeast, trends varied slightly between faulted blocks, using an elongate spheroidal ellipse.
- Monzonite – modelled as an intrusive body using a spherical model.
- Marls – modelled as a tabular layer, using the vein modelling tool.
- Marls_SFD – modelled as a tabular layer, using the vein modelling tool.
- Calcareous S2/S1 unit intensely skarnified (S1_S2) – modelled using the stratigraphic sequence tool.
- Epiclastic unit (SFD) – modelled using the stratigraphic sequence tool.
- Quartzite (S1Q) – modelled using the stratigraphic sequence tool.
- Marble – modelled using the stratigraphic sequence tool.

Figure 14.2 – 3D Oblique View of the Lithology Model



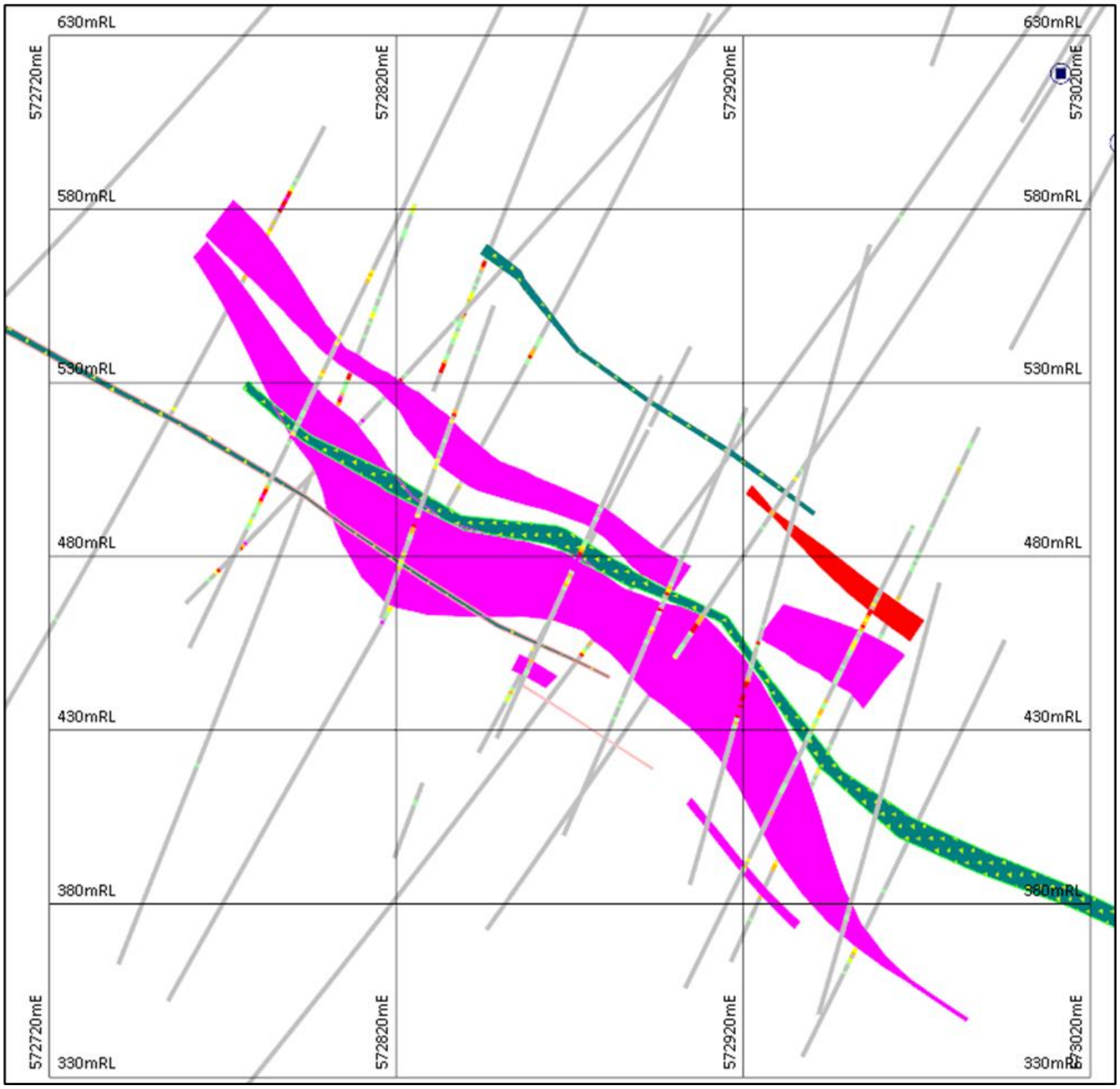
Source: CSA Global, 2023

Figure 14.3 – Cross Section Showing Lithology Units and Drillholes, View Looking North, 4895805mN

Source: CSA Global, 2023

Late-stage crosscutting intrusive sills were modelled using the vein modelling tool. A total of six units were modelled. Modelling was based upon identification of pyroxene porphyry (PXP) which occurred mainly within S2/S1 and frequently crosscuts the mineralisation, these units were identified as plagioclase altered to porcelain-looking rock, with obliterated primary texture where intensive alteration previously led to mis-logging as sedimentary rock. These units were used to define internal dilution within the mineralised skarn model (Figure 14.4).

Figure 14.4 – Cross section (4895780 mN) showing mineralisation (pink/red) and late-stage intrusives (green hatch) and drillholes coloured by gold; view towards north, slice view ±30 m



Source: CSA Global, 2023

14.3.2 MINERALISATION

The Leapfrog Geo model supplied by DPM and verified by the QP contained 3D models of mineralised material as part of the lithology model which were constructed based upon geological observations – mainly skarnification and alteration intensity. The models were built with reference to gold mineralisation (commencing at first instance of grades equal to or above 1 g/t Au and terminating at the last intercept of the same grade) but included significant internal waste material. These models are a useful framework for defining a zone of potential mineralisation but are unsuitable for use as mineral resource domains given the risk of smearing high grades into unmineralised zones.

Cognisant of the presence of coarse gold, and the relatively early stage of development/broad drill spacing, it was considered high risk to constrain mineralisation too tightly without a deeper understanding of what drives the mineralisation when high and low grades can be juxtaposed in core that visually looks similar in terms of alteration intensity/skarnification. It was, however, possible to define and exclude broad zones of internal waste based on the current level of understanding of the mineralisation controls.

Using the mineralisation volumes described above to guide orientations, where there were coherent and continuous zones of internal waste, they were modelled out guided by grade composites generated using Datamine's CompSE process (1 g/t Au grade target over 3 m true width). In addition, the late stage intrusives described in Section 14.3.1 were useful to define unmineralised zones (though these were not always identified in logging due to intense alteration and core was not sampled to geological boundaries). Interpretation was completed under the supervision of QP and in close collaboration with DPM geologists. Three (3) broad zones of higher mineralised intensity were described – one at the basal skarn contact, one (1) just below the main PXP layer and one (1) near the S1_S2 contact with the overlying marl units.

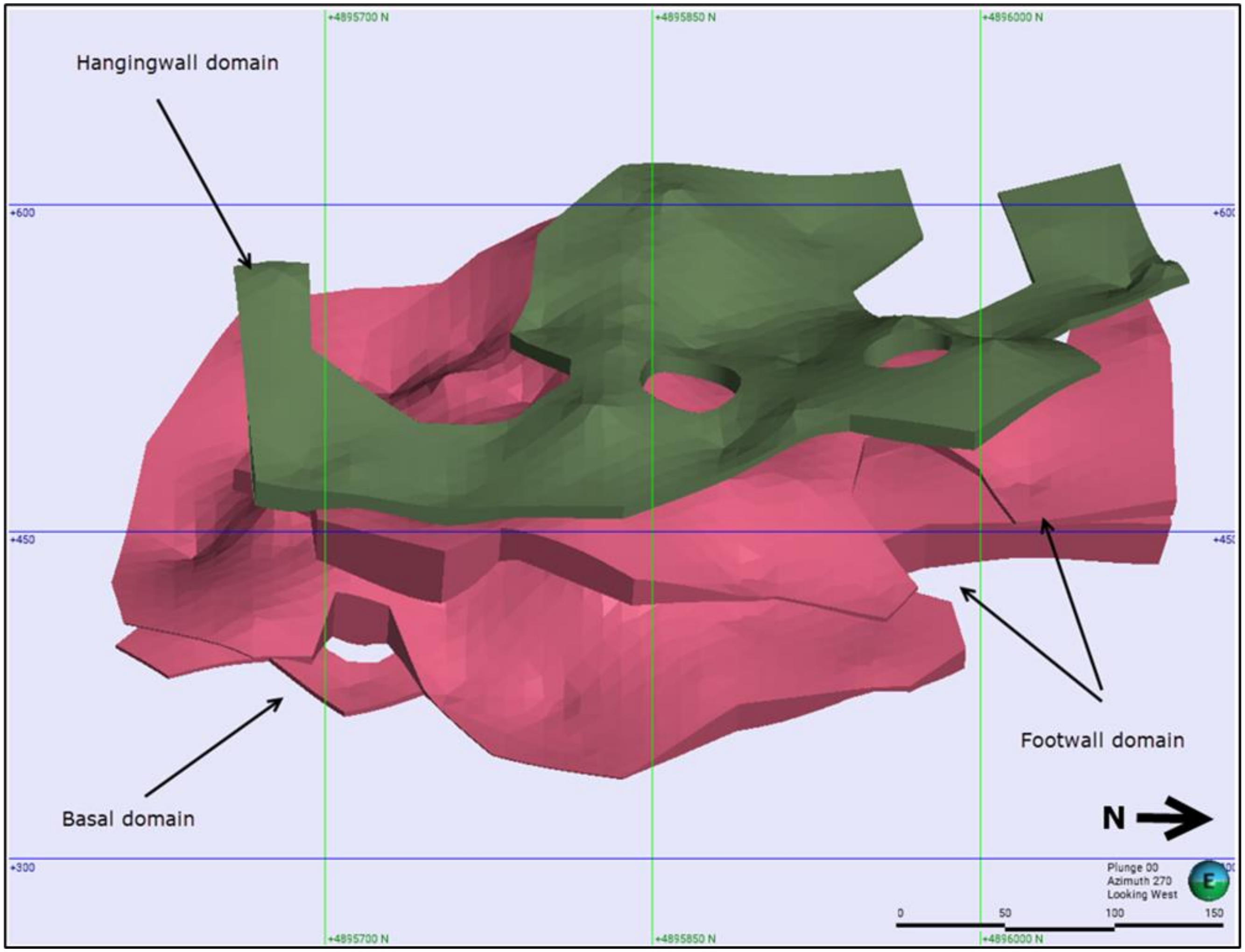
Three (3) large coherent zones were modelled using the vein modelling tool as follows:

- Basal Domain – A small area directly above the southern contact between the S1/S2 sandstone and the early_min_PO.
- Footwall Domain – Two (2) areas of mineralisation combined into 1 domain. One (1) at the contact of the S1/S2 and the Early Min_PO and one (1) just below the initial PXP later stage intrusive.
- Hanging-wall Domain – At the upper contact of the S1/S2 and the overlying SFD and marl units.

Small areas of waste, around single barren holes were excluded from the wireframes. Wireframes were extended halfway between mineralised and barren holes. At the edge of drilling, wireframes were extended to 30 m past the last drillhole.

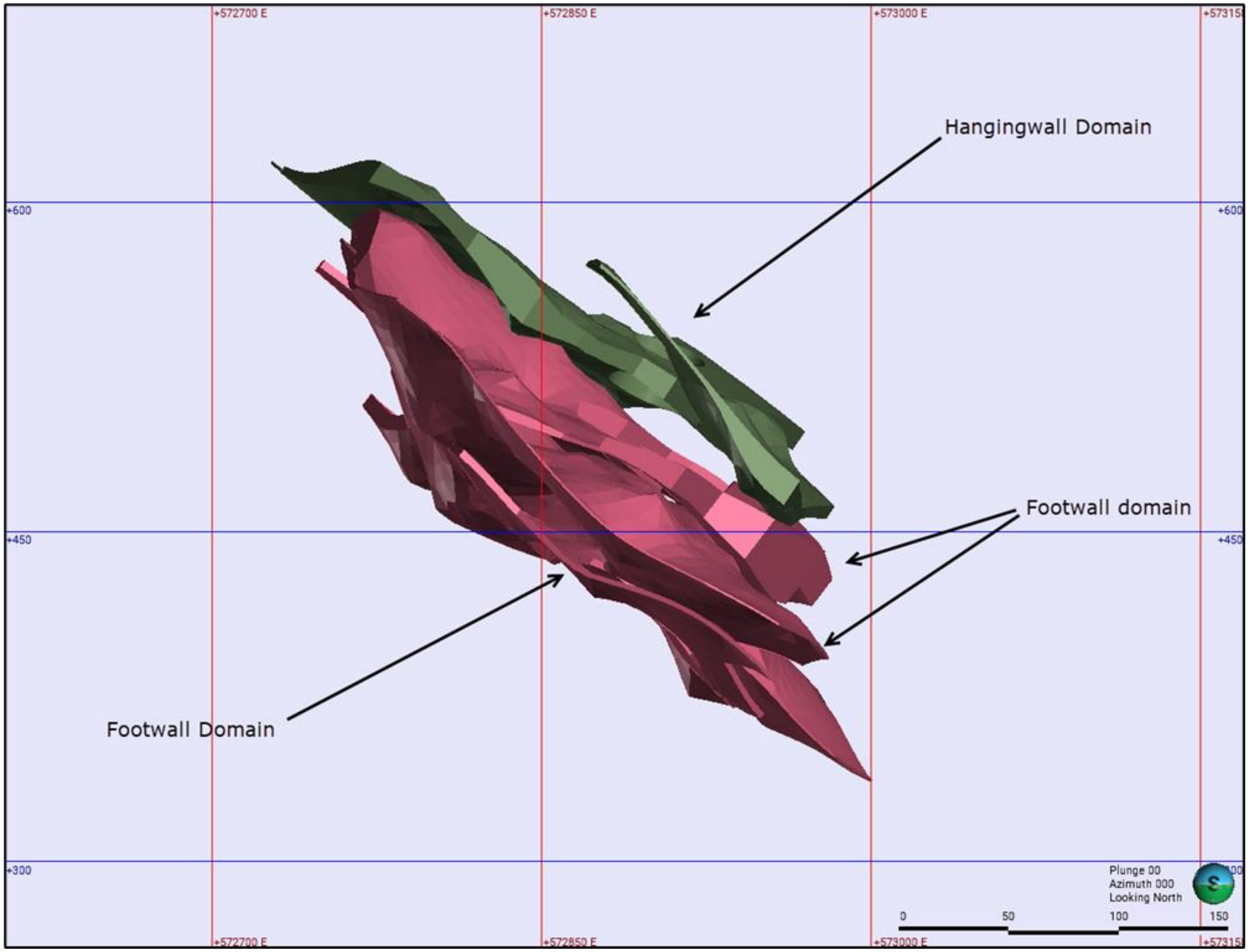
The mineralisation model is shown in Figure 14.5 to Figure 14.7.

Figure 14.5 – Mineralised Domains, View Looking West



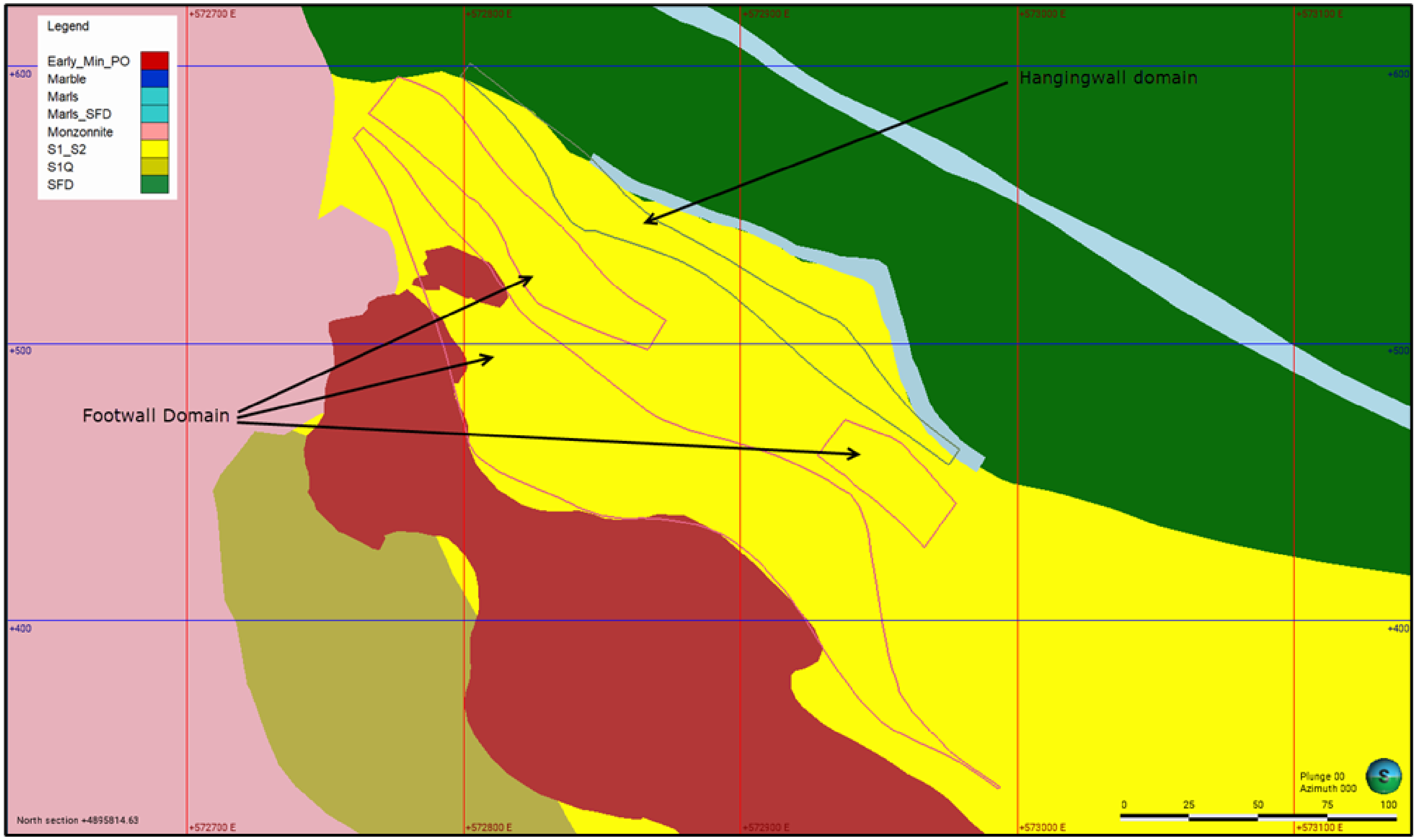
Source: CSA Global, 2023

Figure 14.6 – Mineralised Domains, View Looking North



Source: CSA Global, 2023

Figure 14.7 – Cross Section Looking North (4895815mN) Lithology Model; Mineralisation Overlain as 2D Lines



Source: CSA Global, 2023

14.3.3 LITHOGEOCHEMICAL DOMAINS

Multi-element assay data were reviewed for their suitability to generate lithogeochemical classification to inform geological modelling, resource domaining, and geometallurgical assessments. Data quality and coverage of the multi-element assay database was considered adequate for the purposes of mineral exploration, compositional domaining, and future geometallurgical predictive modelling.

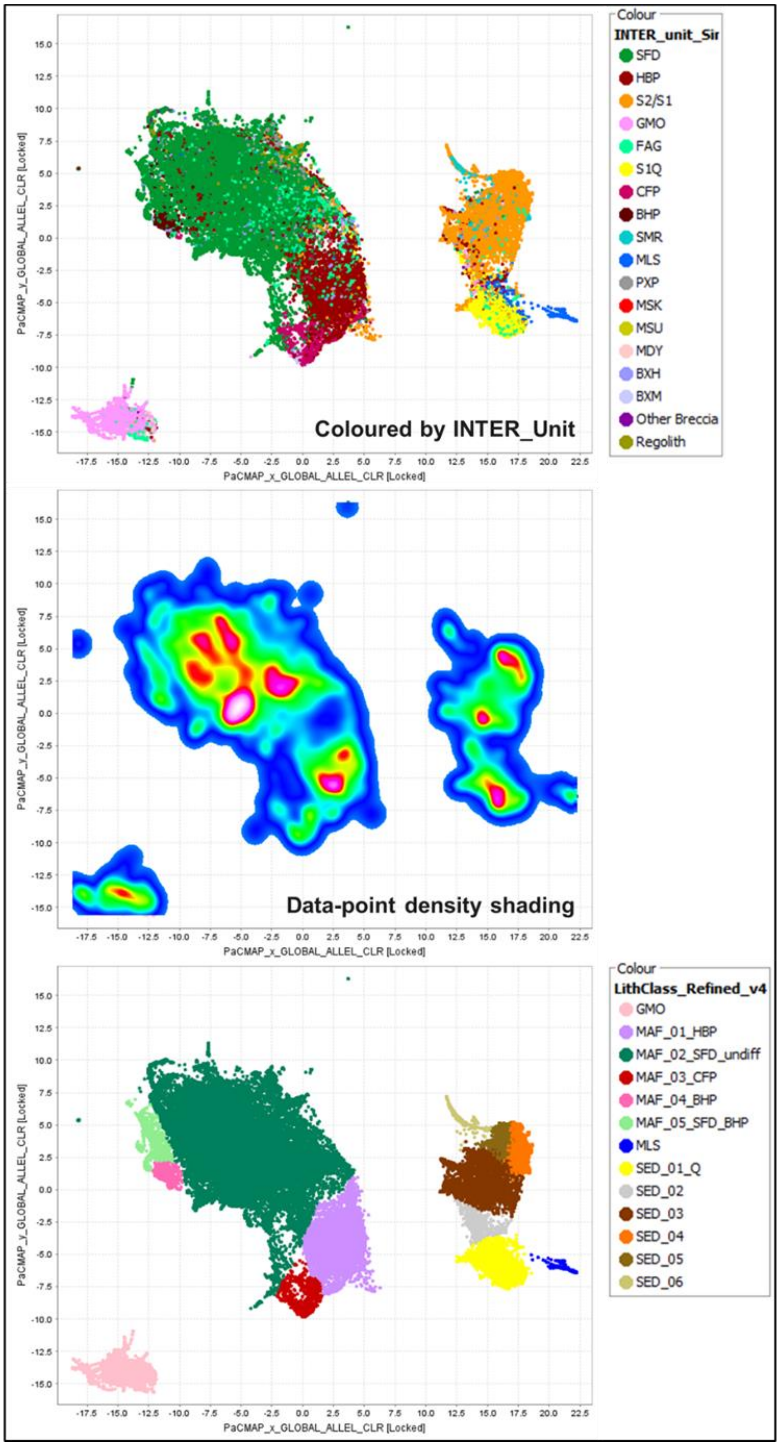
Multi-variate data analysis demonstrates that most lithological domains as well as sub-domains within the S1/S2 unit can be accurately identified based on their composition. Some of the mafic units show overlapping compositions. For example, the composition of epiclastic unit is similar to the compositions of the hornblende and feldspar porphyries, which could suggest a petrogenetic relationships, depending on the relative ages of these units.

Data for all elements with adequate precision and detection limits (Al, As, Ba, Bi, Ca, Ce, Co, Cr, Cs, Cu, Fe, K, La, Li, Mg, Mn, Mo, Na, Nb, Ni, P, Pb, Rb, S, Sb, Sc, Sr, Ta, Tb, Th, Ti, U, V, Y, Zn, Zr) were used to generate a lithogeochemical classification. Compositional groupings in the multivariate space defined by those elements were identified using PaCMAP (Pairwise Controlled Manifold Approximation Projection; Wang et al., 2021) dimensionality reduction and data-density-based cluster analysis (Figure 14.18). Prior to PaCMAP the input features were centred-log-ratio (CLR) transformed to avoid closure effects.

Compositional data may form small subclusters within larger clusters, and it needs to be carefully reviewed in the spatial and geological context, to decide what datapoint clusters to separate and which ones to combine, making the classification an iterative process. The compositional clusters identified in the PaCMAP X-Y projection were reviewed against geological logging and their average compositions to generate a classification (PaCMAP_LithClass), the result of which is shown in the bottom panel of Figure 14.8. Class labels are assigned based on the dominant logged lithology or stratigraphic unit within each compositional cluster, to reference the compositional affinity of each cluster to the visually identified lithologies.

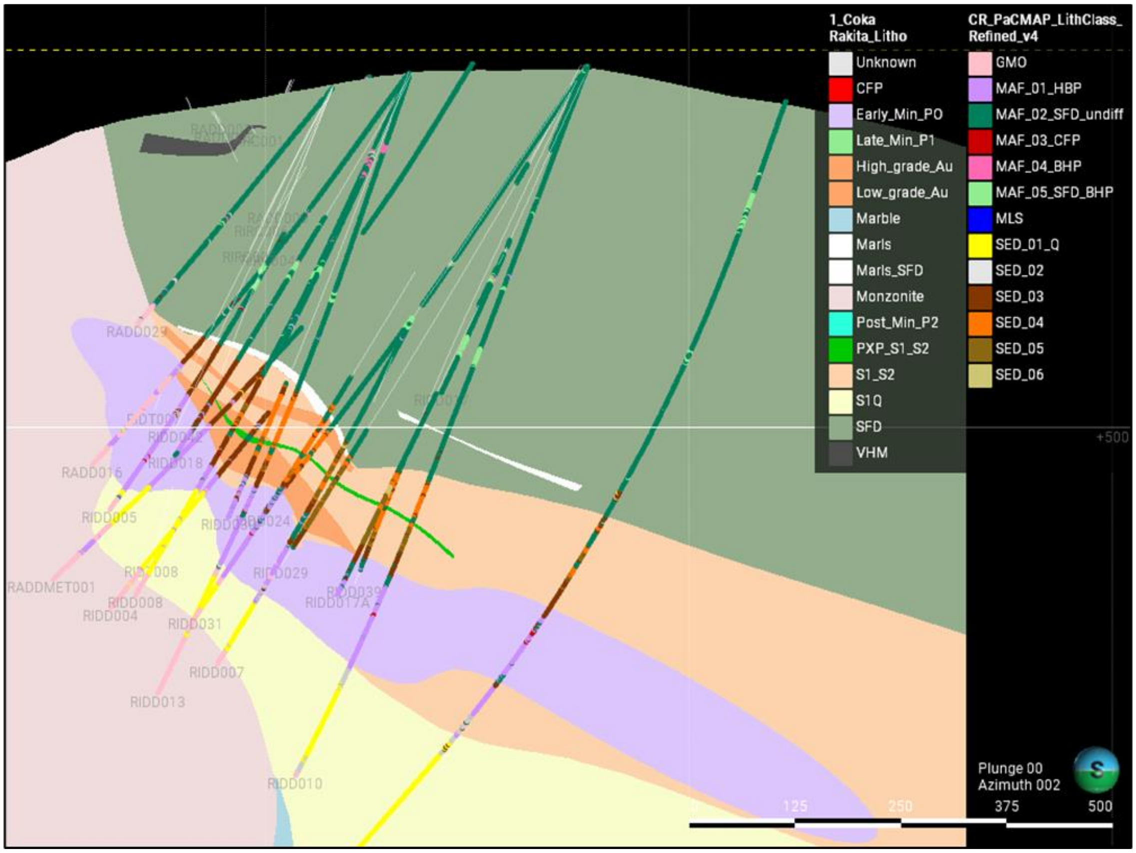
Geological logging and DPM's 3D geological models were validated against the compositional data. A central cross section showing DPM's model and the lithogeochemical classification generated under the supervision of the QP author is shown in Figure 14.9. The logging generally honours compositional differences and main geological units modelled by DPM are in good agreement with the main compositional domains, although the geological model may be refined using the lithogeochemical classifications in tandem with geological logging in the future.

Figure 14.8 – PaCMAP (Wang et al., 2021) x-y Projection of CLR-Transformed Multi-Element Data



Top: Data coloured by lithological logging code (INTER_Unit). Middle: Point-density shading. Bottom: Data coloured by compositional lithogeochemical classification (PaCMAP_LithClass), informed by data-density-based clustering.

Figure 14.9 – Central Cross-Section Looking North and Showing Drillhole Intervals Coloured by PaCMAP LithClass; DPM's 3D Geological Model (Shapes) for Reference



Source: CSA Global, 2023

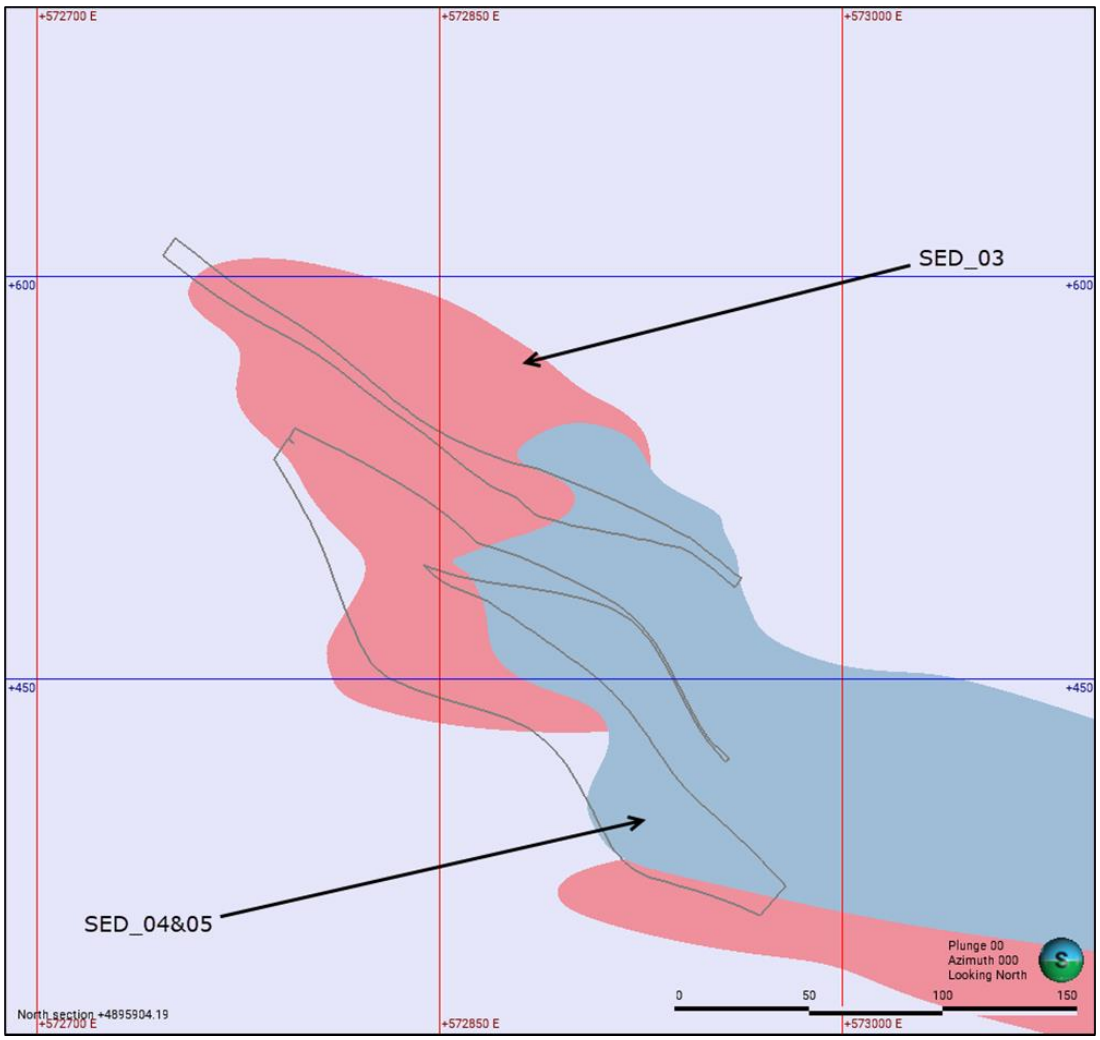
Within the mineralised S1/S2 unit, the following lithogeochemical sub-groups were identified:

- SED_03: Gold-copper exoskarn, possibly higher grossular and epidote relative to andradite, more proximal to GMO (monzonite)and HBP (early mineral porphyry).
- SED_04: Gold exoskarn with low copper and molybdenum, very high iron, high bulk density likely higher andradite and pyrite/pyrrhotite, more distal to GMO and HBP.
- SED_05: Similar to SED_04, but somewhat lower and more variable gold grade, and low iron and sulphur. Boundary to SED_04 may be continuous/fuzzy rather than hard considering the short-range variability where samples of these groups are in contact.
- SED_06: This group appears to comprise the least altered S1/S2 rocks with overall low gold, copper, molybdenum, iron, manganese and sulphur.

Two (2) 3D mineralisation domains were generated (Figure 14.10), with the SED_03 being modelled as a single unit and the SED_04 and SED_05 domains being combined as a gradational boundary between the two (2) units was expected.

Both were modelled in Leapfrog as intrusives with flattened spheroidal trends, using a spheroidal interpolant and no drift. The Sed_04&05 unit overprinted the SED_03 unit in the contact surface chronology, as the SED_04&05 material formed a shell around the central SED_03 material. It is noted that the SED_03 unit is characterised by higher copper grades, compared to the SED_04&05 units. While copper is not reported, grades were estimated using these lithogeochemical domains for use in geometallurgical characterisation.

Figure 14.10 – Cross Section of Lithogeochemical 3D Model Looking North (485905mN)



Note: SED_03 – pink; SED_04&05 – blue; DPM's mineralised domains – grey outlines.
Source: CSA Global, 2023

14.4 Topography

A digital elevation model (DEM) derived topographic surface was provided by DPM for use in the MRE based on the drone topographic mapping described in Section 9.6.

14.5 Domaining

For mineralisation, there were two (2) domains overall – in the southern fault block where most of the mineralisation sits, this comprised the hanging wall (one unit) and footwall mineralisation (three (3) sub-parallel units interpreted to be part of the same mineralisation event); in the northern fault block, the footwall mineralisation comprises three (3) sub-units and were grouped together with those in the southern fault block since the structure is interpreted to be post-mineralisation.

Footwall mineralisation is the largest zone and was coded ESTZON 101 while the hanging wall mineralisation was coded ESTZON 102. Late-stage intrusives were modelled and coded ESTZON 200. While the volumes modelled for late-stage intrusives are considered reasonable, there is uncertainty over the precise locations in some cases where very high grades of gold end up being flagged as late-stage intrusives when using the wireframes. These cases were reviewed visually, and a selection of intercepts were manually re-assigned where the mineralisation flags low grade and the late-stage intrusives flags high grade. Table 14.2 presents the intercepts that were subject to manual flagging.

Table 14.2 – Manual Flagging Intercepts

BHID	From (m)	To (m)	ESTZON
RADD013	520	521	102
RADD013	521	522	200
RADD039	611	612	101
RIDD001	469	470	101
RIDD004	514	515	101
RIDD004	515	516	101
RIDD004	528	529	101
RIDD004	536	537	101
RIDD008	499	500	101
RIDD008	501	502	200
RIDD018	448	449	200
RIDD018	449	450	200
RIDD018	450	451	101
RIDD018	451	452	101
RIDD036	445	446	200
RIDD036	447	448	101
RIDD036	459	460	101
RIDT023	465	466	101

BHID	From (m)	To (m)	ESTZON
RIDT023	466	467	200
RIDT025	468	469	200
RIDT025	470	471	101
RIDT026	477	478	101
RIDT026	478	479	101
RIDT027	450	451	101
RIDT027	451	452	101
RIDT027	452	453	101
RIDT027	453	454	101
RIDT027	454	455	101
RIDT027	461	462	200
RIDT027	462	463	200
RIDT027	483	484	101
RIDT027	485	486	200
RIDT027	486	487	200
RIDT028	405	406	102
RIDT028	408	409	200
RIDT028	452	453	101
RIDT031	424	425	101

Future work will be completed on the modelling of late-stage intrusives to try to improve the spatial modelling of these units following infill drilling currently underway. This is constrained by the nature of sampling which is to 1 m intervals instead of breaking on geological boundaries. While breaking on geological boundaries is preferred and certainly recommended, it is acknowledged that these late-stage intrusives are difficult to identify due to intense alteration and their misidentification as sediments until lithogeochemical analysis has been completed using multi-element analysis.

Waste domains were based on modelled geology, with certain units grouped together where they showed similar gold distributions. Waste domains were prefixed with 30 followed by the lithology code (GEOL). Table 14.3 presents domain codes used for mineralisation and waste.

Table 14.3 – Domain Codes

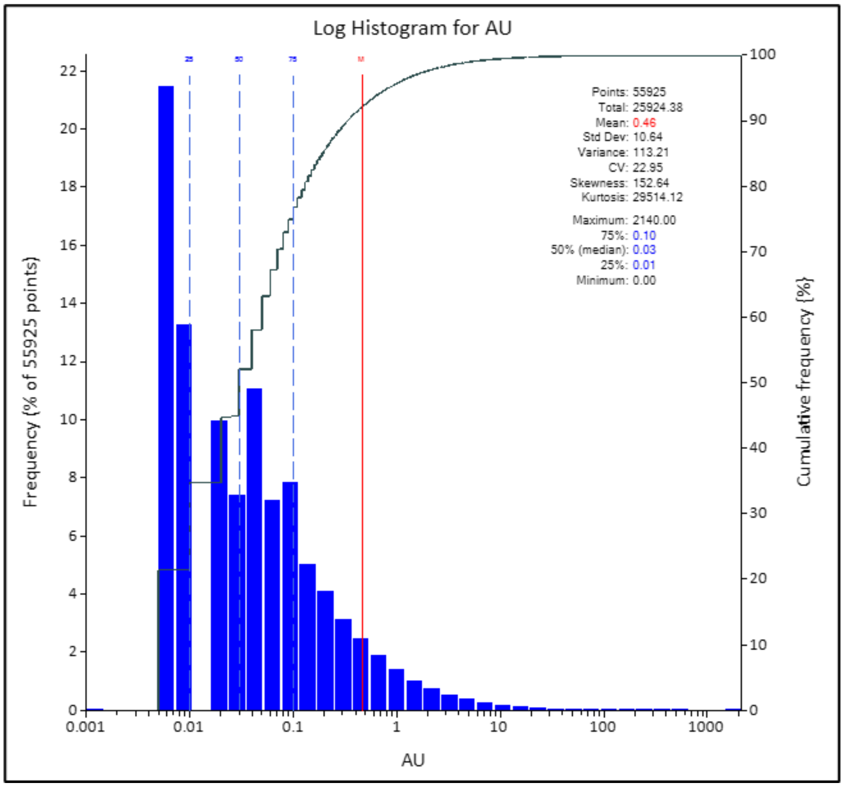
Wireframe	Flagging Field	Flagging Code	ESTZON
cr_mz101_	MINZON	101	101[1]
cr_mz103_	MINZON	101	101[1]
cr_mz104_	MINZON	101	101[1]
cr_mz105_	MINZON	101	101[1]
cr_mz102_	MINZON	102	102[1]
mdy_1	LATE_INT	1	200[1,2]
mdy_2	LATE_INT	2	200[1,2]
pxp_1_block-2_sw	LATE_INT	3	200[1,2]
pxp_2_block-2_sw	LATE_INT	4	200[1,2]
pxp_1	LATE_INT	5	200[1,2]
pxp_2	LATE_INT	6	200[1,2]
pxp_3	LATE_INT	7	200[1,2]
pxp_4	LATE_INT	8	200[1,2]
cr_geo_sfd_	GEOL	1	30137
cr_geo_vhm_	GEOL	2	302
cr_geo_marls_sfd_	GEOL	3	30137
cr_geo_marls_	GEOL	4	3045
cr_geo_s1_s2_	GEOL	5	3045
cr_geo_pxp_s1_s2_ [2]	GEOL	6	-
cr_geo_early_min_po_	GEOL	7	30137

1 37 intercepts were subjected to manual re-coding, as described in Section 14.5.
2 ESTZON 200 is a waste domain (late-stage intrusives) but deviates from the naming convention for other waste domains due to its location largely crosscutting mineralisation.

14.6 Statistical Analysis

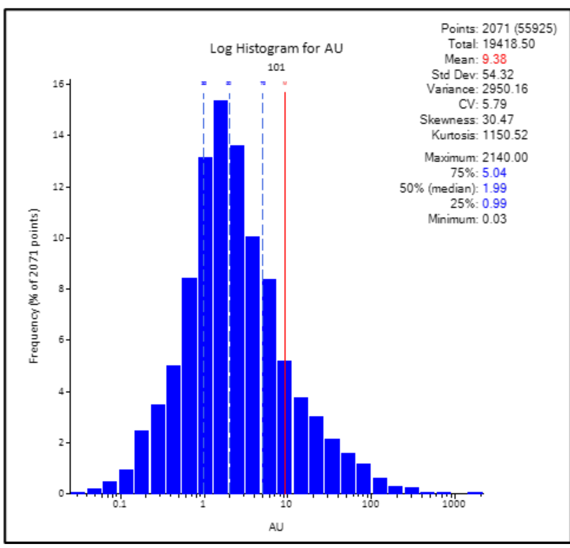
The gold population of the whole dataset is characterised by a positive skew and long tail (Figure 14.11), with mineralised domains shown in Figure 14.12. Histograms for silver (whole dataset) and mineralised domains are shown in Figures 14.13 and 14.14.

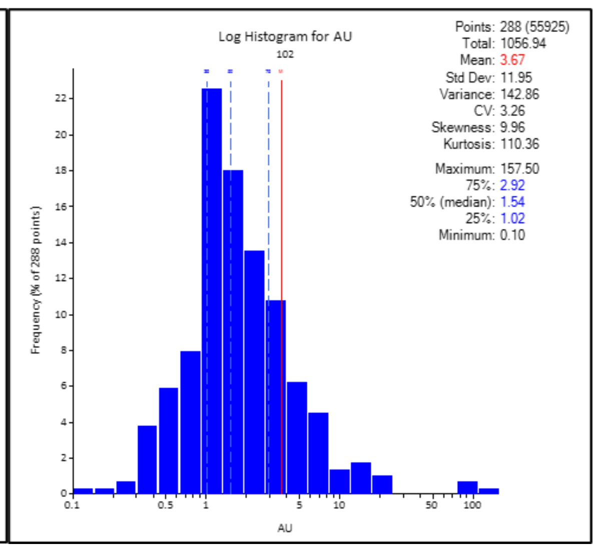
Figure 14.11 – Log Normal Histogram of Gold – Whole Dataset



Source: CSA Global, 2023

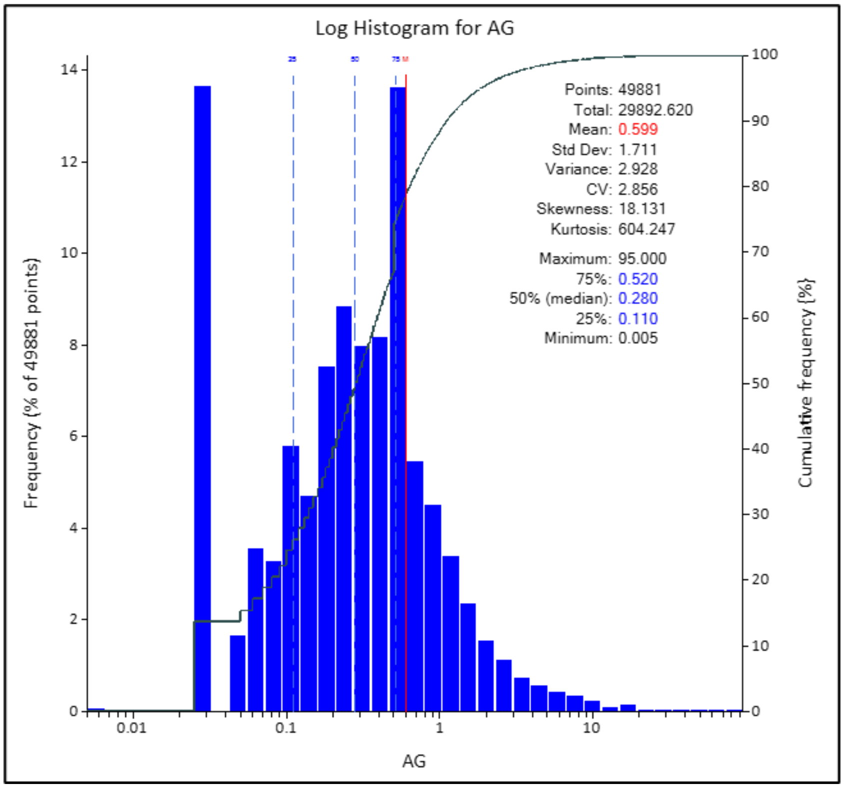
Figure 14.12 – Log Normal Histogram for Gold for ESTZON 101 (Left) and ESTZON 102 (Right)





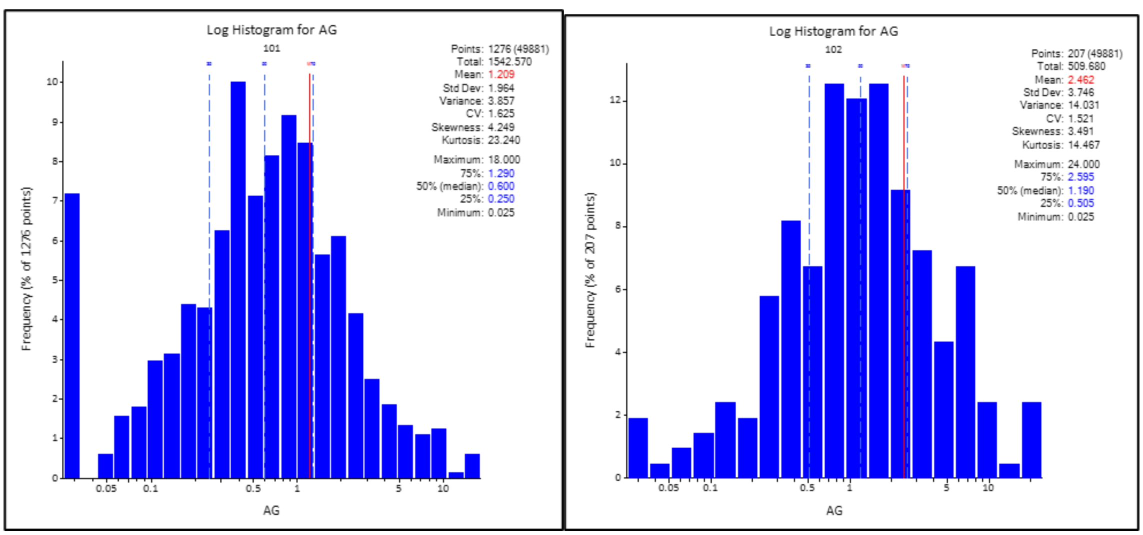
Source: CSA Global, 2023

Figure 14.13 – Log Normal Histogram of Silver – Whole Dataset



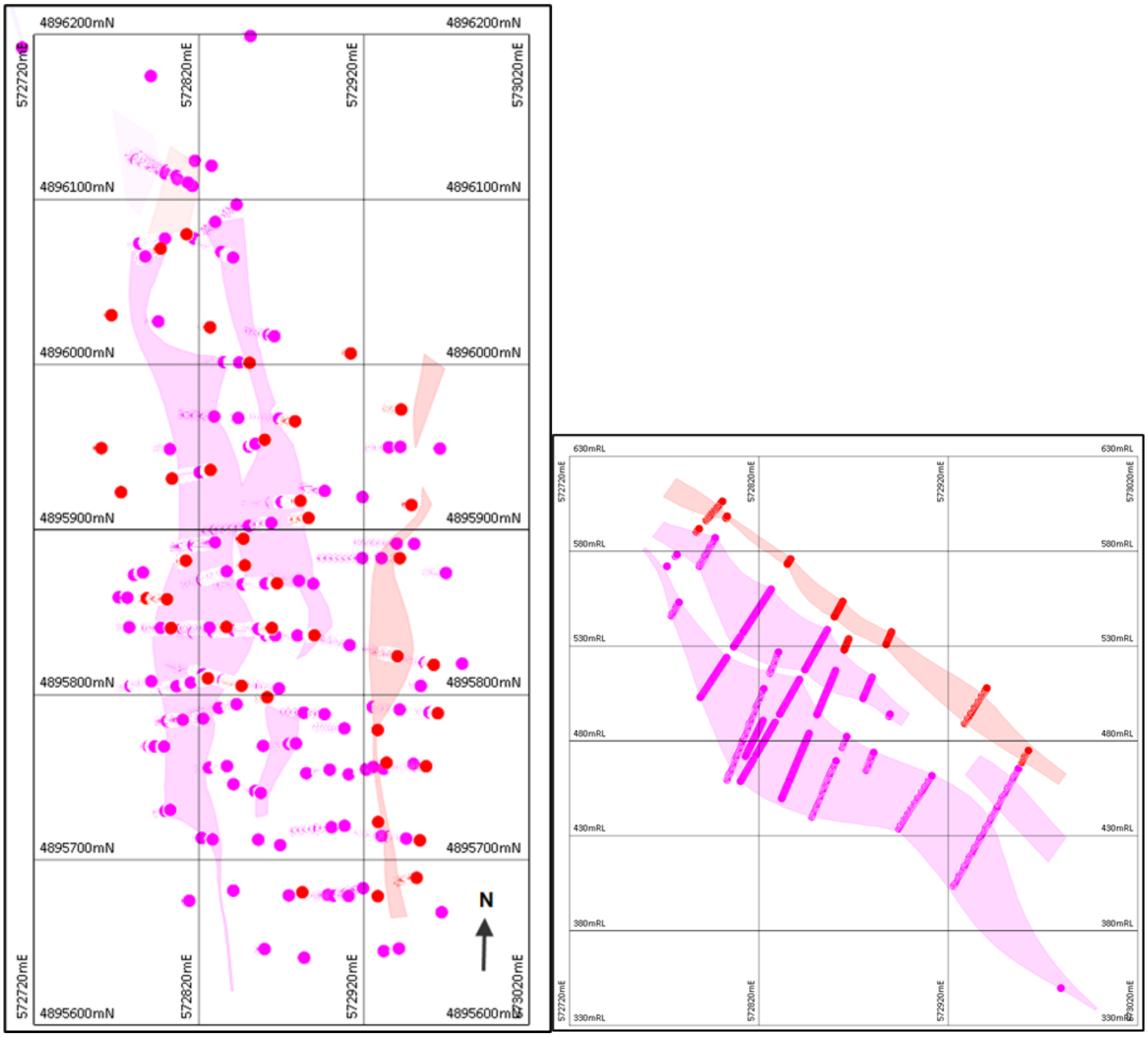
Source: CSA Global, 2023

Figure 14.14 – Log Normal Histograms for Silver for ESTZON 101 (Left) and ESTZON 102 (Right)



Source: CSA Global, 2023

Figure 14.15 – Plan View of Mineralised Intercepts and Cross Section Showing Clustering in ESTZON 101; Cross Section 4895839 mN, View Towards North, ±30 m (ESTZON 101 – pink; ESTZON 102 – red)



Source: CSA Global, 2023

Statistics of raw gold data are presented in Table 14.4 and raw silver data in Table 14.5. Declustering was required for gold in the footwall domain (ESTZON 101), Figure 14.15. No declustering was considered necessary for the hanging wall domain (ESTZON 102), since sample clustering at the Project is not solely to do with drillhole spacing but influenced by drillhole deviation which increases with depth. No declustering was necessary for silver which is a reduced dataset relative to gold.

Table 14.4 – Summary Raw Statistics for Gold – Clustered and Declustered Where Applicable

Statistic	ESTZON 101		ESTZON 102	
	Clustered	Declustered	Clustered	Declustered
Declustering cell size		30 x 30 x 10 m (X x Y x Z)		Not required
Samples	2,071	2,071	288	
Minimum	0.03	0.03	0.1	
Maximum	2,140	2,140	157.5	
Mean	9.38	8.93	3.67	
Standard deviation	54.32	48.7	11.95	
Coefficient of variation	5.79	5.46	3.26	
Variance	2,950.16	2,371.76	142.86	
Skewness	30.47	31.95	9.96	
Log samples	2071	2071	288	
Log mean	0.84	0.82	0.56	
Log variance	2.06	2.03	0.92	
Geometric mean	2.32	2.27	1.76	
10%	0.44	0.43	0.58	
20%	0.78	0.77	0.96	
30%	1.14	1.12	1.09	
40%	1.48	1.45	1.28	
50%	1.99	1.94	1.54	
60%	2.66	2.6	1.92	
70%	4.03	3.93	2.44	
80%	6.57	6.41	3.43	
90%	15.7	15.21	5.57	
95%	34.45	34	8.53	
97.50%	64.03	60.12	13.41	
99%	120.79	107.67	31.55	

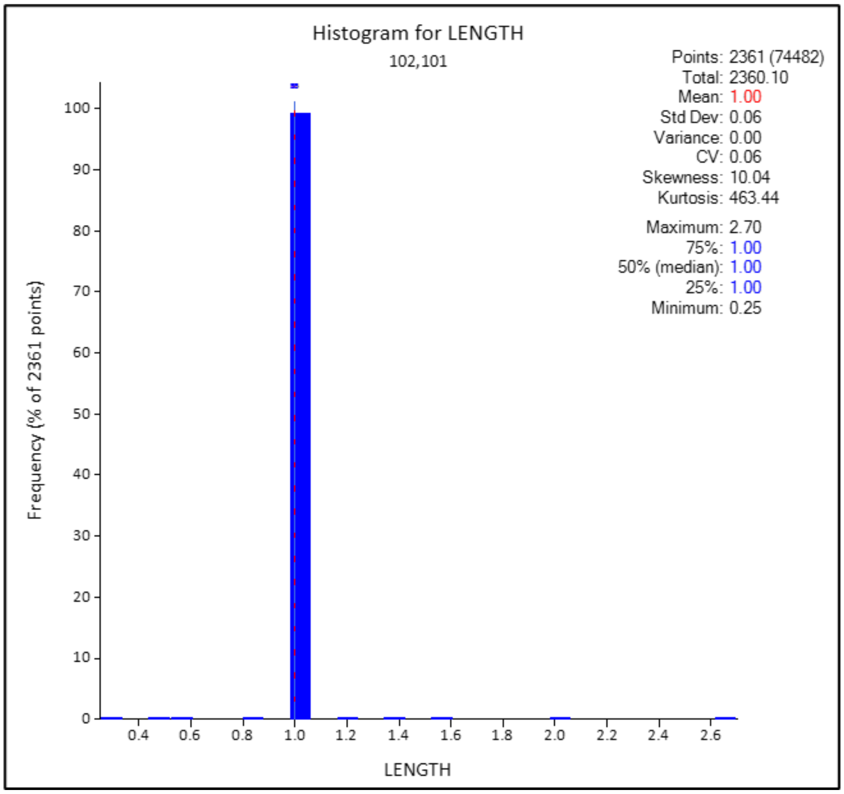
Table 14.5 – Summary Raw Statistics for Silver

Statistic	ESTZON 101	ESTZON 102
Samples	1276	207
Minimum	0.025	0.025
Maximum	18	24
Mean	1.209	2.462
Standard deviation	1.964	3.746
Coefficient of variation	1.625	1.521
Variance	3.857	14.031
Skewness	4.249	3.491
Log samples	1,276	207
Log mean	-0.636	0.123
Log variance	1.914	1.742
Geometric mean	0.529	1.13
10%	0.08	0.227
20%	0.19	0.408
30%	0.31	0.642
40%	0.43	0.85
50%	0.6	1.19
60%	0.81	1.534
70%	1.09	2.224
80%	1.62	3.272
90%	2.694	6.414
95%	4.236	8.425
97.50%	7.471	13.3
99%	9.662	22.65

14.7 Compositing

Compositing was completed to ensure comparable sample support during estimation. Sampling of drillholes has been predominantly on a 1.0 m basis as shown in Figure 14.16; therefore, 1.0 m composite length was chosen with only one (1) residual (0.1 m) which was excluded.

Figure 14.16 – Histogram Showing Length of Raw Data in Mineralised Domain



Source: CSA Global, 2023

14.8 Global and Domain Statistics

Domain statistics of the uncut 1 m composites are presented in Table 14.6. Uncut gold is characterised by a very high coefficient of variation.

Table 14.6 – Summary of 1 m Composite Statistics

Statistic	Gold		Silver	
	ESTZON 101	ESTZON 102	ESTZON 101	ESTZON 102
Declustering cell size	30 x 30 x 10 (X x Y x Z)	Not required	Not required	Not required
Samples	2,069	289	1,273	208
Minimum	0.03	0.1	0.03	0.03
Maximum	2140	157.5	18	24
Mean	8.95	3.71	1.18	2.53
Standard deviation	53.3	11.94	1.85	3.85
Coefficient of variation	5.96	3.22	1.57	1.52

Statistic	Gold		Silver	
	ESTZON 101	ESTZON 102	ESTZON 101	ESTZON 102
Variance	2,841.27	142.65	3.42	14.82
Skewness	29.9	9.94	4.18	3.33
Log samples	2069	289	1273	208
Log mean	0.81	0.57	-0.64	0.14
Log variance	1.83	0.93	1.89	1.77
Geometric mean	2.25	1.77	0.53	1.15
10%	0.5	0.58	0.08	0.23
20%	0.9	0.96	0.19	0.41
30%	1.18	1.09	0.31	0.65
40%	1.47	1.28	0.43	0.85
50%	1.9	1.55	0.6	1.2
60%	2.43	1.93	0.81	1.55
70%	3.51	2.46	1.08	2.25
80%	5.41	3.48	1.61	3.28
90%	12.6	5.74	2.65	6.68
95%	32.2	9.1	4.17	9.14
97.50%	61.76	13.4	7.21	14.75
99%	121.18	30.94	9.35	22.6

14.9 Variables and Correlations

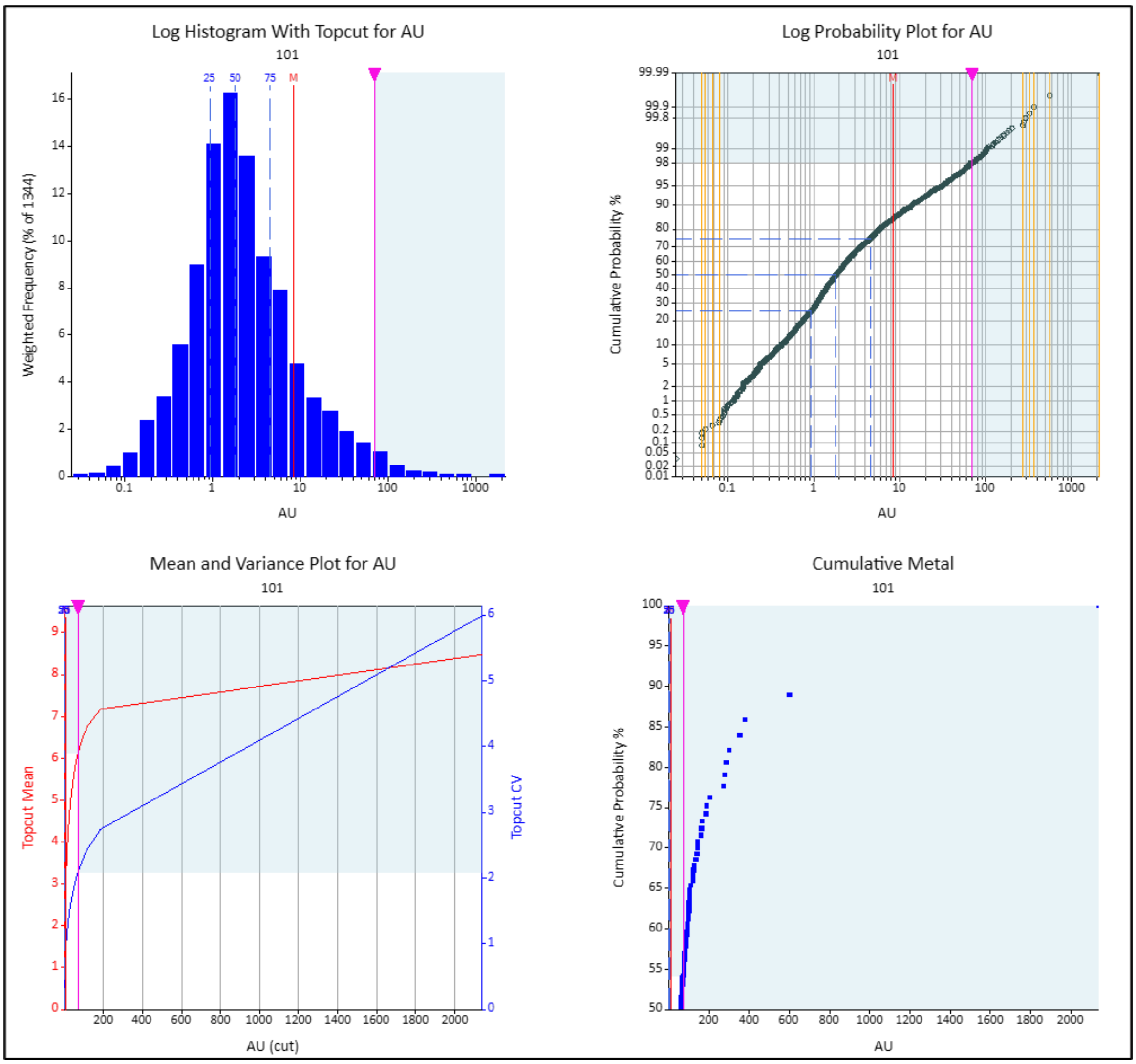
The variables of interest are gold and silver. Gold and silver are not strongly correlated within the skarn mineralisation (correlation coefficient of 0.49).

14.10 Treatment of Outliers (Top Cuts)

The treatment of outliers is one of the key factors that the MRE is sensitive to, given the high grades and known presence of coarse gold particles, and the scattered nature of extreme grades within the domains. This means that the impact of these outliers on the estimated local means would likely be more biased than if they were grouped closer together. Global top cut analysis was completed in Supervisor software and outliers were reviewed spatially prior to a top cut (grade cap) being applied. Global top cut analysis comprises the simultaneous review of the grade histogram, log probability, mean-variance and cumulative metal plots.

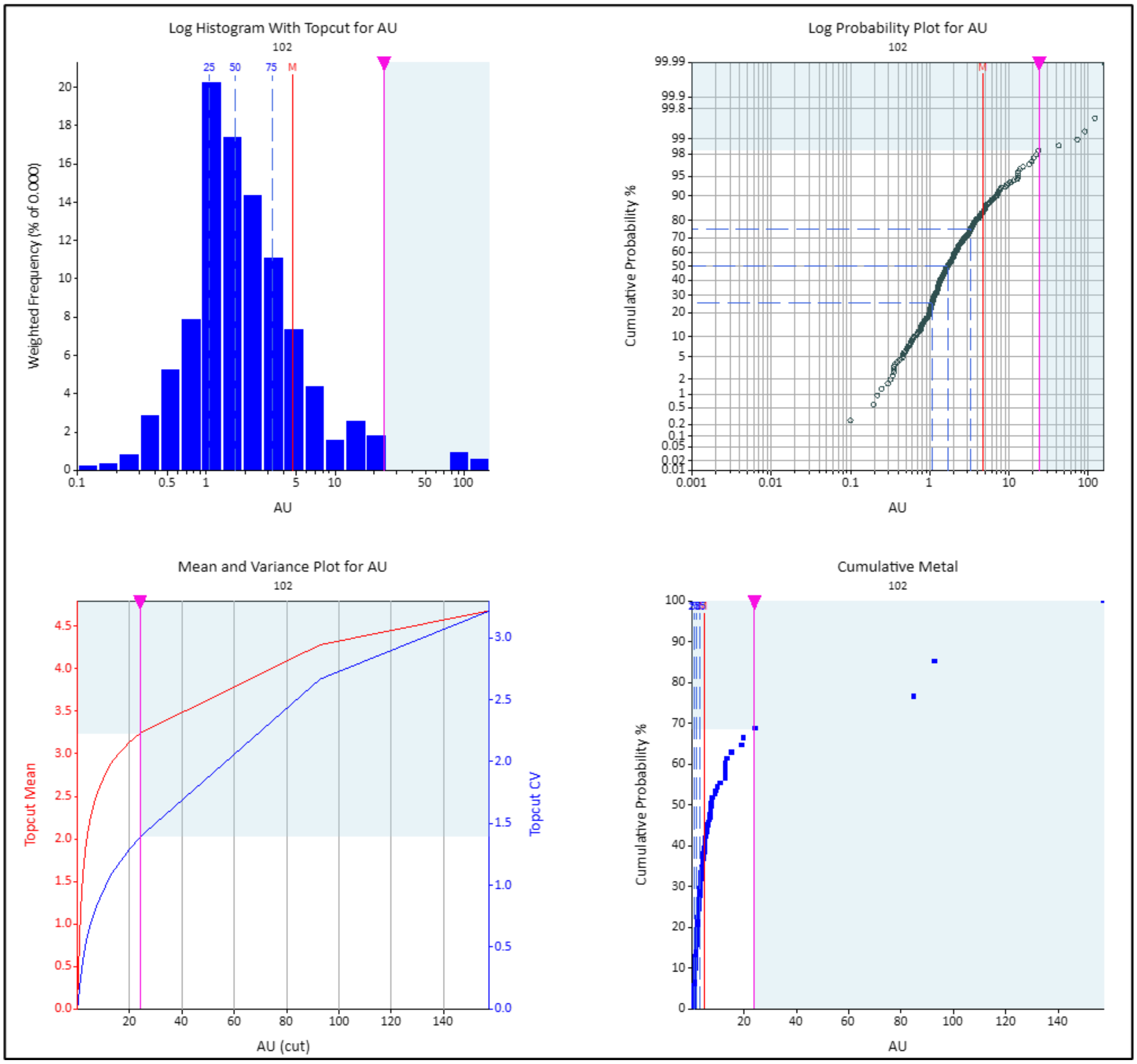
Sensitivity reviews were completed on various top cuts for gold ranging from 70 g/t to 100 g/t Au before a top cut of 70 g/t was chosen. On further drilling, the choice of outlier may become clearer. Outlier values were not removed from the dataset; rather, values exceeding the top cut value chosen were set to that value. Global top cut analysis results are presented for gold and silver in the main mineralisation domains in Figure 14.17 to Figure 14.21, while Table 14.7 presents the top cuts applied to the mineralisation domains. Table 14.8 shows the summary statistics for the top cut estimation composites, showing a reduction in the coefficient of variation for the gold domains.

Figure 14.17 – Global Top Cut Analysis for Gold for ESTZON 101



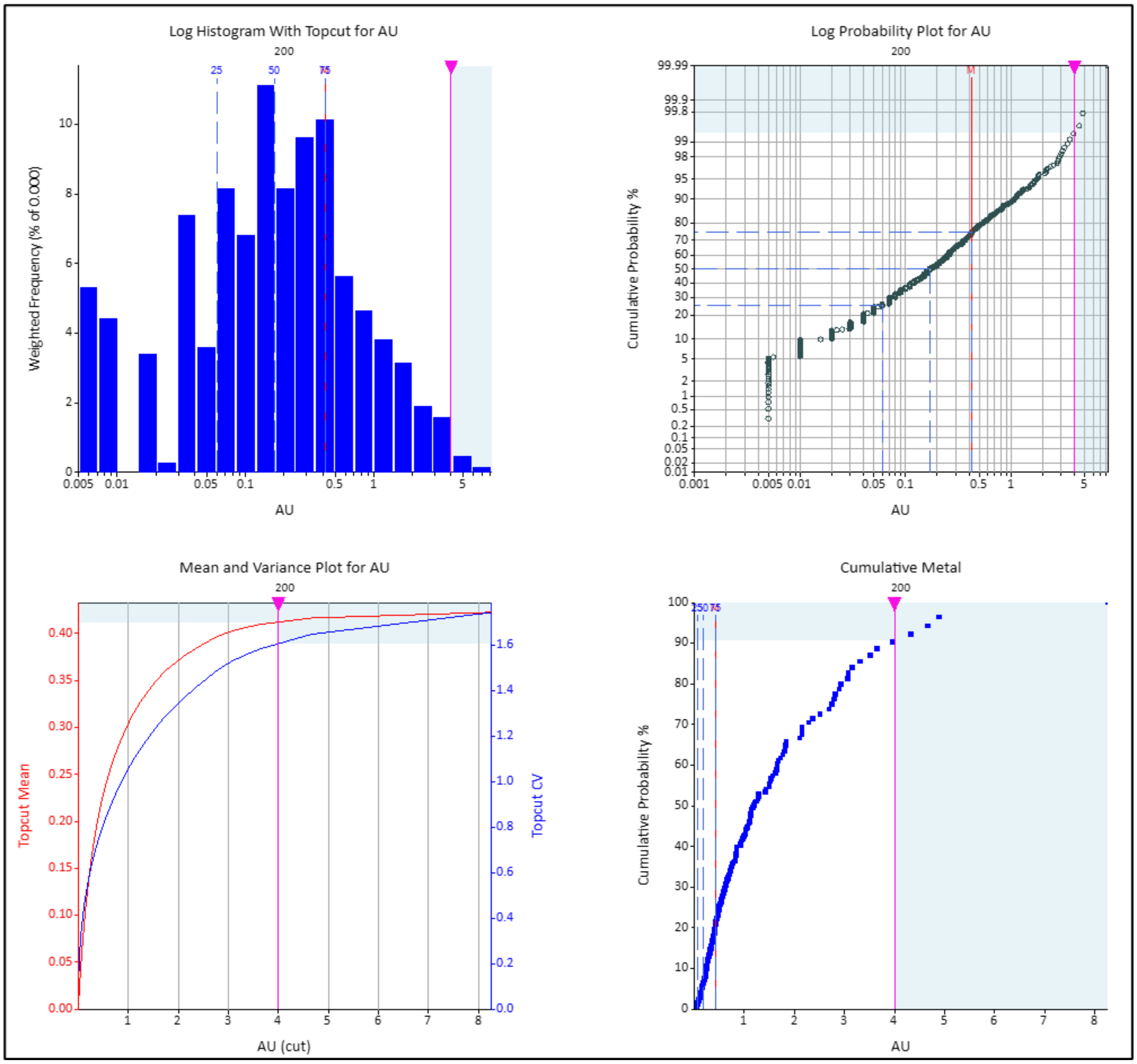
Source: CSA Global, 2023

Figure 14.18 – Global Top Cut Analysis for Gold for ESTZON 102



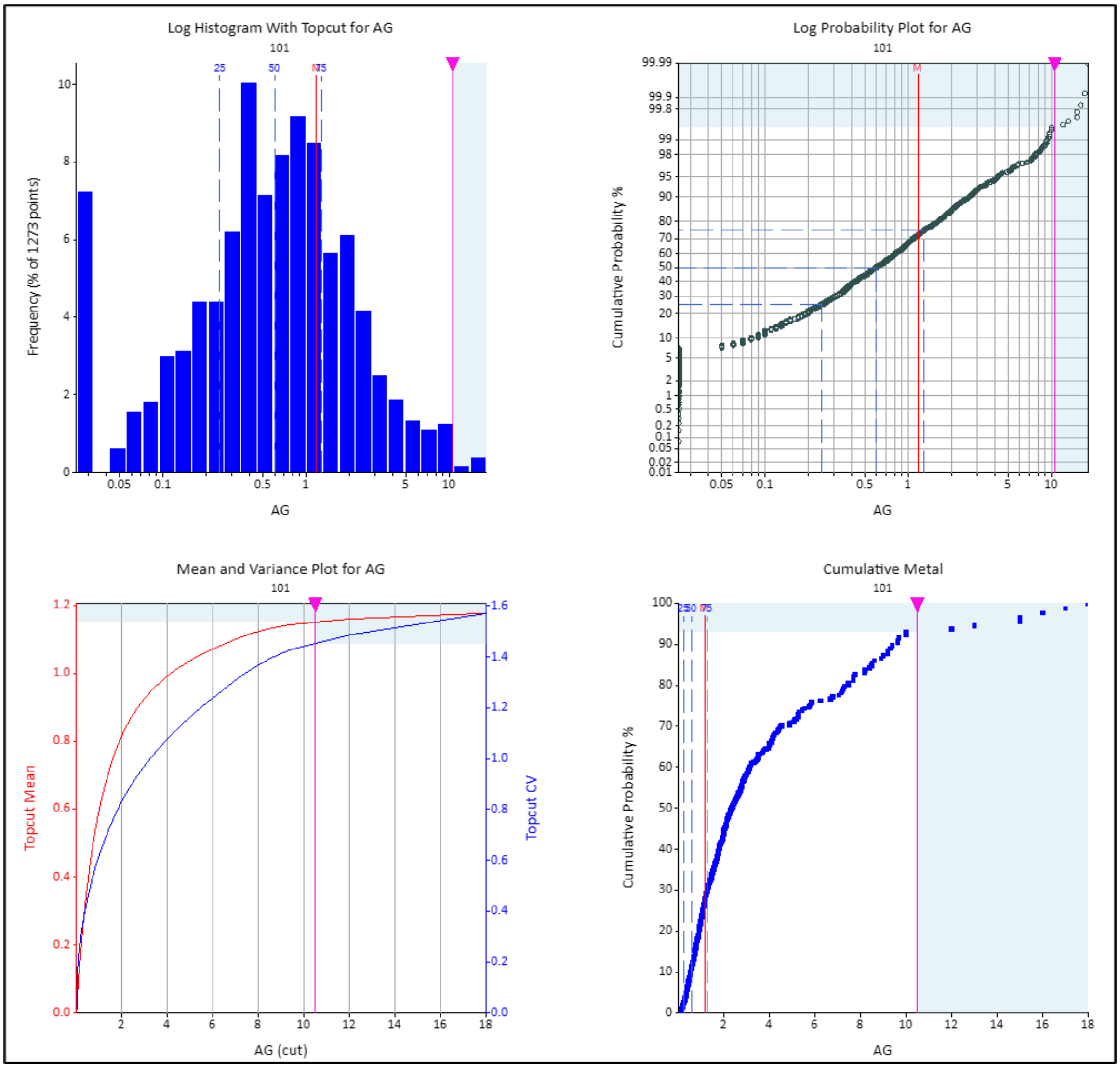
Source: CSA Global, 2023

Figure 14.19 – Global Top Cut Analysis for Gold for ESTZON 200



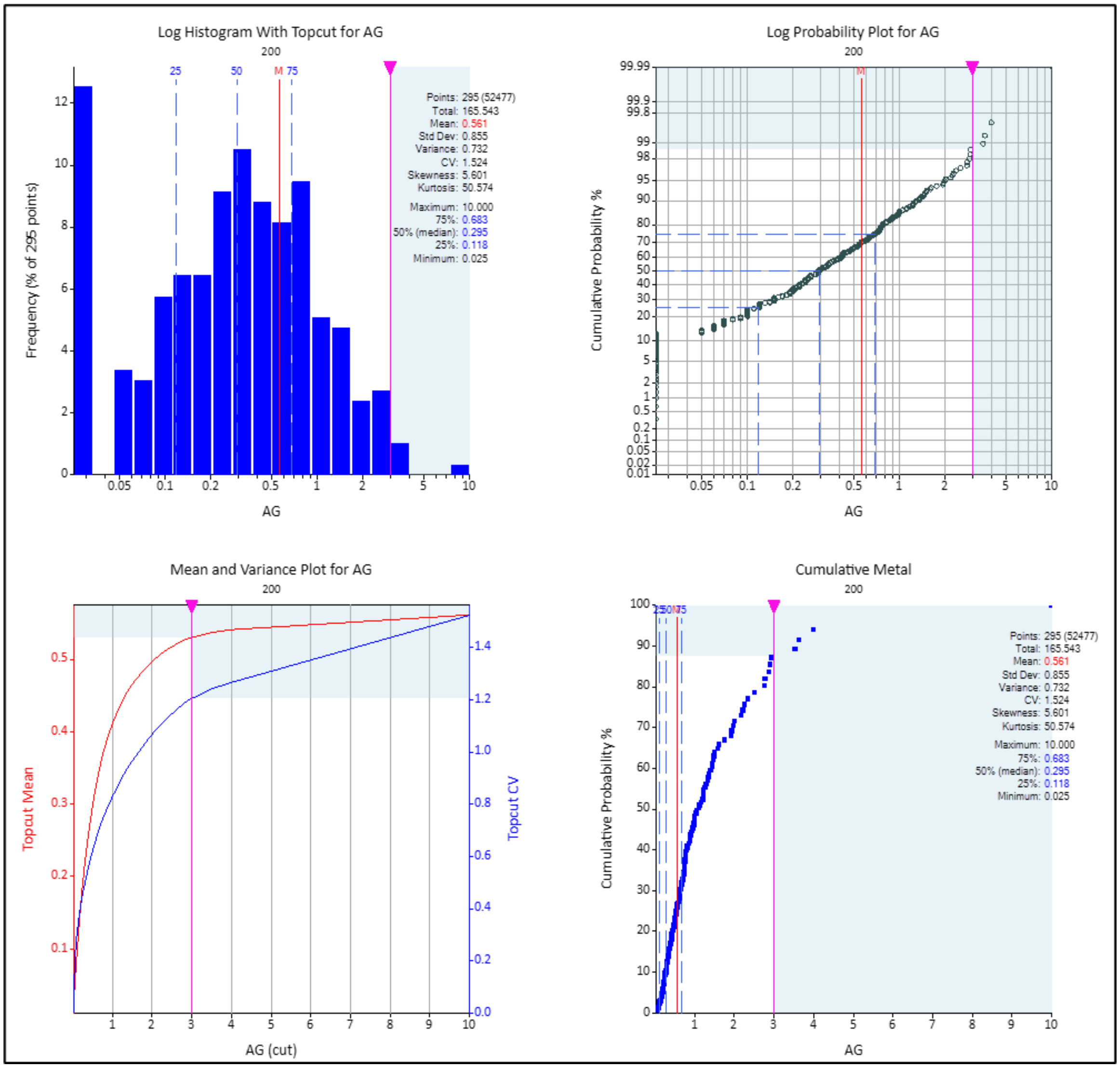
Source: CSA Global, 2023

Figure 14.20 – Global Top Cut Analysis for Silver for ESTZON 101



Source: CSA Global, 2023

Figure 14.21 – Global Top Cut Analysis for Silver for ESTZON 200



Source: CSA Global, 2023

Table 14.7 – Top Cuts Used for Gold and Silver in Mineralisation Domains

Domain	Element	Number (data)	Uncut mean	Top cut	Cut mean	Number (data cut)	% Data cut	% Change mean/metal
101	Gold	2,070	8.48	70	6.12	47	2	28
	Silver	1,273	1.18	10.5	1.15	7	0.5	2
102	Gold	289	4.68	24	3.24	4	1	31
	Silver	N/A	N/A	N/A	N/A	N/A	N/A	N/A
200	Gold	455	0.42	4	0.41	4	1	2
	Silver	295	0.56	3	0.53	4	1	6

Table 14.8 – Summary Statistics for Top Cut Composites

Statistic	Gold		Silver	
	ESTZON 101	ESTZON 102	ESTZON 101	ESTZON 102
Declustering cell size	30 x 30 x 10 (X x Y x Z)	Not required	Not required	Not required
Samples	2,069	289	1,273	208
Minimum	0.03	0.1	0.025	0.025
Maximum	70	24	10.5	24
Mean	6.09	2.8	1.151	2.53
Standard deviation	12.92	3.76	1.673	3.85
Coefficient of variation	2.12	1.34	1.454	1.522
Variance	166.95	14.11	2.801	14.82
Skewness	3.78	3.64	3.25	3.333
Log samples	2069	289	1273	208
Log mean	0.79	0.56	-0.646	0.136
Log variance	1.7	0.83	1.88	1.769
Geometric mean	2.21	1.75	0.524	1.146
10%	0.5	0.58	0.08	0.228
20%	0.9	0.96	0.19	0.412
30%	1.18	1.09	0.31	0.648
40%	1.47	1.28	0.43	0.854
50%	1.9	1.55	0.6	1.2
60%	2.43	1.93	0.81	1.546
70%	3.5	2.46	1.08	2.248

Statistic	Gold		Silver	
	ESTZON 101	ESTZON 102	ESTZON 101	ESTZON 102
80%	5.41	3.48	1.61	3.284
90%	12.6	5.74	2.65	6.68
95%	32.2	9.1	4.167	9.14
97.50%	61.76	13.4	7.211	14.749
99%	70	24	9.349	22.6

14.11 Variography

Experimental semi-variograms (variograms) were generated for gold and silver in Supervisor software for use in grade estimation of ESTZON 101, 102 and 200 using 1 m capped composites (presented in Table 14.9 to Table 14.11). The following approach was used:

- Variograms were generated to determine the major, semi-major, and minor axes of continuity which are perpendicular to each other.
- The variogram in the downhole direction was modelled to determine the nugget to determine the close-spaced variability.
- The major, semi-major, and minor axes of continuity were modelled using two (2) spherical structures.

The modelled orientations were consistent with the geological understanding of the mineralisation (as depicted in Figure 14.22 to Figure 14.25). Variograms for gold in the mineralised skarns were characterised by a moderate nugget (lower than expected given the presence of coarse gold, but highly sensitive to the outliers prior to capping) and a dominant first structure, isotropic in the major and semi-major directions and averaging 50 m. Approximately 90% of the spatial continuity is within 50 m. Silver is characterised by longer ranges and lower nugget.

Table 14.9 – Variogram Parameters in Datamine ZYZ Rotation – ESTZON 101

Element	C0	C1	Rotation			Range			C2	Range		
			Z	X	Z	Major	Semi	Minor		Major	Semi	Minor
Gold	0.34	0.55	90	50	180	54	46	9	0.11	124	100	17
Silver	0.29	0.52	90	50	-160	112	93	18	0.19	182	202	31

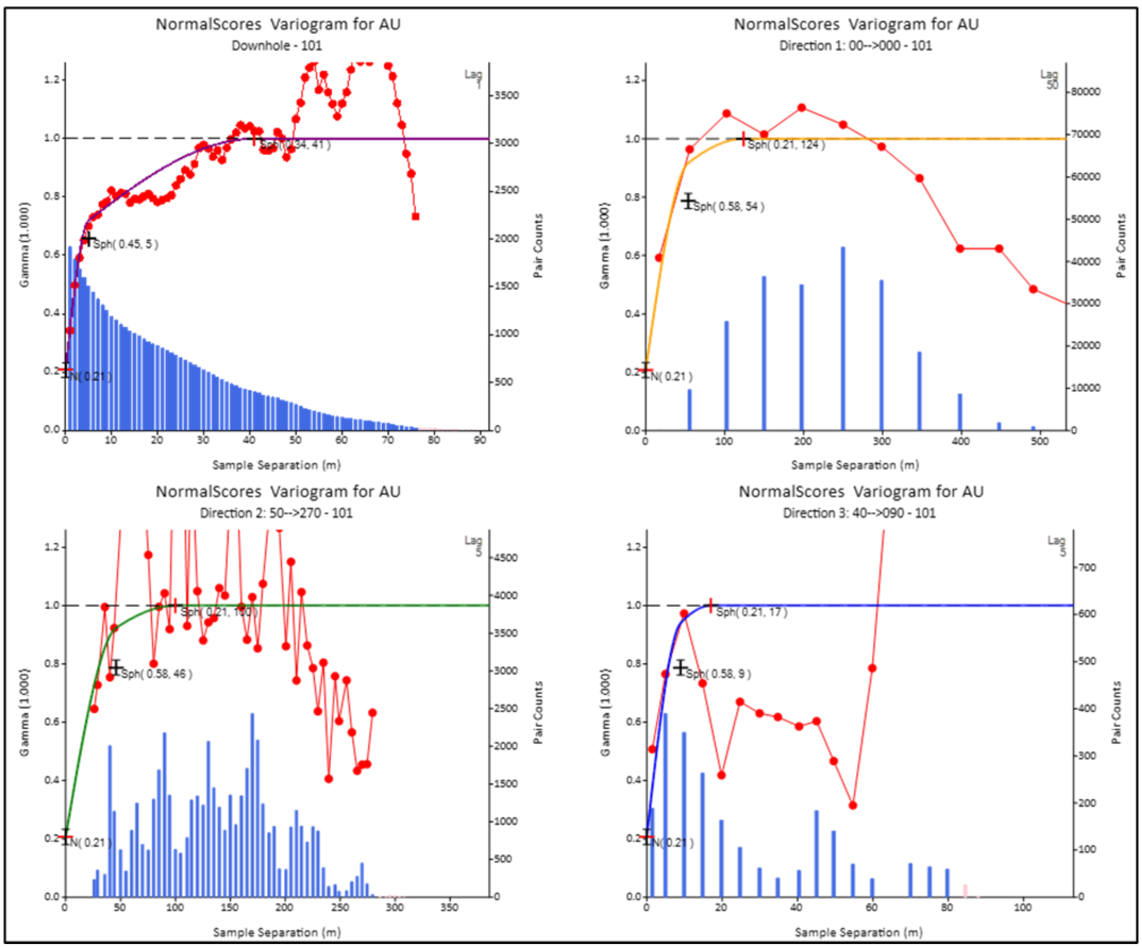
Table 14.10 – Variogram Parameters in Datamine ZYZ Rotation – ESTZON 102

Element	C0	C1	Rotation			Range			C2	Range		
			Z	X	Z	Major	Semi	Minor		Major	Semi	Minor
Gold	0.39	0.50	90	40	180	43	46	9	0.12	61	100	17
Silver	0.22	0.27	80	40	170	77	151	9	0.51	413	193	17

Table 14.11 – Variogram Parameters in Datamine ZYZ Rotation – ESTZON 200

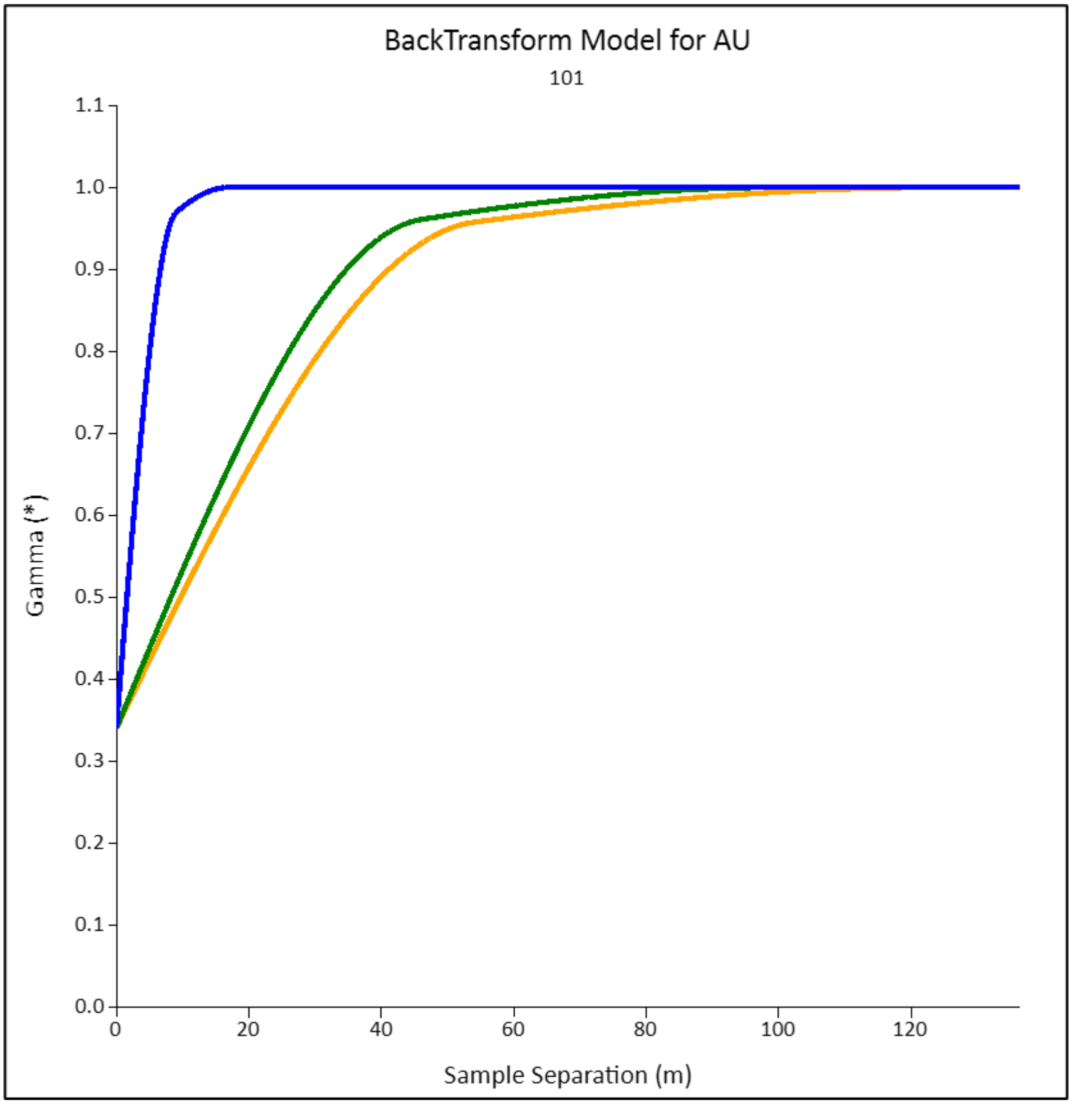
Element	C0	C1	Rotation			Range			C2	Range		
			Z	X	Z	Major	Semi	Minor		Major	Semi	Minor
Gold	0.15	0.57	80	40	180	109	46	24	0.29	191	126	34
Silver	0.45	0.24	60	40	180	167	191	24	0.32	223	318	34

Figure 14.22 – Experimental Semi-Variograms for Gold in ESTZON 101



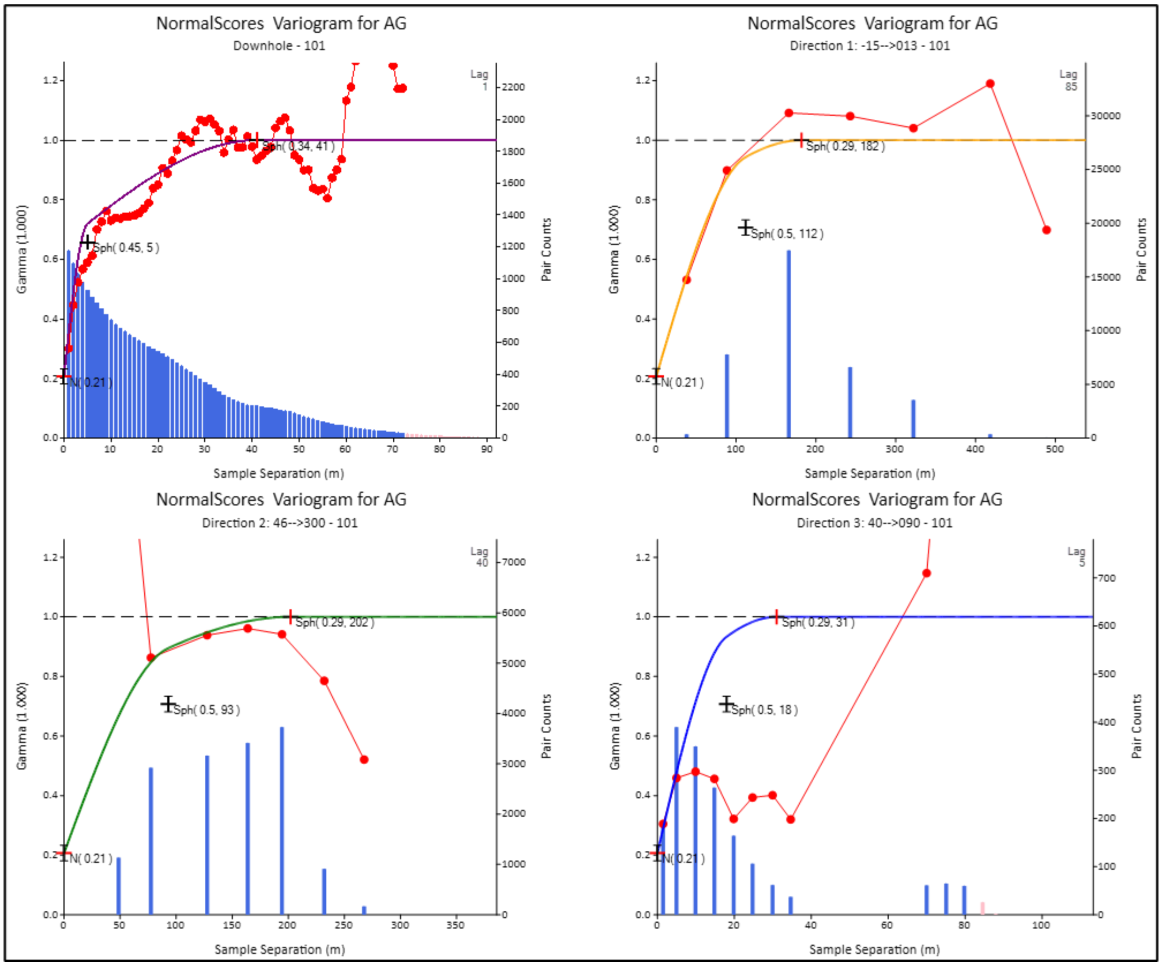
Source: CSA Global, 2023

Figure 14.23 – Variogram Model for Gold in ESTZON 101



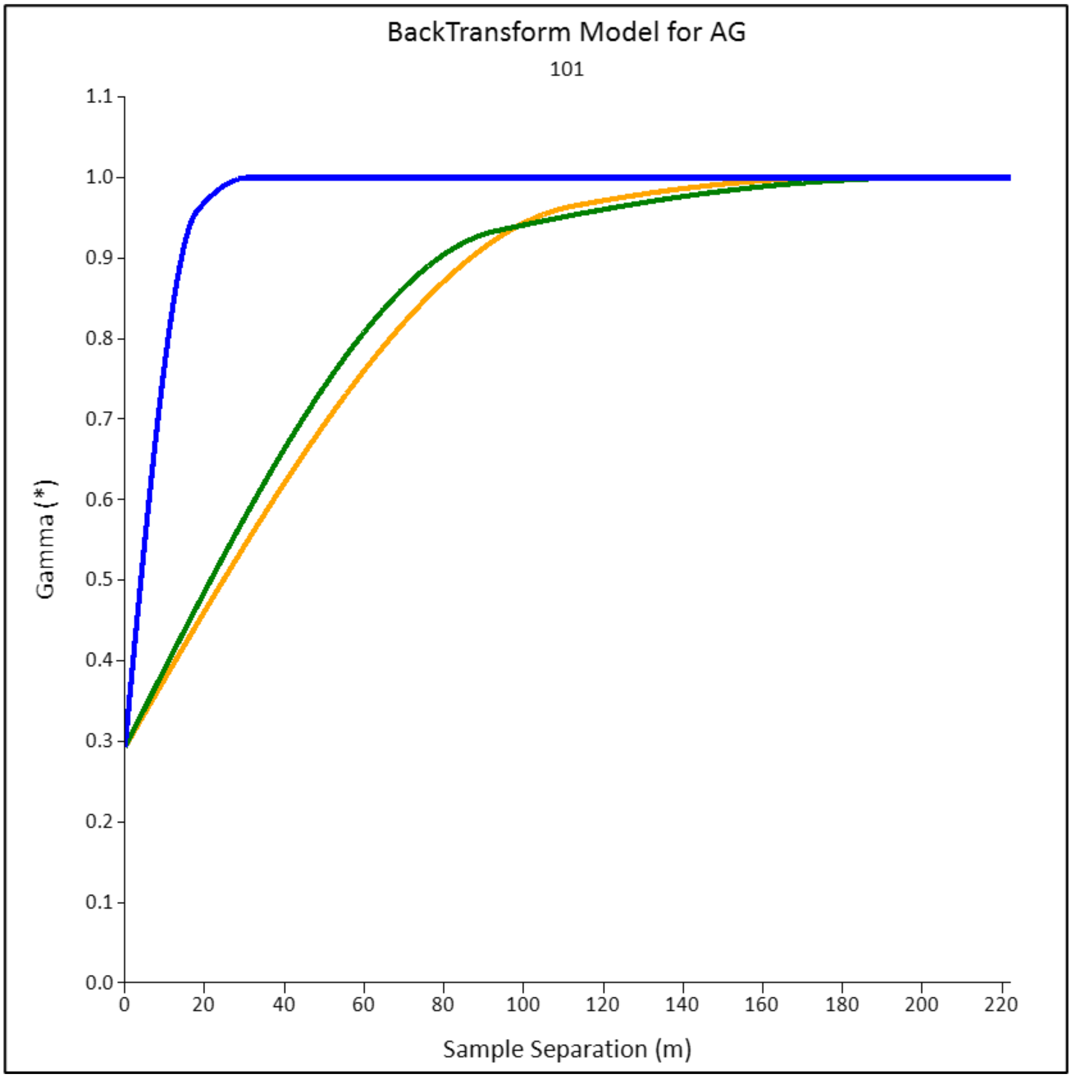
Source: CSA Global, 2023

Figure 14.24 – Experimental Semi-Variograms for Silver in ESTZON 101



Source: CSA Global, 2023

Figure 14.25 – Variogram Model for Silver in ESTZON 101



Source: CSA Global, 2023

14.12 Kriging Neighbourhood Analysis

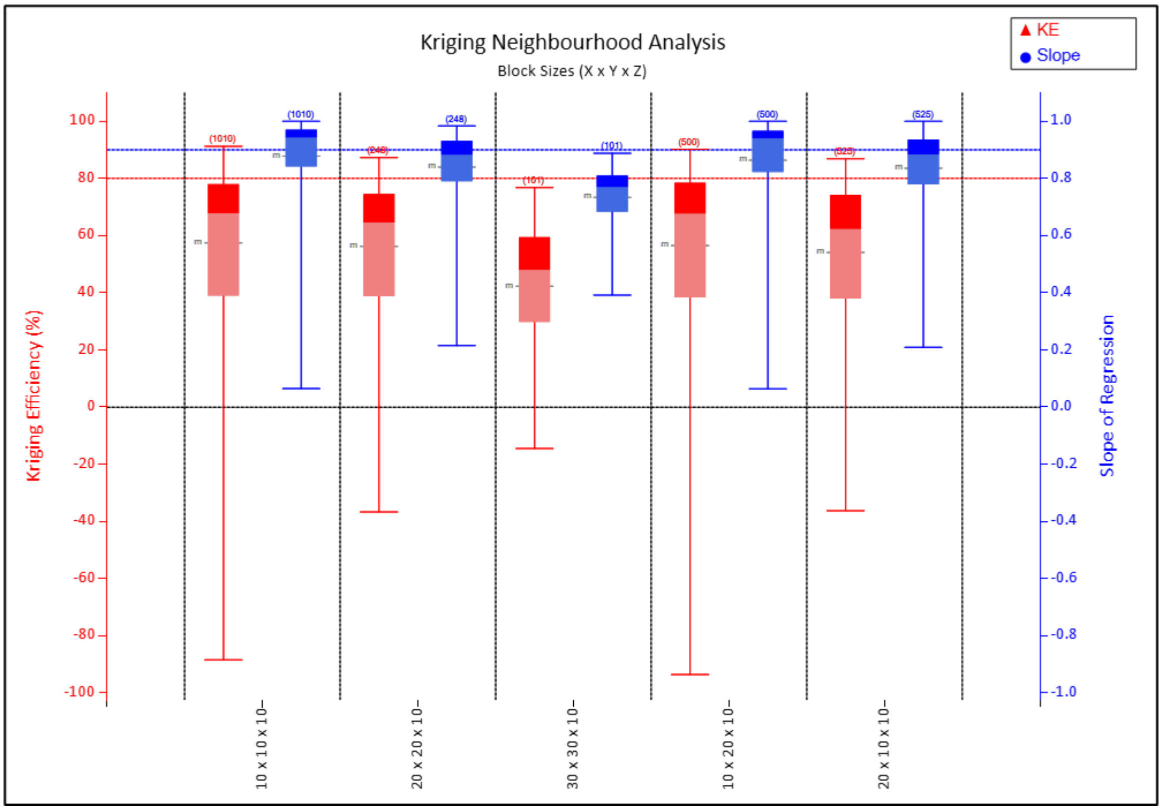
Kriging Neighbourhood Analysis (KNA) was performed for the largest domain (ESTZON 101) in Supervisor software to determine optimal block sizes and to guide and inform the choice of sample search neighbourhoods. Two (2) methods were used: (1) averaging the slopes of regression and kriging efficiencies across a portion of the domain to identify the optimal block size/sample search strategy; and (2) choosing a single block location representing a well-informed block, a moderately informed block and a poorly informed block and reviewing the kriging quality statistics.

A block size of 10 m x 10 m x 10 m was chosen which, although small by comparison with the drilling grid which averages 30–60 m centres currently, resulted in the optimal slope of regression and kriging efficiency when compared to bigger blocks reviewed (Figure 14.26).

Minimum and maximum composite selection of 20 and 40 were chosen (Figure 14.27). High numbers were chosen to intentionally smooth the estimate, given the presence of coarse gold and associated risks in being able to precisely locate blocks above cut-off.

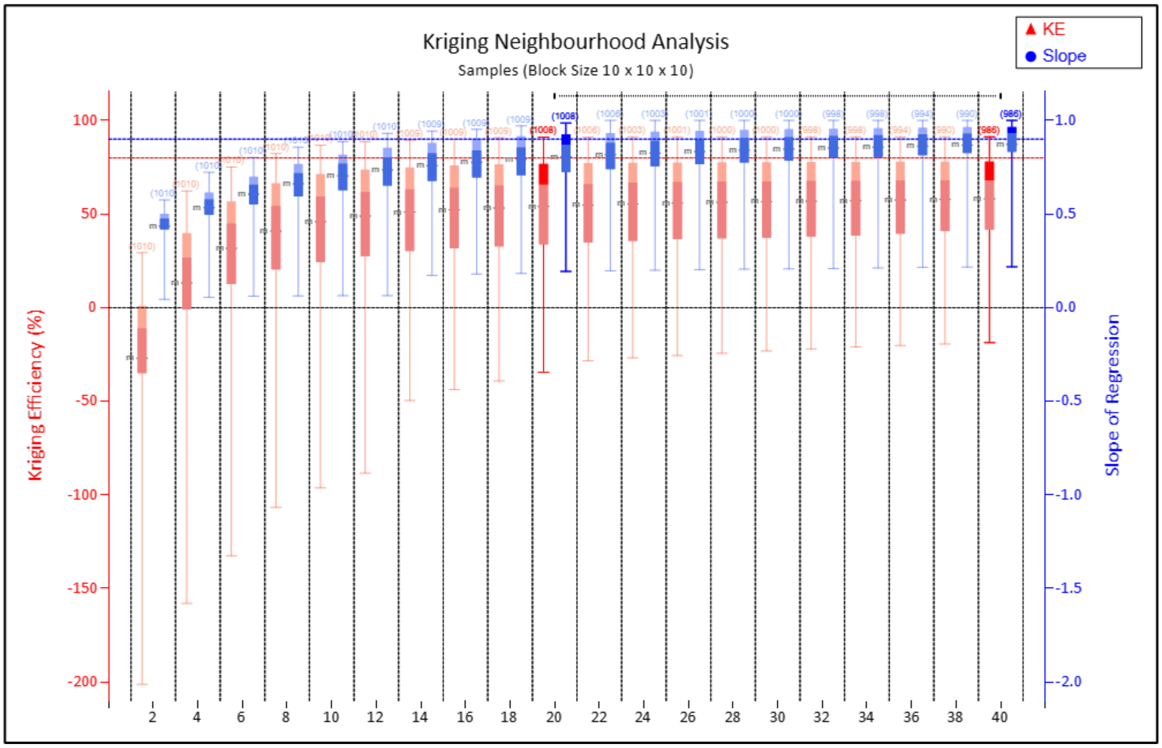
The first search pass was chosen to align broadly with the variogram ranges.

Figure 14.26 – KNA Block Size Review



Source: CSA Global, 2023

Figure 14.27 – KNA Sample Search Strategy Review



Source: CSA Global, 2023

14.13 Block Modelling

A volume block model was built in Datamine Studio RM using the geology, mineralisation, lithogeochemistry and topography wireframes. Block model volumes were validated against wireframe volumes and aligned well. The block model prototype is presented in Table 14.12 and the block model attributes are described in Table 13.11.

Table 14.12 – Block Model Prototype

Dimension	Minimum (m)	Maximum (m)	Extent (m)	Block Size	
				Parent Cell	Sub-Cell
Easting	572,600	573,200	600	10	1
Northing	4,895,500	4,896,400	900	10	1
Elevation	250	1,000	750	10	1

Table 14.13 – Block Model Attributes – Final Model (cr_md231123.mre.dm)

Field Description	Field Name	Type/Unit	Values/Meaning	
Gold estimate	AU	Numeric: g/t	Variable	
Silver estimate	AG	Numeric: g/t	Variable	
Arsenic estimate	AS	Numeric: ppm	Variable	
Sulphur estimate	S	Numeric: %	Variable	
Copper estimate	CU	Numeric: %	Variable	
Mineralisation domains	MINZON	Numeric (Integer)	101	Hanging wall mineralisation
			102	Footwall mineralisation
			999	Waste
Oxidation	OXIDE	Numeric (Integer)	3	Strongly oxidised
			2	Partially oxidised
			1	Fresh
Mineral Resource classification	CLASS	Numeric (Integer)	3	Inferred
			9	Unclassified
Density	DENSITY	Numeric: t/m^3	2.71	Sequence felsic debris flow deposit unit
			2.65	VHM
			2.71	Marls – SFD
			Estimated	Marls
			Estimated	S1/S2
			Estimated	Late Sills
			2.67	Early mineralised porphyry
			2.62	Quartzite
			2.71	Marble
			2.69	Monzonite

Field Description	Field Name	Type/Unit	Values/Meaning	
Estimation zone	ESTZON	Numeric (Integer)	101	Hanging-wall excluding Late Sills
			102	Footwall excluding Late Sills
			200	Late Sills
			302	Waste – VHM
			308	Waste – Quartzite
			309	Waste – Marble
			310	Waste – Monzonite
			3045	Waste – Marls, S1/S2
			30137	Waste – SFD/Marls-SFD/Early min po
Geology	GEOL	Numeric (Integer)	1	Sequence felsic debris flow deposit unit
			2	VHM
			3	Marls – SFD
			4	Marls
			5	S1/S2
			6	Late Sills
			7	Early mineralised porphyry
			8	Quartzite
			9	Marble
			10	Monzonite
Lithogeochemical domain	LGC	Numeric (Integer)	3	SED03
			4	SED04 and SED05
Nearest hole name	BHID	Alpha numeric	Variable	
Mineral Resource	MRE	Numeric (Integer)	1	In MRE (supported by RPEEE)
			0	Outside of MRE (not classified)

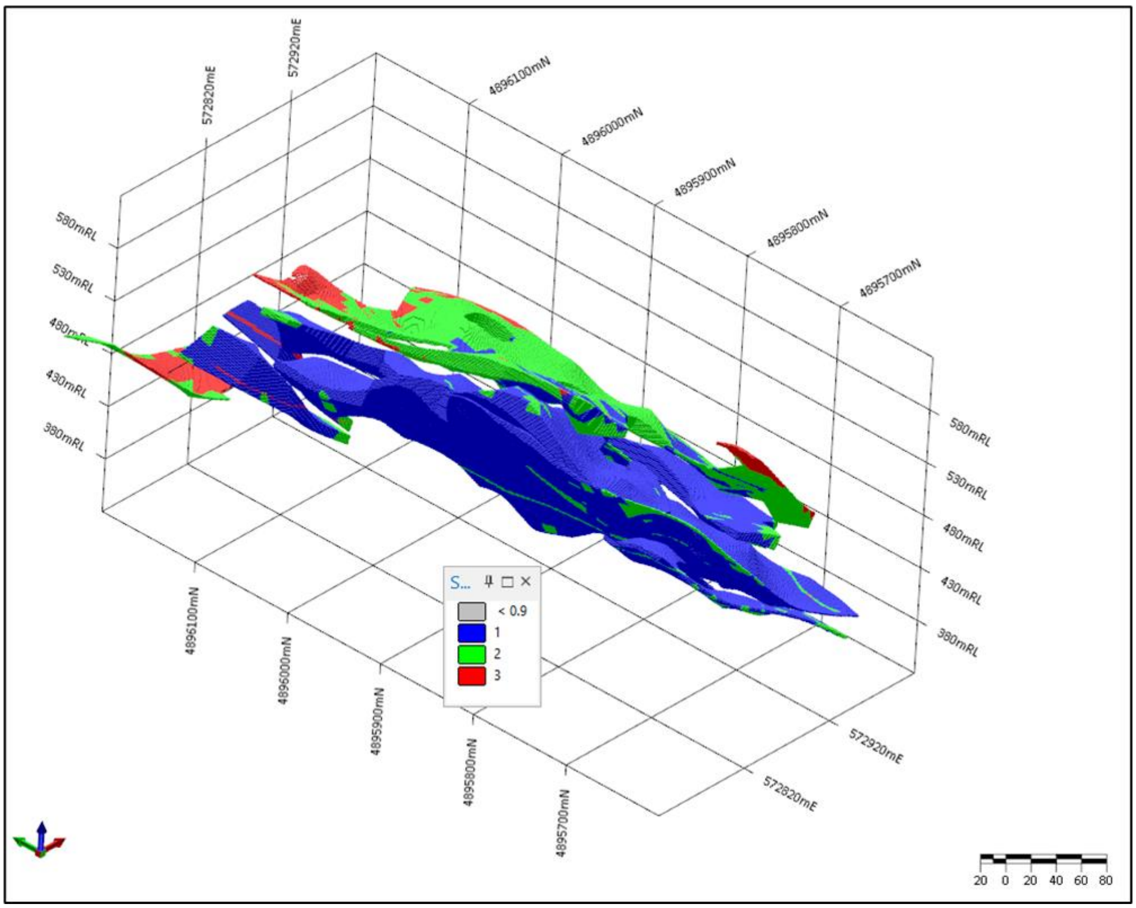
14.14 Grade Estimation

Grades were estimated using Ordinary Kriging in Datamine Studio RM. A three-pass search strategy was used. Table 14.14 shows the percentage of blocks estimated in each search pass and Figure 14.28 shows the model coloured by search pass. Large search ranges and high numbers of composites were used to intentionally smooth the estimate, which is recommended for deposits with coarse gold. Dynamic anisotropy was used to locally guide the rotation of the search ellipse to align with the undulating and variable orientations of the mineralised skarn and post-mineralisation intrusives. Grades were estimated into parent cells only, with sub cells receiving the grade of the parent.

Table 14.14 – Percentage of Blocks Estimated in Each Search Pass

Variable	Search Pass	% Volume
Gold	1	78
	2	19
	3	3
Silver	1	65
	2	30
	3	6

Figure 14.28 – Isometric View of the Model, Looking Northeast, Coloured by Search Pass for Gold



Source: CSA Global, 2023

Discretisation of 10 x 10 x 10 (X x Y x Z) was used – 10 in the Z direction chosen because 1 m composites were used, and best practice is to align composites with the height of the block in the Z direction. High discretisation was used in X and Y so that the block variance stabilised in the estimate.

Kriging and other estimation statistics were written out to the block model including search pass, slope of regression, kriging variance, kriging efficiency, number of samples used to estimate block since these statistics help to evaluate the quality of the estimate. These statistics are removed from the final model provided to engineers for downstream use.

The sample search neighbourhood is presented in Table 14.15. The third search pass was expanded to a very large range to simply fill a small number of blocks at the periphery that remained un-estimated.

Check estimates were also run on uncut (uncapped) variables and using inverse distance weighting squared (IDW^2). Grades were estimated into waste domains using IDW^2.

Table 14.15 – Sample Search Neighbourhood for Grade Estimates

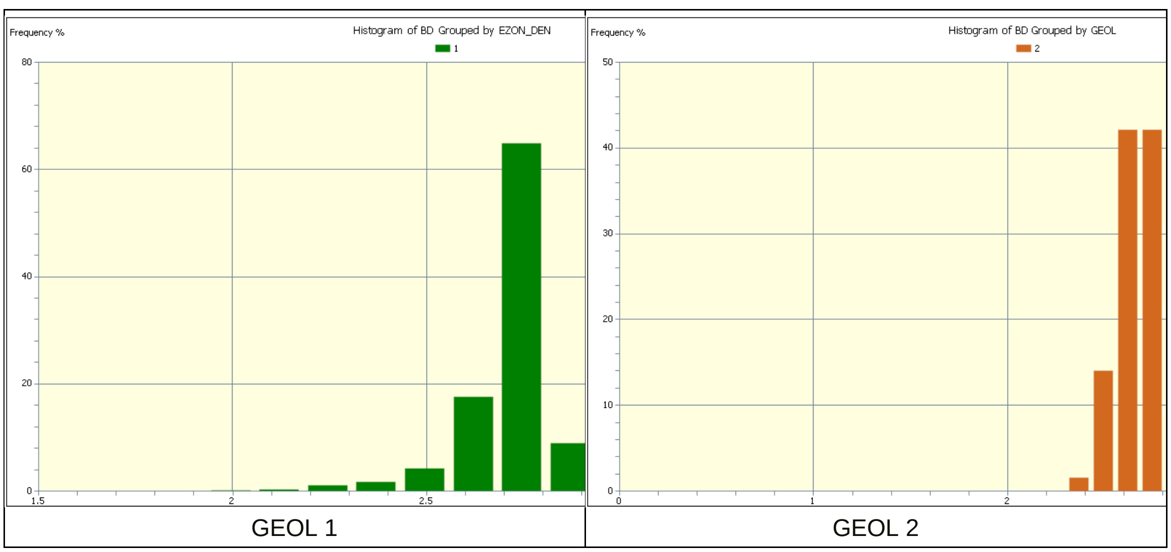
Domains	Search Pass	Search 1	Search 2	Search 3	Minimum Composites	Maximum Composites	Maximum per DH
Gold, Silver Estimate (and Arsenic, Sulphur)							
101	1	125	100	15	20	40	5
	2	250	200	30	20	40	
	3	500	400	60	10	30	
102	1	75	50	10	16	30	4
	2	150	100	20	16	30	
	3	225	150	30	8	30	
200	1	75	50	10	16	30	4
	2	150	100	20	16	30	
	3	375	250	50	8	30	
Waste Domains 301, 302, 308, 309, 310, 3045, 30137	1	125	100	15	20	40	5
	2	250	200	30	20	40	
	3	1,250	1,000	150	10	30	

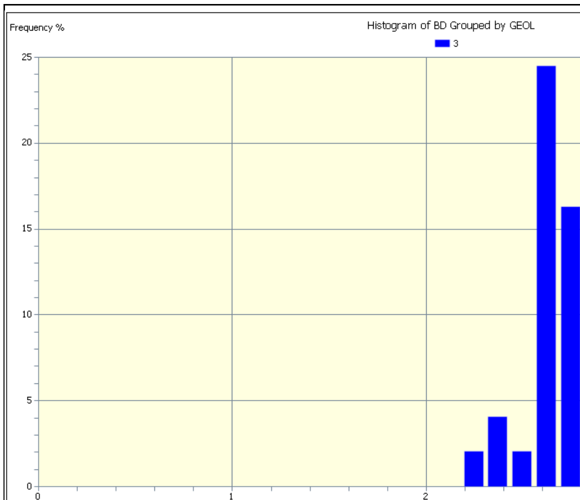
Domains	Search Pass	Search 1	Search 2	Search 3	Minimum Composites	Maximum Composites	Maximum per DH
Copper Estimate							
5101	1	125	100	15	20	40	5
	2	250	200	30	20	40	
	3	1,250	1,000	150	10	30	
5102	1	75	50	10	16	30	4
	2	150	100	20	16	30	
	3	750	500	100	8	30	
5200	1	75	50	10	16	30	4
	2	150	100	20	16	30	
	3	750	500	100	8	30	

14.15 Bulk Density Estimate/Assignment

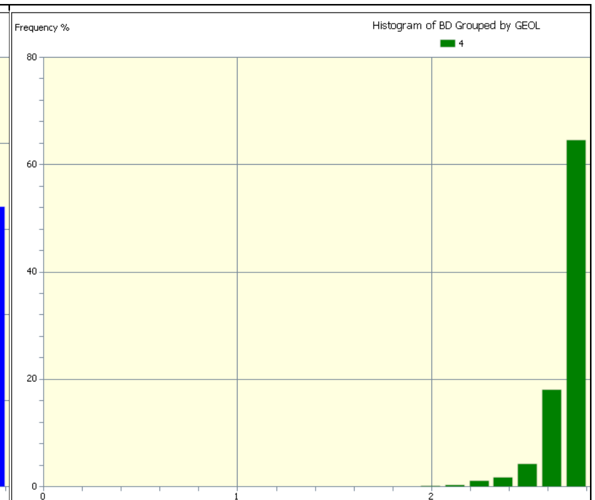
In-situ dry bulk density (BD) measurements were analysed by reviewing histograms by modelled lithology (Figure 14.29). Some zones had a narrow range of measured densities, in which case the mean BD was assigned to blocks within that lithology. Certain lithologies had a wide range of BD values, often reflective of the degree of alteration and in particular skarnification that the rock had been subject to. In those cases, BD was estimated using IDW^2 to reflect the internal variability of the given lithology.

Figure 14.29 – Histograms Showing Measured BD





GEOL 3



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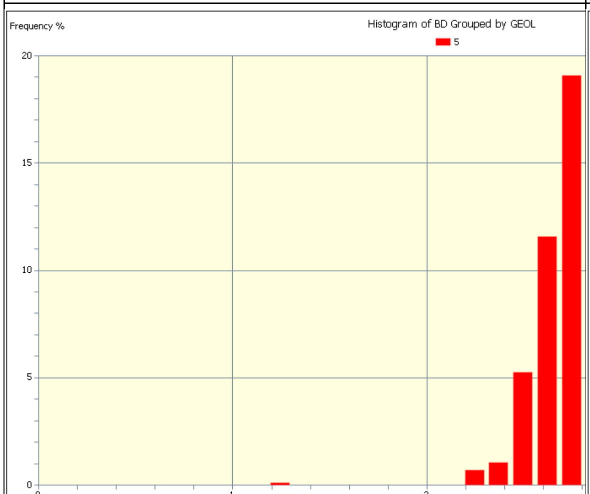
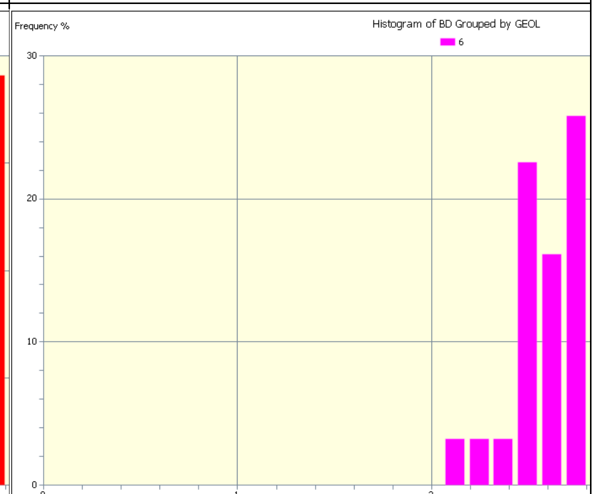
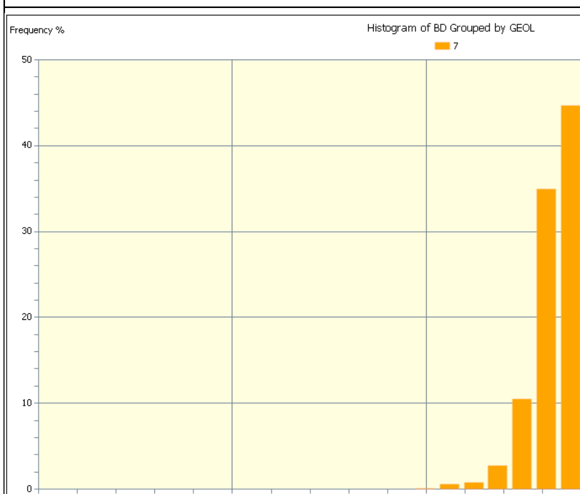


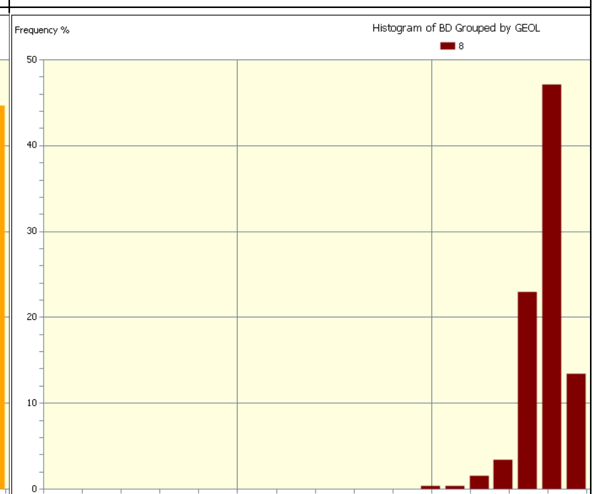
GEOL 5



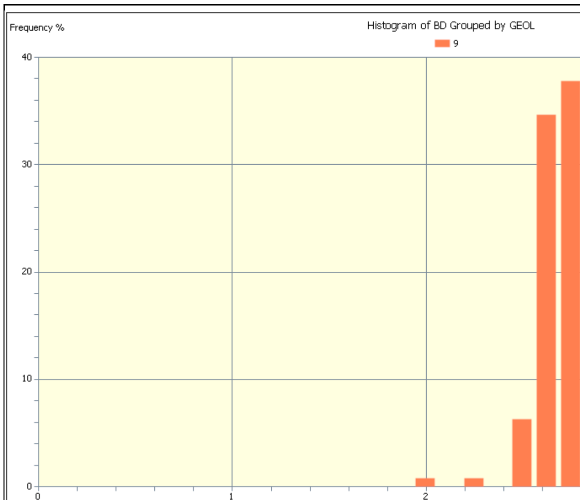
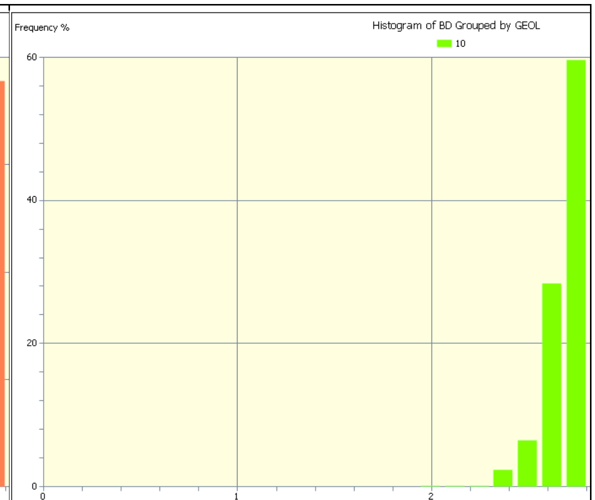
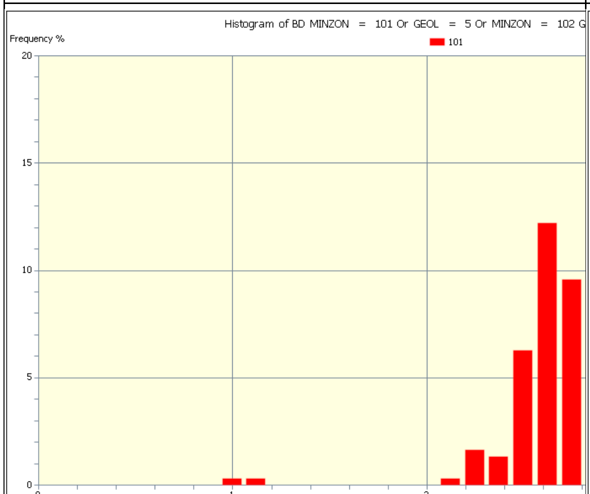
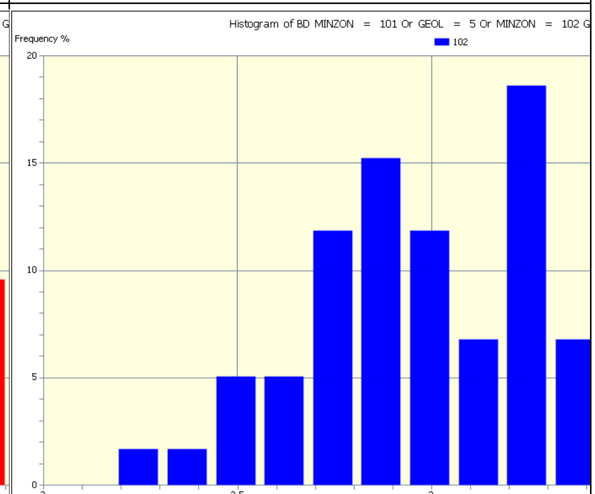
GEOL 6



GEOL 7



GEOL 8

	
GEOL 9	GEOL 10
	
ESTZON 101	ESTZON 102

Source: CSA Global, 2023

Table 14.16 presents the methodology and values (where applicable) used to assign or estimate BD. For all units except post-mineralisation intrusives and mineralised skarn, the mean BD was assigned. Outliers were removed when reviewing histograms and preparing estimation composites, since in most cases they are likely to be measurement errors instead of true outliers.

Table 14.16 – Methods to assign BD by Lithology

Domain	Description	Method	Mean assigned if applicable (t/m^3)
GEOL=1	SFD	Mean BD assigned	2.71
GEOL=2	VHM	Mean BD assigned	2.65
GEOL=3	Marls – SFD	Mean BD assigned	2.71
GEOL=4	Marls	Estimated using IDW^2	2.70 (only assigned to un-estimated blocks)
GEOL=5	S1/S2	Estimated using IDW^2	2.97 (only assigned to un-estimated blocks)
GEOL=6	Post-mineralisation intrusives	Estimated using IDW^2	2.73 (only assigned to un-estimated blocks)
GEOL=7	Early mineralised porphyry	Mean BD assigned	2.67
GEOL=8	Quartzite	Mean BD assigned	2.62
GEOL=9	Marble	Mean BD assigned	2.71
GEOL=10	Monzonite	Mean BD assigned	2.69
ESTZON=101	Mineralised skarn footwall	Estimated using IDW^2	-
ESTZON=102	Mineralised skarn hanging-wall	Estimated using IDW^2	-

For those domains where BD was estimated, the search engaged is presented in Table 14.17. Dynamic anisotropy was used to orient the search ellipse locally and has been described in more detail in Section 14.14. Estimated BD values were validated via statistical checks (Table 14.19), visual inspection and swath plots (Figure 14.36 to Figure 14.38).

Table 14.17 – Sample Search Neighbourhood for BD Estimate

Search Pass	Search 1	Search 2	Search 3	Minimum Composites	Maximum Composites	Maximum per DH
1	125	100	15	20	40	5
2	250	200	30	20	40	
3	500	400	60	10	30	

14.16 Block Model Validation

The estimated block model was validated in the following ways:

- Comparison of volume estimates between the block model and the wireframe volumes (Table 14.18).
- Visual inspection of estimated grades in plan and in cross sections and comparison with the input composites (example cross sections presented in Figure 14.30 to Figure 14.33).
- Check for global bias by estimation pass and by domain – comparison of estimated and declustered composite statistics (Table 14.19).
- Check for local bias considering the supporting information – analysis of local trends in estimates using swath plots (Figure 14.34 to Figure 14.38).
- Checks to ensure the boundary conditions between estimation domains are honoured.

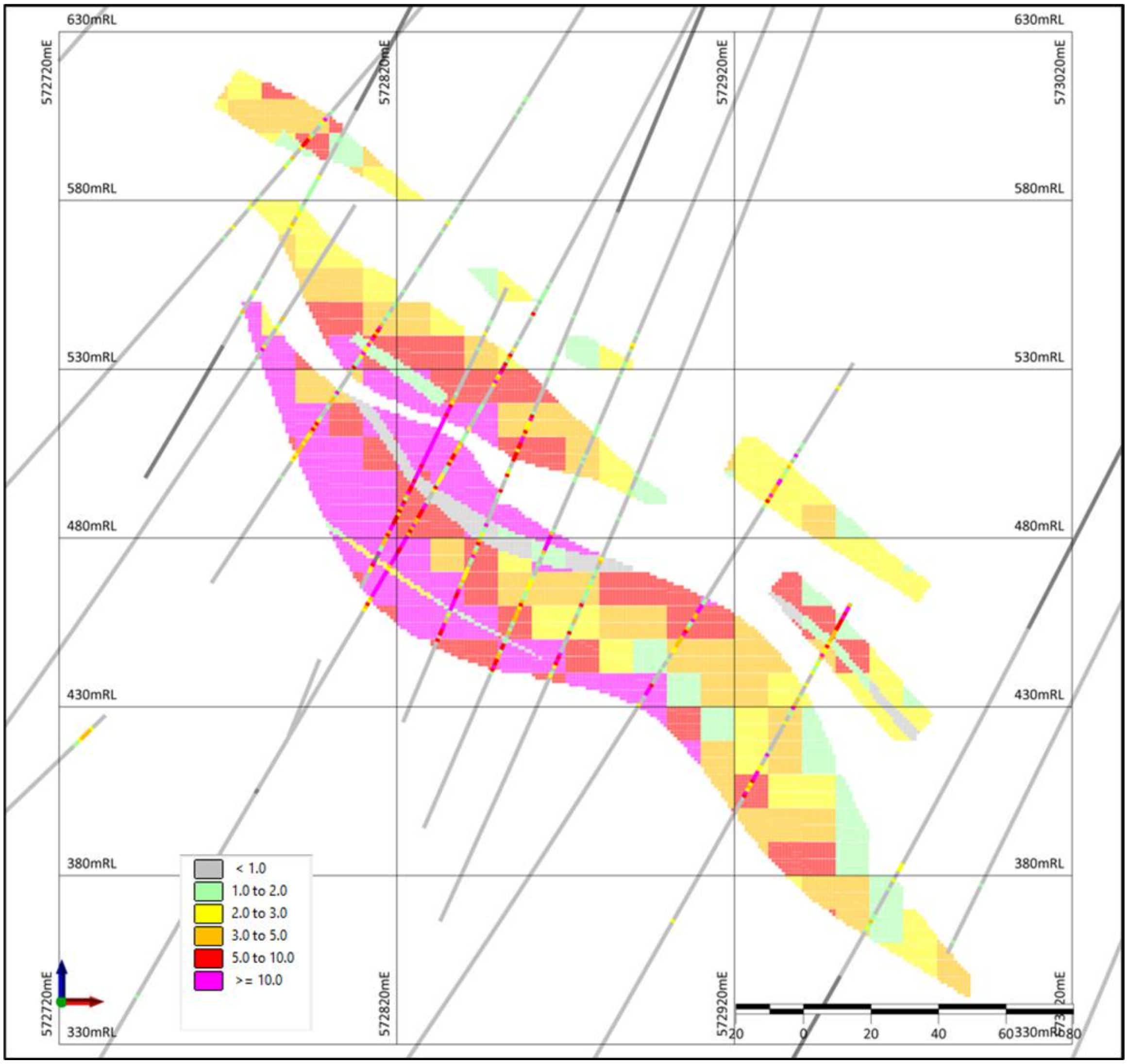
Volume checks show that the model has been built correctly based on the wireframes used. Grades are comparable on a global and domain-by-domain basis between estimation composites and blocks and are well within the 10% threshold that is considered reasonable.

Trends have been reviewed via swath plots to assess semi-local estimates and these also show good comparison for gold and silver. Smoothing is evident visually, in swath plots and in histograms where higher grades are underestimated and lower grades are overestimated. This is to be expected at this stage of resource development and indeed is intentional to mitigate the risks associated with coarse gold where the precise location of high grades is uncertain.

Table 14.18 – Global Statistics – Comparison of Block and Composite Grades

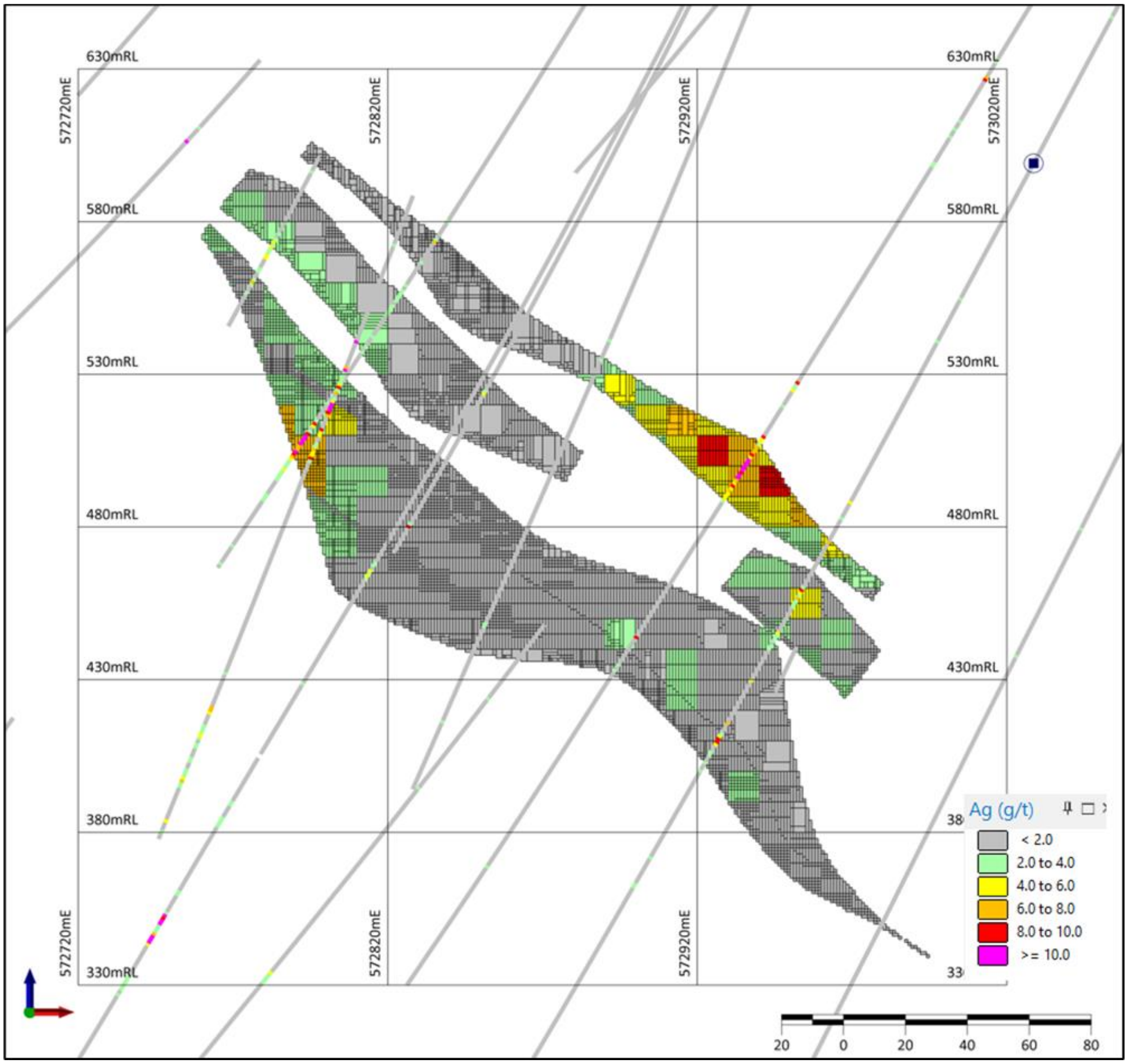
Variable	Domain Name	Domain	Composite Grade	Block Grade	% Difference
Au	ESTZON	101*	6.09	5.97	-2
	ESTZON	102	2.80	2.93	4
Ag	ESTZON	101	1.15	1.09	-6
	ESTZON	102	2.53	2.45	-3
Density	EZON_DEN	4	2.95	2.90	-2
	EZON_DEN	5	2.96	2.96	0
	EZON_DEN	6	2.73	2.74	1
	EZON_DEN	101	3.00	3.01	0
	EZON_DEN	102	3.06	3.08	1

Figure 14.30 – Cross Section at 4895850 m (±30 m) Looking North Showing Estimated Gold Grade and Input Composites



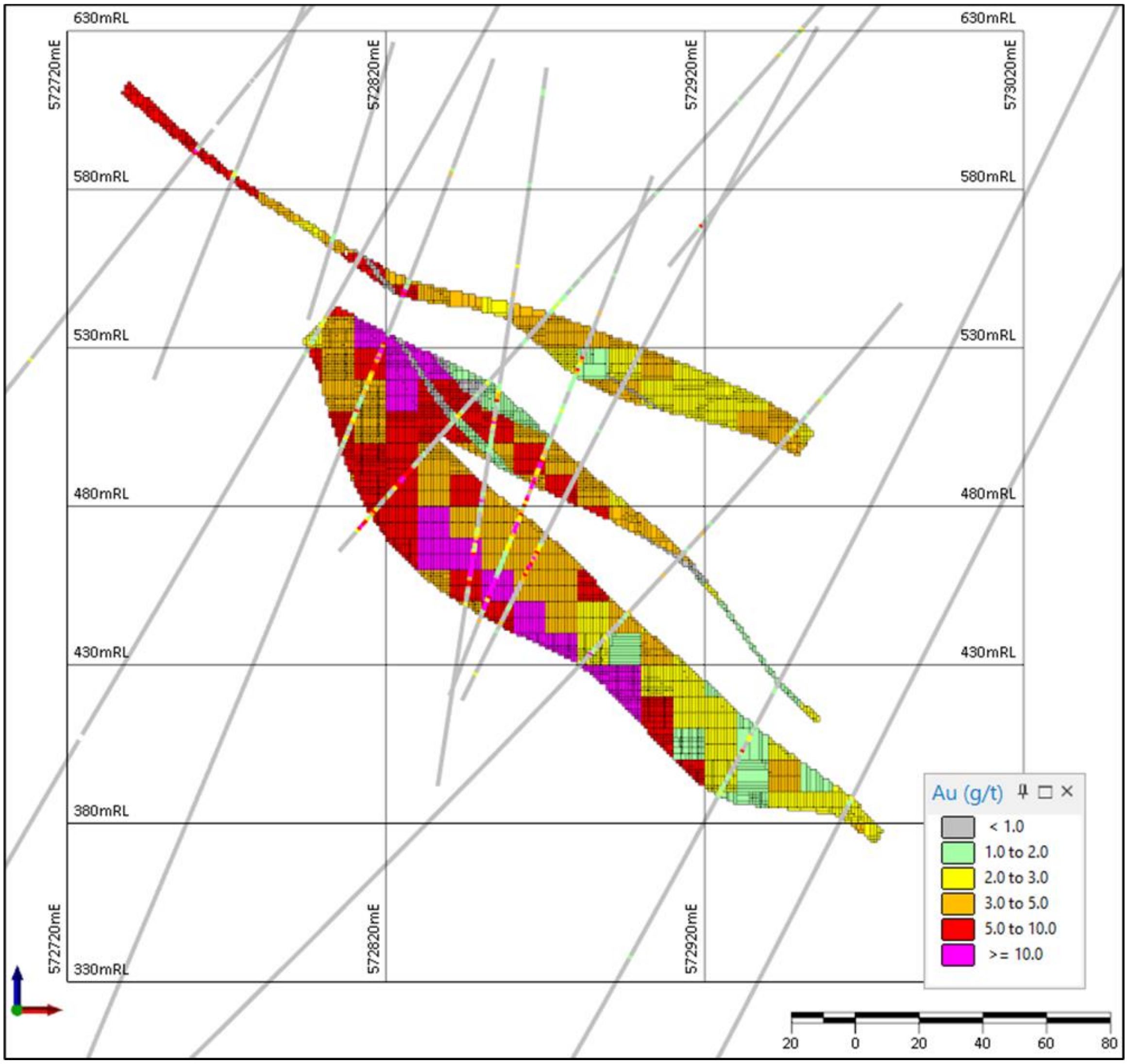
Source: CSA Global, 2023

Figure 14.31 – Cross Section at 4895820 m (±30 m) looking North Showing Estimated Silver Grade and Input Composites



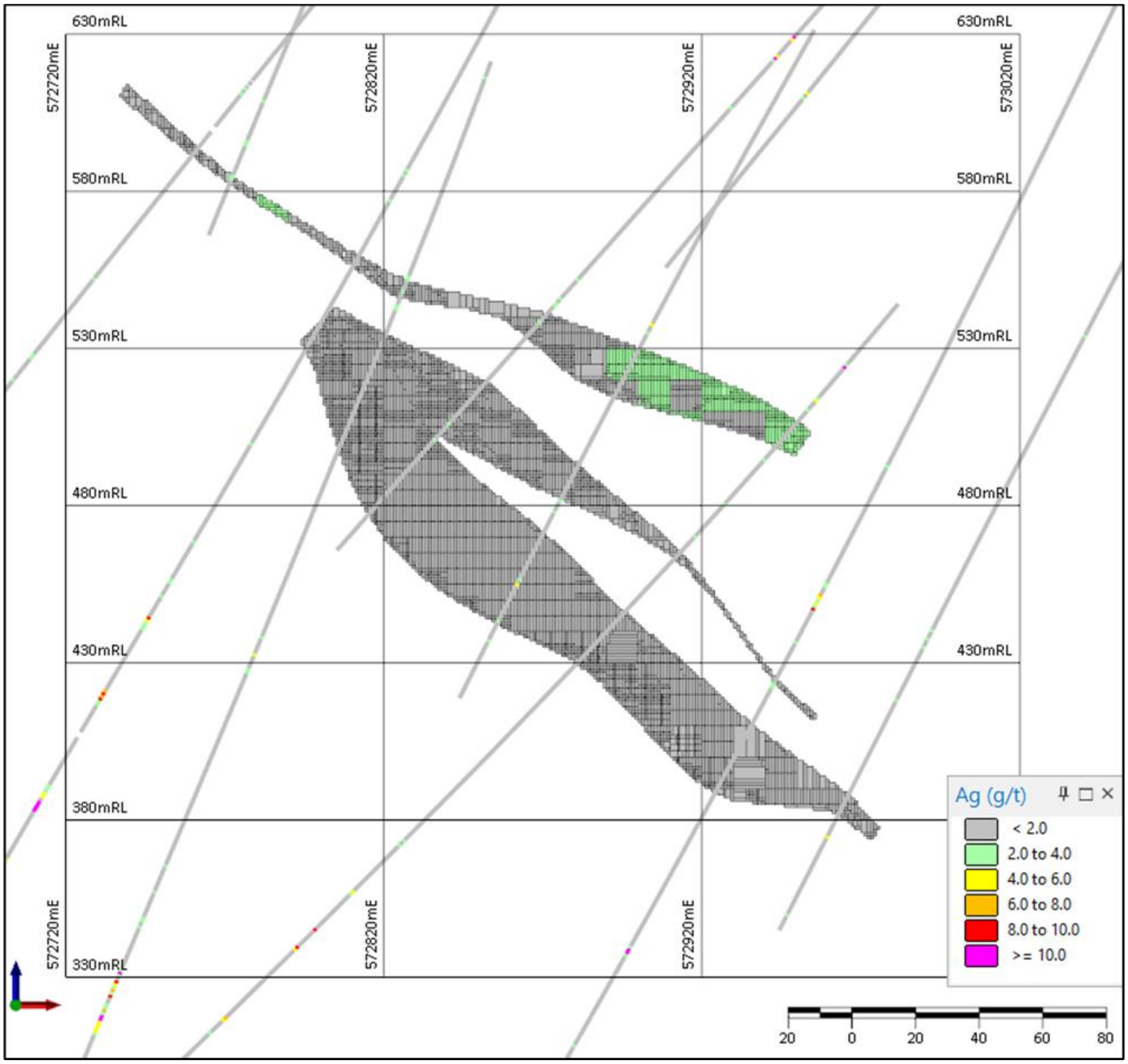
Source: CSA Global, 2023

Figure 14.32 – Cross Section at 4895940 m (±30 m) Looking North Showing Estimated Gold Grade and Input Composites



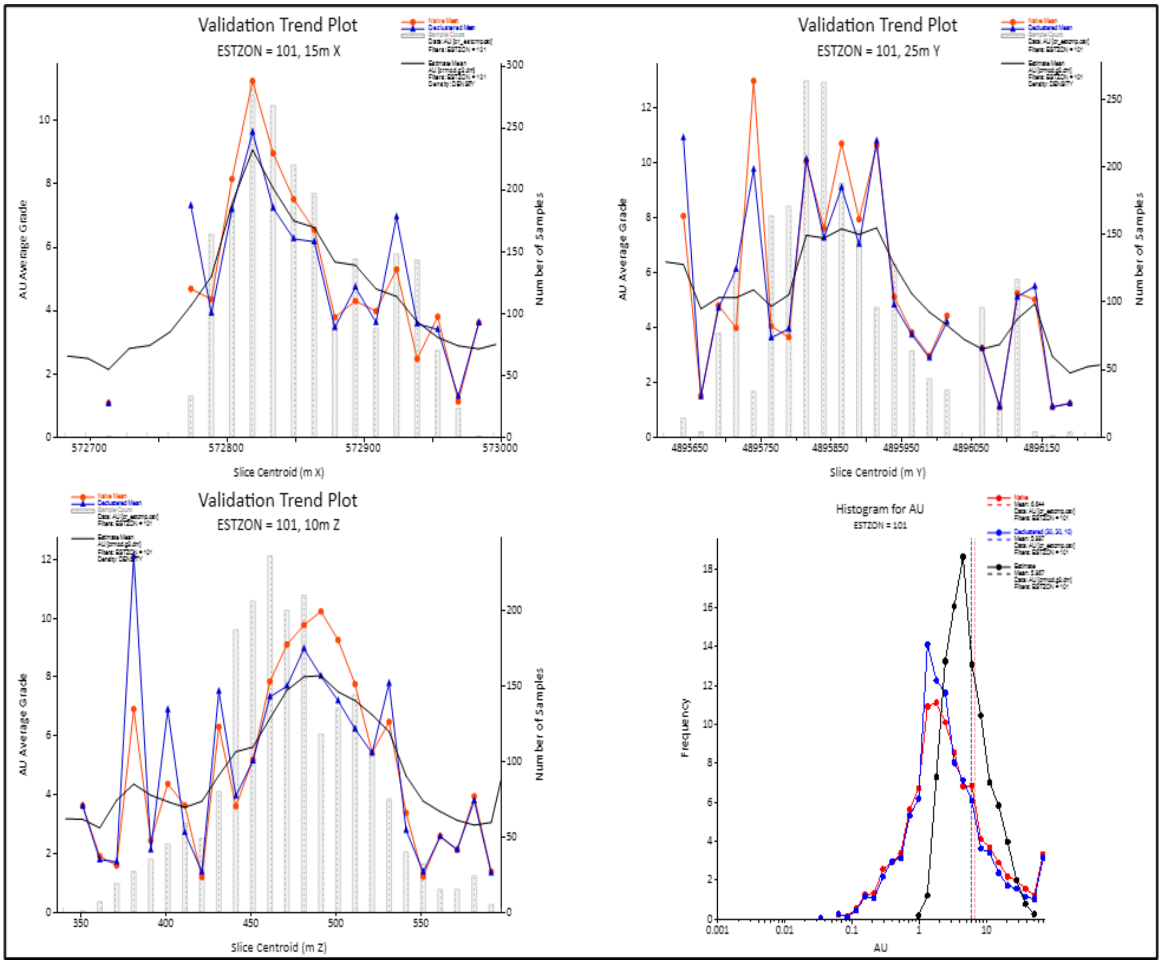
Source: CSA Global, 2023

Figure 14.33 – Cross Section at 4895940 m (±30 m) Looking North Showing Estimated Silver Grade and Input Composites



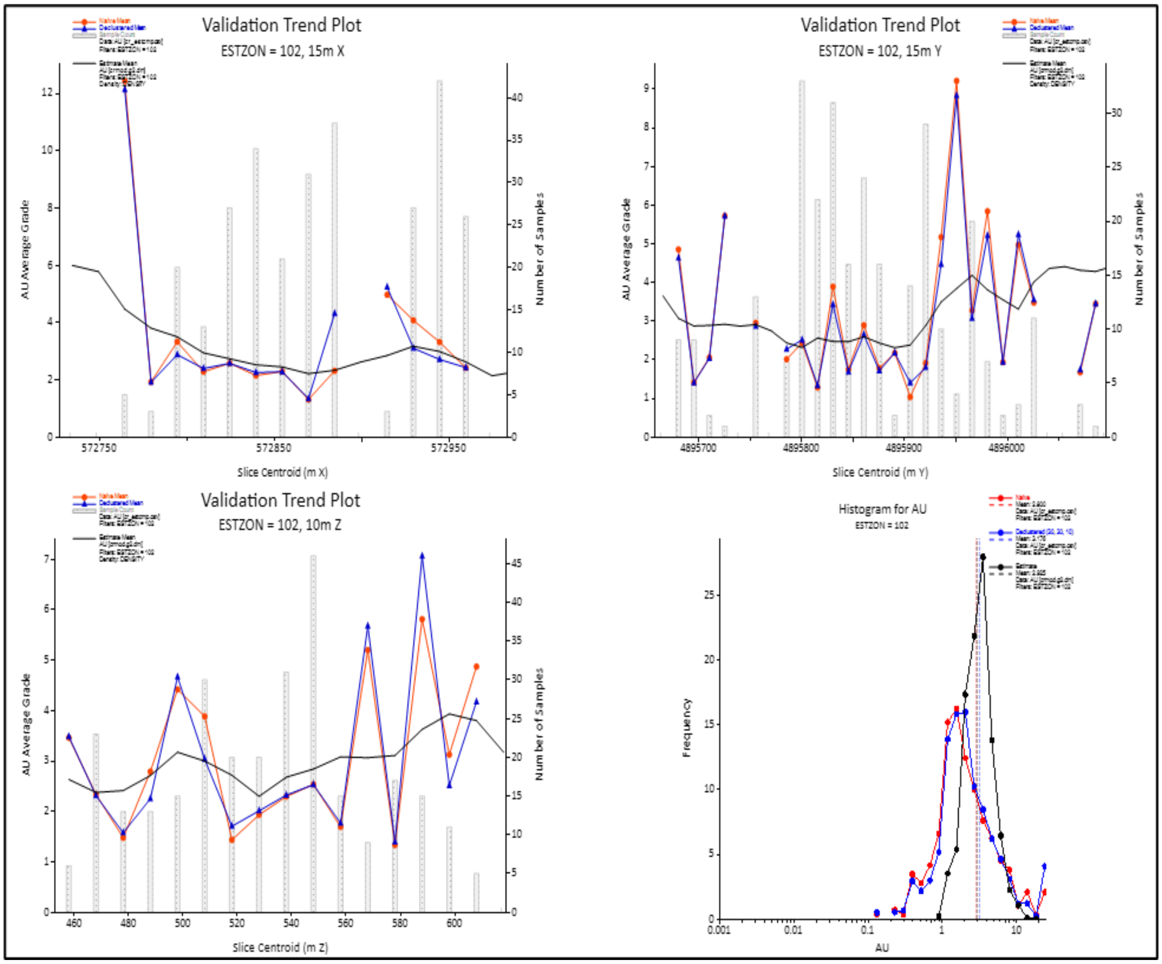
Source: CSA Global, 2023

Figure 14.34 – Swath Plots and Log Histogram for Au ESTZON 101



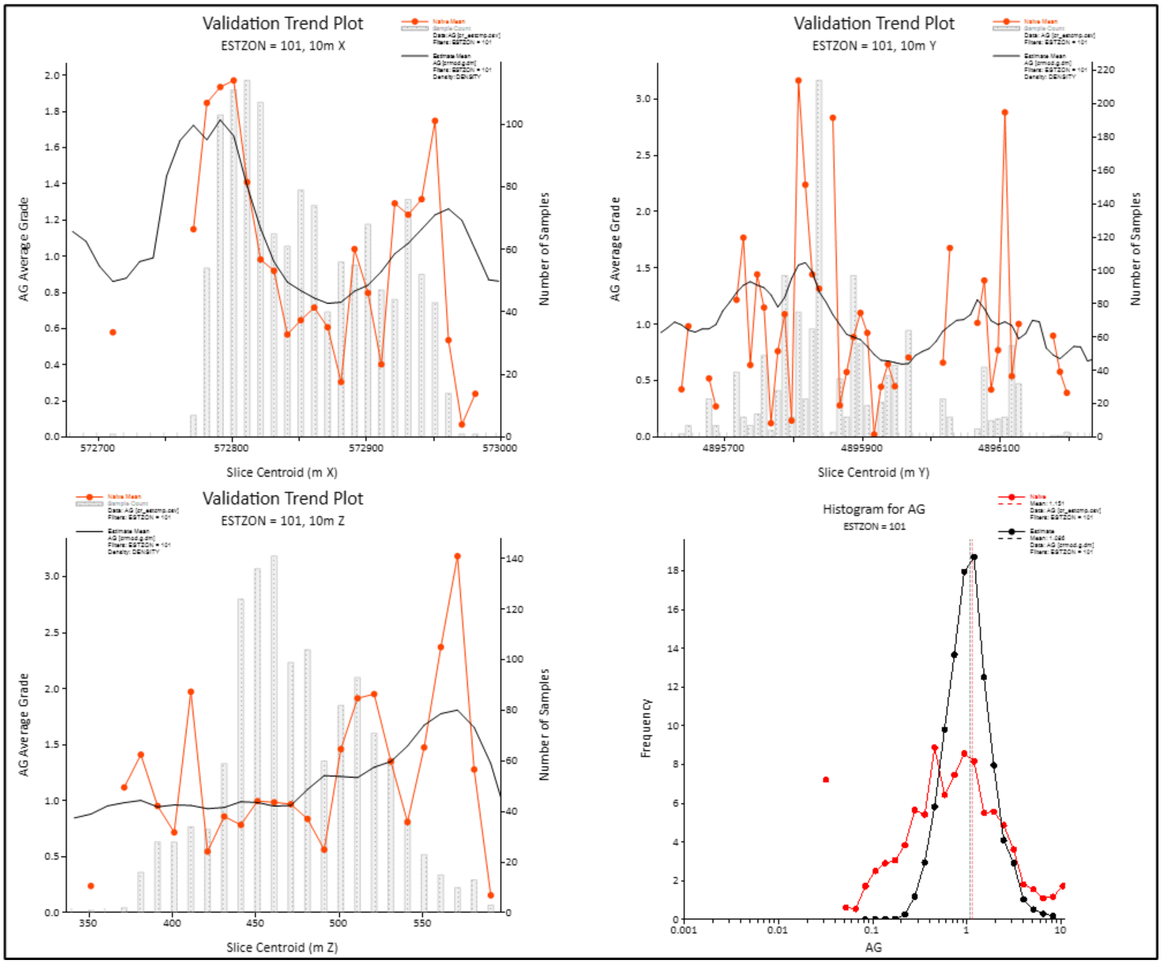
Source: CSA Global, 2023

Figure 14.35 – Swath Plots and Log Histogram for Au ESTZON 102



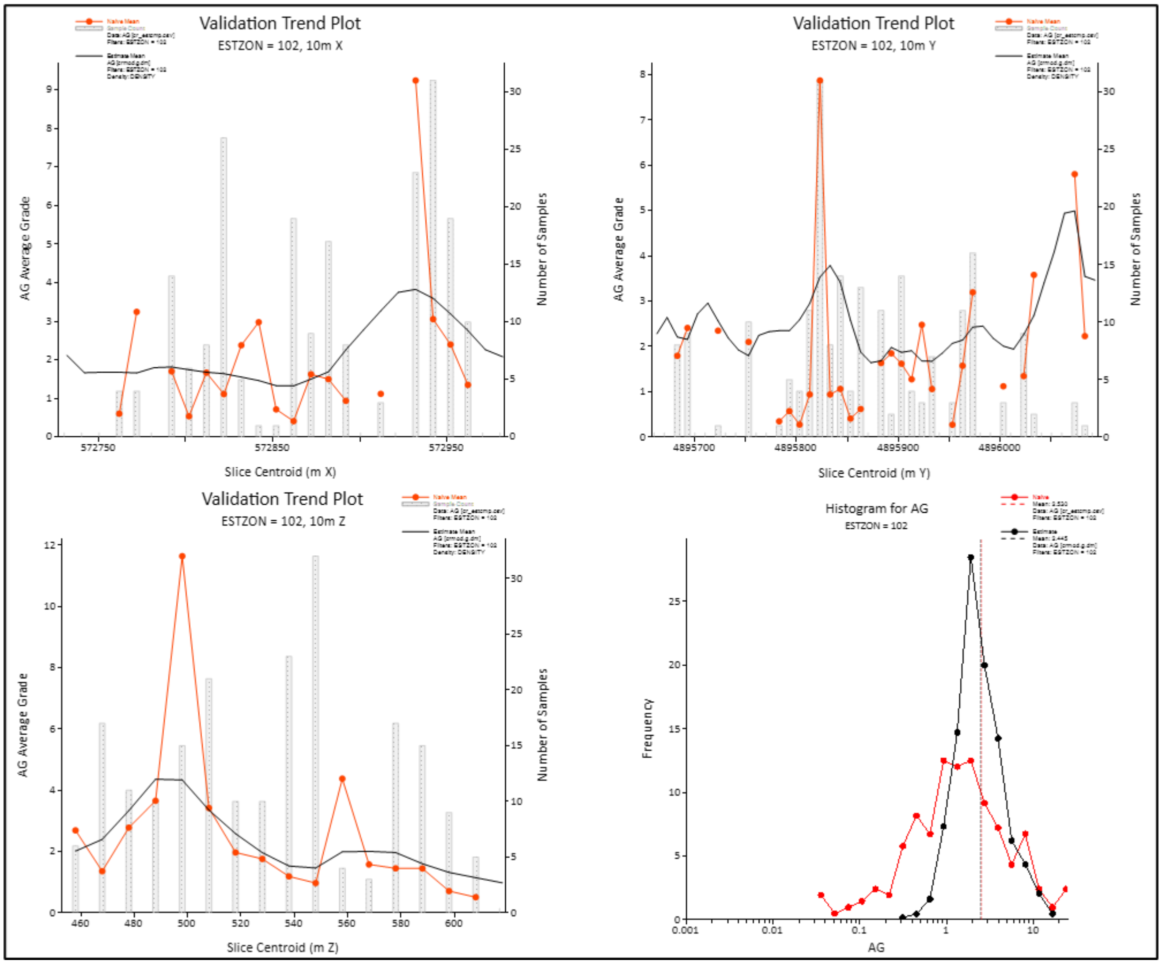
Source: CSA Global, 2023

Figure 14.36 – Swath Plots and Log Histogram for Ag ESTZON 101



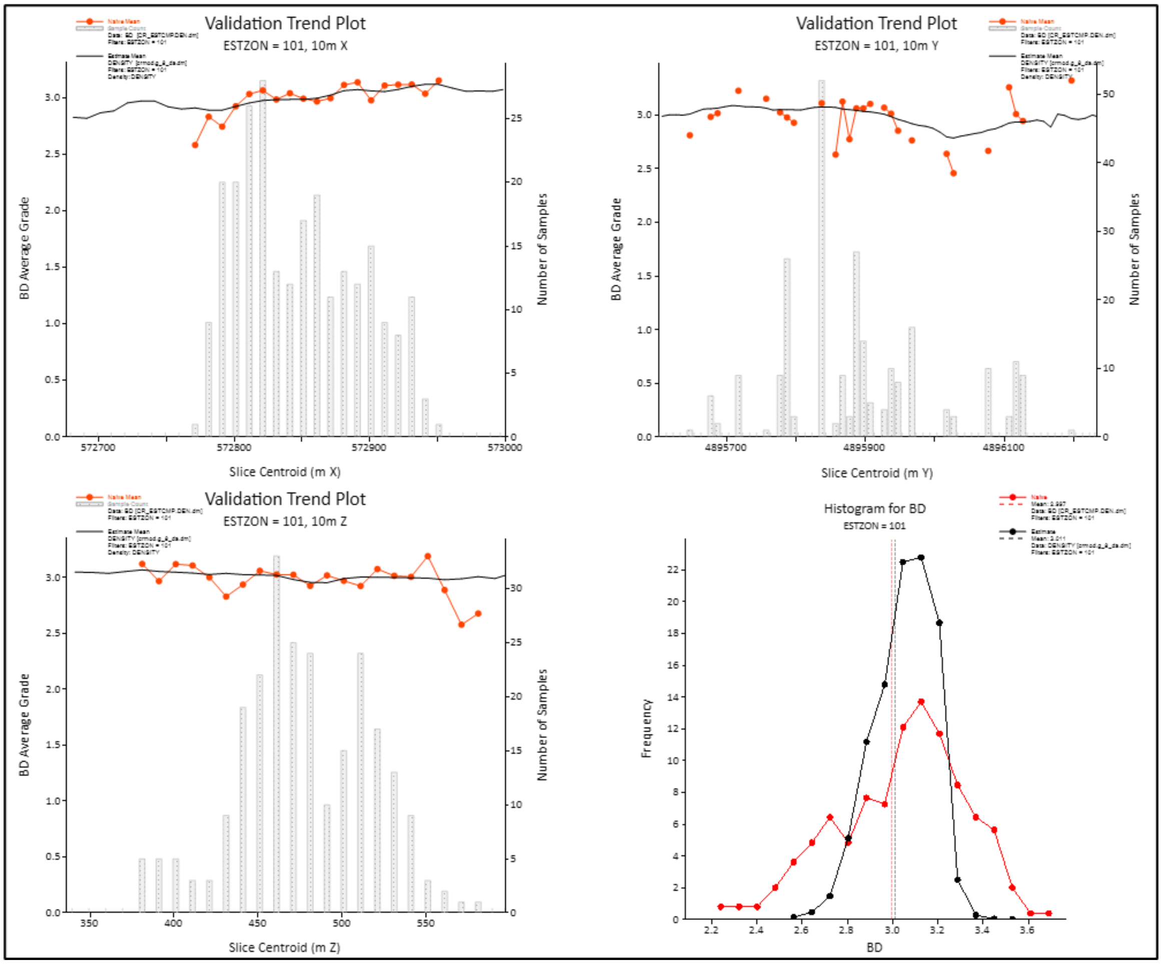
Source: CSA Global, 2023

Figure 14.37 – Swath Plots and Log Histogram for Ag ESTZON 102



Source: CSA Global, 2023

Figure 14.38 – Swath Plots and Log Histogram for Density ESTZON 101



Source: CSA Global, 2023

14.17 Mineral Resource Classification

14.17.1 Determination of Reasonable Prospects for Eventual Economic Extraction (RPEEE)

A breakeven cut-off grade (COG) of 2 g/t Au (rounded from 2.23 g/t Au) at US$1,700/oz of gold and a minimum width constraint of 5.0 m x 5.0 m x 2.5 m was used to define optimised underground potentially mineable shapes using Datamine's Mineable Shape Optimiser (MSO) to determine RPEEE of the block model and classify and report Mineral Resources for the Project. The COG breakdown and cost assumptions are shown in Table 14.19 to Table 14.21. In collaboration with the QP and DPM's Mining Engineers, reasonable parameters were chosen for the MSO process and these are presented in Table 14.22. Figure 14.39 shows the MSO volumes generated, and the resultant smoothed out version used for RPEEE.

Table 14.19 – COG Calculation and Cost Assumptions for Čoka Rakita MRE – Cost per Tonne

Description	Cost / t (US$/t)
Underground mining costs	35.17
Process costs	27.50
G&A costs	16.00
Sustaining capital	7.00
Total $/t	**85.67**

Table 14.20 – Commercial Terms Used Within the COG Calculation

Item	Unit	Cost per gold ounce
Concentrate transportation	US$/dmt	125.00
Concentrate treatment	US$/dmt	200.00
Concentrate refining	US$/dmt	24.11
Concentrate penalty	US$/dmt	0.00
Concentrate grade – gold	g/t	100
Concentrate gold payable	%	97.8
Royalty	%	5
Total US$ per gold ounce		**108.59**
Total US$ per gold gram		**3.49**

Table 14.21 – COG Calculation for MRE

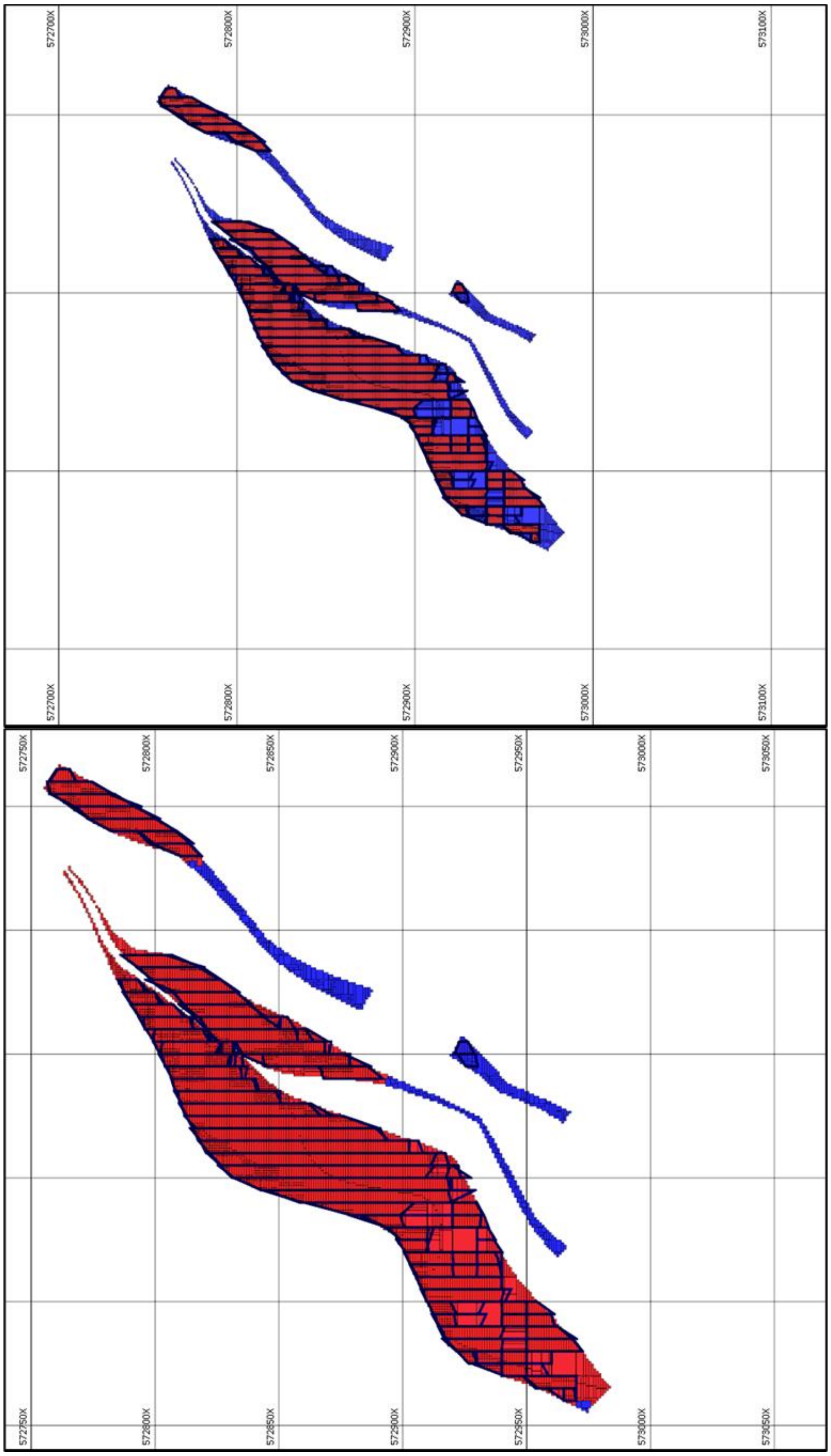
Item	Unit	Calculation
Gold price (ounce)	US$/oz	1,700
Gold price (gram)	US$/g	54.66
Revenue	US$/g	48.11
Less royalty	5% of Sales	($2.41)
Less per gold gram costs	US$/g	(3.49)
Realised revenue	US$/g	42.21
Cost per tonne to produce	US$/t	85.67
MRE cut-off grade	g/t	2.23 (rounded to 2 g/t)

Notes:
- Processing costs: US$21.0/t Process + US$6.5/t Dry Tailings + Paste Fill.
- Transportation cost: Transport cost assumption based on 500 km overland transport to smelter.
- Gold concentrate is subject to 1 gram Au g/t deductible, i.e. 1 g of Au is not payable.
- Revenue calculation assumes gold metallurgical recovery of 90% and gold payability of 97.8%.
- Calculated cut-off grade assumes mining dilution of 10%.

Table 14.22 – MSO Parameters

Parameter Name	Parameter Setting	Value	Unit
Block model settings			
Optimisation field/default value	AU	0	g/t Au
Density field/default value	DENSITY	2.7	t/m^3
Underground mining optimisation method for gold			
Objective	Maximise stope grade/value above cut-off		
Method	Cut-off grade	2	g/t Au
MSO stope parameters			
Framework type	Vertical, mineralised body strike along Y axis		
Section and level intervals	U (Y axis/stope width)	5	m
	V (Z axis/level height)	5	m
Stope depth (Z axis/stope width in MSO)	Minimum	2.5	m
	Maximum	1,000	m
Dilution	ELOS dilution	0	m
Stope dip angles	Minimum	0	°
	Maximum	180	°
Stope strike angles	Minimum	-45	°
	Maximum	45	°
Material configuration	Exclude CLASS 9 material		
Advanced slice options	Ignore pillar requirements between full shapes		
	Ignore pillar requirements in split stope shapes		

Figure 14.39 – Cross Section at 4895880 m (±30 m) Looking North Showing MSO (top image) and Smoothed Version (bottom image)



Source: CSA Global, 2023

14.17.2 Mineral Resource Classification

The maiden MRE for the Project has been classified as Inferred Mineral Resources using the meanings ascribed by the CIM Definition Standards on Mineral Resources and Mineral Reserves (May 2014) and set out below.

An Inferred Mineral Resource is that part of a Mineral Resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity.

An Inferred Mineral Resource has a lower level of confidence than that applying to an Indicated Mineral Resource and must not be converted to a Mineral Reserve. It is reasonably expected that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with continued exploration and drilling.

Mineral Resources for the Project were classified in accordance with the CIM definitions above, and the following was taken into consideration by the QP:

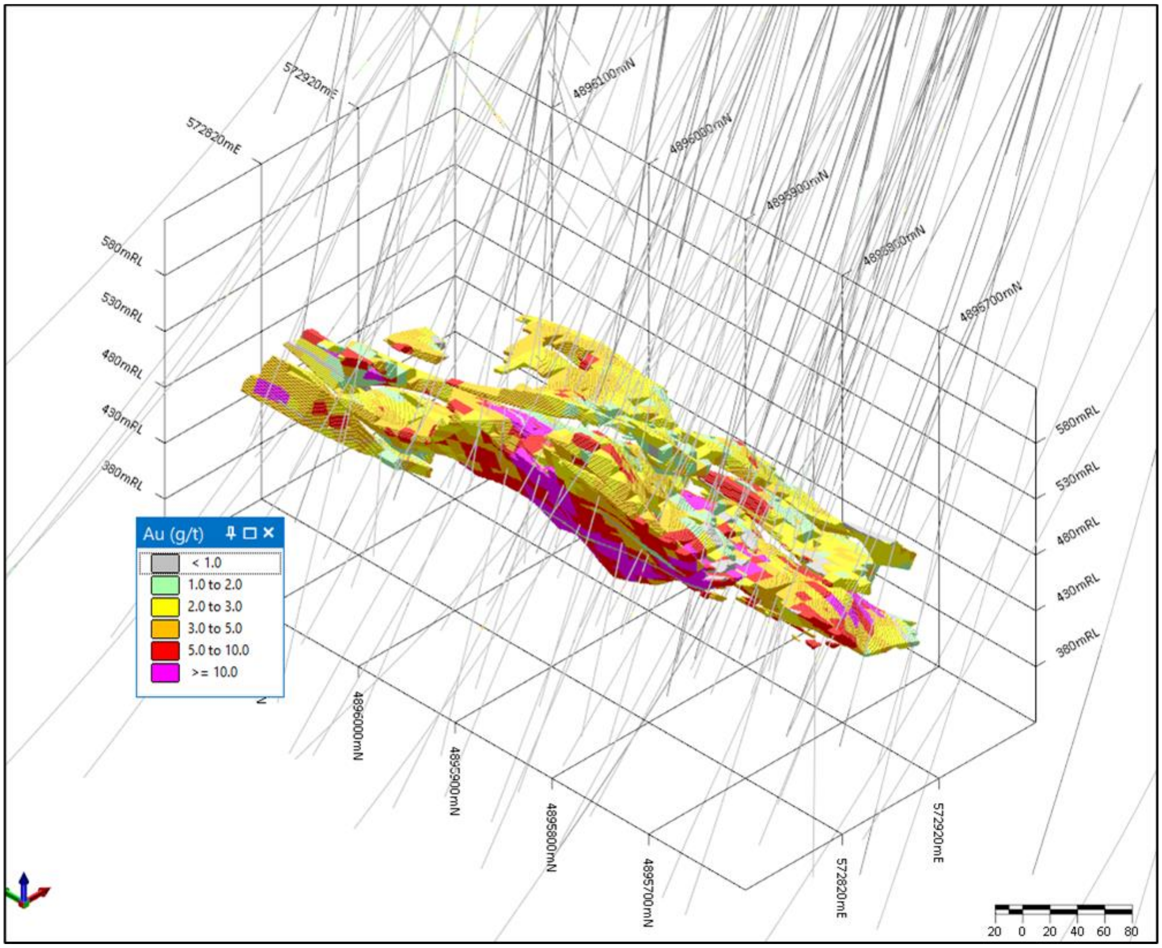
- Geological knowledge and reliability of interpretation;
- Sampling, assaying procedures, QA/QC and database verification;
- Sample support and drill density;
- Grade continuity and variography;
- Ordinary kriging statistics;
- Validation of the estimation of in-situ grades for gold; and
- Validation of the tonnage factors derived from estimation of the in-situ dry bulk density.

There is reasonable confidence in the geological continuity of the mineralisation, given the geological characteristics used in the modelling such as the intensity of skarnification within the S1/S2 sediments. However, the high gold grades and in particular, the presence of coarse gold associated with the higher grades in the deposit has an inherent higher risk attached to factors across the process from sampling and assaying to modelling and grade estimation.

The drill spacing is broadly at 60 m centres, decreasing to 30 m centres in the core. The spacing relative to the variogram ranges is broad, and therefore confidence in the grade continuity as modelled is low by definition.

A 3D view of the Inferred Mineral Resource model is presented in Figure 14.40.

Figure 14.40 – 3D View of the Classified Model, Coloured by Gold (Looking Northeast with Supporting Drillholes)



Source: CSA Global, 2023

14.18 Mineral Resource Reporting

The maiden MRE for the Čoka Rakita Project is presented in Table 14.23.

Table 14.23 - Čoka Rakita MRE using Underground Mining Scenario

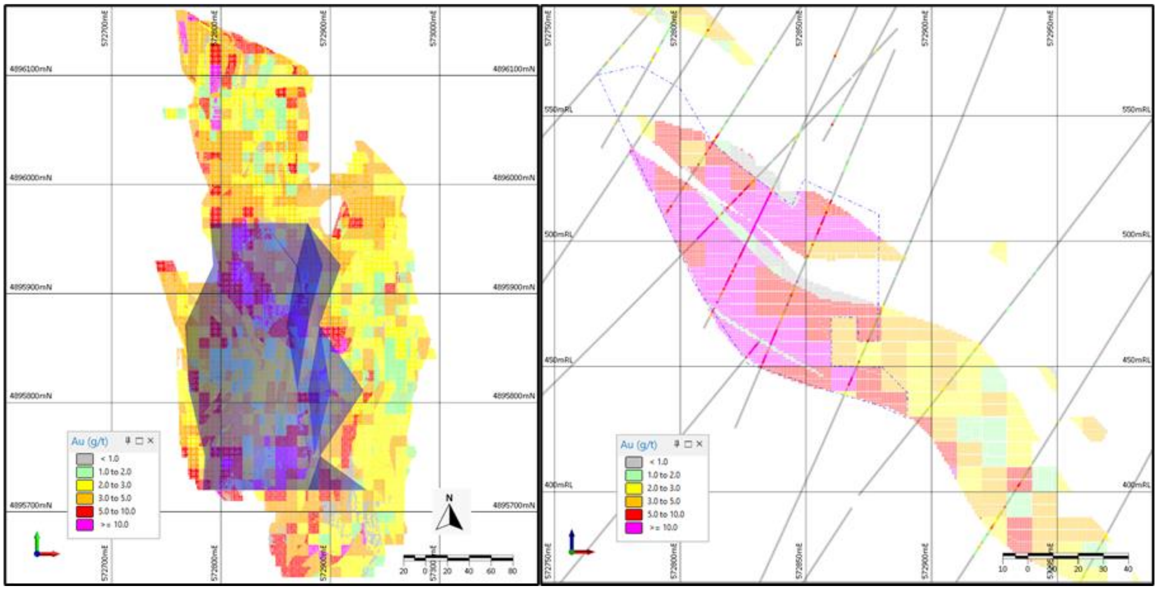
Čoka Rakita Mineral Resource Estimate Effective Date of 16 November 2023					
Mineral Resource category	**Tonnes** (Mt)	**Gold Grade** (g/t)	**Contained Gold** (koz)	**Silver Grade** (g/t)	**Contained Silver** (koz)
Inferred	9.79	5.67	1,783	1.21	382

Notes:

- The cut-off value of 2 g/t assumes US$1,700/oz gold price, 90% gold recovery, 10% dilution, US$79/t operating cost (mining, process and G&A costs), US$7/t sustaining capital cost, as well as offsite and royalty costs.
- Mineral Resources are reported within smoothed MSO underground mining shapes generated at a 2 g/t Au cut-off and a minimum width constraint of 5.0 m x 5.0 m x 2.5 m, to ensure Mineral Resources meet RPEEE. The smoothing process allows for blocks below the cut-off to be included within the final shapes in order to emulate the internal dilution that would be experienced during underground mining as per CIM Estimation of Mineral Resources and Mineral Reserves Best Practices Guidelines prepared by the CIM Mineral Resource and Mineral Reserve Committee and adopted by the CIM Council on 29 November 2019.
- The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that might materially affect the estimate of Mineral Resources, other than those specified in Section 14.19.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Figures have been rounded to reflect that this is an estimate and totals may not match the sum of all components.

Within the Inferred MRE, there is a continuous high-grade core as illustrated in Figure 14.41. A wireframe was digitised around this part of the MRE and it amounts to 2.8 Mt at a 10.1 g/t Au grade for 900,000 contained ounces gold, using the same reporting cut-off grade and RPEEE assumptions described above in Sections 14.17.1 and 14.18. Further infill drilling of this area is required with the objective of increasing the confidence of this part of the MRE since it may impact the potential economics of the Project by accessing higher than average grades during the early years of a potential mine plan.

Figure 14.41 – High Grade Core Wireframe and Mineral Resource Model



Showing plan view (left) and cross section 4895870 m (+/- 30 m) looking north (right)
Source: CSA Global, 2023

14.19 Risk Factors that May Affect the Mineral Resource

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that could materially impact the MRE disclosed in this Report, other than those specified below and in Table 14.24:

- Changes to price assumptions and input values for mining, processing, general and administrative (G&A) costs and metallurgical recovery and other mining assumptions used to constrain the MRE.
- Changes to the deposit scale interpretations of mineralisation geometry and continuity.
- The MRE is very sensitive to the choice of top cut grades; therefore, changes to those values could impact the grade and tonnage above the cut-off grade of the MRE.
- Change to estimation methodology (e.g. to model the high-grade tail) may change tonnage and grade estimates.

The risk attached to other factors identified are summarised in Table 14.24. The overall risk to the Čoka Rakita MRE is reflected in the current resource classification as Inferred Mineral Resources and is considered moderate, which is consistent with the early-stage nature of the Project.

Table 14.24 – Qualitative Risk Assessment

Factor	Risk	Comment
Sample collection, preparation and assaying	Low to Moderate	There are written procedures and data management practices in place. The nature of coarse gold means there is an inherent higher risk relating to and risk associated with sample preparation and analysis, but this is mitigated by the analysis for the vast majority of samples being screen fire assay which requires larger volumes. The majority of the gold is associated with finer fractions, but coarse gold is associated with higher grades.
QA/QC	Moderate	While screen metallics testing is the preferred method for analysing high gold grades in coarse gold environments, the nature of SFAs means that direct quality control is less possible than it is for other methodologies. Quality control review has been performed on FA and has indicated no material issues of concern. Insertion of blind standards, duplicates and blanks is recommended.
Geological model	Moderate	Uncertainty in accuracy of location of late-stage intrusives modelled. The fact that core can look very similar in terms of skarnification and intensity of alteration but have different grade character across short distances, is notable.
Mineralisation model	Moderate to High	The nature of coarse gold means there is an inherent uncertainty in its location and grade since it can be missed in half-core sampling, and variability at close ranges can be high. The mineralisation has been constrained within moderately to intensely skarnified S1/S2 material and guided by grade composites generated at 1 g/t Au. It is important to retain the geological basis of the interpretation and not be guided only by grade since level of selectivity can be low in this kind of environment. This risk can be mitigated by using larger diameter core barrels such as HQ or PQ to collect more sample for assay analyses and a better representative sample. This can also be mitigated through a bulk sample using closely spaced PQ cores.
Treatment of outliers (grade caps)	Moderate	The MRE is very sensitive to the choice of grade cap. Given the early stage of the Project and broad drill spacing, a relatively conservative grade cap was applied, which cuts 2% of the data and c. 30% of the metal. When data is top cut (at 70 g/t Au for the largest domain), variograms indicate nuggets that are moderate and not extreme, indicating grade continuity is not extremely low and grade variability is not extremely high.
Location of post-mineralisation intrusives	Moderate	The location of post-mineralisation intrusives represents a low volume but precise location is uncertain based on current broad drill spacing.
Grade estimate	Moderate	The grade estimate has been intentionally smoothed to reflect the uncertainty of the location of coarse gold. Sensitivity to grade estimation methodology is recommended to assess methodology for improved modelling of the high-grade tail.

Factor	Risk	Comment
Tonnage estimate	Low	The density estimate is considered low risk. The volume estimate is moderate risk, associated with uncertainty in the mineralisation model but not unreasonably so considering the stage of resource development and level of classification.
Permitting Risk	Low to Moderate	A potential risk to the Project is associated with permitting delays. Such delays caused by potential changes to Serbian regulations to align with EU Law, regulator delay, public challenge to the Spatial Plan or EIA and administrative appeals. Similar risks have been experienced by other private sector mining projects permitted in Serbia.
Overall rating	Moderate	The current MRE carries a moderate level of uncertainty and risk which is reflected in its classification as Inferred Mineral Resources.

15 MINERAL RESERVE ESTIMATE

This Section is not required in the Report. Mineral Reserves have not yet been estimated for this Project.

16 MINING METHODS

16.1 General Description of the Deposit and Mining Project

Figure 16.1 provides a 3D illustration of the underground mine planned for the Čoka Rakita Project. The gold mineralisation occurs in a skarn-type deposit approximately 250 m to 600 m below the surface. The mineralisation forms a lens-like shape with a vertical extension of up to 280 m and a strike length of about 500 m. It dips at 40° to 50° towards the east. The thickness of the mineralised zone varies, ranging from less than 20 m at its margins to more than 100 m at its core.

The host rock of the deposit is a calcareous clastic sedimentary rock, with mineralisation occurring in skarn-altered calcareous sandstone. The mineralisation is primarily controlled by stratigraphy and, to a lesser degree, by structural factors. The footwall boundary of the mineralisation closely parallels a sill-like diorite intrusion.

The quality of the sedimentary host rock in the hanging wall generally ranges from fair to good. However, the footwall of the deposit exhibits poor ground conditions. Consequently, DPM plans to position the mine development and underground infrastructure on the hanging wall side of the deposit. This strategy departs from standard mining practice, favouring developing deposits from the footwall side.

DPM plans to mine the deposit using the sublevel stoping. This method is well-suited for mining the deposit considering several key factors:

- It is compatible with the vertical aspect of the deposit configuration;
- The expected geotechnical conditions are favourable for the size of stope openings proposed in this Study;
- As a bulk mining method, it has a lower mining cost than that of a selective method like drift and fill; and
- The cut-off grade largely determines the mining boundaries as the host-rock dilution contains gold, making selectivity less critical.

The targeted production rate of the mine is approximately 2,300 tpd, which aligns with the process plant capacity of 850 kt annually. This output can be substantiated using an empirical approach to determine the optimal production rate. With an assumed resource of 8 million tonnes, the planned output aligns reasonably well with Taylor's Rule and very closely with Long's Relationship.

Taylor's Rule

- Production Rate = 0.0143 * Tst 0.75
- Referential Tonnage = 8,000,000 t
- Short Tons (st) = 8,818,490 st
- Production Rate = 2,314 st/day
- = 2,099 t/day

Long's Relationship for Underground Mines

- Production Rate = 0.297 * $T^{0.563}$
- Referential Tonnage = 8,000,000 t
- Production Rate = 2,287 t/day

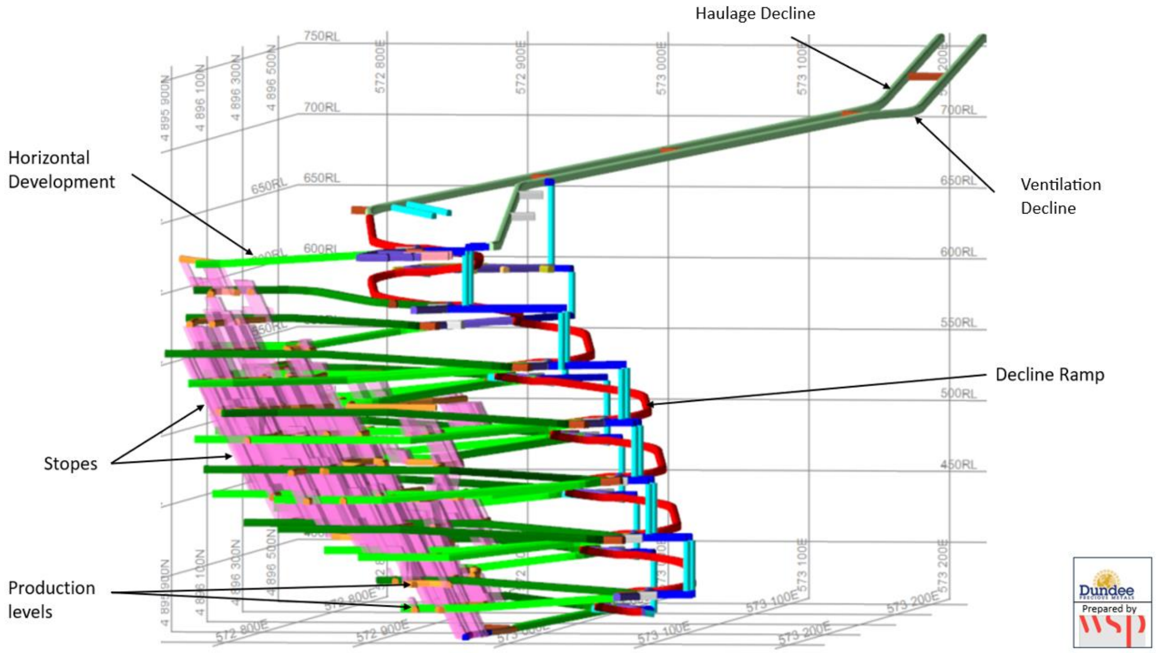
The deposit will be accessed by driving twin declines from the east side of the mineralisation, which will also serve as airways for intake and exhaust ventilation. A spiral ramp will be developed from the end of the decline designated for haulage, providing access to the lower levels of the mine. Additionally, headings will be driven off the spiral ramp to access the sublevels for mining the deposit. These sublevels will be spaced at 20 m intervals.

The QP is of the opinion that developing the mine from the hanging wall side of the deposit is the correct approach, given the current understanding of the ground conditions. However, it is recommended that further geotechnical investigations be conducted to confirm that the footwall is unsuitable for locating development and infrastructure.

WSP recommends conducting a trade-off study to evaluate alternative methods for handling mineralised material and waste compared to truck haulage. One possibility is installing a conveyor system in the ventilation decline.

This Section uses the terms "mineralisation," "mineralised material," and "deposit" in lieu of the term "ore" per Code 7.19 of CRIRSCO International Reporting Template for the Public Reporting of Exploration Targets, Exploration Results, Mineral Resources and Mineral Reserves (2019), which stipulates that the word "ore" must not be used in stating Mineral Resource Estimates.

Figure 16.1 – 3D Representation of the Čoka Rakita Mine



Source: WSP, 2024.

16.2 Geotechnical

WSP completed a Geotechnical PEA level study review for the Project to assist in defining the following key items:

- **Engineering Geology Model (EGM)** – Define the main rock mass quality domains near the deposit (footwall, mineralisation zones, hanging wall) and areas of main infrastructure (twin decline, spiral ramp and hanging wall access drifts);
- **Stope Stability and Dilution Review** – Review mining methods based on the EGM. Stope stability sizes were evaluated using empirical methods Mawdesley et. al (2003) and dilution using Clark and Pakalnis (1997);
- **Drift and Drift Intersection Ground Support Review** – Using Potvin (2017) empirical charts and RocScience Unwedge to define standard and enhanced ground support;
- **Stand-Off Distance for Infrastructure Review** – Compared deposit sizes (width) against distance from the main infrastructure planned in the hanging wall; and
- **Crown Pillar and Temporary Sill Pillar Review** – Scaled Span empirical assessment of crown pillar and simple RocScience RS2 2D numerical model assessment of the temporary sill pillar.

The data available for the PEA included limited geotechnical core logging data for the Project area and information from the nearby Timok Project site.

16.2.1 EGM – Rock Mass Quality

WSP's review of the available data for the general rock mass quality for the Project was estimated between Good (Barton Q'>20) to Fair to Poor (Barton Q'<4). Most of the available geotechnical data was from the hanging wall and the mineralised zone, with limited data reviewed in the footwall. For the PEA study in the hanging wall, mineralised zone and footwall area proximal to the mineralised zone (10 m from the mineralised zone), the rock mass quality has been estimated as Good to Fair with limited sections of Poor rock mass quality.

Observations in a selection of core photos and reviews of RQD core logging data indicate localised Poor rock mass quality zones typically <1 m to several metres wide. These sections were observed either as broken core, lost core, high vein intensity, high alternation and/or faults. DPM has also defined distinct fault zones geometries that intersect the main mineralised zones. The impact of these Poor quality zones in the mine design has been generally addressed as follows:

- Estimated that up to 10% of the infrastructure and stopes could be in Poor quality rock mass zones;
- Enhanced ground support required in permanent and development excavations in Poor quality rock mass zones; and
- Modification to the stope size in Poor quality rock mass zones.

DPM is in the process of completing a PFS-level drilling geotechnical program that WSP assisted in designing. This program is designed to collect sufficient data for the next level of studies to better define the rock mass qualities in the deposit area (hanging wall and mineralised zone), crown pillar and key infrastructure areas (twin decline and spiral ramp). Orientation structural data, field and laboratory rock strength data and hydrogeological data is included in this program.

16.2.2 Stope Stability Review and Dilution

A stope stability and dilution review was completed on the proposed mine design using empirical methods Mawdesley et. al (2003) and Clark and Pakalnis (1997). For the PEA, the mine design is longitudinal open stoping with backfill, with the crosscuts being mined transversely. The following stope design parameters were considered for the stope stability assessment:

- Stope Level Height = 20 m;
- Stope Span Width (hanging wall to footwall) = 15 m;
- Stope Strike Length = 10 m to 30 m;
- Stope Hanging Wall and Footwall Angle = 45°, 60° and 90°;
- Q' ranges: Low = 1 to 4 (Poor), Average =12.5 (Good), High = 20 (Good); and
- Stability Values: A=0.4 (induced stress factor), B=0.25 to 0.4 (orientation of critical structure relative to stope walls) and C=2 to 8 (stope failure mode). The B and C Stability Values were

modified for varying hanging wall and footwall angles where vertical stopes have higher Stability Index Values (N) compared to shallower angled stopes (45° and 60°).

The stope configuration was considered acceptable for design (stable) if greater than 70% probability of failure (POF) was obtained on the plot of N versus stope size (HR=Hydraulic Radius). Based on this assessment the following stope stability parameters are recommended:

- Rock mass quality Q’>20 (Good) = 20 m high x 20 m span and 15 m strike length for a 45° to 90° dipping hanging wall and footwall;
- Rock mass quality Q’=4 to 20 (Fair) = 20 m high x 20 m span and 15 m strike length for a 65° to 90° dipping hanging wall and footwall. Some of these stope areas may require support for the stope hanging wall and stope back; and
- Rock mass quality Q’<4 = 20 m high x 15 m span and 10 m strike length.

At this stage of the Project, the exact location and size of Poor quality rock mass zones are not fully defined. The approach at this level of the study is to ensure that there is some mine design contingency to address Poor quality zones encountered. Methods to address stope stability risks include shortening the stope strike length and/or increasing ground support (e.g., cable bolting) to the stope backs and hanging wall (if required). Additional impacts to the mine design due to Poor quality rock mass zones are the potential for increased stope dilution and increased stope cycle time due to reducing the stope strike length from 20 m to 30 m to less than 20 m.

An estimate of stope dilution was completed using the above mine design parameters against the Clark and Pakalnis stability charts which indicate the following:

- Rock mass quality Q’=20 (Good) or greater result in < 0.5 m of dilution in the stope back, hanging wall and footwall for stope dips of 65° and 90°. This is estimated at 5% total dilution.
- Rock mass quality Q’<20 to 12.5 (Fair) result in = 0.5 m to 1 m of dilution in the stope back, hanging wall and footwall for stope dips of 65° and 90°. This is estimated at 5% to 10% total dilution.
- Rock mass quality Q’< 4 (Poor) has > 2 m of dilution in the stope back and hanging wall and footwall for stope dips of 65° and 90°. This is estimated at 20% total dilution.

At this stage of the Project, and based on the limited geotechnical information, an estimate of 10% of the stopes could be in Poor quality rock mass zones which could result in dilution up to 20% if stope strike length is not reduced and/or additional ground support is applied to the stope backs. At this stage of the Project, it is also reasonable to account for some of Poor quality stope hanging walls and stope backs that can be better defined during the next stages geotechnical drilling program.

16.2.3 Ground Support Review

A ground support assessment was completed based on a range of underground excavation types compared against Potvin (2017) empirical charts, RocScience Unwedge assessment and DPM's current ground support systems employed at the Chelopech Mine, with similar ground qualities to Čoka Rakita Project. The ground support recommendations are based on two (2) ground support rock mass classifications (Good and Poor rock mass quality zones) and standard primary support for excavation sizes (4.5 m to 5.5 m span), intersections (secondary support) for the standard excavations and large excavations. The following are the recommended ground support types:

- Primary Support (4.5 m to 5.5 m span) in Good rock mass: Roof and Wall Support with 2.4 m split sets on 1.5 m x 1.5 m spacing with mesh. Walls split sets installed 1.5 m to 1. 8 m from the floor. In permanent infrastructure resin grouted rebar is to be used instead of split sets in the back of the excavations.
- Primary Support (4.5 m to 5.5 m span) in Poor rock mass: Roof and wall support with 2.4 m resin bolts on 1.5 m x 1.5 m spacing, mesh and 50 mm shotcrete. Walls split sets installed along entire wall.
- Secondary Support (intersections) in Good rock mass: Roof support with 5 m long cable bolts (15.2 mm diameter) on 2.0 m x 2.0 m spacing.
- Secondary Support (interactions) in Poor rock mass: Roof support with 5.5 m long cable bolts (15.2 mm diameter) on 1.7 m x 1.7 m spacing.
- Large Excavation (7.0 m to 9.0 m span) in Good rock mass: Roof and wall support with 3.0 m resin bolts on 2.0 m x 2.0 m spacing, with mesh and 50 mm shotcrete. Wall bolts installed along entire wall. 5 m long cable bolts (15.2 mm diameter) installed on 2.0 m x 2.0 m spacing in roof.

DPM has experience with installing the above ground support systems at their Chelopech Mine except for resin grouted rebar support.

16.2.4 Stand-off Distance from Infrastructure

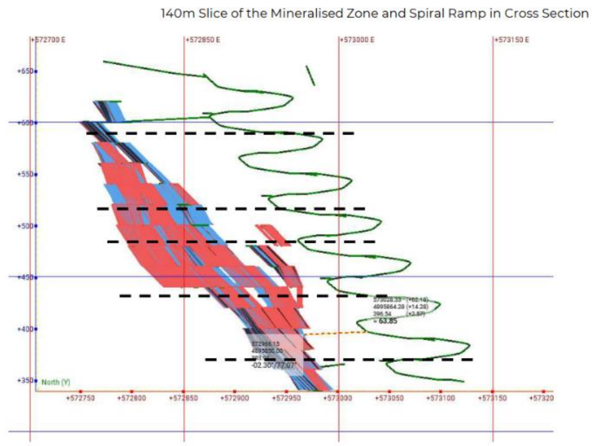
The PEA mine design has all permanent access excavations located in the hanging wall of the mineralised lens (deposit). A spiral ramp is used to access the stoping levels. A simple stand-off assessment was completed comparing the width of the deposit at five (5) elevations of the underground Project to the distance from the spiral ramp. A simple plot and table compare these values in Figure 16.2.

Figure 16.2 – Stand-off Distance Assessment

Mineralised Body/Spiral Ramp Distance

The mineralised body (MB) thickness and the distance from the MB to the spiral ramp were compared to recognize areas on the ramp where increased stresses are expected as mining progresses. A 1:2 ratio between the MB thickness and ramp distance is assumed to avoid concentrated stress.

Location	Mineralised Body Thickness (m)	Closest Distance to Spiral Ramp (m)	Distance Required for 1:2 Ratio (m)
1	20	100	40
2	80	74	160
3	105	30 to HW lens (60 to main mineralised zone)	210
4	80	45	160
5	30	70	60



Source: WSP, 2024

The tabular information in Figure 16.2 indicates that in the widest part of the deposit (80 m to 105 m wide footwall to hanging wall), the distance of the deposit to the spiral ramp is less than 80 m. Therefore, there is the potential that increased mining induced stress, from the mining of widest areas of the deposit, could impact the stability of the spiral ramp in those closer locations. This could also be impacted if in some of these locations “Poor quality” rock mass occurs. As the Project advances, numerical modeling assessments should be completed to identify potential impacts that the mining of the deposit could have on the spiral ramp. Additionally, moving the spiral ramp further from the deposit, and/or increasing the ground support in the wider deposit areas, might be methods to reduce potential instabilities in the permanent spiral ramp infrastructure.

16.2.5 CROWN AND TEMPORARY SILL PILLAR REVIEW

A simple empirical crown pillar stability assessment (Scale Span) and temporary sill pillar excavation assessment using RocScience RS2 was completed to identify potential instability issues with the PEA mine design.

The following were the crown pillar parameters used in the assessment:

- Crown Pillar Bedrock Thickness = 162 m;
- Crown Pillar Span = 17 m;
- Crown Strike Length = 50 m;
- Stope Dip = 48°; and
- Crown Pillar Rock Mass Quality, Q’=12.5 m (Good).

The Scaled Span empirical method indicated a Probability of Failure of < 0.5% which is classified as long-term stable based on the crown pillar parameters and rock mass quality.

Based on the PEA mine plan, a temporary sill pillar is planned to separate the main upper deposit from the lower deposit. The temporary sill pillar is planned to be mined later in the schedule. A temporary sill pillar assessment was completed using the numerical modeling software Rocscience RS2 using the following parameters:

- Sill Pillar Height = 40 m;
- Sill Pillar Span = 25 m;
- UCS strength of the host rock = 100 MPa;
- GSI = 70, m_i=11, D=0; and
- In-situ Stress Case of K=1, 1.2 and 1.3 and K=1, 1.4 and 1.8.

Differential (principal major stress – principal minor stress) stress levels reviewed in the RS2 model for the temporary sill pillar vary between 22 MPa and 63 MPa which indicates the potential for rock bursting and spalling hazards when the deposit is mined above and below the temporary sill pillar. The impact of these mining stresses could be addressed by increasing the ground support in the sill pillar, reducing the mining recovery in the sill pilar (e.g., mine 50% of less of the sill pillar) and/or modify the mining sequence to reduce stress concentrations in the sill pillar. More detailed numerical modeling (3D) will need to be completed after the completion of geotechnical drilling program and the mine design is further progressed.

16.3 Hydrogeology

The overview of current groundwater conditions in support of the PEA level study is based on data collected as part of the hydrogeological studies completed for the Timok property located 3 km northwest from the Project because no site-specific hydrogeological information is presently available for the Project area.

A review of existing hydrogeological studies prepared for the Timok property was undertaken with a focus on information related to groundwater conditions of the area, stratigraphy, and structural geology to support the development of a hydrogeological investigation program to collect information on the hydraulic properties of the hydro stratigraphic units and geological structures within the Project area. The information summarised in this section was obtained from the following documents:

- Groundwater Flow Model and Water Balance Study – Timok Gold Feasibility Study, Wood, February 2023;
- Report on the Infrastructure Hydrogeological Investigations at Timok Gold Project, University of Belgrade Faculty of Mining and Geology, September 2021; and

- Hydrogeological Conceptual Site Model Report – Timok Gold Project (Version 1.0), University of Belgrade Faculty of Mining and Geology, December 2021.

16.3.1 HYDRO STRATIGRAPHY

Based on the results of the hydrogeological investigations, a total of nine hydro stratigraphic units have been identified within the Project area. These were further categorised as low, moderate-low, moderate, and moderate-high permeability units.

Low Permeability Units (aquitards)

- Marl with calcite veins;
- Black laminated siltstone or mudstone; and
- Phyllites and metasedimentary sandstone and shale.

Moderate-low and Moderate Permeability Units (unconfined fractured rock aquifer)

- Volcanic detritus-rich bedded coarse to medium grained sandstone (S2 sandstone);
- Granodioritic intrusions (upper parts weathered);
- Andesitic volcanic and volcanoclastic rocks; and
- Calcareous sandstone to sandy limestone and basal breccia (S1 sandstone).

Moderate-high Permeability Units (unconfined and confined fractured rock and karst aquifer)

- Micritic bedded black to grey limestone, with minor dolomite content (Cretaceous); and
- Micritic limestone, bioclastic and oolitic limestone, dolomite, and limestone-dolomite (Jurassic).

The low-permeability hydro stratigraphic units were characterised as aquitards, and the moderately permeable units as unconfined and confined fractured rock and karst aquifers. Karst morphology, including caves, sinkholes, dry valleys, and dolines, has been identified within the Timok Project area. Two (2) major karst springs are located immediately south and to the north of the Project area, with a minimal flow rate between 10 L/s and 15 L/s.

Unconfined Fractured Rock Aquifer

The hydraulic conductivity of the unconfined fractured rock aquifers developed within the volcanic-volcanoclastic and sandstone sediment sequence was estimated as moderate to low.

The west side of the Timok project area is underlain by the S1 and S2 limestone units developed over the carbonate sequence, while the eastern side of the Project area is characterised by thicker volcaniclastic and sandstone units. In some areas, the upper parts of the epiclastic and diorite intrusions are highly weathered.

Unconfined and Confined Fractured Rock and Karst Aquifer

The hydraulic conductivity of the unconfined and confined fractured rock and karst aquifers developed in Jurassic and Cretaceous limestones was estimated as moderate to high.

The thicker, massive Jurassic limestone unit is more prone to karstification in comparison to the bedded Cretaceous limestone. The Cretaceous limestones are mostly eroded to the west side of the Project area, but to the east, the unit is dipping down under the clastic sediments. The Cretaceous unit is bounded by Paleozoic phyllites to the west and a monzonite intrusion to the south. To the east, the unit is dipping below the volcanic unit and remains open to the north from the Project area.

Low Permeability Bedrock Aquitard

The hydraulic conductivity of the phyllites, shists, interbedded siltstones and sandstones (S1), and Upper Cretaceous marl was estimated to be low. Most of these units are not continuous and were characterised as local aquitards only. The Paleozoic phyllites represent a regional unit that forms a western boundary of the karst aquifer.

Structural Geology

Three (3) structural trends (NW, NE, E-W) are present in the Timok project area. The most recent NW faults and normal E-W faults are of the highest importance to hydrogeology. The normal E-W trending faults have the potential for compartmentalisation of the groundwater flow. The thickness of these fault zones and their characteristics have not been fully investigated.

16.3.2 Hydraulic Parameters

The hydraulic conductivity of the individual hydro stratigraphic units was derived from single-well response tests with packers and pumping tests. The test intervals extended up to 300 m below the ground surface and the majority of the tests were performed within the carbonate units, and the calcareous clastic sedimentary rocks (70%).

The highest hydraulic conductivity of 5E-05 m/s was associated with the highly fractured (karstic) Jurassic limestones, and the lowest hydraulic conductivity of 4E-09 m/s was calculated for the interbedded siltstones and basal breccia. The geometric mean hydraulic conductivity calculated for all in-situ permeability tests at the Timok project was approximately 2E-07 m/s, similar to the geometric mean of the individual hydro stratigraphic units that varied between 2E-07 m/s and 3E-07 m/s. A slight decrease in hydraulic conductivity with depth was observed in most of the hydro stratigraphic units.

The hydraulic conductivity of the geological structures tested varied over two (2) orders of magnitude from 1E-08 m/s to 2E-06 m/s with a geometric mean of approximately 3E-07 m/s, similar to the other hydro stratigraphic units.

16.3.3 Groundwater Levels and Flow Directions

Information collected from the groundwater and piezometer monitoring network established in the Timok project area indicate that the water table is generally a subdued replica of topography, with groundwater encountered at a shallower depth at the valley bottoms and at a greater depth in the upland areas. In the two (2) deposit areas closest to the Čoka Rakita Project (Bigar Hill and Dumitru West), the measured water levels vary from less than 1 m below ground surface in the valleys up to 98 m below ground surface at higher elevations. Groundwater levels vary seasonally mainly due to the high snow and rain precipitation in winter months (January and February) and snowmelt in spring (April). Up to 15 m annual fluctuation in water level was recorded in the area of the Bigar Hill deposit.

Natural groundwater gradients reflect recharge over the mountain tops and groundwater discharge along the valley floor. The groundwater flow direction is generally to the north from the highest groundwater elevations recorded at the south end of the Timok project area, where the ground elevation reaches up to 860 masl, towards the lowest groundwater levels measured at the north end of the study area with ground elevations around 552 masl. Measurements obtained from the limestone units showed deep water levels and strong downward vertical gradients exceeding a gradient of 1.0 at several locations, indicating perched aquifers in the overlying units.

16.3.4 Limitations

Due to the lack of site-specific hydrogeological information for the Project, it was assumed, for the purposes of this Report, that the groundwater conditions are comparable to those observed at the Timok project, located 3 km northwest of the Čoka Rakita Project.

Information obtained from the planned hydrogeological investigations will be used to confirm the hydraulic properties of the hydro stratigraphic units present within the deposit area and update the conceptual groundwater model assumed in this Report. The focus of the investigations will be on delineation of the potentially karstic carbonate sequence within the extent of the proposed underground development and identification and characterisation of geological structures within the Project area, including openness, density, connectedness, and distribution.

16.4 Mining Method

16.4.1 Sublevel Stoping

DPM plans to mine the deposit with sublevel stoping, as illustrated in Figure 16.3. This method divides the deposit into sublevels, and the mineralised material between them is mined as stopes. Each stope is drilled and blasted using longholes, and the blasted material is extracted from the lower sublevel production drive.

Figure 16.4 illustrates the strategy for developing the sublevels, using a plan view of the 500 sublevel as an example. Most of the mineralised material on each sublevel will be mined with parallel longitudinal stopes extending along the strike of the deposit. Each series of longitudinal stopes requires a production drive on the upper sublevel for drilling and loading explosives and a production drive on the lower sublevel for mucking blasted mineralised material. These production drives will be driven from crosscuts extending across the deposit from the hanging wall to the footwall.

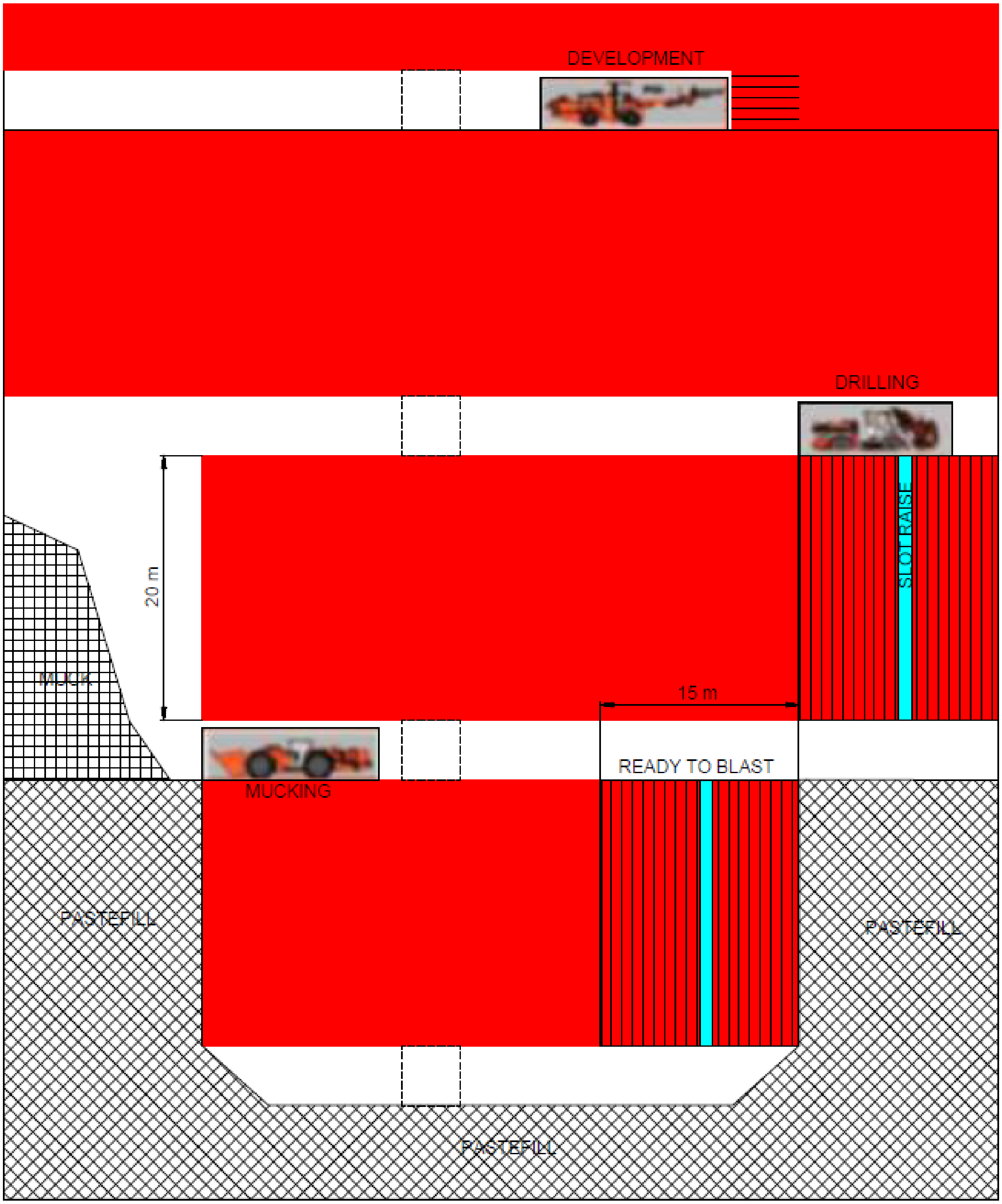
The longitudinal stopes will be mined in a retreating fashion, advancing in series, one after the other, towards the crosscuts. Once longitudinal stoping is completed on a sublevel, the remaining mineralised material adjacent to a crosscut will be recovered as a series of transverse stopes retreating towards the hanging wall. The dimensions of the stopes will be 20 m wide (measured at right angles to the production drive) and 20 m high, corresponding to the sublevel interval. Each stope will be 15 m long, in compliance with geotechnical guidelines.

The QP is of the opinion that sublevel stoping is an appropriate method for mining the Čoka Rakita deposit.

WSP recommends conducting a trade-off study to compare the planned longitudinal method with transverse sublevel stoping, a more typical approach for mining thicker deposits like Čoka Rakita. Mining the deposit with transverse stopes would require developing waste drives adjacent to the hanging wall of each sublevel and, from these headings, extending regularly spaced crosscuts across the width of the deposit to the footwall.

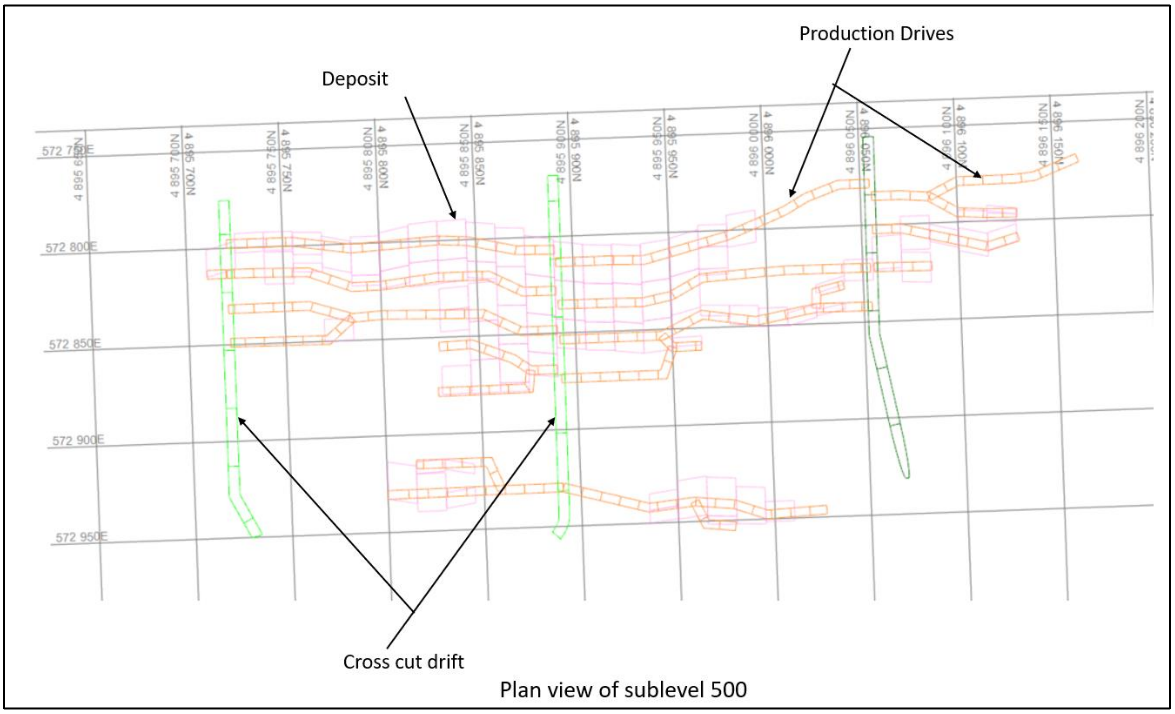
For future resource evaluations, WSP recommends investigating whether drift and fill would be advantageous for mining specific portions of the deposit. This method would provide greater flexibility and selectivity than sublevel stoping for mining the deposit's more complex or irregular zones. Furthermore, it is better suited to managing challenging ground conditions in the mineralisation or adjacent to the hanging wall, as it limits the size of the openings to the width of a drift.

Figure 16.3 – Longitudinal Long-Hole Mining Method – Long Section



Source: WSP, 2024

Figure 16.4 – Example of the Development Required for Mining a Sublevel



Source: WSP, 2024

16.4.2 SEQUENCING

The sublevels will be mined in an ascending sequence, starting from the bottom and working upwards. However, stoping operations will generally occur on more than one sublevel at a time to ensure enough areas are active to meet production targets.

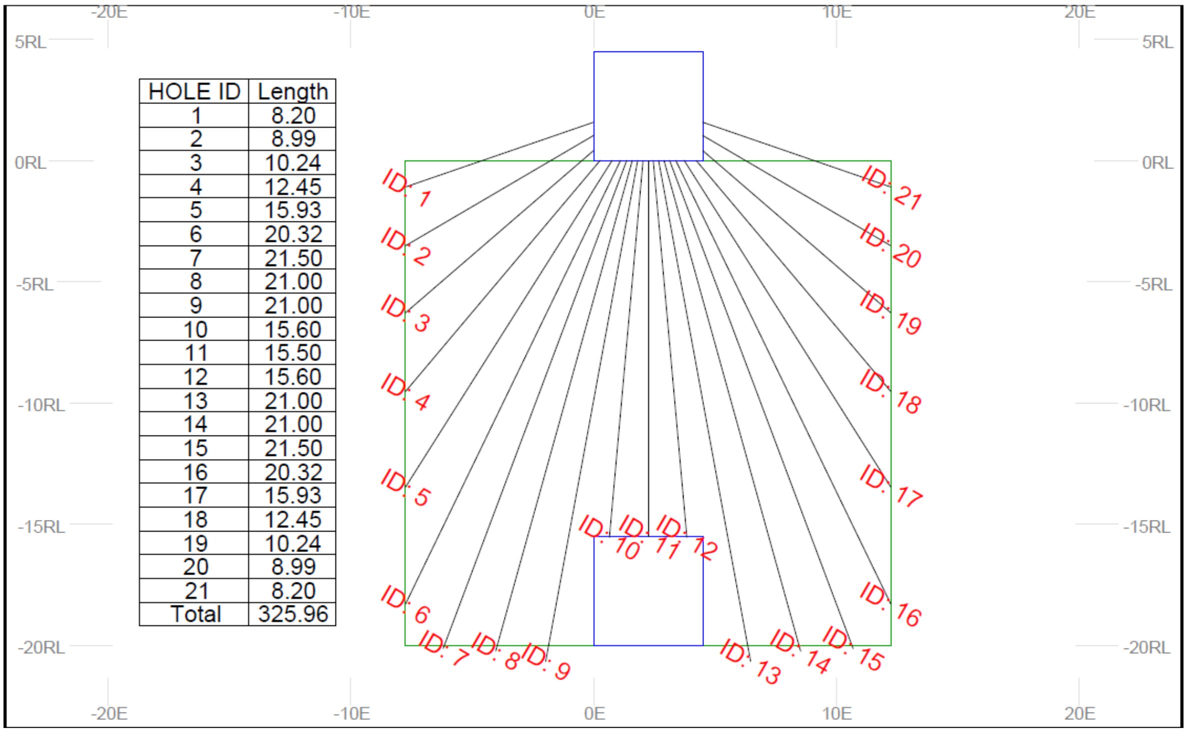
The mineralisation on a sublevel will be mined in a sequence of alternating primary and secondary stopes. The primary stopes will be mined first, leaving pillars of unmined mineralised material between them. These pillars will subsequently be mined as secondary stopes once the adjacent primary stopes have been backfilled and the backfill has had sufficient time to cure and achieve the required strength.

16.4.3 DRILLING

The mine will have two (2) production drill rigs, one tophammer and the other in-the-hole (ITH). In addition to drilling downholes, the tophammer rig will be suitable for drilling in stopes requiring upholes. The ITH rig will be advantageous for applications where minimising hole deviation is a priority and for boring slot raises.

Figure 16.5 illustrates a typical longhole drilling layout in cross-section. The longholes will be drilled in rings angled downwards as inverted fans from a single pivot point at the centre line of the upper production drive. The holes will be 89 mm in diameter, and the toe spacing between them in a ring will be 2.1 m. The burden between rings will be 2.1 m. Additional holes will be required at the stope's far end to create the slot that provides the free face for blasting the rings. A 760 mm diameter borehole, reamed with the boring head of the ITH drill, will create an opening for blasting the slot.

Figure 16.5 – Drilling Layout of a Typical Long-Hole Ring



HOLE ID	Length
1	8.20
2	8.99
3	10.24
4	12.45
5	15.93
6	20.32
7	21.50
8	21.00
9	21.00
10	15.60
11	15.50
12	15.60
13	21.00
14	21.00
15	21.50
16	20.32
17	15.93
18	12.45
19	10.24
20	8.99
21	8.20
Total	325.96

Source: WSP, 2024.

16.4.4 BLASTING

The longholes for stope blasting will be loaded with bulk emulsion explosive. The mine equipment fleet will include two (2) mobile emulsion chargers. Each hole will be primed with a booster and initiated with a detonator, which can be electronic or a Nonel. Given the radial drilling layout of the rings, only about 50% of the metres drilled need to be loaded to achieve a reasonably uniform explosive distribution and the required powder factor.

16.4.5 MUCKING

After each longhole blast, an LHD with a 15 t tramming capacity will muck the broken mineralised material from the lower production drive of the stope. A portion of the material may be extracted with the operator seated on the LHD. However, most of it must be mucked by radio remote control, with the operator controlling the LHD from a safe position in the production drive.

After filling its bucket, the LHD will tram the material out of the stoping area via the production drive and crosscut, transporting it to the level access drifts in the hanging wall. Next, it will either load the material onto a waiting mine truck or dump it into a remuck bay, where it will be temporarily stockpiled. In the latter case, the stockpiled material must be rehandled by an LHD and loaded onto trucks later. After discharging the load, the LHD will return to the stope to extract another bucket.

16.4.6 Haulage

The method for transporting mineralised material at Čoka Rakita will be ramp haulage. Mine trucks with a 45 t payload capacity will haul the material to surface via the spiral ramp and haulage decline. Upon exiting the portal, the trucks will proceed to the run-of-mine (ROM) pad, dumping the material onto the ROM pad stockpile. Subsequently, a wheel loader will rehandle the material, transferring it from the stockpile to the chute of the ROM bin, which feeds the primary crusher.

16.4.7 Backfill

Before mining can proceed in a secondary stope, the voids of the adjacent mined-out primary stopes must be backfilled with paste fill. This paste fill requires adequate time to cure and attain the required strength before an adjacent secondary stope can be mined. Unless it is situated over a sill (i.e., unmined mineralisation), a mined-out secondary stope can be backfilled with either paste fill or rock fill (i.e., waste rock from mine development). However, on a sublevel above a sill, both primary and secondary stopes must be backfilled with paste fill to permit future extraction of the mineralised material beneath them. Before production activities can commence in the upper sublevel of a stope backfilled with paste fill, a layer of rock fill must be spread over the paste fill to create a suitable surface for operating rubber-tired equipment.

16.5 Mine Design

16.5.1 Stope Design

Table 16.1 presents the design parameters for the stopes. The estimated unplanned dilution and mining recovery factors are 10% and 95%, respectively. These estimates were based on benchmarking data from similar projects. For future studies, WSP recommends refining these estimates based on the mining method, geotechnical conditions, and backfill properties.

The stopes designed to be 20 m wide (measured perpendicularly to the production drive) and 20 m high, with a length of 15 m. They will be mined in a retreating sequence, initially advancing longitudinally towards the crosscut and subsequently advancing transversely towards the hangwall contact. The stope dimensions were established through consultation with the WSP geotechnical specialist.

The longholes will be 89 mm in diameter and drilled in rings with a 2.1 m toe spacing and a 2.1 m burden. The slot raise required to provide a free face for longhole blasting will be bored using an ITH drill rig equipped with a boring head. Given to the anticipated presence of groundwater in the mine, bulk emulsion will be used for loading the longholes instead of ANFO. The drilling and blasting design was developed in collaboration with the WSP blasting expert. The paste-fill requirements were determined through consultation with the WSP backfill specialist.

Table 16.1 – Mine Design Parameters – Stopes

Stope Design	Parameter
Mining Method	Sublevel Stoping
Unplanned Dilution	10%
Mining Recovery	95%
Stope Dimensions	
Width	20 m
Height	20 m
Length	15 m
Longhole Drilling	
Drilling Layout	Downhole Fans
Hole Diameter	89 mm
Holes per Ring	21 ea. (typical)
Metres Drilled per Ring	326 m (typical)
Burden between Rings	2.1 m
Toe Spacing between Holes in Ring	2.1 m
Drilling Factor (includes slot)	6.10 t/m-drilled
Slot Raise	760 mm dia. Borehole
Blasting	
Explosive	Bulk Emulsion
Powder Factor	1.79 kg/m^3
Backfill	
Primary Stopes	Paste Fill
Secondary Stopes	Paste Fill or Rock Fill

Source: DPM, 2024

16.5.2 Development Design

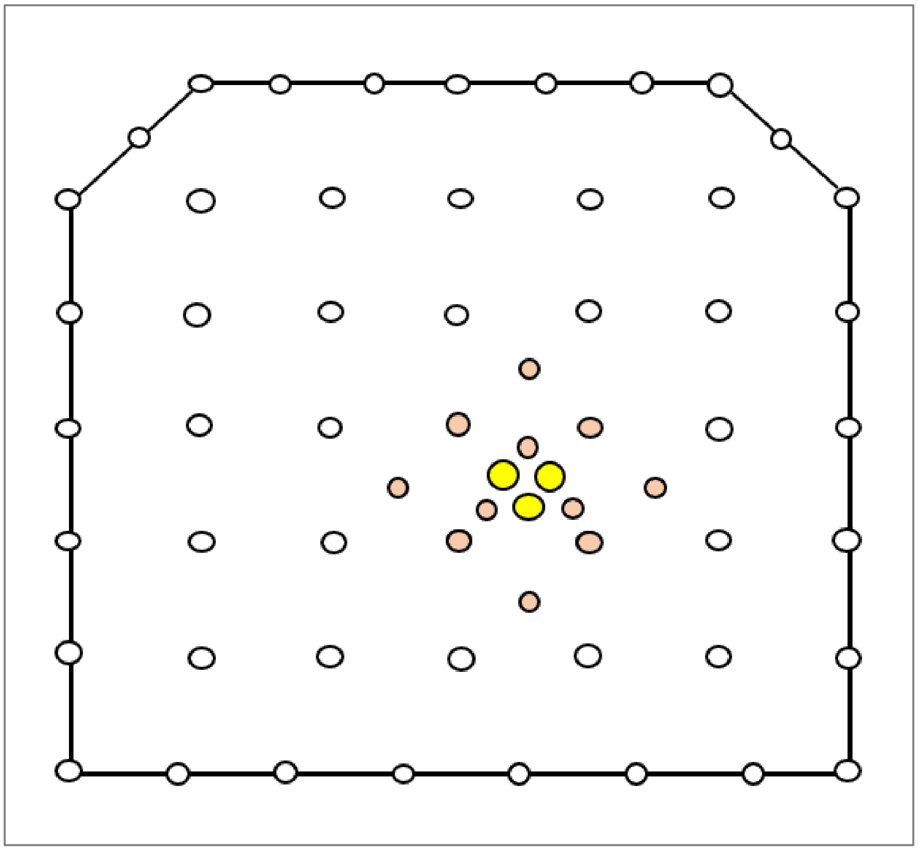
Table 16.2 presents the design parameters for development and infrastructure, including a 10% contingency factor to account for the unlocated poor ground. The declines will have a 5.5 m x 6.0 m (width x height) cross-section, minimising friction loss in the ventilation flow. The spiral ramp, level access headings, and crosscuts will be driven at 5.5 m x 5.5 m, providing adequate clearance for operating the 45-t mine trucks. The production drives will have a 4.5 m x 5.0 m profile, sufficient for the longhole drill rigs and LHDs, even with vent ducting and other services installed overhead. The ground support requirements were established in collaboration with WSP's geotechnical specialist.

16.5.3 Development Rounds

Figure 16.6 illustrates a drilling pattern for a development round. The example in this case is for a 5.5 m x 5.5 m heading, which applies to the spiral ramp, level access development, and crosscuts. The principle is the same for headings with different cross-sections, although the number of holes in the round will vary.

Figure 16.6 presents the parameters for the drilling and blasting design of a 5.5 m x 5.5 m drift round. The design specifies drilling 45 mm diameter holes using 4305 mm drifter rods. Three (3) central holes in the cut are reamed to 89 mm. The holes forming the arched back are loaded with cartridge-type pre-splitting emulsion explosive, designed for perimeter control blasting to minimise rock fracturing. The remaining 45 mm holes are loaded with bulk emulsion explosive, and the three (3) reamed holes are left uncharged. Each loaded hole is primed with a booster and charged at the toe with a stick of detonator-sensitive emulsion explosive. The loaded holes are initiated with Nonel-type LP detonators. The estimated advance per round is 3.8 m.

Figure 16.6 – Drilling Design for a 5.5 m x 5.5 m Drift Round



Source: WSP, 2024

Table 16.2 – Mine Design Parameters – Development and Infrastructure

Description	X-Section Area, m^2	Width, m	Height m	Split Set 2.4 m	Stubby 0.9 m	Resin Bolts 2.4 m	Resin Bolts 3.0 m	Cable Bolt 5.0 m	Meshes	Shotcrete 50 mm	Notes
				pcs/m	pcs/m	pcs/m	pcs/m	pcs/m	sheets/m	m^3/m	
Surface Decline (Main Haulage Ramp)	30.9	5.50	6.00	-	-	7.37	-	-	3.52	1.87	
Surface Decline (Main Ventilation Access)	30.9	5.50	6.00	-	-	7.37	-	-	3.52	1.87	
Spiral Ramp	28.1	5.50	5.50	-	-	6.93	-	-	3.52	1.76	
Ramp Level Access	28.1	5.50	5.50	-	-	6.93	-	-	3.52	1.76	
X-Cut Decline	28.1	5.50	5.50	6.93	3.52	-	-	-	3.52	0.53	30% to be shotcreted
X-Cut Incline	28.1	5.50	5.50	6.93	3.52	-	-	-	3.52	0.53	30% to be shotcreted
Stockpile/Remuck Bay	28.1	5.50	5.50	6.93	3.52	-	-	-	3.52	1.76	
Electrical Substation Cuddy	24.5	6.00	4.50	-	-	6.93	-	-	3.30	1.76	
Paste Fill Cuddy	30.9	5.50	6.00	-	-	7.37	-	-	3.52	1.87	
Horizontal Ventilation Development	23.3	5.00	5.00	-	-	6.60	-	-	3.30	1.76	
Vent Raise	7.6	3.10		-	-	-	-	-	-	1.10	
Water Bay/Sump	23.3	5.00	5.00	-	-	6.60	-	-	3.30	1.76	
Pump Station	23.3	5.00	5.00	-	-	6.60	-	-	3.30	1.76	
Sublevel Production Drift	21.1	4.50	5.00	6.60	3.30	-	-	1.62	3.30	0.53	1. 30% to be shotcreted 2. 30% Cable Bolts 5m 2x2
Explosive Storage	38.6	7.00	6.00	-	-	-	5.94	5.94	5.06	1.87	
Detonator Storage	23.3	5.00	5.00	-	-	-	5.17	5.17	-	1.76	

Description	X-Section Area, m^2	Width, m	Height m	Split Set 2.4 m	Stubby 0.9 m	Resin Bolts 2.4 m	Resin Bolts 3.0 m	Cable Bolt 5.0 m	Meshes	Shotcrete 50 mm	Notes
				pcs/m	pcs/m	pcs/m	pcs/m	pcs/m	sheets/ m	m^3/m	
Maintenance Bay	48.3	9.00	6.00	-	-	-	7.26	7.26	-	2.31	
Welding Bay	23.3	5.00	5.00	-	-	6.60	-	-	-	1.76	
UG Office	23.3	5.00	5.00	-	-	6.60	-	-	-	1.76	
Warehouse	23.3	5.00	5.00	-	-	6.60	-	-	-	1.76	
Fuel and Lube Bay	23.3	5.00	5.00	-	-	6.60	-	-	-	1.76	
Wash Bay	23.3	5.00	5.00	-	-	6.60	-	-	-	1.76	
3-Way Intersection								9.90			
4-Way Intersection								15.40			

Source: DPM, 2024

Table 16.3 – Drilling and Blasting Design for 5.5 m x 5.5 m Drift Round

5.5 m x 5.5 m Drift Round	Value
Drilling Supplies	
Button bit	45 mm
Reaming bit	89 mm
Drifter rod	4,305 mm
Drill Holes	
45 mm holes	62 ea.
Reamed holes	3 ea.
Hole depth	4,050 mm
Blasting	
Total loaded	59 holes
Bulk emulsion	48 holes
Roof perimeter – pre-splitting emulsion cartridge	11 holes
Toe charge – detonator-sensitive emulsion cartridge	59 holes
Booster 25g	59 holes
Nonel-type LP detonator 6.0 m	59 holes
Average break per round	3.8 m

Source: WSP, 2024

16.6 Mine Access and Underground Facilities

The deposit will be accessed by driving twin declines from the east side of the mineralisation. The decline designated as the haulage decline will provide access to the underground for personnel and equipment and function as the return-air exhaust for the mine ventilation system. The second decline, referred to as the ventilation decline, will serve as the fresh-air intake for the mine ventilation.

The development of a spiral ramp will commence from the end of the decline designated for haulage. It will provide access to the lower levels of the mine and enable the transport of mineralised material and waste to surface. Additionally, level access headings will be developed off the spiral ramp to access the sublevels for mining the deposit. These sublevels will be spaced at 20 m intervals.

The underground facilities planned for the mine include a maintenance bay, welding bay, warehouse, fuel-lube bay, explosives magazine, detonator magazine, and an underground office. The mine plan assumes that DPM personnel will undertake all installation and construction work except for raise boring and drilling service holes.

The QP is of the opinion that the underground infrastructure, mine services, and fixed equipment planned for the Čoka Rakita Mine are suitably designed and scaled for the mining operations.

For future studies, WSP recommends incorporating passing bays and safety cuddies into the spiral ramp design to enhance operational efficiency and safety.

16.7 Mine Safety Infrastructure

The Čoka Rakita mine will have the following infrastructure related to mine safety:

- The ventilation raises connecting the sublevels will be equipped with ladderways, providing an emergency escape alternative to the spiral ramp;
- The ventilation decline will provide a second means of egress from the mine in the event that travel via the haulage decline is not feasible;
- Mine-rescue equipment will be provided for use by the mine-rescue team;
- The mine will have a dedicated ambulance for emergency medical response;
- Eight (8) portable mine rescue chambers will be strategically positioned throughout the mine, each designed and equipped to accommodate up to eight people for 36 hours; and
- The mine will have a stench gas system to provide warning in an emergency.

WSP recommends including a permanent mine rescue chamber in the infrastructure design. This facility would have greater capacity than the portable units and serve as a central underground base for mine rescue operations during an emergency.

WSP recommends the implementation of dedicated escapeway raises in the mine rather than installing ladderways in ventilation raises. A ladderway in a ventilation raise reduces its efficiency for airflow by increasing friction loss. Additionally, the high velocity and pressure of the airflow in a ventilation raise render it less practical and safe as a travel way. A ladderway can be installed in a smaller diameter raise bore than is required for ventilation.

16.8 Mine Equipment

The Čoka Rakita Mine will be a mechanised operation utilising rubber-tired diesel equipment for all mining activities. Table 16.4 lists the mobile equipment planned for the mine, indicating the maximum number of units of each type required. Table 16.5 presents a schedule of annual equipment requirements over the life of mine (LOM). The number of units needed per year for each equipment type is based on their estimated productivities and the development and production targets outlined in the LOM plan.

The development jumbos will be equipped with telescopic feeds and other accessories, enabling them to install Split Sets and resin bolts. A cable bolter and a shotcrete sprayer, supported by two (2) transmixers, will be employed for additional ground support requirements. Two (2) emulsion chargers will be utilised for loading explosives for longhole blasting and development rounds. Production drilling will be conducted using two (2) longhole drill rigs, one top hammer and the other ITH. The ITH unit will be equipped with a slot-raise boring head.

The LHDs will have radio-remote-control systems to enable mucking operations inside open longhole stopes. The mine trucks will have a 45-t payload. Service equipment for the mine will include a telehandler, wheel loaders equipped with forks, a mobile rockbreaker, a fuel-lube truck, and a water truck. For tasks requiring work at height, personnel will operate from a platform elevated by a wheel loader.

The QP is of the opinion that the number of units in the planned equipment fleet, as well as their types, makes, and models, are appropriate for the projected production rate, mining method, and development requirements at Čoka Rakita.

WSP recommends conducting a trade-off study to evaluate the implementation of battery-electric mining equipment as an alternative to diesel-powered units. The study would consider various factors such as cost, efficiency, environmental impact, and operational capabilities to determine the most suitable option for mining operations.

WSP recommends investigating opportunities for implementing automated and tele-remote operation of mining equipment at the mine. This technology could be beneficial for extending equipment utilisation during what would otherwise be non-productive periods, such as shift changes and post-blasting smoke clearance. Potential applications for this technology include:

- Automated and tele-remote mucking of longhole stopes;
- Automated longhole drilling; and
- Tele-remote mucking of development rounds between shifts.

As a strategy to reduce capital and operating costs for haulage, WSP recommends exploring alternatives to traditional underground mining trucks. These alternatives, which have been proven effective in underground mines, include:

- Articulated haulers of the type utilised at construction sites;
- Highway-type dump trucks or tippers; and
- Tippers retrofitted for mining.

WSP recommends including a piece of equipment in the service fleet suitable for installing the schedule 80 paste-fill pipe. A practical option for this task could be a scissor lift. When selecting this equipment, it is crucial to consider the weight of the pipe and ensure adequate workspace availability to allow personnel to conduct the installation safely and efficiently.

Table 16.4 – Mobile Mine Equipment – Maximum Units Required

Equipment Type	Equipment Description	Max Units Required
Drilling and Blasting		
Development Jumbo	Two (2) boom, electric hydraulic, telescopic feeds	3
Production Drill, Top Hammer	Rod handler with capacity for 19 MF drill rods	1
Production Drill, ITH w/ Raise Boring Head	Equipped with boring Head for slot development	1
Booster Compressor	Required for ITH drill, 27.6 bar, 26.9 m^3/min	1
Emulsion Charger	Tunnels up to 65 m^2 cross section	2
Mucking and Haulage		
Mine Truck 45 t Payload	45 tonne payload	6
LHD 15 t Tramming Capacity	15 tonne tramming capacity	4
Ground Support		
Cable Bolter	Installs cable bolts up to 25 m long	1
Shotcrete Sprayer	20 m^3/hour capacity	1
Transmixer	5.6 m^3 capacity	2
Service		
Telehandler	4,000 kg capacity	1
Grader	Moldboard, 3658 mm Basic	1
Service Machine - Wheel Loader	Equipped with quick coupler for forks and bucket	2
Fuel Lube Truck	6,000 l capacity of diesel fuel, oils, grease and lubricants	1
Water Truck	6,000 l capacity	1
Mobile Rock Breaker	5000 ft-lb rock breaker	1
Diamond Drill Rigs	Equipped with semi-automatic rod handler	1
Pickup Truck	Diesel, twin cab, 4WD	15
Ambulance		1

Source: DPM, 2024

Table 16.5 – Mine Mobile Equipment – Units Required by Year

Equipment Type / Unit	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Drilling and Blasting												
Development Jumbo	3	3	3	3	3	2	2	2	2	1	1	1
Production Drill, Top Hammer	0	1	1	1	1	1	1	1	1	1	1	1
Production Drill, ITH w/ Raise Drilling Head	0	0	1	1	1	1	1	1	1	1	1	1
Booster Compressor	0	0	1	1	1	1	1	1	1	1	1	1
Emulsion Charger	1	1	1	2	2	2	2	2	2	2	1	1
Mucking and Haulage												
Mine Truck 45 t Payload	1	2	5	6	6	6	6	6	5	5	5	4
LHD 15 t Tramming Capacity	1	1	3	4	4	4	4	4	4	4	4	2
Ground Support												
Cable Bolter	1	1	1	1	1	1	1	1	1	1	1	1
Shotcrete Sprayer	1	1	1	1	1	1	1	1	1	1	1	1
Transmixer	2	2	2	2	2	2	2	2	2	2	1	1
Service												
Telehandler	1	1	1	1	1	1	1	1	1	1	1	1
Grader	1	1	1	1	1	1	1	1	1	1	1	1
Service Machine – Wheel Loader	2	2	2	2	2	2	2	2	2	2	2	2
Fuel Lube Truck	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck	1	1	1	1	1	1	1	1	1	1	1	1
Mobile Rock Breaker	1	1	1	1	1	1	1	1	1	1	1	1
Diamond Drill Rigs	1	1	1	1	0	0	0	0	0	0	0	0
Pickup Truck	11	12	15	15	15	15	15	15	14	14	13	11
Ambulance	1	1	1	1	1	1	1	1	1	1	1	1

Source: DPM and WSP, 2024

16.9 Mine Personnel

Table 16.6 presents the annual staffing requirements for the underground mine over the LOM. This roster includes provisions for absences, vacations, and temporary vacancies. The underground mine will reach its peak workforce of 327 employees in Year 2 of the LOM plan.

The mine will operate on three (3) 7-hour shifts, the workday length stipulated by Serbian legislation. The workforce will be organised into four (4) rotating groups, each working three (3) shifts followed by a period of days off.

Most of the workforce will be recruited locally or from the Serbian labour market. Additionally, DPM plans to bring in experienced personnel from Bulgaria to provide training and ensure safe, and efficient operations. According to the mine plan, all development and production activities will be carried out by DPM employees, except raise boring and drilling for service boreholes, which will be contracted out.

The QP is of the opinion that the proposed personnel structure is appropriate for the requirements of an underground mining operation of Čoka Rakita's scale.

For future resource evaluations, WSP recommends assessing the potential benefits of engaging contractors for specific mine development and infrastructure construction tasks, considering the varying intensity of these activities throughout the mine development program.

Table 16.6 – Annual Personnel Requirements over Life of Mine

Group / Role	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Production												
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Mine Captain Production	0	1	1	1	1	1	1	1	1	1	1	1
Production Supervisors/Shift Bosses	0	5	5	5	5	5	5	5	5	5	5	5
Longhole Drillers	0	5	9	9	9	9	9	9	9	9	9	9
Longhole and Development Blasters	10	10	10	18	18	18	18	18	18	18	10	10
LHD Operators	6	6	14	18	18	18	18	18	18	18	18	10
Mine Truck Operators	7	11	23	27	27	27	27	27	23	23	23	19
Backfill Crew	0	3	13	13	13	13	13	13	13	13	13	13
Cable Bolter Operator	5	5	5	5	5	5	5	5	5	5	5	5
Mine Helpers	4	4	7	9	9	9	9	9	9	9	8	5
Trainer Operations	1	1	1	1	1	1	1	1	1	1	1	1
Mine Clerk	1	1	1	1	1	1	1	1	1	1	1	1
Services												
Services Supervisors	3	3	4	5	5	5	5	5	5	5	5	3
Grader Operators	3	3	4	5	5	5	5	5	5	5	5	3
CAT IT Operators	4	4	7	9	9	9	9	9	9	9	8	5
Service Crew	7	6	13	17	17	17	17	16	16	16	15	9
Diamond Drill Operator	5	5	5	5	0	0	0	0	0	0	0	0
Development												
Mine Captain Development	1	1	1	1	1	1	1	1	1	1	1	1
Development Supervisors/Shift Bosses	5	5	5	5	5	5	5	5	5	4	1	1
Jumbo Operators	14	14	14	14	14	10	10	10	10	6	1	1
Shotcrete Operators	5	5	5	5	5	5	3	3	3	3	1	1
Transmixer Operators	9	9	9	9	9	9	6	6	6	6	1	1

Group / Role	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Trainer Development	1	1	1	1	1	1	1	1	1	1	0	0
Construction												
Construction Supervisor	1	1	1	1	1	1	0	0	0	0	0	0
Construction Journeypersons	6	9	9	1	3	5	0	0	0	0	0	0
Construction Apprentices	6	9	9	1	3	5	0	0	0	0	0	0
Underground Maintenance												
UG Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Senior Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planners	3	3	3	3	3	3	3	3	3	3	3	3
Junior Planners	2	2	2	2	2	2	2	2	2	2	2	2
Electrician Planners	3	3	3	3	3	3	3	3	3	3	3	3
UG Maintenance Supervisors	2	2	2	2	2	2	2	2	2	2	2	2
UG Maintenance Shift Bosses	5	5	5	5	5	5	5	5	5	5	5	5
UG Heavy Equipment Mechanics	10	12	16	18	18	18	18	18	17	16	15	12
UG Light Equipment Mechanics	4	4	7	9	9	9	9	9	9	9	8	5
UG Fixed Equipment Mechanics	4	4	7	9	9	9	9	9	9	9	8	5
Welders	4	4	4	5	5	5	5	5	5	5	5	3
Electricians	7	7	7	9	9	9	9	9	9	9	8	5
Electronics Specialists	4	4	4	5	5	5	5	5	5	5	5	3
Maintenance Apprentices	2	2	3	4	4	4	4	4	4	4	4	2
Safety												
Safety Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Safety Supervisor - Mine	1	1	1	1	1	1	1	1	1	1	1	1
Equipment Trainers - Mine	3	3	3	3	3	3	3	3	3	3	3	3
Safety Apprentices	5	5	5	5	5	5	5	5	5	5	5	5
Mine Rescue and First Aid	10	10	10	10	10	10	10	10	10	10	10	10

Group / Role	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Mine Dry Attendants	5	5	5	5	5	5	5	5	5	5	5	5
Technical Services												
Technical Services Superintendent	1	1	1	1	1	1	1	1	1	1	1	1
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Drill and Blast Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Junior Drill and Blast Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Mining Engineers (Design and Planning)	2	2	2	2	2	2	2	2	2	2	2	2
Junior Mining Engineers	2	2	2	2	2	2	2	2	2	2	2	2
Ventilation Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1
Resource/Exploration Geologist	1	1	1	1	1	1	1	1	1	1	1	1
Database Geologist	1	1	1	1	1	1	1	1	1	1	1	1
Mine Geologists	2	2	2	2	2	2	2	2	2	2	2	2
Junior Geologist	1	1	1	1	1	1	1	1	1	1	1	1
Mine Samplers	5	5	5	5	5	5	5	5	5	5	5	5
Core Logging Geologist	1	1	1	1	1	1	1	1	1	1	1	1
Core Technicians	3	3	3	3	3	3	3	3	3	3	3	3
Core Cutter	1	1	1	1	1	1	1	1	1	1	1	1
Geotechnical Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Junior Geotechnical and Backfill Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Backfill Engineer	1	1	1	1	1	1	1	1	1	1	1	1
Senior Surveyor	1	1	1	1	1	1	1	1	1	1	1	1
Surveyors	5	5	5	5	5	5	5	5	5	5	5	5
Surveyor Assistants	5	5	5	5	5	5	5	5	5	5	5	5
Total Personnel	**221**	**246**	**306**	**327**	**326**	**326**	**310**	**309**	**304**	**298**	**267**	**221**

Source: DPM and WSP, 2024

16.10 Pre-Production Schedule

Figure 16.7 outlines the pre-production schedule for developing and initiating operations at the underground mine. The pre-production phase extends until the middle of Year 1, culminating with the mine attaining full production.

Year -1: The Project's pre-production phase will commence in the year prior to initiating the development of the twin declines. The principal activities during the year will include:

- Excavation of portals for the twin declines;
- Procurement of essential mining equipment, considering the lead time required for delivery from manufacturers; and
- Recruitment and training of personnel for mine development.

Year -2: Year -2 of the schedule will focus on developing the twin declines and initiating the spiral ramp, with additional progress achieved in advancing ventilation drifts, level access headings, and infrastructure excavations. The decline designated for haulage will advance at 5.0 m/day, while the ventilation decline will advance at 4.0 m/day. Achieving these rates will require allocating sufficient personnel and equipment to these headings to effectively develop them as independent single headings in terms of productivity. The development of the spiral ramp will commence in Q3, following the completion of the twin declines.

Year -1: In Year -1, progress will continue in advancing the spiral ramp, ventilation drifts, level access headings, and infrastructure excavations. The focus, however, will shift towards developing the sublevels within the deposit. About half of the year's advance will consist of development required for sublevel stoping, yielding about 46 kt of mineralised material. The spiral ramp will be treated as a priority heading, achieving a rate of 5.0 m/day, while other headings will advance 3.1 m/day on average. Development of the spiral ramp will cease upon reaching the 440 level and remain idle until Year 3. Towards the end of the year, longhole stoping production will commence on the 440 sublevel, generating about 11 kt of mineralised material.

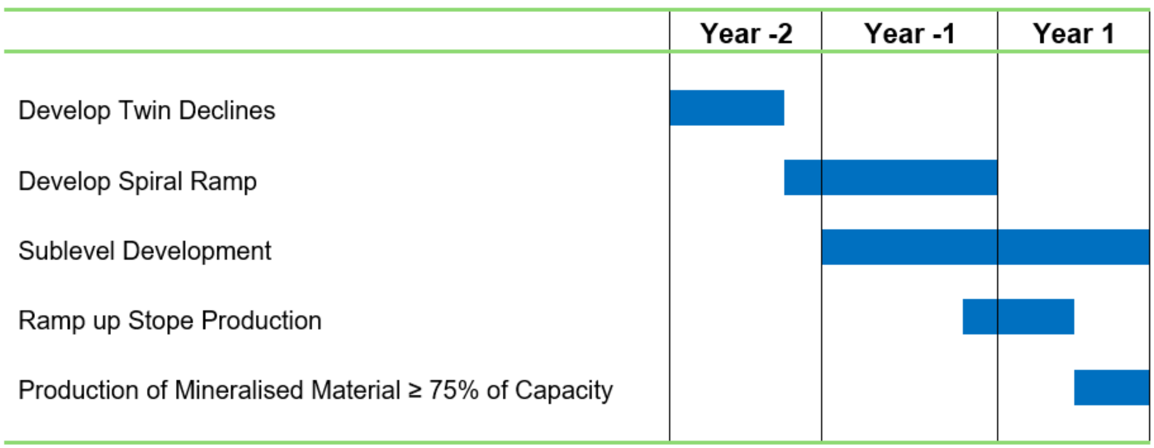
Year 1: Development in waste and mineralisation will continue during Year 1. Stope production will ramp up during the year, accounting for 81% of the mineralised material mined. The pre-production phase will conclude with the mine attaining commercial production (75% of planned capacity) in Q2.

For future studies, WSP recommends verifying that the portal entrances of the twin declines, currently planned with a 30 m centre-line spacing, are sufficiently separated from each other such that the intake of the ventilation decline does not draw return air exhausted from the haulage decline.

In future revisions of the Pre-Production Schedule, WSP recommends enhancing the clarity and detail of the activities required before the initiation of development of the twin declines, including the following aspects:

- Timelines for ordering and delivery of critical equipment;
- Strategies for procurement and logistics;
- Plans for the mobilisation of resources and installation at the Project site; and
- Recruitment and training of personnel.

Figure 16.7 – Pre-Production Schedule



Source: DPM, 2024

16.11 Life of Mine Plan

16.11.1 Production Plan

As indicated in Table 16.7, the LOM production plan extends over 10 years, during which the mine will produce mineralised material at full capacity for eight (8) years. Table 16.8 provides a breakdown of the LOM production by level. Figure 16.8 illustrates the locations of the levels. The potentially viable mineral resources used to develop this LOM plan are based solely on Inferred Resources. They incorporate an unplanned dilution factor of 10% and a mining recovery of 95%.

Year -2: The Project's initial year will focus exclusively on waste development; consequently, there will be no production of mineralised material.

Year -1: In Year -1, the mine will produce a modest amount of mineralised material from sublevel development and stoping operations. Stope production will commence on the 440 sublevel, situated approximately two-thirds of the way down the depth of the deposit. DPM’s strategy is to mine the deposit in two (2) phases, starting with an upper zone to deliver mineralised material to the processing plant per the planned schedule. From the 440 sublevel, stoping operations will progressively advance to higher levels in an ascending sequence.

Year 1: The production of mineralised material from development and sublevel stoping will increase substantially in Year 1, averaging about 65% of the mine's full capacity. Approximately 19% of the output will come from development, with the remainder originating from stope production.

Year 2: The mine will operate at full output during Year 2, producing slightly over 850 kt of mineralised material. Of this total, approximately 12% will come from development, with the remainder originating from stope production.

Year 3: The spiral ramp will reactivate in Year 3, allowing development to proceed in the lower sublevels of the mine. The mine output will remain at just over 850 kt, with sublevel development contributing around 10% of the mineralised material.

Year 4: In Year 4, sublevel stoping will commence on the 360 level, following the completion of the spiral ramp. Production in the lower zone will progressively replace that of the upper zone. The mine output will remain at just over 850 kt; however, sublevel development will decrease substantially, accounting for only around 3% of this total.

Years 5 to 8: The mine will operate at full capacity from Years 5 to 8, with sublevel development contributing only about 4% of the mineralised material produced.

Year 9: In Year 9, the mine will maintain full output with production coming solely from stoping operations.

Year 10: In Year 10, the final year of the LOM plan, production will decrease to approximately 58% of full capacity. The last sublevel to be mined will be the 430 level, recovering the sill left below the 440 level where stope production commenced in Year -1.

Table 16.7 – Life of Mine Production Plan

Description	Unit	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total / Average
Development Mineralised Material	tonnes	-	46,679	108,931	101,684	88,343	29,582	20,088	29,335	37,387	35,989	-	-	498,018
Grade	g/t	-	7.24	7.61	6.56	6.60	4.23	4.84	3.87	4.15	4.68	-	-	6.18
Contained Gold	oz	-	10,860	26,638	21,440	18,752	4,020	3,126	3,646	4,994	5,414	-	-	98,890
Stope Mineralised Material	tonnes	-	10,728	450,325	758,082	764,660	823,462	832,659	823,517	812,690	821,098	856,862	490,701	7,444,785
Grade	g/t	-	5.29	6.47	6.48	7.66	7.47	6.35	5.84	4.36	3.58	4.34	3.74	5.65
Contained Gold	oz	-	1,825	93,715	157,937	188,329	197,888	170,005	154,576	114,037	94,501	119,530	58,993	1,351,335
Total Mined	**tonnes**	**-**	**57,406**	**559,256**	**859,766**	**853,003**	**853,044**	**852,748**	**852,853**	**850,077**	**857,087**	**856,862**	**490,701**	**7,942,803**
Grade	g/t	-	6.87	6.69	6.49	7.55	7.36	6.31	5.77	4.36	3.63	4.34	3.74	5.68
Contained Gold	oz	-	12,685	120,353	179,377	207,081	201,907	173,131	158,222	119,031	99,916	119,530	58,993	1,450,226

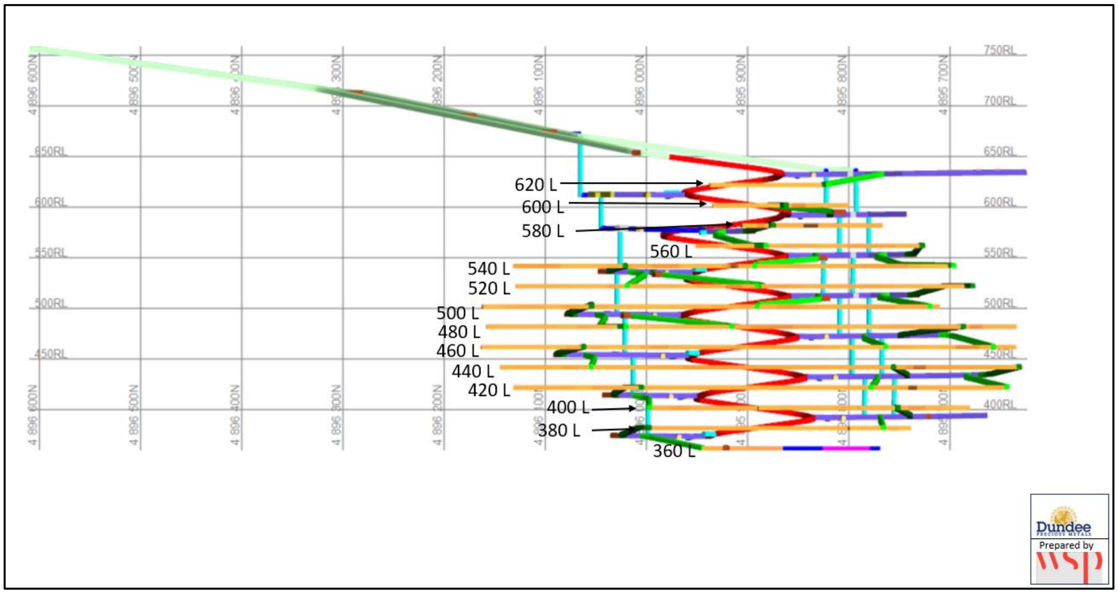
Source: DPM, 2024

Table 16.8 – Life of Mine Mined Mineralised Material Tonnes by Level

Description	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
RL360	-	-	-	-	-	-	3,147	57,762	-	-	-	-	60,908
RL380	-	-	-	-	-	-	1,433	12,883	109,414	163,611	-	-	287,342
RL400	-	-	-	-	-	-	-	-	18,962	154,638	191,026	-	364,626
RL420	-	-	-	-	-	-	-	-	1,319	25,118	369,668	283,822	679,928
RL440	-	22,140	487,044	480,232	294,843	91,504	70,456	21,508	-	-	117,731	-	1,585,457
RL460	-	35,266	44,659	313,510	413,911	369,757	328,306	116,521	-	4,809	36,681	-	1,663,419
RL480	-	-	26,076	39,366	74,745	364,262	273,407	153,419	201,491	34,857	4,875	60,676	1,233,174
RL500	-	-	-	26,659	22,546	6,514	166,273	414,535	67,218	167,416	-	119,801	990,961
RL520	-	-	-	-	46,959	2,298	5,231	72,786	434,566	50,842	48,201	26,402	687,286
RL540	-	-	-	-	-	18,709	2,796	-	-	172,923	-	-	194,428
RL560	-	-	-	-	-	-	1,699	-	8,790	78,384	6,367	-	95,239
RL580	-	-	-	-	-	-	-	683	-	4,489	37,683	-	42,855
RL600	-	-	1,477	-	-	-	-	2,757	8,316	-	44,630	-	57,181
Total Mined Mineralised Material Tonnes	**-**	**57,406**	**559,256**	**859,766**	**853,003**	**853,044**	**852,748**	**852,853**	**850,077**	**857,087**	**856,862**	**490,701**	**7,942,803**

Source: DPM, 2024

Figure 16.8 – East Longitudinal View of the Mine Levels



Source: WSP, 2024

16.11.2 DEVELOPMENT PROCEDURE

Mine development at Čoka Rakita will use drilling and blasting techniques. Development jumbos will drill the drift rounds. Once a round is blasted, an LHD will muck the broken rock, tram it away from the face, and dump it at the nearest muck bay. An LHD will later rehandle the muck stockpiled in the muck bay, loading it onto mine trucks for haulage to surface. However, development waste may also be used as rock fill for backfilling secondary stopes.

DPM plans on using the same development jumbos that drill the drift rounds to scale and install ground support. This multi-functional use of the jumbos is practiced at DPM's Chelopech Mine and is a common approach at underground mines in Australia and Asia. Using a jumbo for drilling, scaling, and installing ground support can reduce non-productive time in the drift-round cycle. Instead of three (3) pieces of equipment requiring time in the cycle for travelling between headings, moving in, setting up, tearing down, and moving out, this non-productive time is limited to just one machine.

As of the date of this Report, DPM was investigating solutions to adapt the proposed make and model of jumbo for installing ground support. The jumbo's fixed feed for 4,305 mm drifter rods measures 5.965 m, too long to install ground support in any of the headings in the mine plan. The jumbo can be ordered with a telescopic feed for each boom. However, when retracted, it can drill to a depth of only 2.22 m, which is too short for the Split Sets and resin rebar specified in the mine design parameters.

For future resource evaluations, WSP recommends conducting a trade-off study to assess whether the benefits of utilising a development jumbo for multiple functions outweigh the efficiency gained through using specialised equipment dedicated to scaling and installing ground support. This study should consider various factors such as operational costs, time efficiency, equipment versatility, operator safety, and overall productivity to determine the most effective approach for the Project.

16.11.3 Development Plan

Table 16.9 presents the LOM development plan for Čoka Rakita. Mine development will be intensive in the initial years of the mine life but diminishes in later years. The horizontal advance will average approximately 4,000 m/year during the initial two (2) years of pre-production development and two (2) years of operations, decreasing to 3,290 m in Year 3. From Years 4 to 8, it will average around 2,000 m annually and cease entirely during the final two (2) years of the mine life.

Year -2: Year -2 will focus on developing the twin declines, the spiral ramp, and level access headings, with progress also achieved in advancing ventilation drifts and infrastructure excavations. The decline designated as the haulage decline will provide access to the mine for personnel and equipment and serve as the ventilation system exhaust airway. The other decline will serve as the intake airway for ventilation and provide a second means of egress from the mine. Development of the spiral decline will commence when the haulage decline reaches its endpoint, affording access to the lower levels of the mine. The ramp will have a -14% grade, flattening to -5% at intersections. Vertical development during the year will consist of 133 m of ventilation raises driven with a raise borer.

Year -1: In Year -1, the focus of development efforts will shift to include crosscuts and production drives within the deposit mineralisation, comprising approximately half of the year's horizontal advance. The development of the spiral ramp will progress until reaching the 440 level, at which point it becomes inactive. The horizontal and vertical ventilation development for the upper levels of the mine will be completed during the year. The remaining development will consist of level-access headings and infrastructure excavations.

Year 1: During Year 1, around 80% of the development will be focused on crosscuts and production drives in preparation for stoping operations. Progress will also be achieved in advancing the ventilation drifts and infrastructure excavations. The spiral ramp will remain inactive in Years 1 and 2.

Year 2: Year 2 development will focus almost exclusively on advancing crosscuts and production drives in mineralisation. Only about 2% of the horizontal advance will be waste development for level-access headings and infrastructure excavations.

Year 3: Development of the spiral ramp will resume in Year 3. Almost 90% of the year’s advance will be dedicated to crosscuts and production drives in preparation for stoping operations. Additionally, development will continue advancing level-access headings and infrastructure excavations.

Year 4: In Year 4, the development of the spiral ramp will continue until it reaches the lowest level of the mine, providing access to the 360 sublevel. Approximately 70% of the year's advance will consist of crosscuts and production drives for mining the lower sublevels. The horizontal and vertical ventilation development required for the lower levels of the mine will be completed during the year.

Table 16.9 – Life of Mine Development Plan

Development Metres by Excavation Type	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Horizontal Capex Development:													
Surface Haulage Decline	771	-	-	-	-	-	-	-	-	-	-	-	771
Surface Ventilation Decline	928	-	-	-	-	-	-	-	-	-	-	-	928
Spiral Ramp	1,054	686	-	-	180	277	-	-	-	-	-	-	2,197
Ramp Level Access	465	588	331	71	166	312	11	-	11	-	-	-	1,954
Stockpile/Remuck Bay	136	-	78	11	60	42	34	31	16	-	-	-	408
Electrical Substation Cuddy	34	83	-	-	10	36	-	-	-	-	-	-	163
Paste Fill Cuddy	28	28	42	14	83	42	-	-	-	-	-	-	236
Horizontal Ventilation Development	407	561	-	-	-	136	-	-	-	-	-	-	1,104
Water Bay/Sump	138	70	-	-	10	20	-	-	-	-	-	-	238
Pump Station	20	37	-	-	-	18	-	-	-	-	-	-	75
Explosive Storage	-	-	254	-	-	-	-	-	-	-	-	-	254
Detonator Storage	-	-	19	-	-	-	-	-	-	-	-	-	19
Maintenance Bay	-	-	50	-	-	-	-	-	-	-	-	-	50
Welding Bay	10	-	-	-	-	-	-	-	-	-	-	-	10
Warehouse	-	-	10	-	-	-	-	-	-	-	-	-	10
Warehouse	-	-	23	-	-	-	-	-	-	-	-	-	23
Fuel and Lube Bay	-	-	20	-	-	-	-	-	-	-	-	-	20
Wash Bay	-	-	20	-	-	-	-	-	-	-	-	-	20
Total Horizontal Capex Development	**3,991**	**2,051**	**848**	**95**	**509**	**884**	**45**	**31**	**27**	**-**	**-**	**-**	**8,481**

Development Metres by Excavation Type	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Horizontal Opex Development:													
Sublevel Drift – MINERALISATION	-	672	1,729	1,728	1,424	543	367	462	661	636	-	-	8,224
Xcut Decline – MINERALISATION	-	30	127	90	116	-	-	-	-	-	-	-	362
Xcut Incline – MINERALISATION	-	90	-	-	-	-	-	-	-	-	-	-	120
Sublevel Drift – WASTE	-	355	566	1,128	871	672	988	851	668	826	-	-	6,926
Xcut Decline – WASTE	-	450	441	686	450	533	448	164	475	-	-	-	3,647
Xcut Incline – WASTE	30	365	310	272	429	242	168	517	191	156	-	-	2,680
Total Horizontal Opex Development	**30**	**1,962**	**3,174**	**3,904**	**3,290**	**1,990**	**1,971**	**2,024**	**1,996**	**1,618**	**-**	**-**	**21,959**
Vertical Capex Development	133	487	-	-	-	161	-	-	-	-	-	-	**781**
Total Development	**4,154**	**4,500**	**4,022**	**3,999**	**3,798**	**3,035**	**2,016**	**2,055**	**2,023**	**1,618**	**-**	**-**	**31,221**

Source: DPM, 2024

Years 5 to 8: From Years 5 to 8, development will focus almost exclusively on crosscuts and production drives, with limited advance in infrastructure excavations. All development activities will be completed by the end of Year 8, with no further advance planned during the final two (2) years of mine operation.

In future studies, WSP recommends redesigning the ventilation system, making the haulage decline and spiral ramp the intake airways. This approach enhances safety by ensuring personnel remain in fresh air when exiting the mine during an underground fire. The downcasting airflow may also enhance engine cooling efficiency for mine trucks hauling up-ramp fully loaded.

WSP recommends evaluating the potential advantages of engaging a mining contractor to drive the twin declines. The contractor could bring in experienced personnel, whereas, with the self-perform approach, DPM would need to hire, train, organise, and manage new crews who may have less experience. Additionally, if the contractor supplies their own equipment, it could allow extra time for the delivery of DPM's equipment, thus providing a buffer in the Project schedule against potential delays.

16.12 Electrical

Power for the underground mine will be supplied from a surface substation that is defined as part of the surface infrastructure. This substation and associated distribution network steps down the site transmission voltage to the distribution voltage of 10 kV and delivers it to the portal. Cabling will be routed down the decline with two (2) mine power feeders installed along messenger wires. This will provide a redundant connection that allows power to be transferred upon failure of one circuit.

At each load centre that supplies power to fixed loads such as mine dewatering and ventilation, an electrical bay ('EBay') will be established with a concrete floor, raised pads, lighting, ventilation and cable management. EBays can be open style or enclosed with a bulkhead, depending on the operational preferences. The open style allows operators to see the equipment as they are passing and identify any issues. However, it requires that each piece of equipment is water and dust-tight and temperature-controlled. It is generally a lower-cost solution than enclosing the room due to the cost of the bulkhead and central air conditioning. The drawback is the dust exposure from material haulage that requires regular changing of filters. It can also be noisier and windier depending on where it is situated in the mine.

Each EBay will house a 10 kV Ring Main Unit (RMU) style of medium voltage switchgear that provides feed-through switches for the backbone feeders that drop down to each level and breakers to supply the step-down transformers for 1000 V or 400 V utilisation. The cable from the EBay above will terminate at the incoming feed-through terminals and feed through a borehole to the EBay below. Two (2) medium voltage feeders will be routed to the bottom of the mine through boreholes between EBays and will form a ring main. If a failure occurs at any point in the network, power can be redirected through the ring main, restoring power to the healthy portions of the system. This is especially important for critical ventilation and dewatering loads and is a requirement of Serbian regulations.

The 1000 V output of the transformer will supply a distribution board that supports the fixed infrastructure and the mining equipment. The benefit of supplying mining equipment at this higher voltage is that the cable runs can be much longer without the need for a Mine Load Centre (MLC) in the production area that steps 10 kV down to 1000 V. Every MLC requires a cuddy, which increases development cost and time.

The breakers in the distribution board, starters and Variable Speed Drives (VSDs) will have ground fault protection to protect the cables. The cables from the distribution board or starters will be supported along messenger wires through the drift until they reach the active mining area. It will terminate at a panel that allows the equipment to be plugged in and safely isolated. The panel could be a custom-built ruggedised panel with integral breakers and starters with ground fault protection and controls to suit the types of equipment used, including mining equipment, face fan and pump. One such product is DEA's Terminator T4, which can support up to 1000-V equipment. The cabling to mining equipment will be trailing cable with a ground check wire that monitors cable integrity.

The pumps are fixed-speed, so they do not require a VSD. They will be connected to Reduced Voltage Soft Starters (RVSS) to reduce the inrush current and improve the stability of the power network. This also reduces the physical strain on the equipment. VSDs are sometimes used for this application, but they require active cooling, which increases the power demand, heat, and maintenance requirements for filter changes. VSDs also affect system power quality because of the harmonics, so they should only be used where necessary. 1000 V-rated motors and VSDs are not common, so consideration should be given to providing 400 V at pump stations with a 10000-1000/400 V dual-winding transformer or stepping down 1000 V to 400 V with individual transformers for each pump.

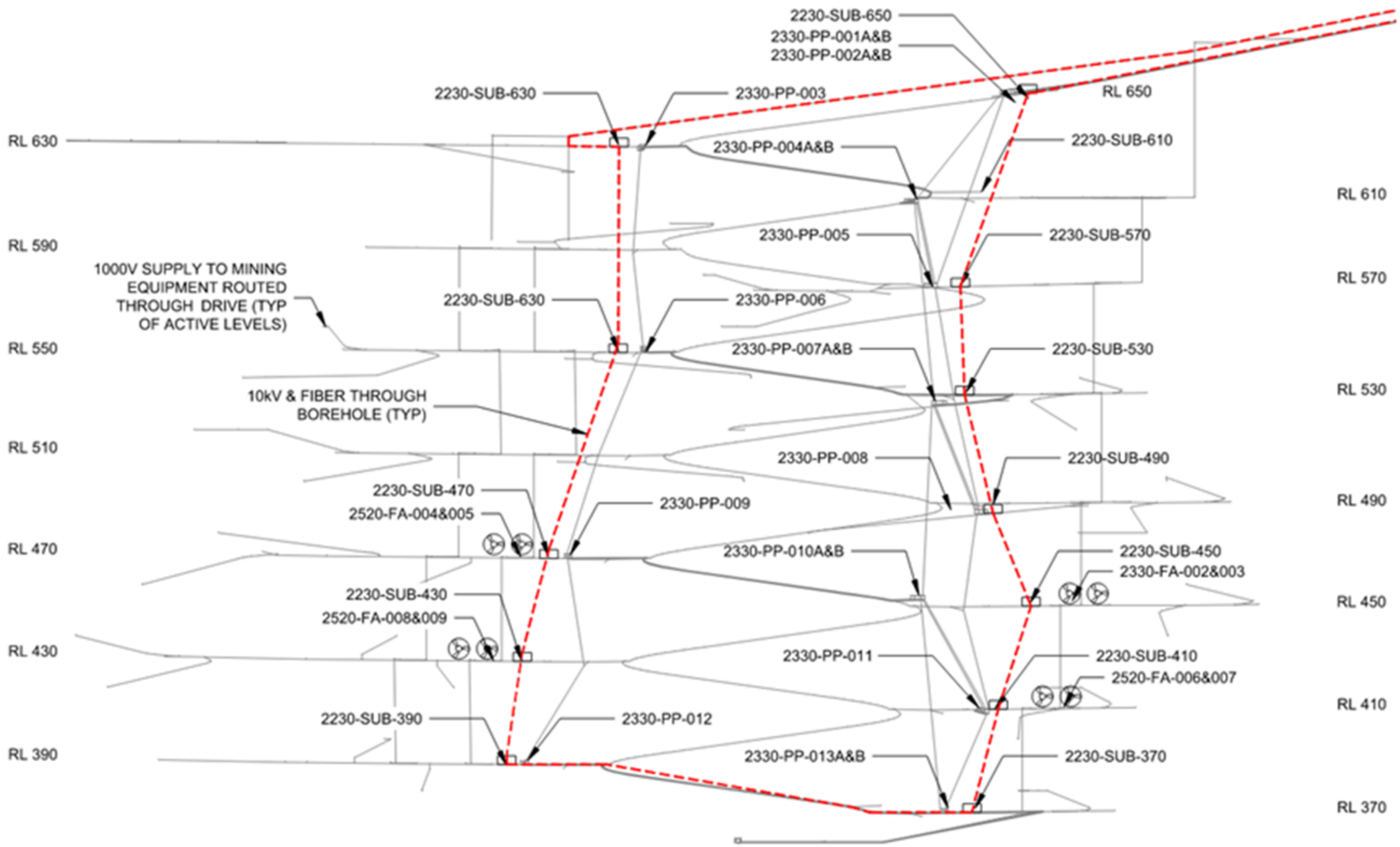
The fans are variable flow, so they will be controlled by VSDs. The same point above applies regarding the challenges with supply of 1,000 V motors.

Figure 16.9 provides an underground electrical reticulation schematic.

16.13 Underground Communication

A fibre optic cable will be routed through the mine, following the main feeders but in parallel boreholes. This cable will connect the network cabinets at each EBay, which forms a communication network throughout the mine. From these network cabinets, Programmable Logic Controllers (PLCs), motor drives, WiFi access points, leaky feeder, CCTV, fire protection systems, and other network devices can be connected.

Figure 16.9 – Underground Electrical Reticulation Schematic



Source: WSP, 2024

16.14 Ventilation

16.14.1 Air Requirements

The airflow requirements were determined according to the planned vehicle fleet and current Canadian regulations in place for Ontario and Quebec and applied to this Project. The QP is of the opinion that European regulations should be applied. Serbia has candidate status for EU membership. Accordingly, the following assumptions were used:

- The airflow requirement for diesel equipment used to ensure adequate dilution is 0.06 m³/s/kW, where the kW is the nominal engine power of the vehicle.
- This regulation is currently in force for Ontario, Canada. This rate was chosen as it is the highest standard for dilution factor used in the industry and is expected to cover the lower exposure limit norms within the European Union. The province of Quebec in Canada has regulated the air requirement based on certified engines, CANMET, which normally have an airflow requirement of 30% or less than the requirement that would be set by 0.06 m³/s/kW.

  For example, for the LH515i used in Čoka Rakita, the CANMET requirement is 5.2 m³/s. On the other hand, applying the 0.06 m³/s/kW regulation to the LHD results in a requirement of 15.4 m³/s. A combination of the use of clean engines (Tier 2 or greater) and the 0.06 m³/s/kW should, therefore, suffice to maintain a safe and healthy environment that is compliant with the regulations in force. With a factor of 0.06 m³/s/kW, the airflow requirement of each vehicle with a motor that is not CANMET certified was determined and then multiplied by the quantity of equipment and utilisation rate to obtain the total airflow requirement. The list of equipment and the utilisation rate were drawn up jointly by DPM and WSP based on the mine's total planned production. A 10 % contingency was used for any requirement that may not have been included.
- A leakage factor of 20% was also used to account for leakages across doors and bulkheads.

The total air requirements are 265 m³/s. The recommended amount of air required at a heading where a truck and an LHD are working together is 22 m³/s. The minimum air supplied to any heading will be 22 m³/s.

16.14.2 Ventilation Layout

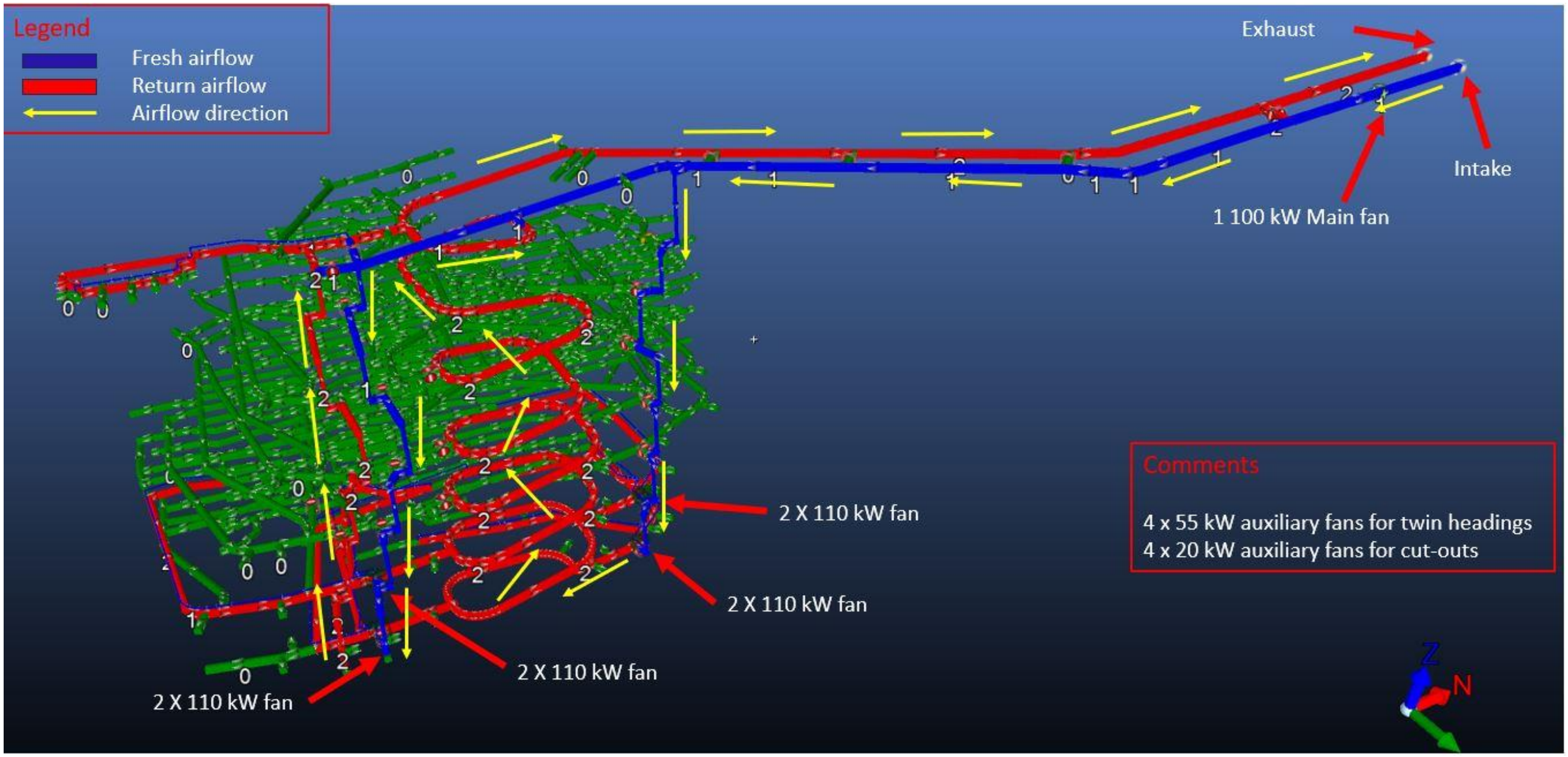
The ventilation layout for the Project was determined based on the airflow requirements and planned infrastructure. The software used for modelling is Ventsim™. The model was designed to meet the requirements of the initial preliminary equipment list, 320 m³/s and is illustrated in Figure 16.10. The model is designed to provide more airflow than required for the current equipment list and will have to be revised in the following study stage of the Project.

Table 16.10 – Total Air Requirements

Mobile Equipment Model	Unit Power (kw)	Unit Air Flow (m^3/s)	Units	Utilisation (%)	Total Air Flow (m^3/s)	Regulation
Jumbo, 2 Boom	124	6.4	3	27	5	CANMET
Production Drill, Top Hammer	110	4.3	1	30	1	CANMET
Production Drill, ITH	110	4.3	1	20	1	CANMET
Emulsion Charger	90	3.9	2	50	4	CANMET
Mine Truck	450	16.4	6	80	79	CANMET
LHD	256	5.2	4	80	17	CANMET
Cable Bolter	110	4.3	1	25	1	CANMET
Shotcrete Sprayer	155	9.3	1	25	2	0.06 $m^3/s/kW$
Transmixer	185	11.1	2	70	16	0.06 $m^3/s/kW$
Telehandler	75	4.5	1	50	2	0.06 $m^3/s/kW$
Grader	93	5.6	1	50	3	0.06 $m^3/s/kW$
Service Machine/ Wheel Loader	111	6.7	2	80	11	0.06 $m^3/s/kW$
Water Truck	185	11.1	1	50	6	0.06 $m^3/s/kW$
Fuel-Lube Truck	185	11.1	1	50	6	0.06 $m^3/s/kW$
Mobile Breaker	147	8.8	1	40	4	0.06 $m^3/s/kW$
Pickup Truck	126	7.6	15	40	45	0.06 $m^3/s/kW$
Ambulance	126	7.6	1	10	1	0.06 $m^3/s/kW$
Subtotal			*44*		*204*	
10% for non-defined requirement (personnel, garage)					20	
Leakage 20%					41	
Total Ventilation Requirement					**265**	

Source: WSP 2024

Figure 16.10 – Ventilation Layout



Source: WSP, 2024

16.15 Mine Backfill

The primary mine backfill will be Cemented Paste Fill (CPF) formulated from mill tailings in the surface paste plant. The CPF will be pumped from the paste plant to the collar of a borehole located remotely from the plant. The Underground Distribution System (UDS) design begins at the collar of the surface borehole and stretches throughout the mine, allowing paste to be delivered into the mine's primary and secondary stopes.

The CPF design criteria and material mass balance were developed based on information received from DPM, provided by other groups working on the Project, and derived from available information and past similar projects.

The information in the design criteria and mass balance was used in the design of the paste backfill UDS for such things as pipeline diameter, materials of construction, and paste flow velocity.

16.15.1 Key Design Components of the Backfill Underground Distribution System

Common design elements within the entire design of the UDS are:

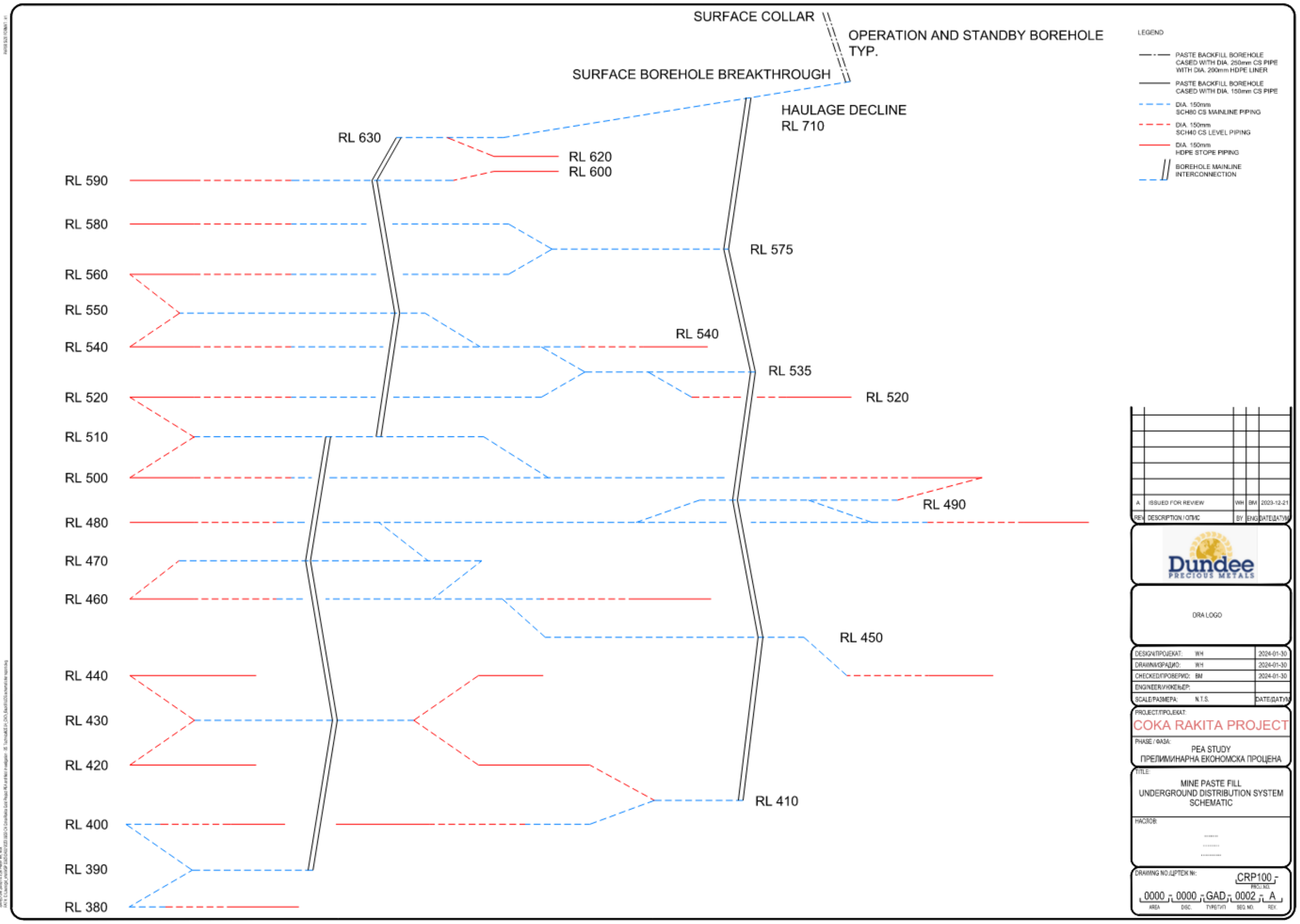
- All boreholes are "twinned" both from surface to the decline, as well as level to level, meaning that there is one operating and one standby borehole in case the operating borehole becomes inoperative. All boreholes are "cased," meaning that oversized holes are drilled within which there is a carbon steel (CS) casing pipe installed due to predicted rock characteristics.
- The boreholes originating on surface have a 250 mm CS casing pipe, which is internally lined with a 200-mm, High-density polyethylene (HDPE) pipe for added longevity of the boreholes. None of the other borehole casings are internally lined.
- The boreholes within the mining areas have a 150-mm CS casing pipe.
- All boreholes originate or terminate in cut-outs (cuddies) to allow drilling equipment operation without interfering with mine vehicle traffic.
- CS pipe sections of either 4 m or 8 m lengths are used and connected together using high-pressure Victaulic style couplings. HDPE pipe of similar lengths is connected together via flange adapters.
- Mainline piping is connected to the boreholes and is constructed of 150 mm CS Schedule 80 pipe to increase its longevity and withstand the possibly higher internal pressures.
- Level piping connected to the mainline piping is constructed of 150 mm CS Schedule 40 pipe due to its decreased longevity requirements and possibly lower internal pressures.
- Stope access piping connected to the level piping is constructed of 150 mm HDPE due to its decreased longevity requirements and lower internal pressures.

Figure 16.12 illustrates the relative position of the various backfill boreholes and level piping with respect to the mine workings. For clarity, the figure shows only one of any twinned boreholes within the system.

The surface boreholes, which dip at approximately 40° from the horizontal, intersect one of the twin declines and extend down the heading to connect to individual primary borehole systems that service each side of the mineralised zone in the Mine. There are two (2) borehole systems to effectively service the development and mining sequence.

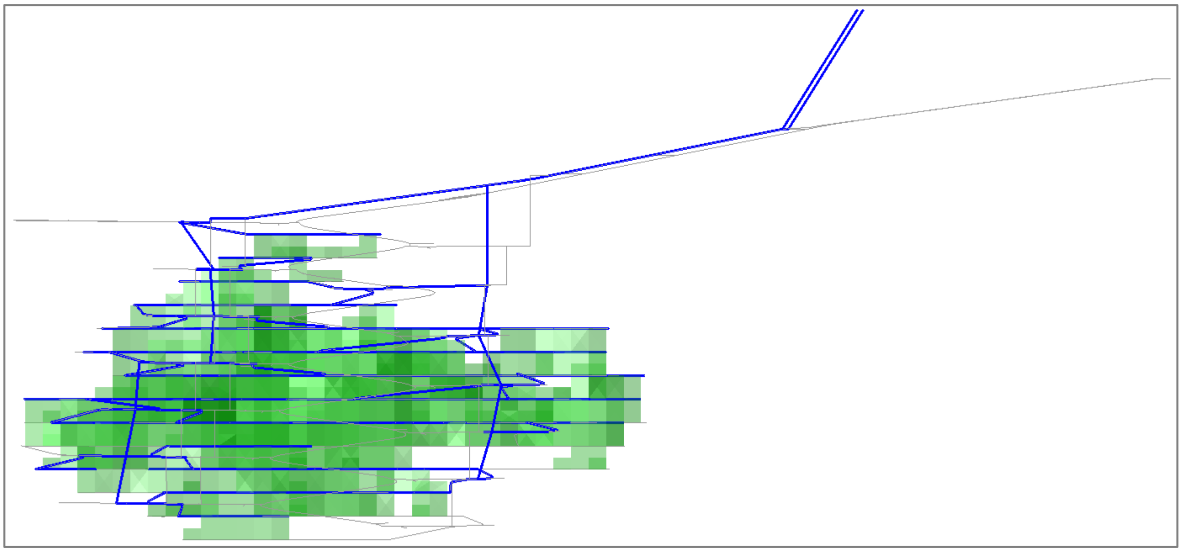
Generally, the interlevel boreholes dip 65° to 70° from the horizontal. The level piping connects to each bore, as illustrated in Figure 16.13.

Figure 16.11 – Mine Paste Fill Underground Distribution System Schematic



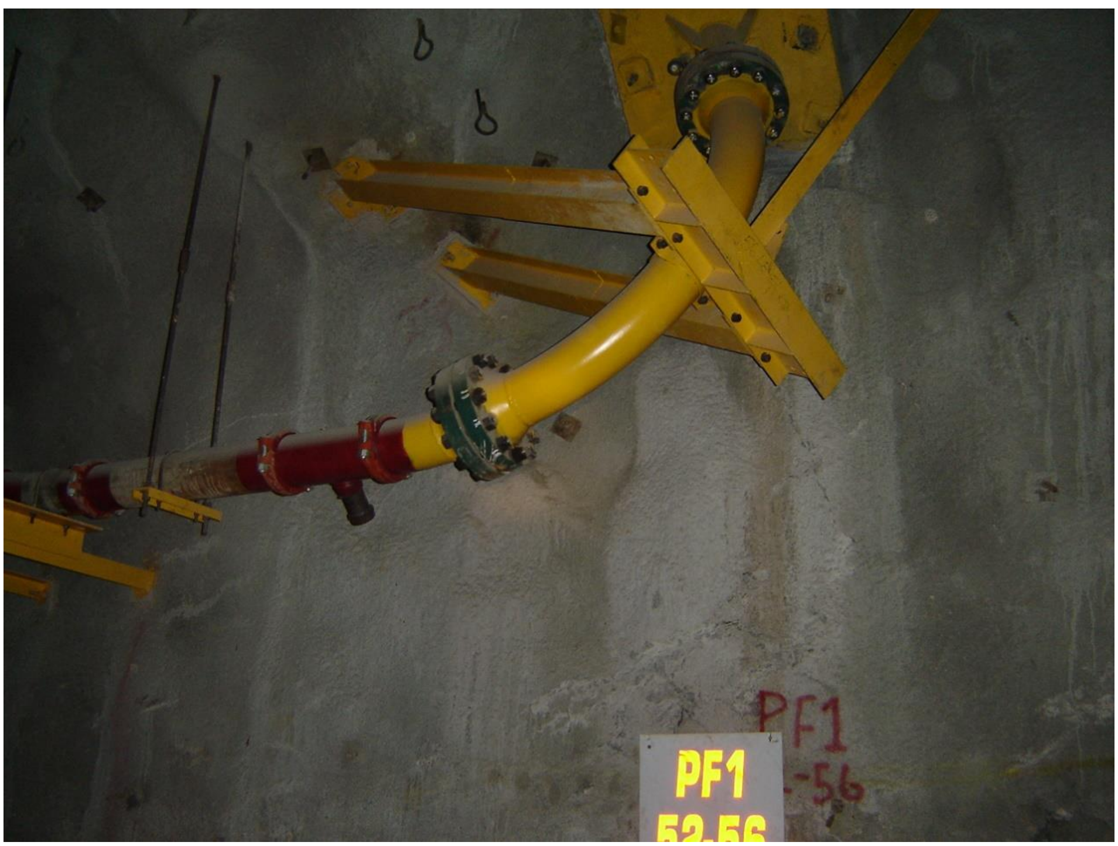
Source: DPM, 2024

Figure 16.12 – Backfill Underground Distribution System on a Longitudinal Section



Legend: Blue – UDS system; Green – Stope
Source: DPM, 2024

Figure 16.13 – Typical Borehole to Level Piping Connection Configuration



Source: WSP 2024

16.16 Mine Dewatering

DPM provided WSP with their preliminary mine dewatering concept, which included examples of pumps used in another DPM mine. WSP refined the concepts presented to a PEA-level design by determining the pump and pipeline sizing appropriate for the anticipated quantity of water to be discharged from the Mine to ponds located on surface. The pump sizes provided by DPM were found not to be suitable in this application.

It is important to note that the battery limit for the mine dewatering study is the underground mine water collection and water delivery to the mine portal.

The mine dewatering design criteria were based on the water balance, which included:

- Groundwater seepage;
- Paste backfill contained water release at placement and system flushing; and
- Water required for mining operations, including drilling, dust suppression when mucking, and other ancillary water consumption.

The water balance for this Study did not consider water generated from diesel fuel combustion emissions, water leaving the Mine with mineralised material and waste, and water introduced or expelled via the ventilation system.

16.16.1 System Design Inputs

The following sections explain the inputs used to develop the dewatering system design.

16.16.1.1 Ground Water Seepage

No geotechnical drilling has yet taken place at the Čoka Rakita site, so the estimate of the quantity of water from groundwater seepage was based on information provided by other engineering groups working on that portion of the Project and information on the available data from the nearby DPM Timok Mine.

The hydrogeological group provided Table 16.11 while completing their SOW under Section 16.3 of this Report. The hydrogeological team presented this summary table to DPM, and DPM agreed to use it as the groundwater seepage quantity to be included in the dewatering assessment.

Table 16.11 – Ground Water Seepage Estimate

Elevation (m asl)	Depth (m)	Inflow based on different K values (L/s)		
		3x Lower	Base Case	3x Higher
750 - 700	0-50	0	0	0
700 - 650	50-100	2	5	15
650 - 450	100-300	2	6	18
450 - 350	300-400	1	2	6

Source: WSP 2024

Through discussion and agreement by all parties, the base case scenario inflows illustrated in Table 16.11 were used in the dewatering system design: 5 L/s at 700-650 masl, 6 L/s at 650-450 masl, and 2 L/s at 450-350 masl.

16.16.1.2 *Paste Backfill Contained Water Release and System Flushing*

The amount of water originating from paste backfill was estimated for paste having a 178 mm (7") slump. Most of the water contained in the paste backfill is consumed in the cement hydration process, and it was estimated the excess water released into the Mine dewatering system would be 4 L/s. Flushing water was estimated to be 2 L/s, based on past projects.

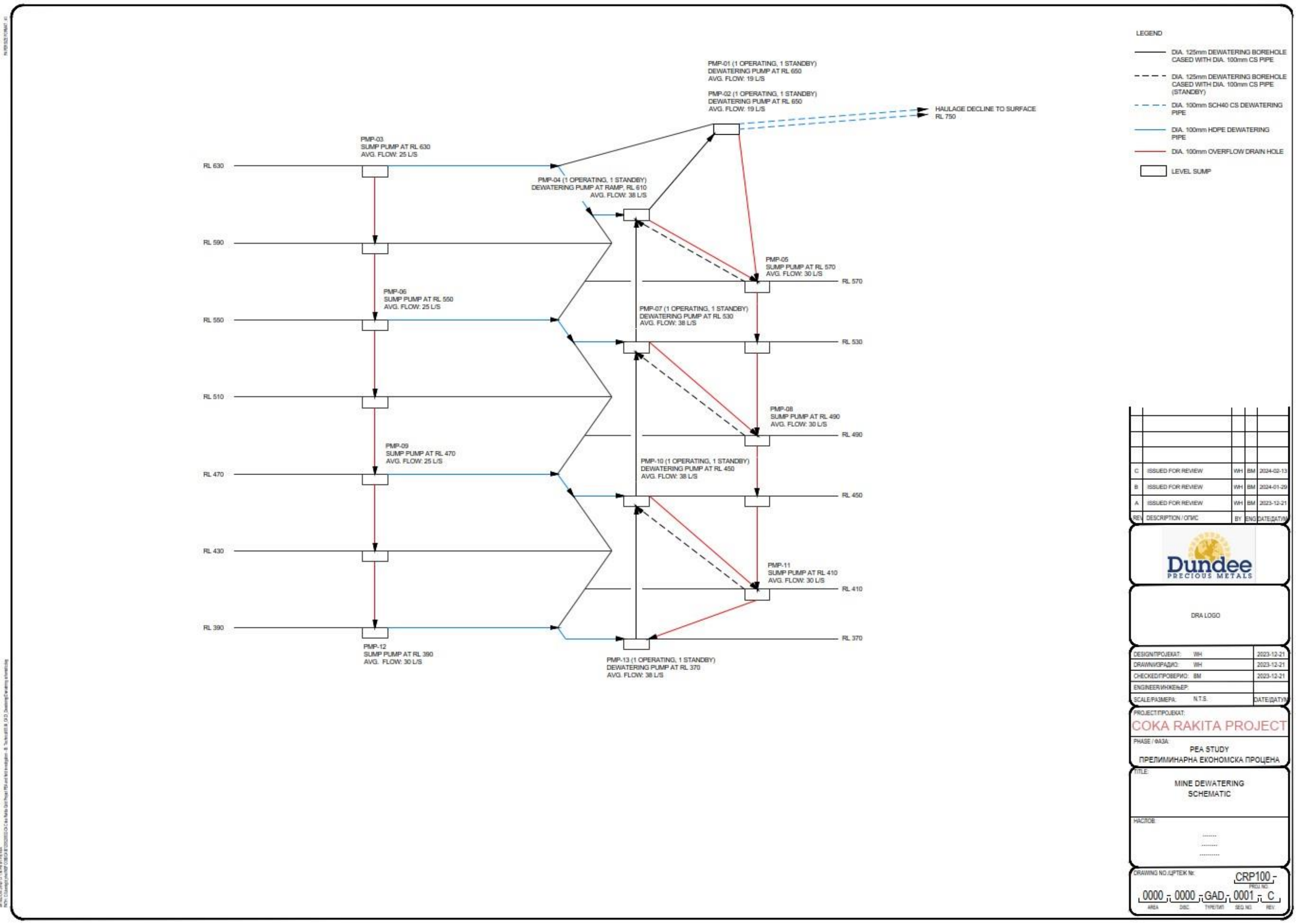
16.16.1.3 *Water Required to Conduct Mining Operations*

The water reporting to the mine dewatering system from mining operations was estimated to be 18 L/s. Based on the design inputs, the total estimated quantity of water reporting the dewatering system was 37 L/s, which was the basis for the design of dewatering system components, including dewatering pumps, boreholes, and pipelines.

16.16.2 KEY DESIGN COMPONENTS OF THE MINE DEWATERING SYSTEM

Figure 16.14 illustrates the dewatering system design, which includes the level sumps, drain holes, pipelines, and pump locations on the various levels of the mine.

Figure 16.14 – Mine Dewatering Schematic



Source: WSP 2024

Table 16.12 summarises the design locations of sumps and pump specifications required at each location within the dewatering system.

Table 16.12 – Mine Dewatering System Location of Sumps and Pump Specifications

Location	Pump #	Unit	Flowrate (L/s)	Power (kW)	Pipeline Diameter (mm (inches))	Total Dynamic Head (m)	Number of Standby Pumps
RL 650	PMP-01	Main pump	19	75	100 (4)	160	1
RL 650	PMP-02	Main pump	19	75	100 (4)	160	1
RL 630	PMP-03	Sump pump	25	19	100 (4)	33	0
RL 610	PMP-04	Main pump	38	56	100 (4)	63	1
RL 590	-	Level sump	-	-	100 (4)	-	-
RL 570	PMP-05	Sump pump	30	37	100 (4)	56	0
RL 550	PMP-06	Sump pump	25	19	100 (4)	37	0
RL 530	PMP-07	Main pump	38	93	100 (4)	113	1
RL 530	-	Level sump	-	-	100 (4)	-	-
RL 510	-	Level sump	-	-	100 (4)	-	-
RL 490	PMP-08	Sump pump	30	37	100 (4)	53	0
RL 470	PMP-09	Sump pump	25	15	100 (4)	30	0
RL 450	PMP-10	Main pump	38	93	100 (4)	107	1
RL 450	-	Level sump	-	-	100 (4)	-	-
RL 430	-	Level sump	-	-	100 (4)	-	-
RL 410	PMP-11	Sump pump	30	37	100 (4)	54	0
RL 390	PMP-12	Sump pump	30	30	100 (4)	49	0
RL 370	PMP-13	Main pump	38	93	100 (4)	107	1

1: "-" indicates not applicable i.e., drain hole used in lieu of a pump
2: Main pumps are centrifugal pumps and Sump pumps are submersible pumps

Source: WSP 2024

The current design incorporates a single dirty water settling sump design (cuddy) instead of a dirty/clean water sump design used at some other mining operations. Therefore, the system can be considered a "dirty water" pumping system whose pump designs will be further refined in subsequent phases of the Project.

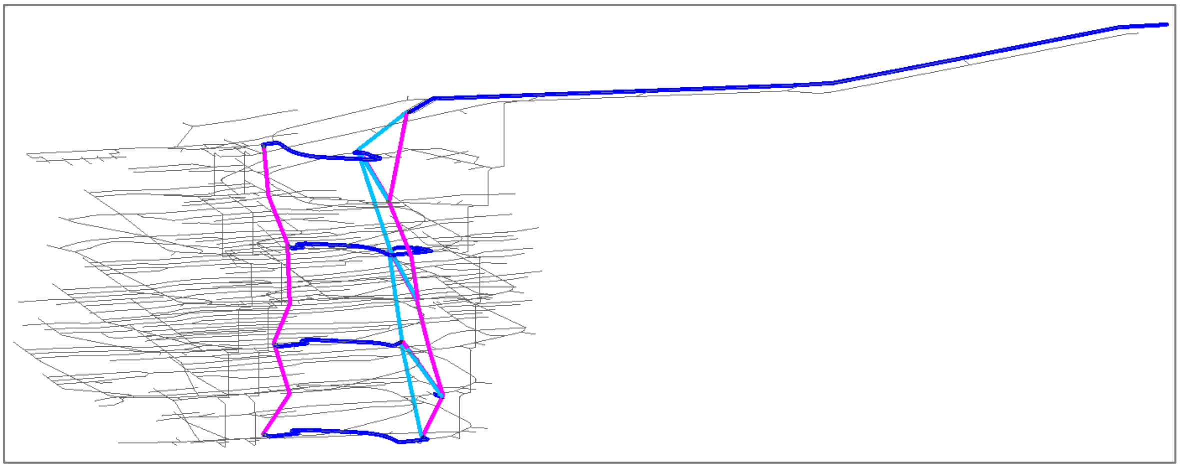
The intent of the current PEA design is to locate the sumps and associated lengths of pipelines and boreholes and to establish the type, size, and power requirements for the pumps within the system.

The design incorporates the following:

- The main mine dewatering sumps, PMP-01 and PMP-02, collect all water generated in the mine sumps and is located at the top of the deposit / bottom of the twin declines.
- Per legislative requirements, the main dewatering pumps at the main mine dewatering sump consist of a duty and standby pumping group, and within each group, there are operating and standby centrifugal pumps.
- There are two (2) 100 mm Schedule 40 carbon steel pipelines fitted with Victaulic couplings originating at the main dewatering sump at the top of the deposit and running up the decline to the portal, one pipeline operating and one standby.
- The main mine dewatering sump receives water from the PMP-04 sump, which receives water from the PMP-07 sump, which in turn receives water from the PMP-10 sump, which in turn receives water from the PMP-13 sump located on the lowest level of the mine. The pump and sump arrangement mentioned here form the “backbone” of the dewatering system. The design incorporates a single sump arrangement at each location and includes a drain hole in the event of a sump overflowing. All the pumps at these locations operate at the same flow rate and dynamic head.
- Boreholes associated with the system are not “twinned”.
- All boreholes connected to pumps are 125 mm in diameter and contain a 100 mm CS casing to mitigate poor host rock conditions.
- All level sumps contain a borehole that will direct water to the sump on the level below it, should the sump begin to overflow due to a pump failure or an inundation of water.
- Level pipelines are 100 mm HDPE.
- Submersible pumps are installed in the level sumps, while centrifugal pumps are installed at all other locations.

Figure 16.15 presents an overview of entire mine dewatering system.

Figure 16.15 – Mine Dewatering System Illustrated on a Longitudinal Section



Legend: Dark Blue – Dewatering pipeline; Light Blue – Dewatering drain hole; Cyan – Dewatering borehole
Source: WSP, 2024

17 RECOVERY METHODS

The process plant design is derived from data and design criteria provided by DPM, DRA, testwork, vendor data and regulatory/permitting requirements. The testwork results are provided in Section 13 of the Report and constitute the basis of process design as defined in the process design criteria, process mass balance and process area flowsheet.

The crushing and grinding design is based on the peak (design) throughput requirements and measured material competency and hardness. The sizing of grinding mills was conducted using peak throughput and nominal material hardness. Usually, a combination of nominal throughput and peak hardness (75th percentile of hardness) is used. However, the three (3) composites tested yielded very similar hardness indicators which would not have resulted in sufficient design margin.

The design of the flotation circuit is based upon the testwork results, DRA's experience, and calculations. These provided the basis for recovery and grade calculations and residence times.

Concentrate and tailings products are dewatered using conventional plate-and-frame pressure filtration. The filtration design is based on pressure filtration testwork and common design practices.

17.1 Trade-Off Studies

Three (3) process development trade-off studies (ToSs) were conducted to evaluate options for gravity concentrate treatment, comminution and flotation. These ToSs were carried out simultaneously with the development of the PEA.

For gravity concentrate treatment the initial assumption was to sell primary concentrate in its current state to a third party after dewatering and bagging. This assumption was maintained throughout the PEA, although a more detailed ToS will be performed in the next phase of the Project to reassess other options for gravity concentrate beneficiation.

The base case flowsheet for the comminution ToS consisted of an open circuit SAG mill, followed by a vertical stirred mill and hydrocyclones. The ToS indicated that the SAG mill from Ada Tepe would adequately meet the necessary throughput requirements. Consequently, the single-stage SAG (SSAG) mill alternative was selected for cost analysis purposes when finalising the PEA.

The flotation unit operation was based on a flowsheet that included conventional tank cells, with a rougher stage followed by two (2) cleaner stages. This selection was made primarily due to testwork completed for conventional flotation cells at the start of the PEA. However, following a preliminary ToS, it was determined that Jameson cells would be the optimal choice, which were subsequently selected for the purposes of cost evaluation. More work is required to confirm the selection.

Details of each ToS analysis are summarised below.

17.1.1 Gravity Concentrate Treatment Options

This section presents an overview of the gravity concentrate treatment options that were considered during the ToS. To explore these options a literature survey was conducted that compiled a list of possible treatments, including:

- **Option 1**: Intensive cyanidation or other lixiviant followed by electrowinning and smelting.
- **Option 2**: Secondary upgrading of the Knelson concentrate using a Mozley table or equivalent.
- **Option 3**: Direct smelting of secondary gravity concentrate to produce doré bars.
- **Option 4**: Amalgamation using mercury.
- **Option 5**: Acid digestion of entire concentrate using aqua regia followed by solvent extraction and electrowinning then smelting.
- **Option 6**: Destruction of pyrite using acid followed by roasting and smelting.
- **Option 7**: Destruction of sulphides using POX, BIOX or Albion (ultrafine grinding followed by atmospheric oxidation) followed by smelting to produce doré bars.
- **Option 8**: Selling the primary concentrate without any further upgrading.

Option 8 was the base case assumption for the PEA. Options 3 to 7 were rejected by DPM mainly due to their associated environmental risks. During the next phase of the Project only the selected three options (1, 2, and 8) will be explored in more detail.

17.1.2 Comminution Options

The process design criteria currently considers a target product size P_{80} of 53 µm for the flotation feed stream. Initial testwork indicates that the material is of average competency. For this scenario, the following flowsheet options have been considered:

- **Option 1**: SAG and Ball milling (SAB) – no pebble crush;
- **Option 2**: Single stage SAG Mill (SSAG);
- **Option 3**: SAG mill and vertical stirred mill(s) (SAV) (new); and
- **Option 4**: Refurbishment of a used SAG mill and two (2) Vertimills from existing DPM mine.

For all options, pebbles will be recycled back to the SAG mill feed chute, but a pebble crusher has been omitted because the material is not competent in nature.

For all four (4) flowsheet options, the grinding section is preceded by a jaw crusher which reduces ROM material to 80% passing 120 mm. The comminution section is designed to treat 127 tph fresh feed and to produce a product with 80% passing 53 µm. The material is of average hardness with a Bond Ball Mill Work Index of 13 kWh/t and an Axb value of 54.2.

For the SAB configuration the pinion power drawn by the SAG and Ball mills was estimated at 882 kW and 1,550 kW respectively. A standard size SAG mill, as offered by Metso, of 5.5 m diameter by 2.44 m long (EGL) was selected as the SAG Mill for this option. The ball mill is also a standard 4.3 m diameter by 6.1 m EGL ball mill.

For Option 2, the single stage SAG mill need to draw 3072 kW compared to 2560 kW for the SAB circuit. In this instance, a 6.7 m diameter by 4.26 m EGL SAG mill was selected to provide all the grinding energy required to produce the 53 µm product. It is noted that the mill dimensions are identical to the SAG mill that DPM currently owns.

For Option 3, the ball mill of Option 1 is replaced by two parallel vertical stirred mills. The flowsheet remains identical to that of Option 1 but for the splitting of cyclone underflow to two parallel vertimills as opposed to a single ball mill. For this scenario, the SAG mill is identical to that of the SAB configuration (Option 1). It was assumed that the grinding energy required for the two vertical stirred mills are 20% lower than that of the comparable ball mill.

Option 4 considers the refurbishment of three (3) mills from another DPM mine which is nearing the end of its life. The potentially available SAG mill is a 22 ft diameter by 14 ft length mill driven by a 3 MW motor. Two (2) VTM1250-WB Vertimills as well as a Sandvik CH420 pebble crusher will also potentially become available. The SAG mill as identical in size to the single stage SAG mill identified in Option 2, consequently this option has been re-evaluated as a SSAG application.

17.1.2.1 *Capital Expenditure Comparison – Comminution Options*

Prices of two sets of mills which include trommels and VFDs were obtained from recent quotations from Metso and New Concept Projects. The cost of the mills in this ToS was estimated by comparison of its internal volume to that of the mills in the DRA database. Direct as well as indirect costs were estimated by applying established factors to the main equipment costs.

The summary for the Capex options is detailed in Table 17.1, listing all costs in US$ unless otherwise specified.

Table 17.1 – Installed Capex Summary of Comminution Options

Option 1 SAB	Option 2 SSAG	Option 3 SAV	Option 4 Refurbished SAG
22,119,060	16,399,427	34,163,455	11,143,417

Note: All figures are in US$.
Source: DRA, 2024

Based on the analysis, refurbishing the mill from Ada Tepe is estimated to result in cost savings of approximately $5 M compared to purchasing a new mill, with the savings being solely attributed to capital costs. However, if the cost of refurbishment exceeds the 25% allowance, procuring a new mill may become more economical.

17.1.2.2 *Opex Comparison – Comminution Options*

Table 17.2 – Opex Summary ($/a) of Comminution Options

Option 1 SAB	Option 2 SSAG	Option 3 SAV	Option 4 Refurbished SSAG
3,413,708	3,916,888	3,763,200	3,916,069

Source: DRA, 2024

a. Net Present Costs

The Discounted Cash Flow (DCF) technique excluding revenue was used to estimate a Net Present Cost (NPC). The NPC over the LOM period of ten (10) years includes the Capex, which was considered to be executed in Year 0, and the Opex, starting in Year 1. Based on DRA's in-house experience for similar studies, a discount or hurdle rate of 8% was used.

The refurbished SSAG (Option 4) projects to be the least expensive at an NPC of $34.6 M. The new SSAG (Option 2) is next at $39.5 M. The high upfront cost of the stirred vertical mills renders Option 3 by far the most expensive of the four options at an NPC of $55 M. Option 1 is close to the two leading options as its lower Opex almost offsets its higher Capex over the ten year LOM.

The refurbished SSAG (Option 4) is recommended.

17.1.3 Flotation Options

This Čoka Rakita PEA flowsheet originally considered using conventional tank cell flotation technology for both the rougher and cleaner bank duties.

The base case flotation circuit flowsheet consists of bulk sulphide rougher flotation, two (2) stages of cleaning and a cleaner scavenger stage after the first cleaner. Rougher tails and cleaner scavenger tails both report to a tailings thickener. Cleaner scavenger concentrate and second cleaner tails are recycled back to the first cleaner stage without regrinding.

DRA assessed the following flotation options:

- Conventional Tankcell flotation (Base case);
- Staged Flotation Reactors (SFRs); and
- Jameson cells.

Jameson cells are pneumatic flotation cells similar to column cells. SFRs are a hybrid where mixing and flotation occur in different cells. SFR cells currently used on another DPM Plant will become available for use in the Čoka Rakita plant and would require refurbishment only.

The conventional tankcell design features five 100 m^3 cells as roughers, five 10 m^3 cells in the first cleaner bank and two 10 m^3 cells in the second cleaner bank.

The SFR design requires eight rougher cells, three first cleaner cells, six cleaner-scavenger cells and six second cleaner cells. It is noted that the Ada Tepe plant does have eight rougher cells which are sufficient for Čoka Rakita but four additional new cells would need to be bought for the cleaners and cleaner scavenger banks.

The Jameson cell design requires three cells only. The first E3432/8 unit would be used as a rougher scalper while a similar cell would be used as rougher scavenger. A single E1714/2 cell would suffice as a cleaner.

17.1.3.1 Efficiency Comparison

It is expected that the SFR and Jameson cell circuits will yield better recoveries and concentrate grades when compared to conventional tank cell technology due to their superior recovery performance within the finer size ranges (<53 µm). However, the testwork conducted to quantify the expected benefits yielded inconclusive results due to the presence of coarse gold affecting the overall recoveries significantly.

For the purposes of this ToS, recoveries were assumed to be similar regardless of the flotation technology used. Additional testwork is required to accurately quantify the benefit (if any).

17.1.3.2 Capital Expenditure Comparison for Flotation Trade-off Study Options

The flotation cells prices were based on quotations received from suppliers and previous ToSs performed by DRA.

Due to higher complexity of the SFR technology the flotation cell installation labour cost is higher when compared to conventional and Jameson cell technologies. An additional 10% was added to the installation labour estimate for the SFR cells only. Similar adjustments were made to account for footprint and complexity issues based on DRA's experience.

Based on Capex comparison (shown in Table 17.3), the overall installed Capex for the Jameson cell option is similar to conventional tankcells, but the SFR option is notably more expensive.

Table 17.3 – Capex Comparison for Flotation Cell Options

Conventional	Jameson	SFR
6,283,208	**6,226,052**	**8,695,854**

Note: All figures are in US$.
Source: DRA, 2024

17.1.3.3 Opex Basis

Based on the Opex comparison shown Table 17.4, the operating cost for the SFR plant is slightly lower than for the conventional and Jameson options

Table 17.4 – Opex Comparison for Flotation Cell Options

Unit	Conventional	Jameson	SFR
US$/a	**2,433,907**	**2,147,710**	**2,070,076**
US$/t	**2.86**	**2.53**	**2.44**

Source: DRA, 2024

17.1.3.4 Net Present Cost and Net Present Value

NPCs were calculated for all three options. The NPC for the Jameson cells plant is estimated to be $2.05 M less than that of a circuit consisting of conventional cells. The NPC for a circuit consisting of the mix of refurbished and new SFR cells would be $0.3 M less than the conventional cells circuit.

Jameson cells are recommended and were adopted for the costing of the PEA study.

17.2 Gold Production Schedule

A gold production schedule was established based on the latest available mine plan and assumptions derived from testwork conducted to date. In terms of inputs the following apply:

- Testwork indicated that there is a constant upgrade ratio from the gold head grade to the gravity concentrate grade. For this study a value of 150 was adopted;
- A 17% scale-up downgrade was applied to the gravity recovery. This is based on industry experience in scaling laboratory gravity recoveries up to full-scale plant recoveries;
- Based on testwork results an empirical correlation was derived for gravity recovery as a function of gold head grade, as follows:

  Gravity Recovery = 41 x Gold Grade ^ 0.14;
- Gold recovery from gravity tails to rougher concentrate was consistent at around 88%;
- Mass pull from gravity tails to rougher concentrate can be described as:

  Rougher Mass Pull (%) = 12.62 - 0.386 x Gold Grade;
- Gold recovery from rougher concentrate to second cleaner concentrate was consistent for the three locked-cycle tests at around 91%;
- Mass pull from rougher concentrate to second cleaner concentrate was also somewhat constant at around 45%;
- For the first year of operation (Year 2), recoveries to gravity concentrate and rougher concentrate are expected to be about 20% below par due to operator inexperience and optimisation / commissioning issues.

With the above assumptions a production profile was determined and is presented in Table 17.5.

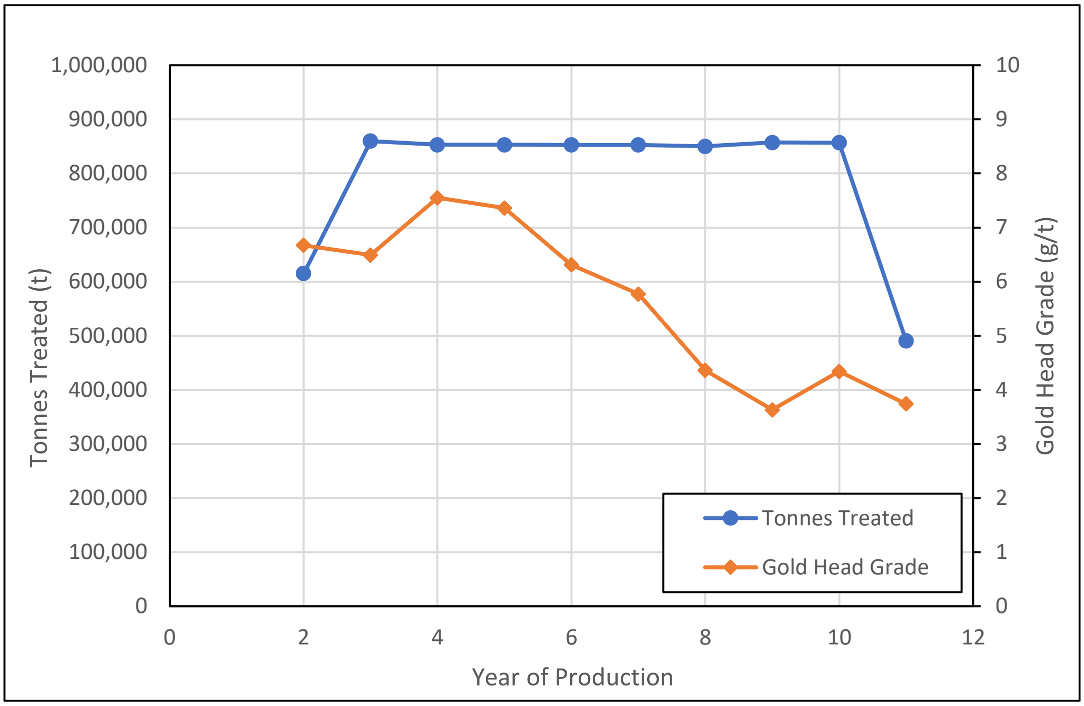
Table 17.5 – Gold Production Profile

Description	Unit	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11
Tonnes Treated	t	615,184	859,766	853,003	853,044	852,748	852,853	850,077	857,087	856,862	490,701
ROM Au Grade	g/t	6.67	6.49	7.55	7.36	6.31	5.77	4.36	3.63	4.34	3.74
Gravity Conc Au Distribution	%	44.4	44.2	45.2	45.0	44.0	43.5	41.8	40.8	41.8	40.9
Gravity Concentrate Grade	g/t	1000.6	973.5	1132.5	1104	946.5	865.5	654	544.5	651	561
Gravity Au Content	oz	58,560	79,322	93,511	90,837	76,192	68,813	49,834	40,773	49,969	24,151
Flotation Conc Au Distribution	%	35.6	44.7	43.9	44.0	44.8	45.3	46.6	47.4	46.6	47.3
Flotation Conc Grade	g/t	52.7	63.9	76.1	73.9	61.9	56.0	41.4	34.2	41.2	35.3
Flotation Au Content	oz	47,006	80,141	90,927	88,903	77,523	71,592	55,517	47,452	55,730	27,910
Overall Au Recovery	%	80.0	88.9	89.1	89.0	88.9	88.7	88.4	88.2	88.4	88.2
Overall Au Produced	oz	105,566	159,462	184,438	179,741	153,714	140,404	105,351	88,225	105,699	52,061

Note: Totals may not add due to rounding.
Source: DRA, 2024

Figure 17.1 shows the ROM feed profiles over the LOM. It was assumed that all material mined in Year 1 would be stockpiled and treatment would commence in Year 2 only.

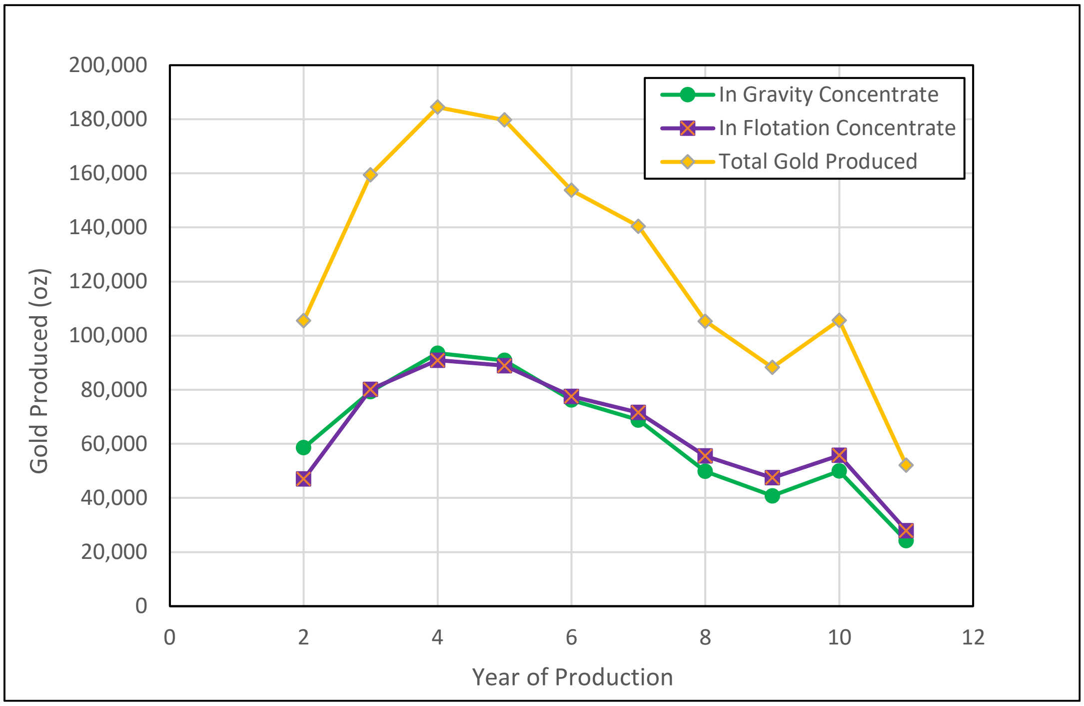
Figure 17.1 – Plant Feed Tonnage and Gold Grade Profiles



Source: DRA, 2024

Figure 17.2 shows the ounces of gold produced in total and the contributions by each of the concentrates produced. It is evident that the gravity and the flotation steps are expected to contribute approximately the same amount of gold to the total. For the higher grade (Years 3 and 4), the gravity concentrate will contain more gold than the flotation concentrate while this will be reversed towards the final years of operation due to the declining head grades.

Figure 17.2 – Gold Production Profiles



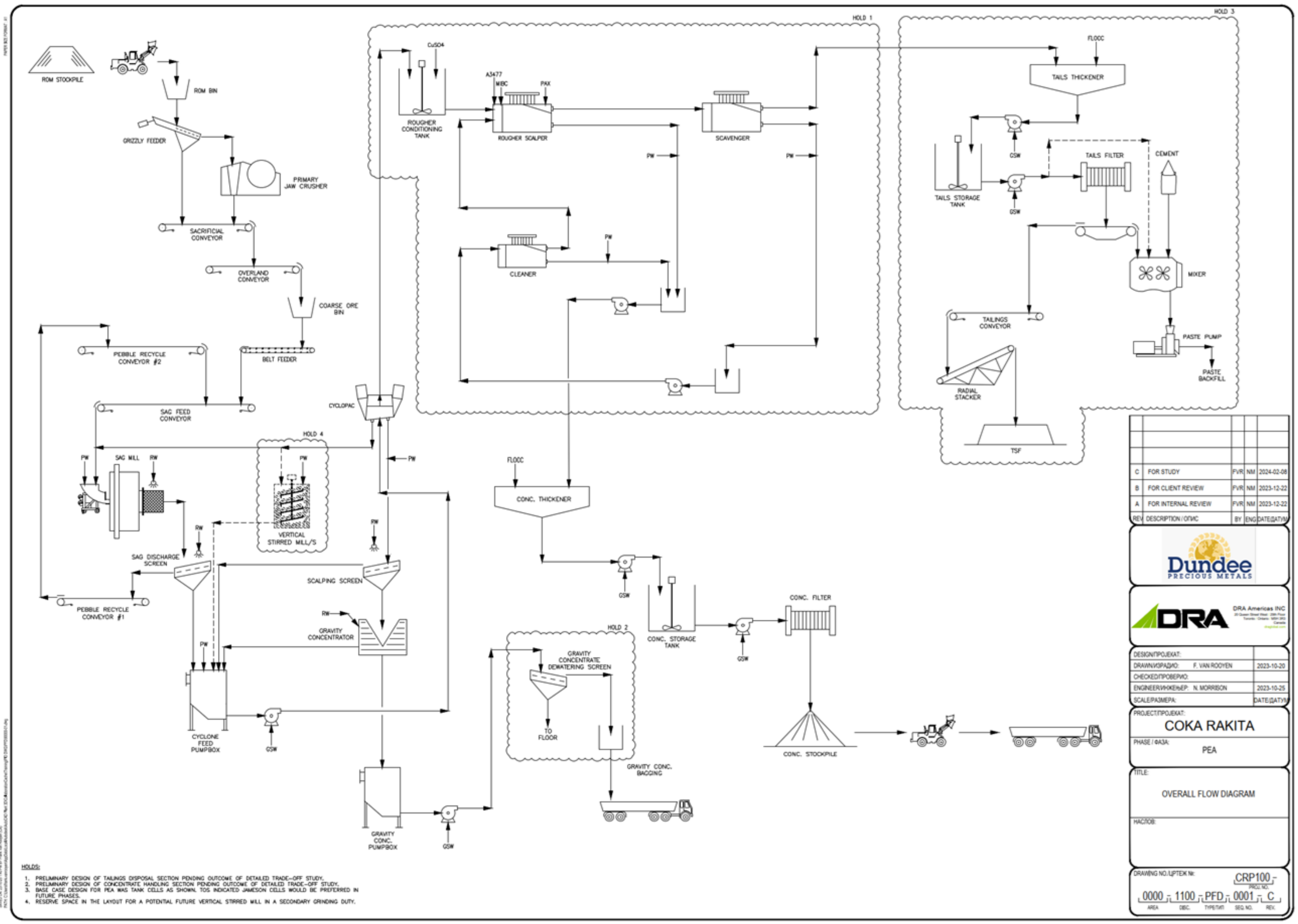
Source: DRA, 2024

17.3 Process Flowsheet

The Čoka Rakita deposit has high gold-grade, gold-copper skarn type mineralisation roughly 250 m below surface. Based on its grades, copper is not expected to contribute significantly to the revenue stream. Preliminary metallurgical test results indicate that the mineralisation is amenable to flotation and gravity concentration and has the potential to produce clean gold concentrates with overall gold recoveries around 90%. The proposed process plant consists of surface facilities including: crushing, grinding, gravity concentration, Jameson flotation cells, thickening, filtration and a paste backfill sections. The proposed process plant will be producing a gravity concentrate and a bulk flotation concentrate. Tailings will be disposed of underground as paste backfill or above ground as filtered cake in a dedicated Tailings Storage Facility (TSF).

Figure 17.3 illustrates the simplified flowsheet for the Project.

Figure 17.3 – Simplified Process Flowsheet



Source: DRA, 2024

17.4 Process Design Criteria

Table 17.6 summarises the Process Design Criteria (PDC) to nominally process 850,000 dry tpa of ROM material from the mine.

Preliminary metallurgical test results indicate that the mineralisation is amenable to flotation and has the potential to produce a clean gold concentrate with overall gold recoveries around 90%. The proposed process plant consists of surface facilities including: jaw crusher, SAG mill, gravity concentrators, flotation cells, thickeners, filtration, and paste backfill.

Table 17.6 – Process Design Criteria Summary

Description	Unit	Nominal	Design
Average ROM Throughput	dry tph	97	116
ROM Throughput-Plant	dry tpa	850,000	1,020,000
Feed Grades			
ROM Material Head Grade	Au g/t	5.68	10.0
Material Hardness			
Bond Ball Mill Work Index (BWi)	kWh/t	13.3	13.4
Abrasion Index (Ai)	g	0.138	0.146
Operating Schedule			
LOM	years	8.3	12
Crushing Plant Availability	%	75	75
Milling Plant Availability	%	92	92
Crushing			
Feed Rate	dry tph	129.4	155.3
Required Power	kW	51	56
Feed Size F_{80}	Mm	500	
Product Size P_{80}	Mm	92	
Primary Grinding			
Feed Rate	dry tph	105	127
Installed Power	kW	3000	3000
Mill Type	-	Single-Stage SAG Mill	
Mill Dimensions	-	6.71 m Diam × 4.26 m EGL	
Grinding Cyclone Cluster			
Target Product Size P_{80}	µm	53	

Description	Unit	Nominal	Design
Cyclone Diameter	mm	381	
Gravity Concentration			
Scalping Screens Dimensions		1.83 m L × 0.92 m	
Slurry Flow to Screens (combined)	m^3/h	160	
# of Parallel Gravity Circuits	#	2	
Slurry Flow to Concentrators	$m^3/h/unit$	83	
Flotation Design: Jameson Cells			
Rougher Scalper Model	-	E3432/8	
Rougher Scalper - Fresh Feed Slurry Flow	m^3/h	367	
Rougher Scavenger Model	-	E3432/8	
Rougher Scavenger - Fresh Feed Slurry Flow	m^3/h	366	
Cleaner Model	-	E1714/2	
Cleaner - Fresh Feed Slurry Flow	m^3/h	49	
Gravity Concentrate Handling			
Slurry Flow Rate to Screen	m^3/h	0.56	1.1
Flotation Concentrate Handling			
Thickener Fresh Feed Slurry Flow	m^3/h	20.8	25.0
Thickener Underflow	m^3/h	3.8	4.5
# of Flotation Concentrate Storage Tanks	-	1	
Concentrate Storage Tank - Storage Time	h	24	
Concentrate Storage Tank – Live Volume	m^3	91.4	109.0
Tailings Handling			
Tailings Thickener Fresh Feed Flow	m^3/h	324.4	393.5
Tailings Thickener Underflow	m^3/h	101.6	123.2
# of Tailings Filter Feed Storage Tank	-	1	
Tailings Filter Feed Storage Tank-Live Volume	m^3	48	
Number of Filter Presses	-	2	
Reagent Storage / Mixing			
Days of On-Site Reagent Storage	day	30	
Flocculant Consumption Rate (tailings)	g/t tailings	50	60
Flocculant Consumption Rate (concentrate)	g/t conc	15	25
Copper Sulphate Consumption	g/t ROM	250	250
Cement Addition Rate	%	5	5

Description	Unit	Nominal	Design
Potassium Amyl Xanthate (PAX)	g/t ROM	165	206
Methyl Isobutyl Carbinol (MIBC)	g/t ROM	70	87.5
A3477 Collector	g/t ROM	200	250

Source: DRA, 2024

17.5 Process Description

Run-of-mine (ROM) material delivered to the crushing section is assumed to have a moisture content of 5% and an indicated average gold grade of 5.68 g/t. The mined material will be stored on a 3-day capacity ROM stockpile in the vicinity of the primary crusher. The ROM material, with a top size of 500 mm, will be hauled with a front-end loader from the stockpile and dumped into a ROM bin with a live volume of 47 m³. It will then pass through a Vibrating Grizzly Feeder with an aperture size of 64 mm. The grizzly oversize will pass through the crusher while the undersize will report directly to the crusher discharge conveyor. A 580 mm x 930 mm primary jaw crusher will crush the feeder oversize from the top size of 500 mm to a P_{80} of 92 mm.

The crusher discharge conveyor transfers the grizzly undersize and the jaw crusher product to the coarse material bin. The primary jaw crushing section is expected to operate at a utilisation of 75%. The coarse material bin has a live volume of 949 m³, equivalent to 12 hours of SAG mill fresh feed. One belt feeder will deliver the crushed material to the SAG mill feed conveyor.

The grinding operation consists of a primary grinding circuit only featuring a single SAG mill, which will grind the material to the final product size of 80% passing 53 µm. The SAG mill dimensions are 6.71 m inside diameter with an equivalent grinding length of 4.26 m and is driven by a 3 MW motor. A vibrating screen is used to scalp off pebbles, which will be recycled back to the SAG feed conveyor. The circuit is closed-out by a cyclone cluster consisting of nine 15” cyclones (5 operating under nominal conditions). A portion of the cyclone’s underflow will be diverted to feed two parallel gravity concentration circuits.

The gravity concentration circuit consists of two parallel lines each starting with a scalping screen followed by a centrifugal gravity concentrator. Spray water will dilute the Cyclopac underflow in the launder, to improve flowability, before it reports to the scalping screen feed box prior to passing through the scalping screen. Additional water sprayed onto the screen will assist with screening. The screen panels will have an aperture size of 3 mm ×10 mm. Screen oversize will return to the grinding circuit.

Each scalping screen underflow feeds its dedicated centrifugal gravity concentrator. The expected cycle times of the concentrators will be nine minutes each, but this can be adjusted in operations to reduce the volume of concentrate produced. These gravity concentrators will separate heavy gold containing particles from the associated gangue minerals. The dense particles that have accumulated in the concentrator will be flushed at the end of a cycle from the bowl to a concentrate pump box and then pumped to the gravity concentrate bagging facility. A dewatering screen will be used to remove the bulk of the water while the hanging bags will be used to drain excess surface moisture. The final product moisture content is expected to be approximately 18 to 20 % w/w. Gravity tailings will flow, by gravity, from the centrifugal concentrator to the mill discharge pump box.

Bulk sulphide flotation will be used to generate a sulphide-rich concentrate. The flotation plant consists of three Jameson cells (a rougher scalper, rougher scavenger and a cleaner). Flotation concentrate and tails streams will be pumped to a dedicated concentrate thickener and a dedicated tails thickener (respectively) prior to filtration.

Cyclopac overflow with 26.8% solids will flow, by gravity, to the rougher conditioning tank with a volume of 39 m³ where it will be conditioned for five minutes prior to discharging to the rougher Jameson cell feed pumpbox. The rougher scalper cell will be a model E3432/8 (rectangular shaped 8 downcomers). Rougher scalper flotation concentrate will be transferred via a launder to the concentrate thickener, while the rougher scalper tails will flow by gravity to the rougher scavenger cell feed pumpbox. The scavenger cell will also be a model E3432/8 and it will ensure that the overall recovery of the circuit is maximised by recovering any valuables that may have escaped the scalper. Tailings from the scalper will be the final circuit tailings reporting to the tailings thickener. Flocculant is added to the thickener feed at a dose of 25 g/t to increase the rate of settling.

Rougher scavenger flotation concentrate will be cleaned in a single E1714/2 Jameson cell. Cleaner concentrate will join scalper concentrate enroute to the 5 m diameter concentrate thickener where excess water will be removed prior to pumping thickener underflow to the agitated storage tank. The surge/storage tank provides a process break ahead of a plate-and-frame concentrate filter.

Scavenger cleaner tails will be recycled back to the rougher conditioning tank.

The expected moisture content of the filter cake product will be 12% and it will be stockpiled on a 7-day capacity stockpile. A front-end loader will load filtered concentrate from this stockpile into a concentrate transport truck.

Tailings from the rougher flotation section will be dewatered in a tailings thickener with an approximate diameter of 14 m. Flocculant will be added to the thickener feed at a dose rate of 25 g/t to increase the rate of settling. Thickener underflow, at 55% w/w solids, will be pumped to the mechanically agitated tailings storage tank which provides three times the filter volume as surge capacity. Stored concentrate will be pumped to the horizontal pressure filter, which dewaters it to a cake with an average moisture content of 14% w/w. Over the life-of-mine about 60% of this cake will be conveyed to the TSF, while the remaining 40% will be transported to the paste backfill plant.

The paste backfill plant will operate when there is a demand for paste. The paste backfill plant receives 40% of the dry filter cakes and some slurry from the tailings feed tank. A mixer will mix the filtered cake and slurry to a predetermined solids density together with binder (cement). This mixture will be pumped into the paste backfill distribution tank.

17.6 Reagents and Consumables

The following reagents are used throughout the process plant:

- Flocculant – gold concentrate and tailings thickeners;
- Copper sulphate – flotation reagent;
- Potassium amyl xanthate (PAX) – flotation collector;
- Cement – for paste plant;
- Methyl isobutyl carbinol (MIBC) – flotation frother; and
- A3477 – flotation collector.

Reagent mixing will be completed in a designated area within the plant. The design of this area includes features such as bunding, with dedicated sump pumps. The layout and general arrangement of the reagent area accounts for the need to prevent contact of incompatible reagent types. Separate onsite long-term reagent supply storage is provided at a safe distance away from the process plant.

Reagents are usually made up with fresh water where possible – for Čoka Rakita fresh water may be replaced by treated water obtained from the effluent treatment plant.

Reagents are dosed using positive displacement pumps which allow for accurate dosing at a controlled rate to the process.

A summary of the reagents and consumables usage can be found in Table 17.7.

Table 17.7 – Reagents and Consumables

Description		Consumption	
	Unit	Nominal	Design
Reagents			
Flocculant – Concentrate Thickener	g/t concentrate	15	25
Flocculant – Tailings Thickener	g/t tailings	50	60
Collector (A3447)	g/t ROM	200	250
Collector (PAX)	g/t ROM	165	206
Frother (MIBC)	g/t ROM	70	87.5
Grinding Steel			
SAG Mill Media	kg/t ROM	0.65	0.67
SAG Mill Liners	kg/t ROM	0.078	0.080

Source: DRA 2024

17.7 Utilities and Services

17.7.1 Water Services

Raw / fresh water will be harvested from surface run-off and stored in the non-contact water pond. Downstream users of raw water include fire hydrant, reagent mixing and gland water. Given the low moisture content of the products leaving the plant; it is expected that the facility will have a positive water balance i.e., excess water will need to be treated and discharged to the environment. A portion of the treated water will be pumped to the raw / freshwater system as make-up to minimise discharge to the environment. It is expected that treated water will constitute the bulk of the make-up water supplied to the raw / freshwater system.

17.7.2 Air Services

All plant air compressors are housed in the open, within acoustic enclosures. Low pressure blower air will provide air to the flotation section. Higher pressure air will also be required for the filters and as instrument air.

17.7.3 Electrical Services

Electricity will be sourced from the local grid via overhead power lines. The total installed power for the site is estimated at 7.3 MW.

17.8 Process Control

The Project will have an adequate level of automation and semi-remote-control facilities. Instrumentation will be provided within the plant to measure and control key process parameters and to provide a safe work environment.

All key process and maintenance parameters will be available for trending and alarming on the process control system (PCS). The PCS will control the process interlocks and PID control loops for non-packaged equipment. Control loop set-point changes for non-packaged equipment will be made at the operator interface terminals (OIT).

The control of most of the vendor packages should be remote from PCS OIT and Ethernet connection to control system network. General equipment fault alarms from each vendor package will be monitored by the PCS and displayed on the OIT. Fault diagnostics and troubleshooting of vendor packages will be performed locally.

A historian will be provided for logging data and changes made to the system. This will have the ability to perform trending of all measured data as well as system variables.

18 PROJECT INFRASTRUCTURE

18.1 General

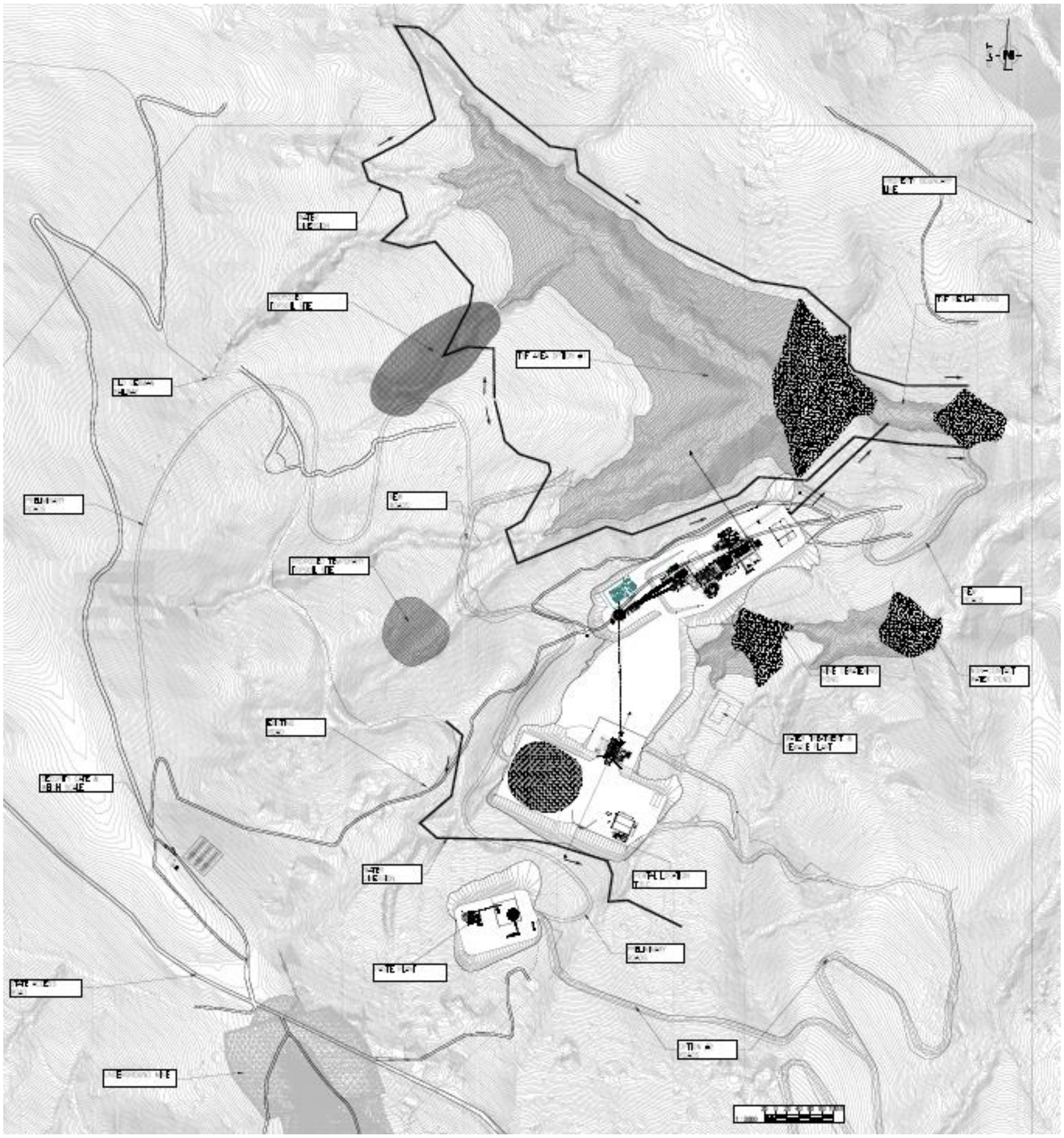
This section describes the infrastructure, building and other facilities such as access roads and power supply, that are required for the operation of the Project. It consists of the following infrastructure, which are all located on-site:

- Roads and terraces;
- Mine and process plant supporting infrastructure;
- Mine waste rock management;
- TSF;
- Water management;
- Power supply and distribution;
- Automation and communication; and
- Services and utilities.

No geotechnical investigations have been conducted.

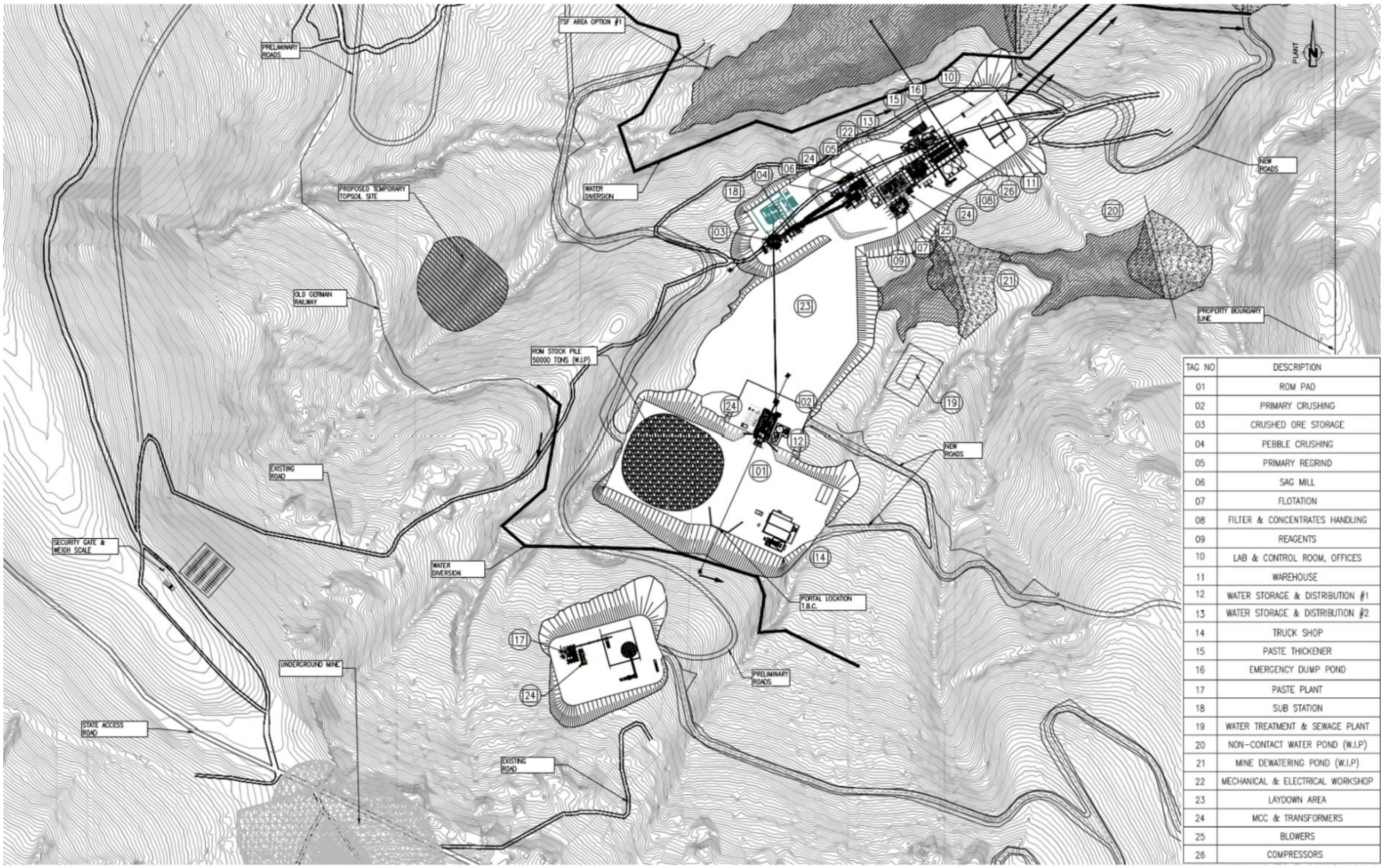
The proposed overall site layout and access is shown in Figure 18.1, and Figure 18.2 illustrates the proposed details of the plant layout.

Figure 18.1 – Proposed General Site Layout



Source: DRA, 2024

Figure 18.2 – Proposed Process Plant Layout



TAG NO	DESCRIPTION
01	ROM PAD
02	PRIMARY CRUSHING
03	CRUSHED ORE STORAGE
04	PEBBLE CRUSHING
05	PRIMARY REGRIND
06	SAG MILL
07	FLOTATION
08	FILTER & CONCENTRATES HANDLING
09	REAGENTS
10	LAB & CONTROL ROOM, OFFICES
11	WAREHOUSE
12	WATER STORAGE & DISTRIBUTION #1
13	WATER STORAGE & DISTRIBUTION #2
14	TRUCK SHOP
15	PASTE THICKENER
16	EMERGENCY DUMP POND
17	PASTE PLANT
18	SUB STATION
19	WATER TREATMENT & SEWAGE PLANT
20	NON-CONTACT WATER POND (W.I.P)
21	MINE DEWATERING POND (W.I.P)
22	MECHANICAL & ELECTRICAL WORKSHOP
23	LAYDOWN AREA
24	MCC & TRANSFORMERS
25	BLOWERS
26	COMPRESSORS

Source: DRA, 2024

18.2 Main Access Road

The Project site will be accessed via a new road coming from an existing public road 105 to the administration area near the process facility. The road will be oriented east-west and will provide gravel access with an estimated width of 8 m. Alternate routes will be reviewed during the next phase of the Project when geotechnical work is completed. Table 18.1 indicates estimated lengths of roads for this Report.

Table 18.1 – Estimated Length of Roads

Item	Approx. Length (m)
Access To Process Plant from Public Road	1,500
Paste Backfill Access from Process Plant	710
Process Plant Terrace to Crusher Terrace	470
Access To Tailings TSF from Process Plant Terrace	570

The main access road will terminate at the guard house ahead of the administration building. All vehicles and personnel will be inspected prior to entering the Project site.

18.3 Site Roads

A series of roads will be constructed and give access to various areas surrounding the Project site. Access roads will be sized for the underground mining trucks exiting the mine to the stockpile, for the mobile equipment requirements surrounding the Project site, access roads to tailings handling facilities, the paste backfill plant and other infrastructure. All site roads except for mine roads will be sized to 8 m wide.

The construction of the new roads consists of the following:

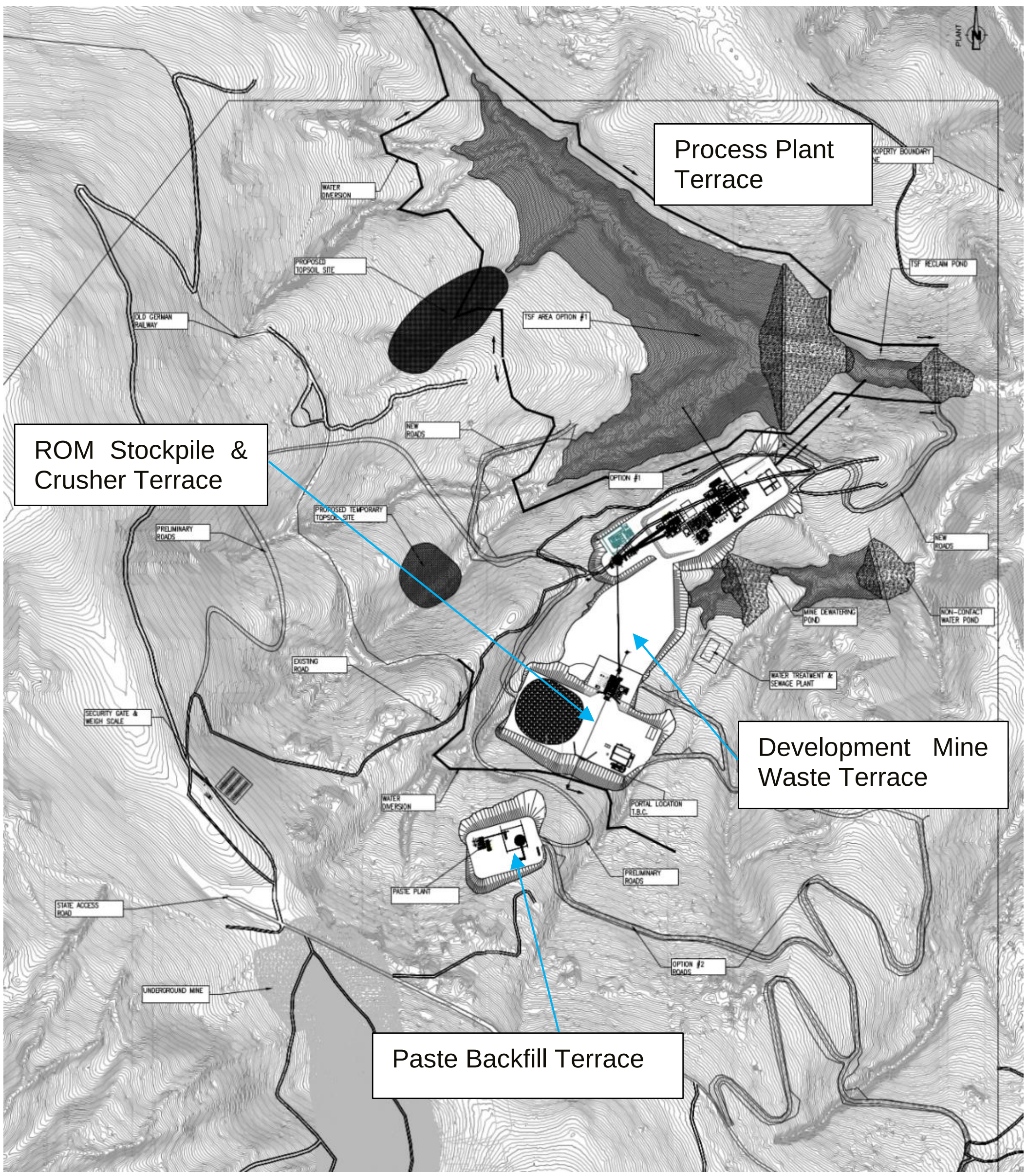
- Access to Process Plant.
- Paste Backfill Access.
- Process Plant to Crusher Access.
- ROM Pad Access.
- Access to Process Plant Option 1.
- Access to Tailings 1.
- Access to Tailings 2.

18.4 Terraces

The Project site requires terraces areas to accommodate the Project infrastructure and equipment. Terraces have been designed through an iterative process between Project element layout and suitable topography and terrain.

The terraces are designed with grading and a drainage system that will allow the runoff water to reach the collection ponds and natural drains in accordance with the site water management requirements.

Figure 18.3 – Terraces



Source: DRA, 2024

18.5 Camp Site Accommodations

No accommodation camp will be present on-site. Sufficiently established towns are located within reasonable distance and travel time from the Project site. Operational staff will be required to relocate to a local town of their choice.

18.6 Buildings

18.6.1 Process Plant Configuration

The purpose of the PEA is the development of crushing, grinding and floatation process and supporting infrastructure associated with treatments of sulphide minerals.

Various design elements were progressed to understand the layout requirements of the Project. These elements include the:

- Process facility layout;
- Potential topsoil dump locations;
- Waste rock dump locations; and
- Tailings facility placement and configuration.

18.6.2 Plant Location

The process plant site location was determined by multiple criteria, including:

- Proximity to the TSF;
- Proximity to mine portal;
- Proximity to land ownership boundaries; and
- Condemnation drilling.

Available topographical areas that will require a practical amount of earthworks for the establishment of plant terraces and connecting of roads and minimise haulage.

Due to the mountainous terrain’s limitation and condemnation drilling, two (2) plant location options were reviewed as shown in Figure 18.4. Option 1 was chosen as the study's plant location closest to the TSF due to the favourable practical access and reasonable terrace earthwork provided by the terrain. Table 18.2 provides a qualitative summary of both options.

Table 18.2 – Site Options - Location and Terracing Advantages and Disadvantages

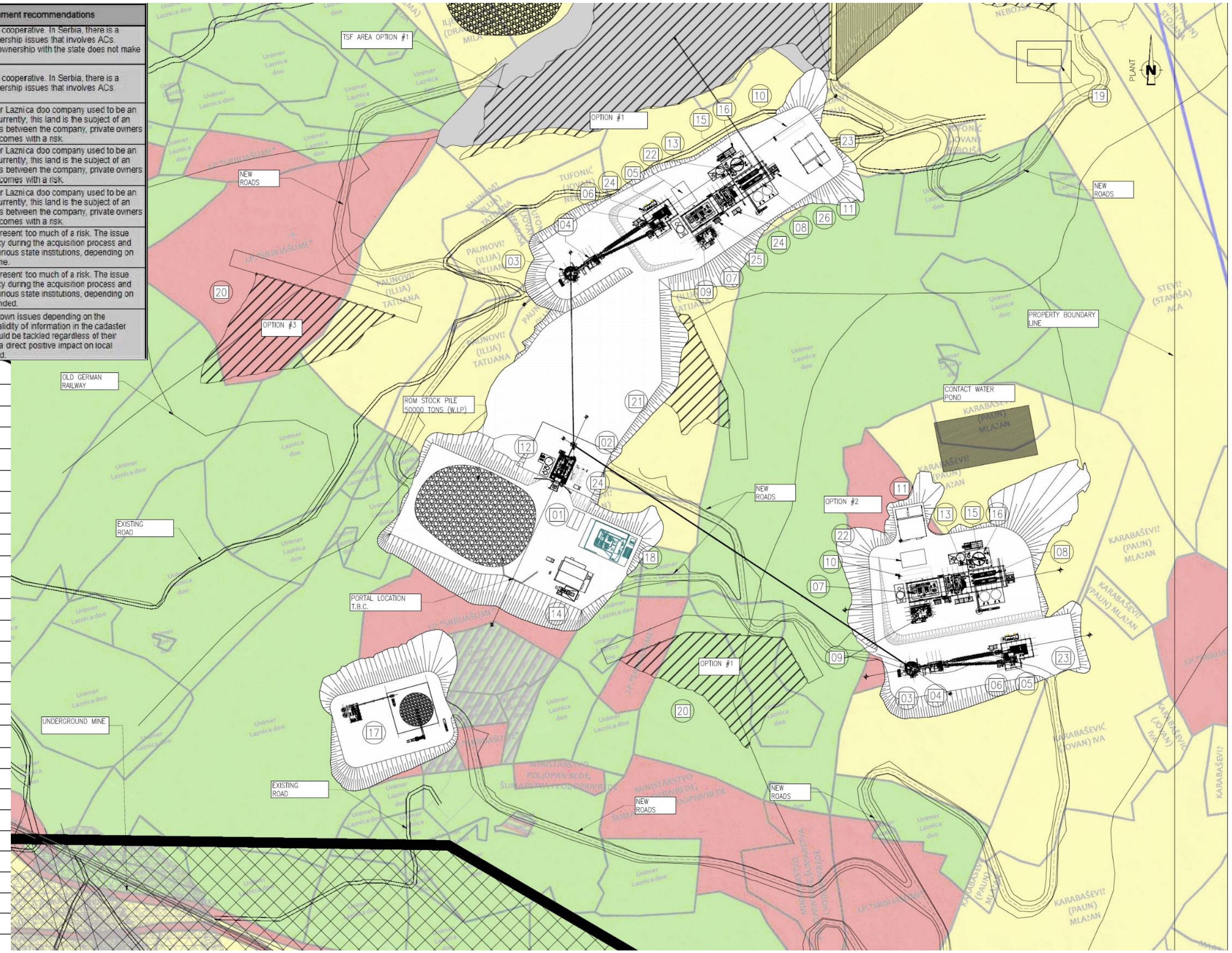
	OPTION 1	OPTION 2
Reference Drawings	CRP100-1000-1310-PLN-0001 CRP100-1000-1310-PLN-0002	CRP100-1000-1310-PLN-0003 CRP100-1000-1310-GAD-0007
Advantages	Located closer to TSF. Less traffic and noise to surrounding area. Located on flattest area found during site visit. Less earthworks than Option 2. Located in lower risk permitting region which is preferred.	Terrace and road access can be made through lower risk land ownership and permitting areas which is preferable.
Disadvantages	Road Access through more condensed land ownership permit areas which may require a longer acquisition process.	Located further from TFS. Pumping and/or filtered tails trucking requires more energy, roadworks creating traffic and noise in surrounding areas. Slightly larger terracing requirements for the process plant facility. A review was made to observe a portal location closer to Option 2. The result was higher decline costs and challenges on feed conveyor to plant coarse material bin. The position of Option 2 from the portal is further and requires more energy to deliver ROM to process plant coarse material stockpile.
General	DRA has reviewed the mine portal location chosen by DPM. DRA recommends further review of ROM area terracing to optimise portal location, and to determine the best location for waste and ROM feed stockpile. Once geotechnical information becomes available to assess topsoil and rock depths, additional work can be carried out to determine the best terracing strategy and estimate associated costs.	

Figure 18.4 – Option1 for Site Location

Landowners	Facility placement recommendations
Zemljoradinicka zadruga Homoljac +State (1)	Homoljac is an agriculture cooperative. In Serbia, there is a history of problematic ownership issues that involves ACs. Avoidable if possible. Co-ownership with the state does not make it easier.
Zemljoradinicka zadruga Homoljac (9)	Homoljac is an agriculture cooperative. In Serbia, there is a history of problematic ownership issues that involves ACs. Avoidable if possible.
Unimer Laznica doo + State (2)	The land owned by Unimer Laznica doo company used to be an agriculture cooperative. Currently, this land is the subject of an ongoing long court process between the company, private owners and the state. Acquisition comes with a risk.
Unimer Laznica doo + private owner (5)	The land owned by Unimer Laznica doo company used to be an agriculture cooperative. Currently, this land is the subject of an ongoing long court process between the company, private owners and the state. Acquisition comes with a risk.
Unimer Laznica doo (200)	The land owned by Unimer Laznica doo company used to be an agriculture cooperative. Currently, this land is the subject of an ongoing long court process between the company, private owners and the state. Acquisition comes with a risk.
State + private parcels (2)	State parcels should not present too much of a risk. The issue can be the state's efficiency during the acquisition process and possible involvement of various state institutions, depending on the ownership. Can be done.
State parcels (84)	State parcels should not present too much of a risk. The issue can be the state's efficiency during the acquisition process and possible involvement of various state institutions, depending on the ownership. Recommended.
Private parcells (1004)	Private parcels have their own issues depending on the ownership situation and validity of information in the cadaster office. These can and should be tackled regardless of their complexity. Also, this has a direct positive impact on local community. Recommended.

PRODUCTION PROCESS

TAG NO	DESCRIPTION	DRAWING No:
01	ROM PAD	.
02	PRIMARY CRUSHING	.
03	CRUSHED ORE STORAGE	.
04	PEBBLE CRUSHING	.
05	PRIMARY REGRIND	.
06	SAG MILL	.
07	FLOTATION	.
08	FILTER & CONCENTRATES HANDLING	.
09	REAGENTS	.
10	LAB & CONTROL ROOM, OFFICES	.
11	WAREHOUSE, MECH &	.
12	WATER STORAGE & DISTRIBUTION #1	.
13	WATER STORAGE & DISTRIBUTION #2	.
14	TRUCK SHOP	.
15	PASTE THICKENER	.
16	EMERGENCY DUMP POND	.
17	PASTE PLANT	.
18	SUB STATION	.
19	WATER TREATMENT & SEWAGE PLANT	.
20	NON-CONTACT WATER POND (W.I.P)	.
21	MINE DEWATERING POND (W.I.P)	.
22	MECHANICAL & ELECTRICAL WORKSHOP	
23	LAYDOWN AREA	
24	MCC & TRANSFORMERS	
25	BLOWERS	
26	COMPRESSORS	



Source: DRA, 2024

18.6.3 Building Designs

All structures will be conventional type insulated structures. Concrete foundations will be designed for soil conditions following geotechnical work. Where possible, lightweight dome-type structures could be utilised if economically feasible. These domes could be used for the warehouse. This will be explored further in the PFS.

18.6.4 Process Facilities

18.6.4.1 Primary Crusher Building

The primary jaw crusher and ancillary process equipment will be housed in a conventional structure measuring 18 m by 33 m by 22 m high. The structure will comprise the crusher, vibrating grizzly feeder, compressor, lubrication units and dust collection system. The crushed material exits the crusher building to the storage facility via a conveyor.

18.6.4.2 Coarse Material Storage

The coarse material bin has a live volume of 510 m^3, equivalent to six (6) hours of SAG mill feed time. One belt feeder will deliver the crushed material to the SAG mill feed conveyor. The coarse material bin is fitted with dust extraction/suppression.

18.6.4.3 Process Areas

The process area facilities comprise the equipment and services to transform the ROM into the desired product. These facilities are conventional insulated structures with the full complement of dust collection systems (as required), HVAC systems, service cranes, process water and gland water distribution systems, fire protection and detection systems, compressed air, interconnected process and service piping systems, and power and control systems.

The paste plant consists of a mixer and a paste pump. Dry filter cake and some slurry from the filter feed tank will be mixed with cement and pumped to the paste backfill holding tank at the underground mine. More details are contained in Section 18.7 of this Report.

18.6.5 Administrative Complex

The administration building will accommodate a sufficient number of offices for administration, staff, visitors, and sub-contractors. Provision has also been made for a first aid station as well as a conference room and a lunchroom for employees.

Area parking will be provided with area lighting for safety.

18.6.6 Gate House

The gate house will be a single storey building. All access to and from the site will be monitored and an electric gate will be used to restrict access to and from the site.

18.6.7 Mine Equipment Maintenance Building

The mine equipment maintenance building is sized at 33 m by 25 m and 14 m high and is located adjacent to the mine portal. The building will comprise maintenance bays for repair of equipment, a wash bay to clean the equipment, and mine offices, mine dry, compressor room, workshop area and storage for sensitive equipment spares. The wash bay will be equipped with a pressure washing system and an oil/water separator would be included.

18.6.8 Warehouse

A lightweight warehouse, measuring 20 m by 30 m and 13 m high, is provided for storage of mechanical equipment parts. The end walls will each have a truck door as well as a man door. Racking will be provided for storage of small equipment and parts and sufficient storage will be provided for major items. All sensitive equipment and parts will be stored in the concentrator.

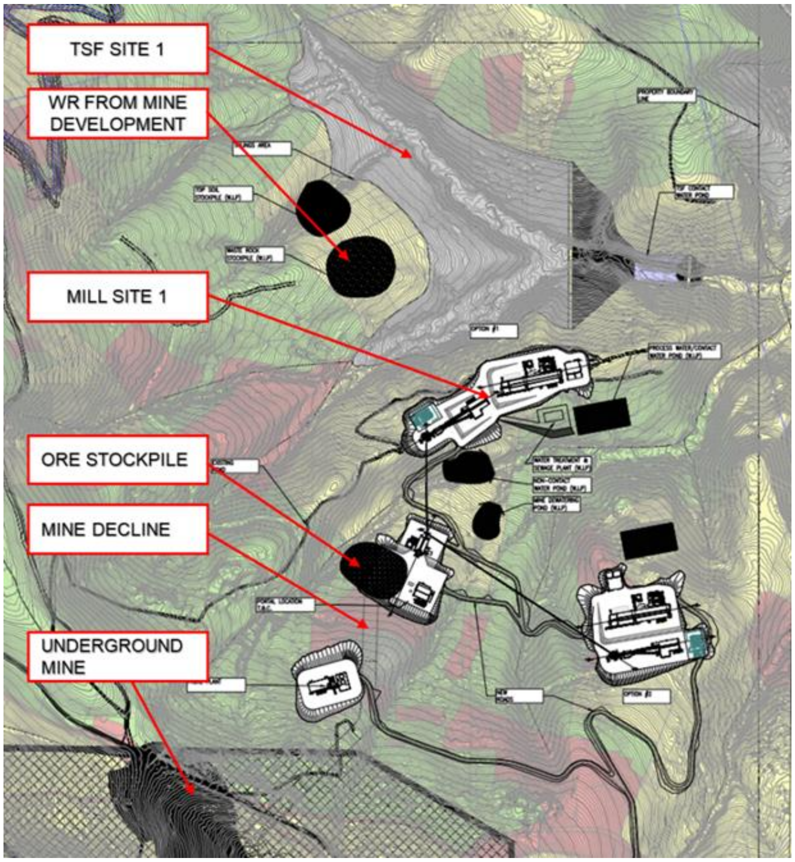
18.7 Paste Backfill System

18.7.1 Introduction

RMS and SLR were retained to assess tailings surface disposal strategies and the underground paste backfill system at a conceptual level of design for the Project.

The recent mine design relies on the mine decline and ramp positioned on the eastern side of the body of mineralised material, along with the ventilation system. Long hole open stoping was selected as a mining method that will be supported by backfill. Mineralised material is planned to be hauled from underground to surface and stockpiled prior to comminution and flotation processes. The gravity separation and flotation process were envisioned to separate the bulk concentrate and tailings stream. The preliminary site plan was developed by DRA and is illustrated in Figure 18.5.

Figure 18.5 – Site Plan



Source: RMS, 2024

18.7.2 Design Criteria

The basic paste backfill plant design parameters are summarised in Table 18.3.

Table 18.3 – Basic Design Criteria

Description		Unit	Throughput
Operating Data (LOM – Year 9)			
Mineralised Material Production		Mtpa	0.8
Underground Voids		Mm^3/a	0.286
Voids Backfilled with Paste Backfill		Mm^3/a	0.254
Voids Backfilled with Waste Rock		Mm^3/a	0.032
Paste Backfill Paste Parameters			
Backfill Type		100% FPT	FPT/CWR[1] = 85/15
175 mm Slump	%	76	77
Consolidated Wet Density	t/m³	1.98	2.01

Description		Unit	Throughput
Consolidated Dry Density	t/m³	1.51	1.55
250 mm Slump	%	74	75
Consolidated Wet Density	t/m³	1.93	1.96
Consolidated Dry Density	t/m³	1.43	1.47
Paste Backfill Plant Parameters			
Backfill Type		100% FPT	FPT/CWR[1] = 85/15
Tailings Balance: Underground/Surface	%	41 / 59	36 / 64
Utilisation	%	37	36
Solid Tailings Throughput	tph	114	114
Crushed Waste Rock Throughput	tph	-	20
Total Dry Solid Backfill Throughput	tph	114	134
Binder at 5%	tph	6	7
Paste Backfill Transport Velocity	m/s	1.1 – 1.2	1.3 – 1.4
UDS[2] Pipe Diameter	mm	150	150

[1] CWR = Crushed Waste Rock;
[2] UDS = Underground Distribution System.

Source: RMS, 2024

According to the initial LOM plan, the underground mine would produce about 0.8 Mtpa of mineralised material and approximately 0.29 Mm^3/a of voids. However, the recently developed LOM plan shows increased production to about 0.85 Mtpa. The tailings dewatering circuit and the paste backfill plant would be designed to process the full plant tailings throughput of about 114 tph on dry solid basis, while an equipment design factor of 1.15 has been included to satisfy upset conditions.

According to the LOM plan, development waste rock and paste backfill are to be used to support underground operations. Based on the preliminary geotechnical and geochemical results, the waste rock appears to be acid generating (AG). Therefore, this waste rock was planned to be used for backfilling and stored underground. Likewise, the FPT were estimated as non-acid generating (NAG) and could be deposited on the TSF impoundment, which complies with environmental requirements. The waste rock development would be intensive during the initial few years of the LOM, which is dictated by the mine growth. As the mine development progresses towards the end of the LOM, less waste rock quantities would be generated and used for backfilling.

In accordance with the LOM plan, the paste backfill plant was envisioned to backfill the remaining underground voids that were not backfilled with the waste rock. However, due to possible logistical issues that can occur underground, the paste backfill plant would be designed to backfill 100% of the underground voids without waste rock, to maintain underground production and to meet the mining schedule. This scenario marginally increases the annual paste backfill plant utilisation to about 42%.

18.7.3 DEWATERING AND PASTE BACKFILL PLANT LOCATIONS TRADE-OFF

DRA provided the following three (3) locations (Figure 18.6) for the dewatering and paste backfill plant concurrently considering the process plant, run-of-mine stockpile and mine decline:

- Site 1 – Dewatering and paste backfill plant to be within the process plant perimeter and adjacent to the TSF;
- Site 2 – Dewatering and paste backfill plant to be within the process plant perimeter and located south-east from the TSF; and
- Site 3 – A hybrid option, where the dewatering plant would be located within the process plant perimeter at Site 1, while the mixing (paste) plant would be located at Site 3 close to underground mine. This hybrid option aims to facilitate tailings disposal to the TSF and paste backfill transport to underground mine.

Site 1

Site 1 appears to be the preferred option as it offers tailings disposal "in valley". However, installing both dewatering and paste backfill infrastructure at Site 1 and routing the UDS throughout the mine decline calls for a paste pumping circuit at the paste backfill plant and an additional underground booster station. An additional overland pipeline would be required from the paste backfill plant located at Site 1 to mine decline entrance. For these reasons, Site 1 was not deemed the preferred solution.

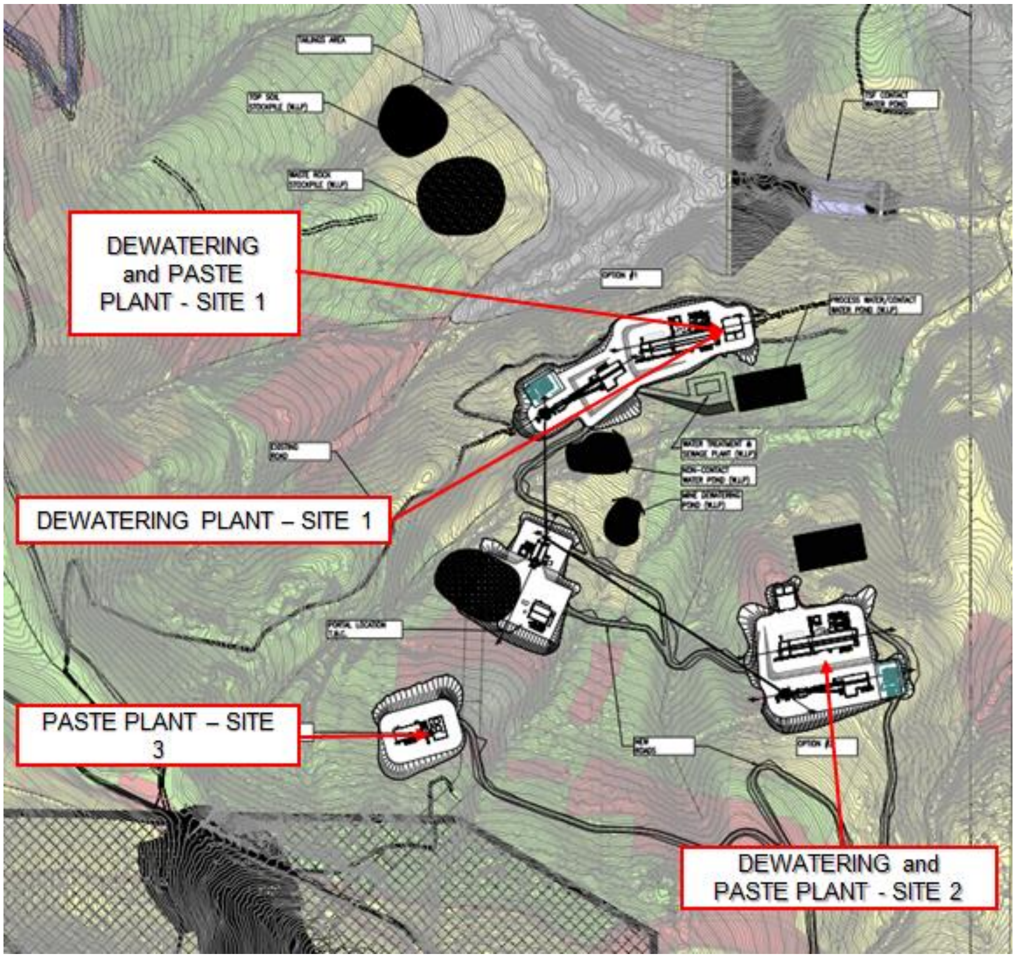
Site 2

The preliminary findings show that Site 2 would require > 1.0 km long overland conveyor to transport the filter cake from the process plant at Site 2 to the TSF. As well, the paste backfill transport from Site 2 to underground stopes would call for a paste pumping circuit at the paste backfill plant (Site 2) and an additional underground booster station. An additional overland paste backfill transferer pipeline would be required along with the entire environmental protection system. For these reasons, Site 2 was not deemed the preferred solution.

Site 3

Installing a dewatering arrangement at Site 1 and the paste backfill mixing system along with the filter cake containment at Site 3 appears to be the optimal solution. When backfill is required, the paste backfill would be transported by a single pumping station from Site 3 to the underground stopes. Likewise, the tailings could be easily disposed to the TSF impoundment from Site 1, when backfill is not required. Therefore, this hybrid solution appears to be the preferred site.

Figure 18.6 – Dewatering and Paste Backfill Plant Locations



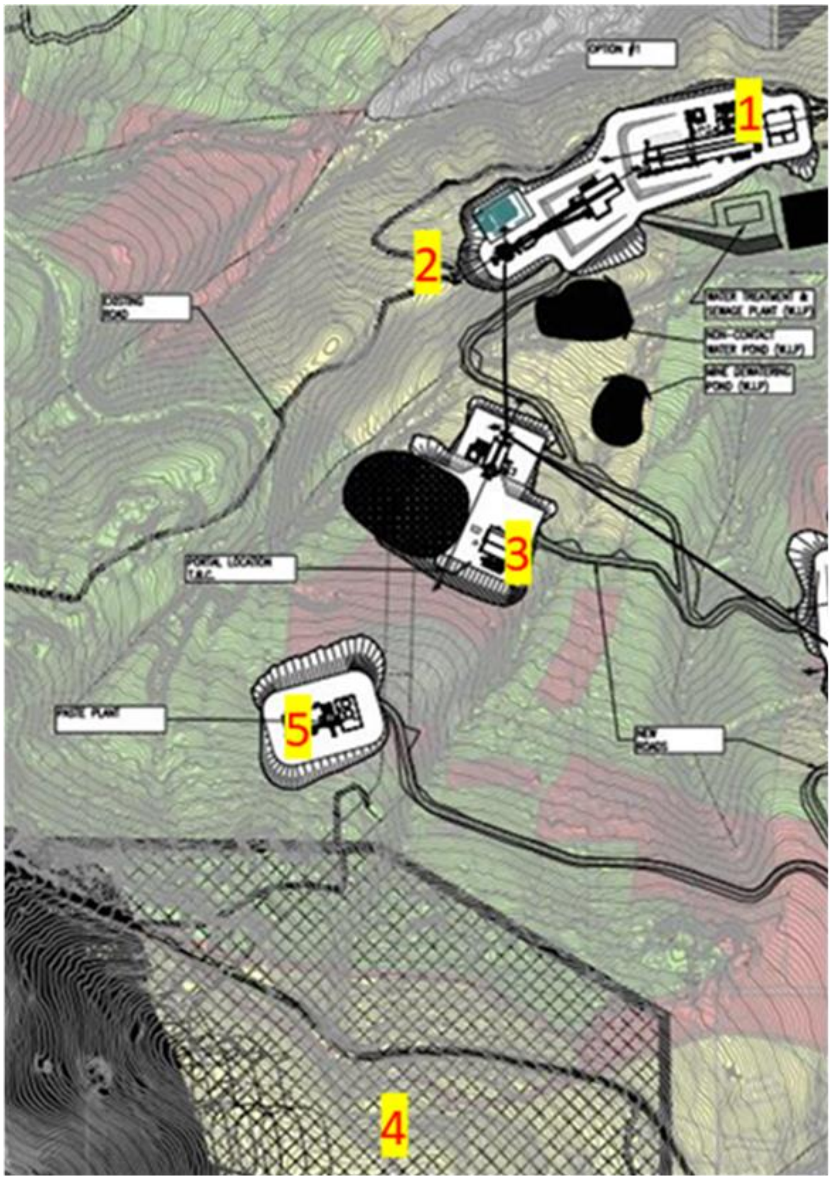
Source: RMS, 2024

18.7.4 OVERVIEW OF POTENTIAL OPTIONS

It should be noted that the initial options on the tailings disposal and paste backfill plant locations were evaluated using three (3) different options for tailings disposal: thickened tailings, paste tailings and dry stacking as well as paste backfill that can be prepared either with 100% FPT or as a blend of Full Plant Tailings / Crushed Waste Rock (FPT/CWR) in the ratio of 85/15.

In order to find the optimal location for installing dewatering (thickening and filtration) equipment and the paste backfill plant, a few more locations that are displayed in Figure 18.7 were explored.

Figure 18.7 – Expanded ToS on Dewatering and Paste Backfill Plant Locations



Source: RMS 2024

Two (2) base cases and five (5) scenarios were investigated in this expanded ToS. DPM selected Option 4a/4b (initial ToS) as the Base Case 1 and 2 that would be compared with the other scenarios. At the beginning of the PEA, RMS recommended Site 4 as the best location for placing the paste backfill plant as this would allow gravity supply of the paste backfill to underground stopes, keeping dewatering arrangement within the mill perimeter (Site 1) to facilitate filter cake disposal to the TSF impoundment. As DPM does not possess the land at the Site 4, this site was excluded from the initial assessment. However, DPM would like to explore the possibility of utilising Site 4.

RMS included an additional scenario (Scenario 5) that includes the dewatering infrastructure at Site 2 and the paste backfill plant at Site 5. It should be noted that base cases and scenarios are evaluated using dry stacking surface disposal and paste backfill that can be either 100% FPT or a blend of FPT/CWR in the ratio of 85/15.

Table 18.4 – Backfill Types

Option	Backfill Type	Paste Backfill
1	Paste Backfill	100% FPT
2	Paste Backfill Blend	FPT/CWR = 85/15
3	Co-disposal[1]: Paste Fill and Waste Rock	100% FPT and Waste Rock
4	Co-disposal: FPT/CWR and Waste Rock	FPT/CWR = 85/15 and Waste Rock

[1] Co-disposal would involve separately placing waste rock into underground stopes and pouring the paste backfill (250 mm Slump) on top to fill the voids among the waste rock.

Source: RMS, 2024

18.7.4.1 *Trade-Off Study – Quick Overview and Option Selection*

Options 1, 2, and 3 would not be suitable solutions for delivering the paste backfill to underground stopes due to terrain conditions (i.e., low overland pipeline points and uphill sections), pipeline operating and design complexity (i.e., leak protection system, containments and long-distance transport etc.) that might cause paste slaking process and lead to pipeline blockages. Additionally, due to the higher operating complexity and additional pumping requirements associated to tailings surface disposal, Options 1 and 2 were identified as unpreferred solutions. They appear to be uneconomical and unfeasible solutions due to their complexity in terms of the earthwork required for plant installation, interference with ROM stockpile and the other infrastructure. Also, placing the filtration arrangement at Site 3 (Scenario 4) requires installation of the overland tailings transfer pipeline from Site 1 to Site 3 which is associated with leak detection and protection systems and containment to prevent environmental spillages. Besides, the overland conveyor that spans from Site 3 to the TSF (Scenario 4) makes this Scenario more complex for design and operation.

The Base Case 2 is both economically and technically more viable than Base Case 1. This is due to the fact that paste backfill can be delivered to underground stopes via gravity, making it more cost-effective. Base Case 2 also allows both backfill types (100% FPT and FPT/CWR) to be delivered by gravity from the Site 4 to underground stopes. However, Base Case 2 comes with additional permitting efforts, if it is granted the necessary permits, it would be advantageous to proceed with a more detailed level of engineering.

The abovementioned options and scenario did not have qualities to be studied with more detailed level of engineering.

18.7.4.2 Options 4a and 4b (Base Case 1 and 2) – Site 1: Dry Stacking Tailings Disposal and Site 3: Paste Backfill

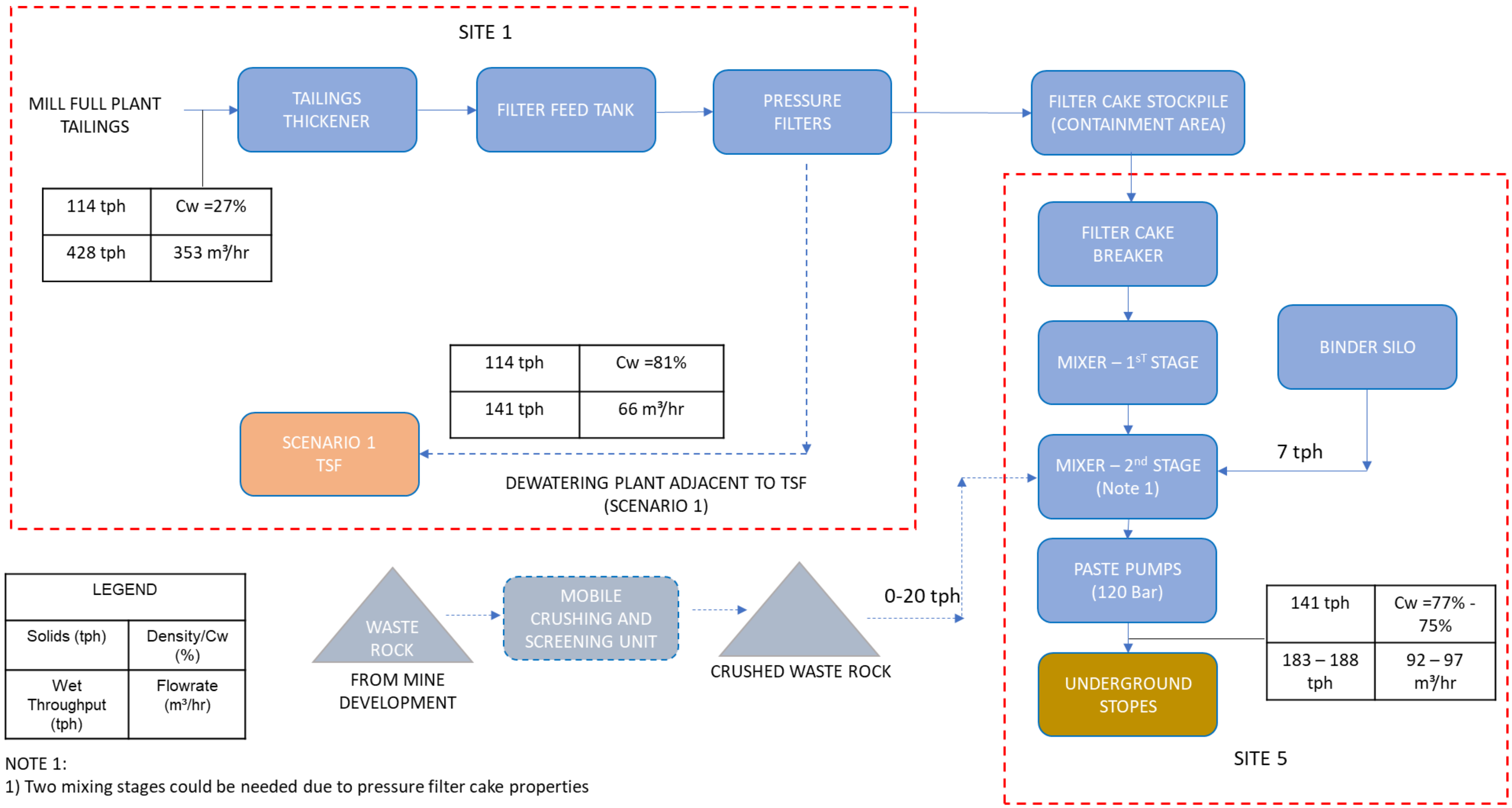
The block flow diagram and mass balance are presented below in Figure 18.8.

This option relies on the dewatering arrangement installed at Site 1 to produce a filter cake that would be conveyed and stacked on the TSF. Concurrently, the paste mixing plant would be installed at Site 3 along with the filter cake handling system and storage containment.

When backfill is required, the filter cake would be produced by a dewatering circuit and conveyed to the storage silo with a live bottom system at Site 1. Concurrently, the silo would prevent dusting allowing gravity loading the trucks with the filter cake which, in turn, reduces the operational expenditures (Opex) and CO_2 emissions. Once loaded at Site 1, the filter cake would be trucked and stored to containment area at Site 3 (Figure 18.9). The filter cake containment area is sized to provide 24h live storage capacity that offsets three (3) filter feed tanks assigned for thickened tailings storage described in Option 1. The filter cake handling, and radial stacking system is assigned to receive and to stockpile the filter cake within the containment footprint.

A single front-end-loader (FEL) would be used to load the filter cake to a receiving bin with a live bottom system that would break the filter cake chunks, facilitating the upstream mixing process. The pressure filter cake would be dropped onto the transfer conveyor that would be fitted with a scale and transferred to a mixing system and mixed with a process water and binder. The mixing system would consist of two (2) continuous twin shaft mixers that would produce paste with a targeted density. Thus prepared, the cemented paste would be conveyed by the piston pumps to underground stopes without utilising the underground booster stations.

Figure 18.8 – Block Flow Diagram and Mass Balance

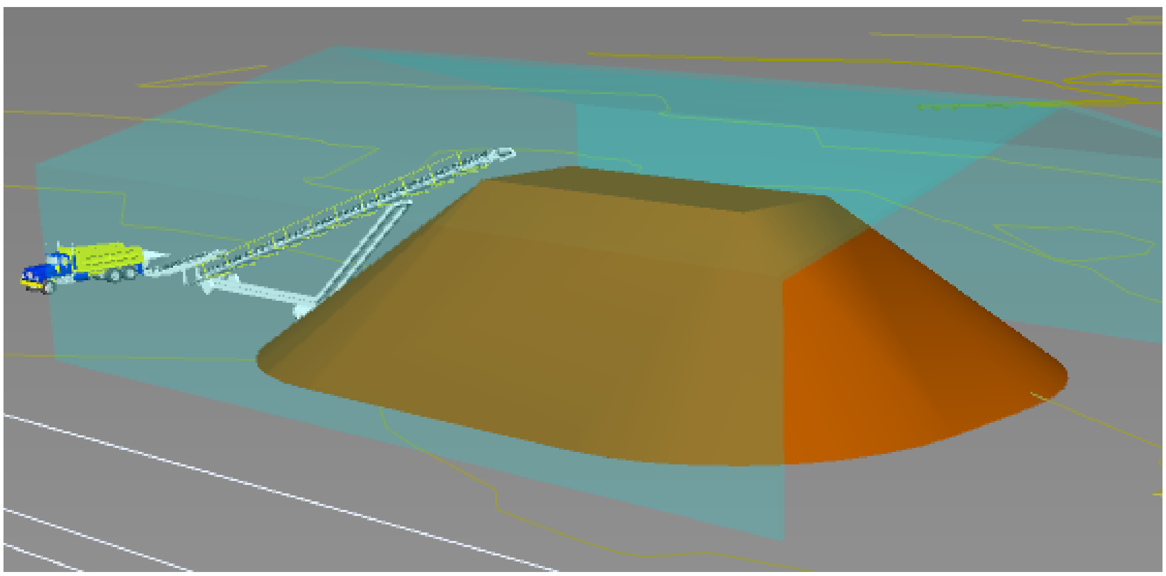


Source: RMS, 2024

In order to minimise the environmental impact and to improve the backfill strength, the crushed waste rock would be blended with the filter cake, process water and binder in the same paste backfill plant at Site 3. The mobile crushing unit would be used to crush the waste rock to a size of about 12 mm that would be suitable for mixing and transporting along the underground distribution system (UDS). In accordance with the backfill schedule, the same UDS with nominal pipe diameter of 150 mm would be utilised for conveying either the paste backfill blend FPT/CWR = 85/15 or paste backfill with 100% FPT, when required.

When backfill is not required, the filter cake would be delivered to an overland transfer conveyor and stacked on the TSF.

Figure 18.9 – Filter Cake Containment and Filter Cake Handling System



Source: RMS, 2024

18.8 Mine Waste Management

Mine waste developed through the development and operation of the mine include mine development and tailings generated through the milling operations. Both mine waste and tailings are planned to be used to backfill the mined out underground workings, with any excess stored on surface. The following will be produced over the LOM:

- 1.1 Mm^3 of mine rock; and
- 7.9 Mt of tailings.

During the initial two years of operation, all mine rock will be sent to surface and then gradually surface waste rock will taper off until Year 5. Tailings produced in the mill will be used to produce a paste and pumped underground with an estimated average utilisation of 37% of the total tailings feed.

After accounting for underground backfill, the following quantities are planned to require surface disposal:

- 0.34 Mm^3 of mine rock from the decline development; and
- 5.0 Mt of tailings (3.0 Mm^3 at estimated dry density of 1.65 t/m^3).

Mine rock sent to surface will be used as a construction fill for the terrace and tailings will be deposited in a surface TSF.

18.8.1 Geochemistry

The geochemical testing completed to date indicates that the tailings are non acid generating and the mine rock is potentially acid generating. For the purposes of this study, it is assumed that tailings are non-potentially acid generating (non-PAG) while all of the mine rock is considered potentially acid generating (PAG). It is currently understood that the onset time for acid generation of the mine rock is relatively short, in the range of two years. It is also understood that further testing is ongoing, and the results may change these assumptions.

To mitigate acid rock drainage (ARD) of the PAG mine rock, a liner system will be installed at the subgrade to prevent infiltrating groundwater. Runoff from the terrace will drain by gravity to the lined mine dewatering pond. Similarly, the TSF will have a basal liner that will prevent infiltration of contact water into the groundwater.

18.9 Tailings Storage Facility

A Tailings Storage Facility (TSF) will be constructed to store tailings produced in the mill. A portion of tailings will be used to backfill the underground workings and the remainder would be deposited in the surface TSF.

18.9.1 Tailings Management Trade-Off Study

A tailings management ToS was conducted to evaluate potential sites and tailings dewatering consistencies. Three (3) sites were proposed by DPM and a fourth was added by SLR. All of the sites were in valleys due to the location of the Project in a mountainous region of rugged topography and high relief. The valleys are V-shaped, narrow, and deep, providing little in terms of natural containment and generally requiring high dam fill to tailings ratios (ranging from 3 to 6). This makes a valley-fill type of tailings facility uneconomical in favour of a valley-sloping type of tailings facility.

The ToS evaluated each site with tailings dewatering consistencies consisting of thickened, paste, and filtered. The Project intends to backfill the underground mine with paste tailings and will therefore produce streams of thickened and filtered tailings, with paste being produced from the filtered tailings that are rewetted.

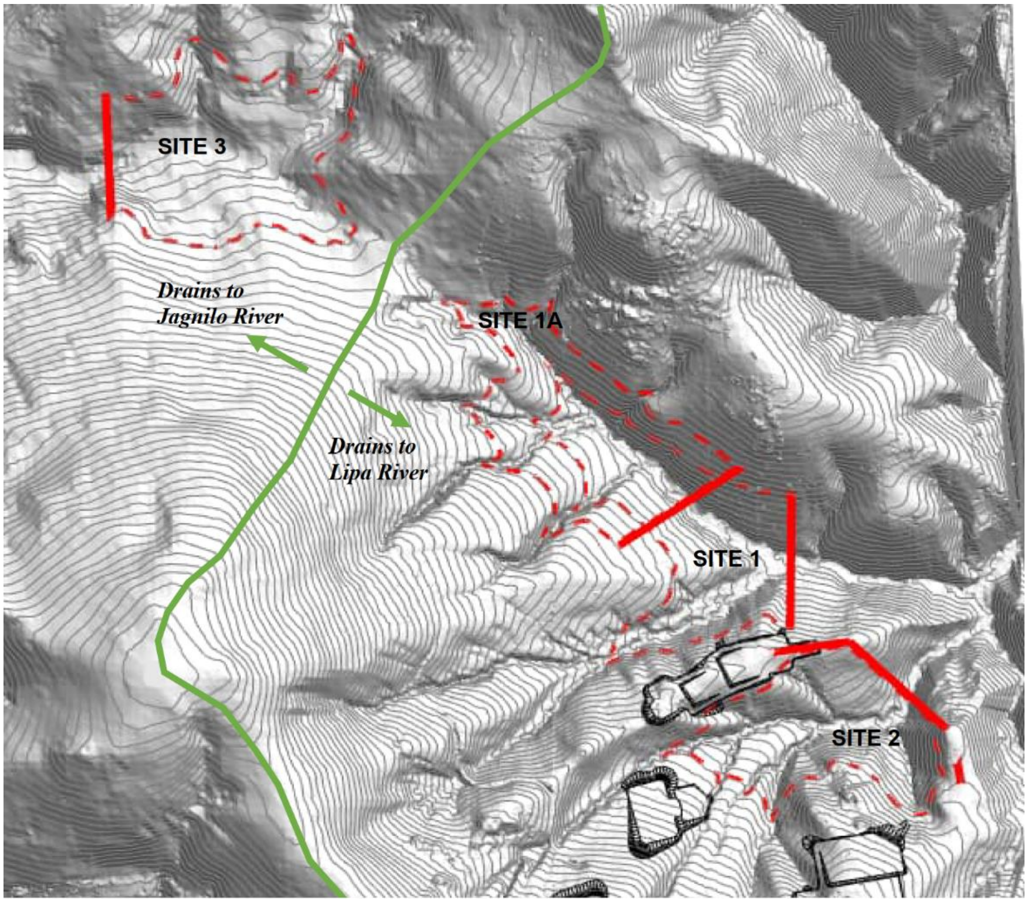
The ToS found that a filter cake TSF at Site 1 or 1A (Figure 18.10) was the most economically advantageous. Due to the poor natural containment characteristics of the sites, containment dams required for hydraulically placed thickened or paste tailings added a considerable capital cost. A filter stack facility is also amenable to co-disposing mine rock, which was identified as an objective of DPM.

The alternative selected for the PEA was the 'base case', a filter cake TSF located at Site 1. Further analysis of alternatives will be carried out during the PFS including potentially depositing mine development rock in the TSF by co-disposal to suit the paste backfill requirements.

Table 18.5 – Tailings Trade-Off Study Summary

	Filtered Tailings	Paste Tailings	Thickened Tailings
Sites evaluated	1, 1A, 3	1, 1A	1, 1A, 3
Capital cost	$29M to $40M	$35M to $38M	$35M to $48M
Operating costs	$3.05 to $3.30 per t	$1.75 per t	$0.65 per t
Flexibility / expandability	Readily expandable by stacking tailings higher Waste rock can be combined to reduce project footprint		
Operational issues	Off-spec area required Dust mitigation Long haul distance for Site 3	Maintaining tailings consistency for higher deposition slopes	More water lost and recirculated with discharged tailings Long distance pipelines for Site 3.
Environmental & social risks	Dust generated from unsaturated tailings Site 3 in separate watershed	Public perception of dam safety for large tailings dams Site 3 in separate watershed	Public perception of dam safety for large tailings dams Site 3 in separate watershed
Design & approvals risks	Lined filter stack, reported easiest to permit	Larger dams or earlier raises would be required if target beach slope not attained	Sourcing suitable materials in required quantities for large dams

Figure 18.10 – Proposed TSF Sites



Source: SLR, 2024

18.9.2 Design Criteria

Design of the TSF will be in accordance with Canadian Dam Association (CDA) guidelines. The criteria used for the TSF are based on the dam hazard potential classification (HPC) of 'High'.

The technical design criteria for the TSF are fully detailed in the TSF Design Report which was prepared for DPM during the PEA.

18.9.3 Tailings Management

Tailings will be dewatered by thickening followed by pressure filtration to produce tailings having a filter cake (filtered) consistency that will be deposited in a filter stack TSF. A filter stack TSF is well suited for this Project over alternative methods because of the following:

- Soil-like tailings consistency allows for steep deposition slopes that are ideal for the valley topography of the TSF site;
- Lack of natural containment mean hydraulically placed tailings would require significant containment dams;
- Water recovered in the mill obviates the need for large tailings ponds for mill make-up water, especially during winter months when significant amount of water lost to ice;
- The mill process rate allows for filtered tailings to be handled and deposited using minimal resources;
- The facility is readily expandable by stacking higher or expanding the footprint without significant dam construction;
- Amenable to progressive reclamation and immediate closure as the filtered tailings do not require drying prior to installation of closure cover; and
- Reduced risk of containment loss result of low water content tailings.

The main risks to successfully developing a filter cake TSF that will need to be carefully managed are:

- Off-specification tailings that are not sufficiently dewatered by filtration will require additional rework, drying, or re-processing before they can be deposited; and
- The design deposited dry density may not be achievable during the winter months and may require temporary storage until spring when the thawed tailings may be compacted.

18.9.4 TSF Conceptual Design

The TSF comprises the following design components:

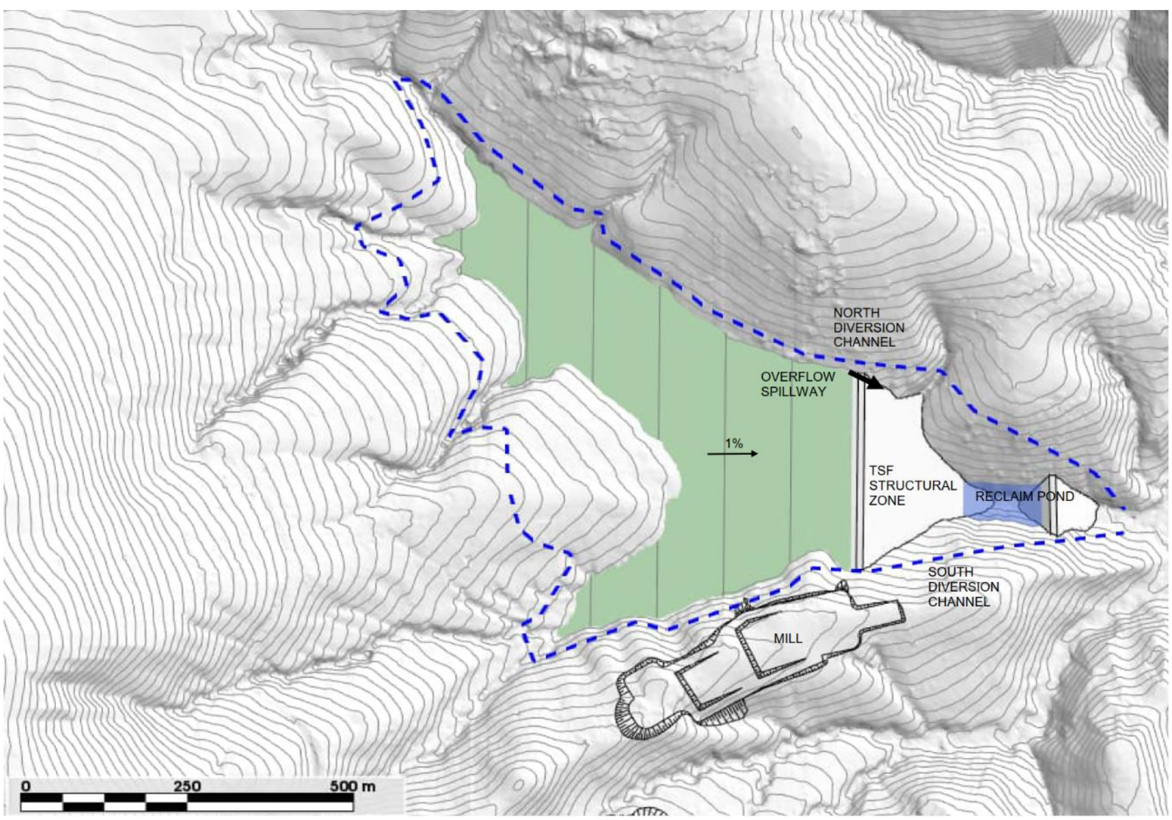
- Collection system, consisting of a composite liner system, to minimise seepage to environment;
- Structural zone to provide containment of the tailings;

- Reclaim pond to store excess contact water;
- Internal surface grading to convey contact water to the reclaim pond; and
- Diversion channels to convey fresh, or non-contact water, around the TSF.

The design and operation of a filter stack TSF is dependent on tailings to the specified consistency, i.e., filtering to near optimum moisture content to allow for placement and compaction. Additional rework of the tailings may be necessary to achieve the optimum moisture content and design dry density. Tailings samples were tested in the laboratory including compaction testing and filtration testing. The results indicate that the optimum moisture content for compaction is 12.7% with a dry density of 1,963 kg/m^3 while the filtration testing had achieved moisture contents of about 15 to 16%. Some additional drying may be expected during transport and placement, allowing for compaction to achieve >95%.

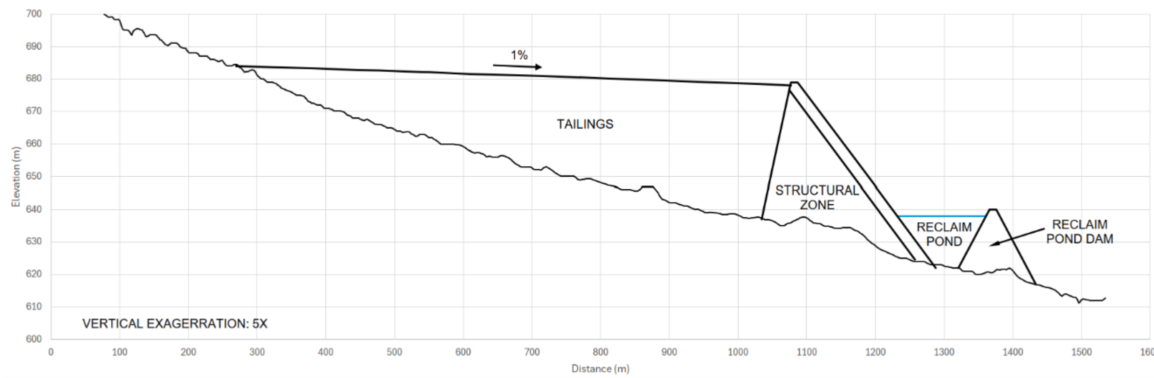
If adequate and consistent filtering cannot be achieved, the system may not work. A dedicated off-spec tailings area is required at a minimum where tailings can be temporarily stored to air dry or reprocess to attain a suitable moisture content. Tailings used to construct the structural zone will meet all specifications and be placed during ideal weather conditions.

Figure 18.11 – TSF Plan View



Source: SLR, 2024

Figure 18.12 – Cross-Section Through the TSF



Source: SLR, 2024

18.9.4.1 *Storage Capacity*

The TSF has been sized to store approximately 63% of the LOM tailings, estimated to be 5.0 Mt. The estimated volume of tailings to be disposed in the TSF is 3.0 Mm^3 with an estimated dry density of 1.65 t/m^3.

18.9.4.2 *Containment*

Containment and structural support of the TSF is provided by a structural zone consisting of compacted filter cake tailings, an outer shell of rockfill to prevent erosion from runoff and wind, and filter zones between the tailings and rockfill to prevent internal erosion. The filter and rockfill zones will be constructed as berms and raised progressively in the upstream direction, ensuring that the base of each berm is founded on rockfill or compacted (structural) tailings. The rockfill shell and filter zone has an average width of 15 m with 3.5(H):1(V) exterior side slope. The structural zone will be constructed in annual lifts during the summer/fall construction season.

18.9.4.3 *Diversions*

Freshwater diversion channels will be constructed upstream of the ultimate footprint of the TSF to reduce runoff from the upstream catchment. The diversion channels will collect runoff and drain by gravity to downstream of the TSF.

18.9.4.4 *Collection System*

A collection system consisting of a composite liner and drainage layer is provided on the base of the TSF to prevent groundwater contamination and promote porewater pressure dissipation within the tailings. The deposited tailings will be unsaturated, fine grained, and compacted, therefore minimal internal drainage may be expected. Similarly, precipitation falling on the filter stack will mostly runoff and be conveyed to the reclaim pond by surface grading and channels.

Grading of the base of the TSF will mimic the existing topography such that drainage will occur towards the centre of the valley and then downslope by gravity towards the to the Reclaim Pond. Contact water will be recirculated to the mill or pumped to the water treatment plant prior to discharge.

The collection system comprises soil bedding and drainage collection layers, and a composite liner.

The collection system currently is preliminary and will be refined in the next study phase after site investigations have been carried out.

Shear keys will be required to anchor the containment system to the foundation to prevent the TSF from sliding downslope. Shear keys will consist of trenches excavated perpendicular to the axis of the valley, installing the liner through the trench, and then backfilling with suitable free-draining fill.

18.9.4.5 Tailings Placement

Filtered tailings will be transported from the process plant to the TSF by an overland belt conveyor and distributed inside the TSF by a radial stacker and up to five (5) grasshopper conveyors. The tailings will then be spread by dozers into lifts and compacted using a conventional vibratory roller.

Tailings placement and compaction within the structural zone should only be carried out under suitable weather conditions (i.e., dry and warm) with tailings that meet the target moisture content. Compaction testing will be required for the structural zone to ensure fills meet specification for level of compaction and moisture content. Additional reworking of the filtered tailings, including discing or harrowing, may be required reduce the moisture content prior to compaction to meet these requirements.

During the winter months, where freezing temperatures can limit the effectiveness of compaction, the filtered tailings will be placed internally within the TSF and away from the structural zone. Re-compaction of tailings placed during the winter will be carried out in the spring following thawing. Consideration and planning will be required to ensure adequate footprint is available to spread the tailings that have accumulated over the winter.

An area will be designated to temporarily store off-spec tailings (i.e., tailings that have a higher water content than specified and may not be able to be compacted per Project specification) to allow for future rework/drying before placing in permanent storage in the TSF.

18.9.4.6 Final Surface Grade

The top of the TSF will be graded at a minimum 1% slope from the top of the valley to the structural zone to promote shedding of precipitation and avoid accumulation or ponding of water that would adversely affect the consistency of tailings. The structural zone will be outfitted with a spillway channel at the abutment to convey runoff to the reclaim pond.

18.9.4.7 Reclaim Pond

Contact water resulting from precipitation on the TSF will drain by gravity to the reclaim pond downstream of the TSF. A spillway channel will convey contact runoff down the outer TSF slope into the reclaim pond, where water will either be used for mill make-up supply or treated and discharged.

The filtered tailings are relatively dry with an estimated gravimetric moisture content of about 18% which expected to result in very little free water to drain out. Rainfall is expected to be the main source of contact water, and some infiltration is expected to report as tailings seepage via the collection system.

The reclaim pond was sized to contain a volume of 59,000 m³, assuming a 4,000 m³ operating volume and the runoff from an EDF, defined as the 100-year 24-hour event. SLR calculated the EDF volume based the footprint area of 41 ha for the process plant and TSF (after diversions) and a runoff factor of 100%.

SLR assumed that the containment berms for the reclaim pond will be constructed by placing and compacting soils excavated from within the reclaim pond footprint and cut soils from site development, pending suitability for reuse. The reclaim pond will be constructed with a similar composite liner system as the TSF to prevent solution seepage into the groundwater.

An emergency spillway to prevent overtopping will be constructed with a rip rap protected channel and energy dissipation downstream of the of the pond and will be sized to safely pass the IDF event, defined as the 1/3 between the 1,000-year 24-hour event and the probable maximum flood (PMF). The water discharged by the spillway will drain overland to the Lipa River.

18.9.5 Closure Considerations

At the mine closure, the TSF will be fully covered with a soil growth medium and vegetated with similar plant species found naturally in the area. As the tailings will be deposited in an unsaturated state, the TSF surface cover can be constructed immediately upon cessation of operations. In the active phase of closure, water treatment and environmental monitoring will be required until it is demonstrated that the site conditions meet pre-mining conditions. As the site reached passive phase of closure, no additional treatment or monitoring would be expected.

18.9.5.1 Water Considerations

At closure, the contact water, collected in the reclaim pond, will continue to require treatment until water quality meets regulatory requirements. Once the water quality requirements have been met, the pond can be drawn down and the dam breached such that a smaller pond volume is retained to allow sedimentation of the runoff flowing off the TSF.

18.9.5.2 Progressive Reclamation

The outer slope of the TSF will be progressively reclaimed during operations as the TSF is filled and the basin level rises. Rockfill berms will be constructed at the toe of the compacted tailings structural zone in upstream raises, staying ahead of the internal elevation of deposited tailings. The rockfill slope cover will protect the deposited tailings from erosion and is considered suitable for closure.

18.10 Water Management

18.10.1 Hydrology

The proposed Čoka Rakita operations lie within the headwaters catchment of the Ogašu Lu Gjori Stream, a tributary of Lipa River. All above-ground Project facilities, including the TSF, the ROM pad, the underground mine portal and the waste rock stockpile, lie within the catchment or on top of the tributaries of Ogašu Lu Gjori Stream.

The Ogašu Lu Gjori Stream drains to the east into Lipa River, which drains north before it merges with Veliki Pek River. The flows ultimately report to the Danube River. The natural streams in the Project area are impacted by snow precipitation, experiencing low flow conditions during the winter months until a major thaw that typically occurs around March and April. Many watercourses within the Project area are ephemeral (per Section 20).

18.10.2 Description of the Water Management System

The water within the basins where the mine facilities will be located is classified into two (2) categories, “contact” and “non-contact” water. Contact water is surface water that has been exposed to excavated materials (e.g., ROM, tailings and waste rock) or mining process facilities (e.g., water within the Process Plant and the Underground Mine). Non-contact water is surface runoff that has not been in contact with any disturbed surface within the Project area and is diverted around the mine facilities. Any non-contact water that mixes with contact water becomes contact water.

The water management system of the Project includes the following main components:

- Non-contact water diversion ditches;
- Contact water collection ditch for the Process Plant terrace;
- Non-contact water collection pond; and
- Contact water management ponds (the mine dewatering pond and the TSF reclaim pond at the base of the TSF).

Figure 18.13 illustrates the water management concept proposed for the operating phase of the Project. The overall water management concept is to divert non-contact water to reduce the amount of contact water to be managed at the Project site and collect the contact water for conveyance to water management ponds. The contact water collected is used to meet water demands onsite, with excess discharged to the receiving environment. A non-contact water collection pond is proposed to provide water supply contingency for the Process Plant.

Figure 18.1 shows the footprints of the mine facilities as well as the water management infrastructure, including non-contact water diversion ditches, the non-contact water collection pond, and contact water management ponds. The contact water management ponds will be equipped with an emergency spillway to prevent dam overtopping. An operational spillway will be constructed at the non-contact water collection pond to allow passive discharge of flows to the environment during normal operations. It will be also equipped with an emergency spillway to prevent dam overtopping.

The non-contact water diversion ditches will collect and divert surface runoff from undisturbed areas upstream of the Project facilities to reduce the amount of water to be managed on site.

Non-contact water will be collected in the diversion ditches around the TSF and the south of the Process Plant terrace where the ROM Pad is located. The collected water will be directed away from the Project facilities towards the downstream sections of Ogašu Lu Gjori Stream that is undisturbed by Project operations. The Ogašu Lu Gjori Stream is a tributary of the Lipa River.

The mine dewatering pond will collect surface runoff from a portion of the Process Plant terrace (including the ROM Pad) by gravity drainage, as well as groundwater seepage pumped from the Underground Mine. It is noted that the mine rock taken to surface from underground mining will be used to construct the Process Plant terrace without need to deposit the mine rock in a stockpile.

Surface runoff from the north portion of the Process Plant terrace, where the majority of the buildings will be located, will be conveyed by gravity with a channel to the TSF Reclaim Pond (Figure 18.1). The TSF Reclaim Pond will collect surface runoff from the TSF footprint and the north portion of the Process Plant terrace.

The water collected in the mine dewatering pond and the TSF reclaim pond could be pumped to the Process Plant (Figure 18.1) to meet processing water requirements if needed (or pumped for other water uses such as dust suppression). However, there is no need for process make up water for Čoka Rakita. The only net industrial water demand is the freshwater requirement at the Process Plant for reagent mixing and gland water to be sourced from the effluent treatment plant (Figure 18.8). Excess water collected in the ponds will be pumped to the effluent treatment plant prior to its release to the receiving environment (Ogašu Lu Gjori Stream).

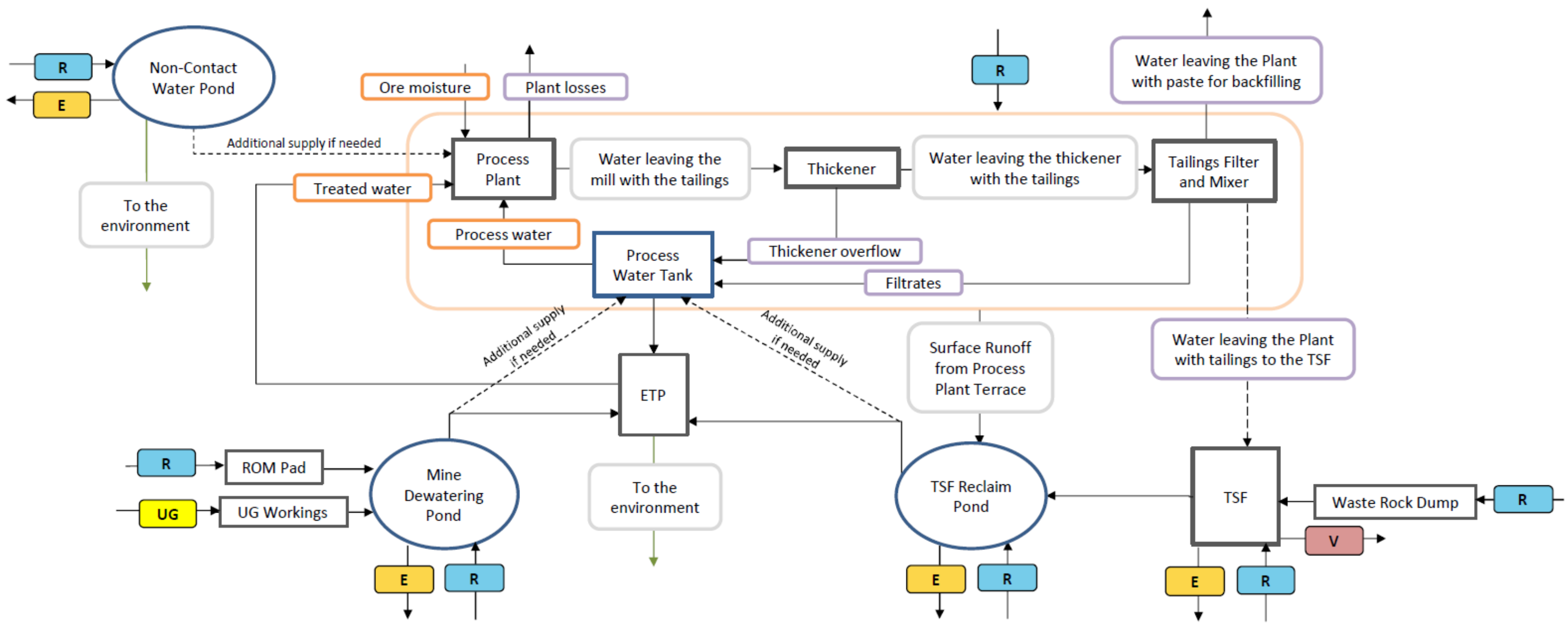
18.10.3 Effluent Water Treatment

Based on the available information when developing the water balance, an allowance has been made to treat mine dewatering and/or tailings reclaim water. Further quantification of the mine dewatering capacity and quality required to define the scope of the effluent treatment plant (ETP).

Figure 18.13 – Flow Logic Schematic

Legend

R	Surface runoff or direct precipitaion onto pond surfaces
E	Evaporation from pond surfaces
UG	Underground dewatering flow
V	Water retained in tailings voids



Source: SLR, 2024

18.10.4 Site-Wide Water Balance

Deterministic water balance modelling developed in Excel spreadsheets was carried out for the operating stage to:

- Estimate the water availability from surface runoff (collected from the Project facility footprints, taking into account the implementation of water diversion ditches) and groundwater flows collected in the Underground Mine;
- Verify that the processing water demands can be met; and
- Simulate the transfer of water between water management ponds to either store it, use the water collected on site to support mining activities (process, dust suppression, etc.), or discharge excess water to the receiving environment.

The PEA site-wide water balance considers ultimate facility footprints and the proposed water management strategy for the operations phase. Sequencing of construction and commissioning of Project facilities has not been defined and was not considered in the water balance simulations (i.e. no multiple snapshots were modelled for the PEA).

The site-wide water balance simulations were developed for average annual precipitation conditions as well as dry and wet annual precipitation conditions associated with the 25-year return period. The modelling results show that water surplus is expected under the three precipitation conditions simulated including the dry condition (i.e., annual precipitation below average), resulting in flows that must be discharged to the receiving environment.

It is noted that there is no need for process make up water for Čoka Rakita (Figure 18.13). The only net industrial water demand is the freshwater requirement at the Process Plant for reagent mixing and gland water that will be supplied as treated water from the effluent treatment plant. The water balance analysis indicates that sufficient water would be collected from surface runoff and Underground Mine dewatering to meet the freshwater demand at the Process Plant without the need to withdraw water from natural streams or aquifers through groundwater wells, even under dry annual precipitation conditions.

The design storage capacity up to the overflow spillway invert for each contact water and non-contact water pond proposed for the Project are summarised below. The storage volumes were defined according to design criteria as discussed in Section 18.10.5.

- TSF Reclaim Pond 59,000 m^3;
- Mine Dewatering Pond 36,000 m^3;
- Non-Contact Water Pond 66,000 m^3.

18.10.5 DESIGN CRITERIA FOR WATER MANAGEMENT FACILITIES

The design criteria of the water management components are as follows:

- The diversion ditches required to divert non-contact water are designed for closure for the 1 in 100-year, 24-hour rainfall event.
- The Non-Contact Water Pond to be used as a contingency source of freshwater supply for the Process Plant is designed with storage capacity to maintain two months of freshwater supply. In addition to the required storage capacity, the dam crest elevation has been defined assuming a 1 m freeboard in order to prevent dam overtopping.
- The storage capacity of the contact water collection ponds (i.e., Mine Dewatering Pond and TSF Reclaim Pond) accounts for the following:
- Dead storage allowance (10% of operating pond volume);
- Operating pond volume (live storage) calculated as 3 days of storage for the month with highest inflow (April) under the 1 in 25 years wet annual precipitation conditions;
- The EDF, which corresponds to the runoff volume associated with the 1 in 100 year, 24-hour rainfall storm event;
- In addition to the required storage capacity, the dam crest elevation for all contact water collection ponds has been defined assuming a 1 m freeboard in order to prevent dam overtopping;
- The design rainfall storm event depth was increased 20% to account for climate change effects in the EDF volume estimates and the design peak flows for the non-contact water diversion ditches; and
- It has been assumed 12-month discharge from the ETP to the Lipa River.

The EDF is defined as the runoff resulting from the largest storm event that can be stored without discharge to the environment through the emergency spillway. Retention of water during the EDF requires storage capacity above the normal operating water level (CDA, 2014). The contact water collection ponds will be equipped with emergency spillways. Runoff events exceeding the EDF will be discharged through the emergency spillway.

18.11 Power Supply and Distribution

18.11.1 POWER SUPPLY OPTIONS

The Project has two (2) options for powering the Project.

- The first option is to construct a 35 kV line from the city of Zagubita. The Zagubita substation will need to be expanded to accommodate the necessary equipment.

- The second option is to construct a 110 kV line from the city of Bor. In this case, the respective adjustments will also need to be made to expand the technical capacity of the substation.

In case of emergency, two (2) diesel generation plants, each with an effective capacity of 1.6 MW, will be provided to supply the demand of the mine and critical processes within the plant.

A power system study using the ETAP tool was conducted for each of the two (2) options, with the total plant demand fixed at 7.85 MW, leading to the following results and conclusions:

For the first option (Zagubita 35 kV), the line would be approximately 11 km long, originating from the Zagubita electrical substation to the Project's substation. The study resulted in a voltage drop of 6.4%, indicating that this option is not viable without the installation of a high-voltage regulator within the Project's substation. This voltage regulator aims to compensate for the voltage drop and set a voltage level of 35 kV at the power transformer terminals, with an available short-circuit current of 2.412 kA.

The second option involves providing a transmission line from the city of Bor at a voltage level of 110 kV and approximately 28 km. This option provides a voltage drop of no more than 2%, making it viable from a voltage regulation standpoint. There are no issues regarding voltage regulation, and the available short-circuit power 4.099 kA would be higher if greater power is required.

In summary, at this stage of the Project, two (2) options for electrical supply have been identified, each with its own advantages and disadvantages. In the next phase of the Project, it will be crucial to conduct a more detailed analysis from both technical and economic perspectives to determine which of the two options is most suitable for the Čoka Rakita Project. The Zagubita option was selected as a base case for the PEA study.

18.11.2 Power Distribution

The primary power system will feed a pair of transformers, each with a capacity of 16 MVA. These transformers, with a delta-star configuration, will be responsible for supplying 9 MW of power. These transformers should operate at no more than fifty percent (50%) of their capacity in order to provide redundancy.

The operating voltage level at the primary will depend on the choice of the supply source. For the secondary power distribution, 10 kV will be used to supply the switchgear in the medium-voltage electrical room.

18.11.3 Electrical Rooms (E-Rooms)

The electrical distribution considers the installation of a minimum of four (4) prefabricated electrical rooms (E-Rooms), equipped with all auxiliary services and containing Motor Control Centers (MCCs), variable speed drives at different power levels, communication equipment, and automation. To supply power to each of these electrical rooms, the installation of underground wiring is proposed from the medium-voltage electrical room. This will feed a system of transformers of different capacities according to the needs of each of these rooms, operating at a voltage of 10/0.4 kV.

18.11.4 Emergency Power

An emergency power system will be provided as a standby source of power to feed essential services (emergency and exit lighting, fire pumps, etc.) as well as critical process loads in the event of power loss from the power grid.

18.11.5 Site Load

The total power demand is estimated at 9.0 MW with 5.0 MW for the process plant, 2.5 MW for the underground mine and ventilation facility, and 1.5 MW for the waste dewatering and non-process facilities. The non-process facilities power is required to service the following: Administration, Offices, Mechanical Shop, Laboratory, Electric Rooms, Truck Maintenance, Guard House, heating of the Concentrator as well as losses in transformers and feeders. The process power demand was estimated based on data from the Mechanical Equipment List prepared for the Project.

Table 18.6 presents a summary of electrical load consumptions.

Table 18.6 – Project Summary of Power Consumption

Item	Description	Power Consumption (kW)
1	Process Plant	5000
2	Underground Mine	2500
3	Waste Dewatering	1000
4	Non-Process	500
	Total Estimated	**9000**

18.12 Automation

18.12.1 Control System Philosophy

For the purposes of this PEA, it is assumed that the production facilities are controlled and supervised from the central control room equipped with a SCADA control system located in the process plant.

The control system philosophy is based on the utilisation of Programmable Logic Controller (PLC) in all key areas of the plant. The ring topology is proposed to reduce the risk of downtime.

18.12.2 Local Control System and Instruments

The proposed control system includes local push button stations for all motors and the mainstream on/off valves for maintenance and safety.

The push button stations include a local start/stop station for all motors but no selector switch manual/ automatic in the field. The manual / automatic function is accessible only at the SCADA operating station and is programmed by area. The push button station can only be activated when the plant operator has selected which process area to change to manual mode for the push button function start. The stop function is always functional.

For the critical equipment, an extra push button (Emergency Stop) is added directly connected to the motor starter.

All the control loops are integrated and controlled by the PLC. For complex instrument or equipment supplied with PLC, a communication link is added to get remote status and diagnostics for the plant supervision control system.

All the field instruments and switches are wired to the PLC through junction boxes and digital and analogue input/output modules mounted in automation panels located in area electrical rooms. The standard 4-20 mA signal is the standard for instrumentation. The control logic is performed by the PLC.

The proposed PLC wiring includes junction boxes for instrument power supply, digital signals and analogue signals. The junction boxes will be located and installed in all the process areas. The junction boxes are interconnected to the remote Input/Output rack panel by multi-conductor cables.

18.12.3 Fibre Optic Network

An Ethernet network will be installed in the Processing Plant.

The proposed network communication system includes one fibre optic cable (16 fibres) with patch panels for the PLC and operator stations. The PLC communication network and the operator stations used different fibers from the same cable.

The Ethernet protocol communication system for a PLC application is fast, reliable and is the industry standard. All PLC manufacturers support the Ethernet protocol.

The remote facilities such as the fresh and reclaim water pump houses will have local PLC control link to the main control system by radio communication system with antenna.

18.12.4 System Server / Software

For the processing plant, the proposed system includes a redundant system server, one historian server and two operator workstations located in the central control room and remote operator stations in the field. The redundant server insures Network availability and data protection.

An Engineering station is also supplied for the system programming and the maintenance debugging. The station will be in plant electrical rooms or in the central control room.

The proposed system is designed with PLC and the equipment is supplied with standard PLC programming software and standard software for the supervisory and control system (SCADA). This type of equipment is available from any major PLC supplier.

The SCADA system includes a development license and run time licenses for the supervision and control of the entire plant operation and has the capacity to communicate with management's computer network.

The electrical power supply for all PLC and servers will include Uninterruptible Power Supply (UPS) units located in pressurised electrical rooms.

18.13 Communication

Communication for personnel within the process plant and mine area will utilise handheld radios and mobile phones, leveraging services provided by the local wireless cell phone provider.

Moreover, internet access and data transmission will be facilitated through the services of the local wireless provider. It's imperative to establish a connection with a local wireless provider early in the construction phase to ensure internet and VPN network connectivity necessary for construction activities. Additionally, once the dedicated Project hardwired connection is operational, the local wireless network will serve as a backup internet connection, ensuring network redundancy.

18.14 Site Services

18.14.1 Raw Water Supply

Process related water supply can be fed from multiple sources to optimise water usage and reduce water treatment as noted in the Site Wide Water Balance (Section 18.10.4).

18.14.2 Compressed Air

An allowance for compressed air distribution has been included. Compressed air will be individually produced at the process plant and mine workshop terrace areas using dedicated compressors to avoid the requirement for buried piping.

18.14.3 FIRE SYSTEMS

The plant raw water tank will be segmented to provide a suitable volume of dedicated fire water, attached to suitably arranged duty/standby fire water pump manifold.

Suitably insulated fire hydrants will be present throughout the site and distributed terraces, connected to the underground/surface distribution system.

Buildings will be equipped with fire hoses and fire extinguishers at every floor.

18.14.4 SEWAGE SYSTEMS

A centralised packaged sewage treatment plant has been included at the effluent treatment plant terrace. The sewage treatment plant will process sewage and grey water run-off. The use of a septic tank system will be evaluated in the next phase toward optimising the design. An allowance has been included for a sewage treatment plant system of sufficient capacity for the site personnel compliment.

18.14.5 WASTE DISPOSAL

A suitable domestic waste disposal site has not been identified as part of this Report. A waste management strategy will be developed as part of the next Project phase as to the correct on and off-site disposal for waste. It is recommended that a potential domestic waste dump facility be identified early on in the next study phase. Specialised third-party contractors will be identified that can safely dispose of hazardous waste.

Industrial and hazardous waste will be stored in suitably contained areas for off-site disposal. It is assumed suitable facilities are available in the region for disposal of hazardous waste due to ongoing heavy industry in the region. It is recommended to identify and negotiate with such facilities during the next Project phase.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

The gold market is mature globally with numerous reputable refiners located throughout the world. As such, a market study for gold products or for the expected price of gold was not undertaken.

The terms contained within any future sales contracts are expected to be typical and consistent with standard industry practice and contracts for the supply of filtered concentrate elsewhere in the world.

Table 19.1 presents the smelter terms, including the percentage of the payable metals, treatment charges, refining charges, and penalties on a Carriage and Insurance Paid To (CIP) destination port basis. The terms are based on DPM's discussions with international smelters, refineries and traders for similar quality concentrates and can be taken as representative of the expected terms for concentrates produced by the Project. From a logistics perspective, the mine would need to absorb all freight costs from the mine to the relevant destination port, with the buyer bearing all remaining transportation costs from the arriving vessel at discharge port to the receiving smelter.

Table 19.1 – Smelter Terms

Item	Unit	Flotation Concentrate	Gravity Concentrate
Gold Payability	%	97.5[1]	99.8
Silver Payability	%	95.0	95.0
Copper Payability	%	100.0	0.0
Moisture	%	12.0	20.0
Treatment Charges	US$/dmt	150.00	170.00
Refining Charges			
Au	US$/oz	7.50	6.00
Ag	US$/oz		0.75
Penalties	$/dmt	-	-

Note:
1. Gold payability in flotation concentrate is subject to a -1 g/dmt minimum deduction if the gold grade is less than 40 g/t.

19.2 Commodity Price

DPM has adopted the following price projection for the PEA financial model base case as presented in Table 19.2.

Table 19.2 – Commodity Pricing

Element	Unit	Financial Model
Au	US$/oz	1,700.00

19.3 Contracts and Off-Takes

No sales contracts have been established to-date for either of the Čoka Rakita concentrate qualities.

Contract terms and treatment charges will be negotiated with international smelting and refining companies. It is expected that any future contract terms will be typical of sales contracts for similar concentrate qualities. Metal prices in the sales contracts will reference prices published in the London metals exchanges.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

As the Project is located in Serbia, it will be permitted to operate and regulated by Serbian authorities to Serbian standards. DPM is operating with the permission of the Ministry of Mining and Energy (MME) in conjunction with the Ministry of Construction, Transport and Infrastructure (MCTI). Mining structures are permitted under the Law on Mining and Geological Explorations, non- mine Objects are separately permitted under the Law on Planning and Construction. There is a range of other approvals and permissions required under ministries including the Ministry of Agriculture, Forestry and Water Management (MAFWM), Ministry of Environmental Protection (MEP), Ministry of Interior (MI), and Institute for the Preservation of Cultural Heritage and the Institute for Nature Conservation of Serbia.

Serbia was granted candidate status in March 2012 and legislation and frameworks are being harmonised progressively to those of the EU. Therefore, the Project utilises standards that relate to EU environmental laws, such as the Environmental Impact Assessment Directive (2014/52/EU), Water Framework Directive (2000/60/EC) and Waste Framework Directive (2008/98/EC) and Industrial Emissions Directive (2010/75/EU). Few private-sector mining projects of this scale have passed through the entire Serbian permitting process in recent years, although Cukaru Peki copper-gold mine nearby is now two years into operation; but there are limited other precedents to inform the permitting process as it currently stands.

In addition, DPM is developing the Project in alignment with EU requirements and good international practice, such as the performance requirements of the EBRD Environmental and Social Policy and World Bank Group Environmental, Health and Safety Guidelines (International Finance Corporation World Bank Group, 2007). Further, DPM has committed to develop all of its projects in compliance with EBRD Environmental and Social Policy.

20.2 Permits

The Project has prepared a Permitting Plan (DPM, 2024) and this document sets out the principal permits and approvals required for exploration, construction, operation and closure of the Project.

Crni Vrh Resources d.o.o., a Serbian corporate entity and wholly owned subsidiary of DPM, is the holder of all licenses and permits for further exploration on the Project.

20.3 Environmental and Baseline Studies and Issues

The Project is located in the Region of Southern and Eastern Serbia, in the Braničevo District, and respectively, in the Municipality of Zagubica, on the mountain range between Bor to the southeast and Žagubica and Laznica to the west. Mjajdanpek and the Danube River are to the north. State roads 164 and 161 run east-west close to the Project area. There are no designated protected areas for biodiversity or cultural heritage in or around the infrastructure that make up the current Project. The area is characterised by wooded valleys and seasonally grazed pastures with isolated settlements.

Key environmental and social risks are similar to those associated with other gold mining projects and include safeguarding rivers, groundwater and biodiversity and mitigating permanent effects, as well as risks associated with acquiring land. During operations, rivers may potentially be impacted by dewatering, diversions and discharges, and permanent infrastructure will overlie several hundred metres of river channel within the headwaters of the Ogašu Lu Gjori and Dumitrov streams, within the Lipa River catchment. Access to seasonal farming, hunting and tourist amenity in the project area will also be restricted during operations, although closure planning could include the means to reinstate these after operations cease. Completion of the baseline survey program is required to fully understand and manage these impacts.

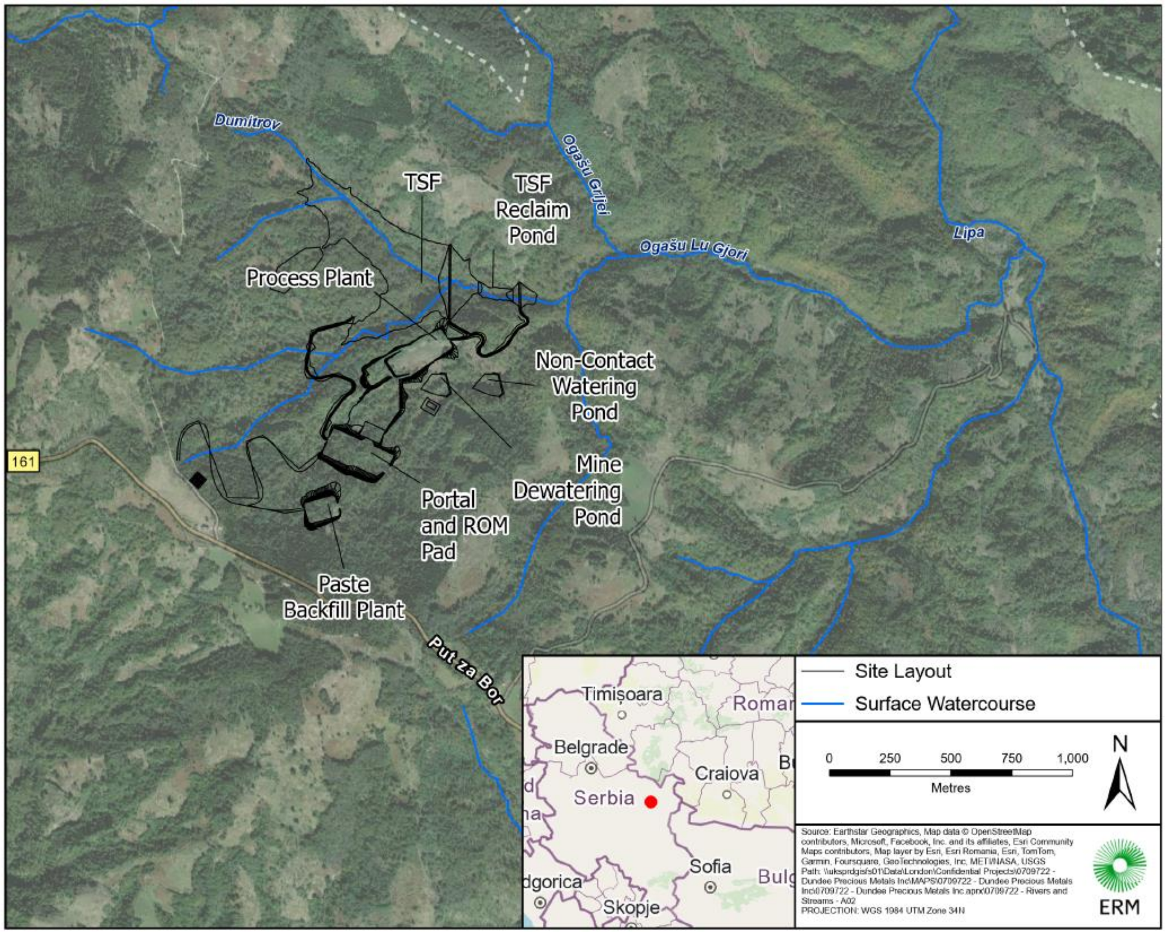
Environmental and social risks associated with the Project were identified through a risk review in November and December 2023, and periodically reviewed throughout the PEA. These risks will be further investigated and assessed as part of the EIA process. The key risks are around surface water and groundwater during operation and especially in the closure phase, the impact from loss of several hundred metres of riverine habitat and consequently on biodiversity, dewatering and diversions during operations affecting springs, wells and streams, including in adjacent catchments to the south, and economic displacement associated with land acquisition.

20.3.1 WATERCOURSES AND GROUNDWATER

A network of rivers and streams runs through the Project area (Dumitrov stream, Ogašu Lu Gjori Stream, Ogašu Grljei Stream, Valja Saka River and the Lipa River Figure 20.1). Many of the uppermost catchment streams in the project area are ephemeral and are likely seasonal watercourses fed by spring snow melt. Watercourses in the area drain to the Danube River, which is internationally protected. Serbia is a signatory to the International Commission for Protection of the Danube River and projects which may affect water quality of the Danube could trigger the need for transboundary engagement between the Serbian and neighbouring governments (Romania and Bulgaria).

The TSF will be situated over the Dumitrov Stream, while the process plant, UG mine portal, ROM pad and paste backfill plant will be located on elevated ground within the headwaters of the Ogašu Lu Gjori Stream. Diversion channels have been planned to redirect surface water flow around the TSF, the TSF reclaim pond and the south of the process plant terrace where the ROM pad is located.

Figure 20.1 – Drainage Map of Čoka Rakita PEA Area



Source: ERM, 2024

20.3.2 BIODIVERSITY

No internationally recognised sites for biodiversity were identified within 5 km of the Project; the closest protected area is the Nature Park Kučaj – Beljanica (located within the Crna Reka catchment) which is currently in the process of being upgraded to National Park status. This area is a forested mountainous region which hosts a number of endgandered and protected species, likely to also be present in the project area. The boundary of Nature Park Kučaj – Beljanica is 3 km southwest of the mineral resource (at surface) and 5 km from the nearest project infrastructure.

It is not anticipated that the protected sites identified will be impacted by the Project. However, downstream effects including the potential for a reduction in water flows caused by underground mine dewatering will need to be assessed.

In the adjacent Timok study area, the presence of International Union for Conservation of Nature and Natural Resources (IUCN) Red List Vulnerable species, and species listed on Annex IV of the

Habitat Directive meet the current criteria for the European Bank for Reconstruction and Development (EBRD) priority biodiversity features and critical habitat. If these are also present in the Project area, they will require further assessment to understand the potential effects of the Project on the priority biodiversity features and critical habitat, and the development of dedicated management plans to mitigate them.

These assessments and management plans will be undertaken as part of the EIA process. The mitigation hierarchy will be applied and, if required, additional conservation actions or biodiversity offsets will be investigated. The importance of the Project site in relation to ecosystem services will be assessed as part of the social assessments undertaken.

DPM has planned further baseline studies, including surveys for bats, mammals, birds, protected flora and aquatic ecology which will further inform the environmental impact assessment.

20.3.3 Noise, Vibration, Nuisance Dust and Greenhouse Gas Emissions

Baseline noise surveys began in 2022, two (2) seasonal campaigns are planned, and the summer survey was completed in 2022. Both campaigns are two (2) weeks long with attended and unattended measurements at identified receptor locations over and around the project footprint. The winter campaign is to be completed late 2024 or early 2025. Additional monitoring locations to the northeast of the Project area should be identified and included in future campaigns.

Baseline vibration surveys are yet to take place.

Baseline air quality monitoring commenced in December 2021 at 11 locations in the adjacent area (associated with the Timok project) (PM_{10} and airborne metals at six (6) locations, $PM_{2.5}$ at five (5) locations). Results indicated that the area adjacent to the Project airshed is considered undegraded with regards to carbon monoxide (CO), SO_2, NO_2, Volatile Organic Compounds (VOCs), benzene and dust deposition, PM_{10} and $PM_{2.5}$. (particulate matter). Occasional exceedances of the daily limit were recorded in certain locations for PM_{10} and $PM_{2.5}$; therefore, these emissions and associated impacts may require special attention. PM_{10}-bound levels of arsenic, cadmium, lead, nickel, manganese and copper are all below annual limit values. Therefore, the adjacent area is also considered undegraded with regards to these metal levels.

Monitoring of $PM_{2.5}$ and PM_{10} will continue at three (3) locations, monitoring of NO_2 and SO_2 will continue at five (5) locations throughout 2024 and 2025. Additional monitoring locations to the south and east of the Project area should be identified and included in future monitoring.

The Project will generate greenhouse gas emissions that will need to be measured and managed in line with international good practice.

20.3.4 Soil and Geology

As part of the exploration activities soil sampling was undertaken across the Project area. Assessment of the soil in the context of the baseline of agricultural capability and chemical quality has not been undertaken to date. Consistent with the PEA planned layout, where topsoils are to be disturbed, the principle will be to conserve these in stockpiles for ultimate reuse as a rehabilitation medium.

No geotechnical studies have been undertaken to date, but it is noted that many of the Project infrastructure locations, especially access roads and the TSF, are on very steep terrain. Risks from landslides and other ground instability will need to be assessed by detailed geotechnical investigations in the future.

20.3.5 Cultural Heritage

There are no protected areas for cultural heritage or known registered archaeological features within the Project footprint. The region is well-known to be home to some of the earliest metallurgical technology in Europe and it is possible that there are buried remains in the Project area.

The Project area is important to the Vlach community, an ethnic community in Serbia with its own language, dress and culture. Transhumance is a fundamental element of Vlach culture, with grazing on higher ground, including that in the Project area, in the summer. The remains of this are widespread in the form of many isolated farms, mills and other structures, most of them abandoned.

The route of the historic Žagubica-Bor railway crosses the Project area. This was built by forced labour during the Second World War. This narrow-gauge line, preserved in cuttings and embankments, runs along a meandering route to the west and south of the TSF, process facility and portal location. Relatively little is known of its history other than it was constructed during the German occupation in very difficult conditions by forced labour, many of them Hungarian Jewish prisoners based in camps at Bor. The railway has historical significance, notably to those communities whose ancestors were forced to build it. Small sections of the railway are likely to be disturbed by the Project infrastructure.

The environmental impact assessment will address potential impacts on both tangible and intangible heritage. Any rural and railway structures that are likely to be disturbed shall be recorded archaeologically.

Land acquisition is required for the development of the Project, access to the site and the construction of associated facilities. Gaining access to this land requires purchasing land from private and public owners. Proof of land ownership is a prerequisite to obtaining key permits such as the Approval for Construction of Mine Structures. Habitation within the wider Project area is sparse and typically restricted to summer seasons. There is currently a land dispute underway in the Serbian court system, where the siting of mine infrastructure has avoided those areas under dispute.

The Project will be required to develop a Land Acquisition Strategy in line with national legislation and international good practice, to manage this complex and sensitive process. This will lay the foundation for the development of a more detailed Land Acquisition Plan once the Project design is completed.

20.4 Social and Community Engagement

DPM has worked to establish good relationships with the local community since 2007 and communications are managed through a DPM's communication plan. DPM expanded its resources and conducted training in 2019 to facilitate transparent and meaningful community engagement. The Project team maintains a map of stakeholders and has earmarked vulnerable groups which will require targeted engagement.

20.5 Mineral and Non-Mineral Waste

20.5.1 Mineral Waste

Prior and during mining activities at Čoka Rakita, rock will be produced from the excavations in the underground mine, as well as from construction activities, such as road cuttings. About 60% of the rock generated during mine development is planned to be used as mine backfill, and the remainder will be used to construct an operational platform in the process area. This terrace platform will be lined and run-off directed to lined ponds and ultimately on to the ETP. There will be no other waste rock facilities required by the operation or in construction. Furthermore, DPM targets to store 40% of the filtered and cemented tailings underground with the remainder stored in a TSF at surface.

To determine the risk for metal leaching and acid rock drainage (ML/ARD) of excavated rock, DPM selected 29 composite samples for an initial testing campaign. Each composite sample comprises material from five to ten (in most cases consecutive) 1-m drill-core intervals from the same drill hole. The 29 samples were selected to represent main lithologies that are projected to be disturbed by mine development and underground excavation: the sandstone (S1/S2 unit), diorite (EMPO), marls (SMR) and mafic volcanogenic epiclastics (SFD).

The S1/S2 and marl samples show highly variable NPR and NNP, with about half of those samples being uncertain or PAG. Specifically, the skarn-altered S1/S2 and marl are likely to be PAG, because the metasomatic processes precipitated sulphides while primary carbonates were destroyed. The less altered sedimentary rocks contain more primary carbonates and less sulphides which is reflected in high NPR and NR values for samples of those rocks. Of the 29 rock samples tested, 8 classify as PAG, 13 as uncertain, and eight as not PAG.

A representative set of samples of all rock and tailings materials will be subject to long-term kinetic testing. Excavated rock used for construction will undergo humidity cell tests to determine geochemical behaviour under atmospheric conditions. Column tests to determine geochemical behaviour of rock and tailings materials to be stored underground and of potentially underground-exposed wall rocks in permanently water-saturated conditions are also under consideration.

Guided by the static and kinetic testing results, an ML/ARD management plan will be developed to mitigate potential adverse effects on the receiving environment from excavated rock and tailings and elevated metal concentrations in surface and groundwaters. Proactive planning by DPM can integrate ML/ARD testing results with multi-element drill-hole assays in order to spatially model and predict the ML/ARD characteristics of rocks and tailings across the deposit.

20.5.2 Non-Mineral Waste

Non-mineral wastes will include non-hazardous and hazardous materials such as packaging, used oil, batteries, food, medical waste and sewage. The Project will develop a waste management inventory as part of the design process and a strategy for disposal of each waste stream, following the waste hierarchy (reduce, reuse, recycle, treat, dispose) and in line with Serbian regulations and international good practice. Suitable third-party waste carriers and treatment/disposal sites will be identified and the details of the approach for storage, transportation, treatment and disposal of each waste stream will be set out in the Project waste management plan.

20.6 Closure

The Project will advance closure planning and costing exercises throughout the planning stages and operational life, with the next revision at the EIA stage.

21 CAPITAL AND OPERATING COST ESTIMATES

The Capital Cost Estimate (Capex) includes the material, equipment, labour and freight required for the mine pre-development, processing facilities, tailings storage and management, as well as all infrastructure and services necessary to support the operation. DRA developed the Capex and Opex related the process plant, plant infrastructure and on-site infrastructure for the Project scope described in this Report. DRA also consolidated external sources, including:

Table 21.1 – Other Capex/Opex Responsibilities

Consultant	Scope
DPM	Provided Owner's G&A Costs, Taxes and Duties; and Closure costs
WSP	Prepared underground mining and mining infrastructure costs for initial and sustaining Capex and Opex
SLR	Prepared construction quantities for the Filtered TSF, Water Management, and DRA applied rates to complete the Capex
RMS	Prepared Capex and Opex for Paste Backfill Plant, where DRA only estimated bulk earthworks, concrete, and structural steel

The estimate is within the range -30%+60% and was prepared in accordance with the AACE International (Association for the Advancement of Cost Engineering) Class 5 estimating standard. Although some individual elements of the Capex may not achieve the target level of accuracy, the overall estimate falls within the parameters of the intended accuracy.

All Capex and Opex costs are expressed in United States Dollars (US$ or $) and are based on Q1 2024 pricing.

21.1 Capital Cost Estimate

21.1.1 Capital Cost Summary

The Capex consists of direct and indirect capital costs as well as contingency. Provisions for sustaining capital are also included mainly for mining and tailings storage expansion. Amounts for mine closure and rehabilitation of the site are estimated as well.

21.1.1.1 Pre-Production Initial Capital Cost

Table 21.2 presents a summary of the initial Capex and sustaining Capex, which is distributed over the LOM and is separately indicated from the initial Capex.

Table 21.2 – Capital Cost Summary

Area	Description	Initial Capex (US$ M)	Sustaining Capex (US$ M)	Total (US$ M)
2000	Underground Mine	76.2	49.5	125.7
4000	Processing Plant	91.1	1.2	92.3
5000	Filtered Tailings / Water Treatment Facilities	31.3	18.0	49.3
6000	On-Site Infrastructure, Site-Wide General	25.6	0.0	25.6
7000	Off-Site Infrastructure	9.7	0.0	9.67
8000	Operational Readiness	11.5	0.0	11.5
9000	Indirect Costs	52.8	3.4	56.2
9100	Owner's Costs	13.5	0.0	13.5
9170	Closure and Rehabilitation[1]	0.0	19.3	19.3
9900	Project Contingency	69.2	20.3	89.5
	Total Major Area Capex	**380.9**	**111.7**	**492.6**

Numbers may not add due to rounding.
1. Closure costs do not include the non-recoverable VAT of approximately $3 M.

21.1.1.2 *Sustaining Capital*

The sustaining Capex commences upon production of concentrate and continues throughout the LOM. The sustaining costs are included in the overall Project Capex.

Sustaining capital costs cover several areas, as itemised in Table 21.2.

21.1.1.3 *Closure and Rehabilitation Costs*

At the end of the Project life, it is required that all disturbed areas are rehabilitated, and equipment and buildings are disposed of. Closure costs are included in the sustaining capital. This excludes the non-recoverable VAT of approximately $3 M.

Based on site layouts, a provision is estimated for closure and rehabilitation of the mine site.

21.1.1.4 *Working Capital*

Requirements for Working Capital were estimated as 60 days of receivables, 30 days of payables, and 5 days of inventory This working capital amount is required at start of production and accounted for in the economic analysis.

21.1.2 Major Assumptions

The Capex is based on the following key assumptions:

- All relevant permits are obtained in a timely manner to meet the Project schedule.
- Quotes from Vendors for equipment and materials are valid for budget purposes.
- Suitable backfill material is available locally. Soil conditions are adequate for foundation bearing pressures.
- Engineering and construction activities will be carried out in a continuous program with full funding available including contingency.
- Bulk materials such as cement, rebar, structural steel and plate, cable, cable tray, and piping are all readily available in the scheduled timeframe.
- Capital equipment is available in the timeframe shown.
- A 28-month construction and commissioning period is required from receipt of construction permit, including mine decline and infrastructure development followed by a three month process plant ramp-up.

21.1.3 Major Exclusions

The following items are not included in the Capex:

- Provision for inflation, escalation, currency fluctuations and interest incurred during construction;
- Schedule delays and associated costs;
- Scope changes;
- Unidentified ground conditions;
- Extraordinary climatic events;
- Force majeure;
- Labour disputes;
- Schedule recovery or acceleration;
- Cost of financing, property taxes, corporate and mining taxes, duties; and salvage values (although these are considered in the Economic Analysis).

There is no construction camp and catering in this Capex. It is assumed that local contractors can find local accommodation.

21.1.4 Currencies

The base currency for this PEA is United States Dollars (US$). The currency exchange rates were determined by referring to the OANDA.com website. The exchange rates for the various currencies were ascertained as of Q1 2024 and are listed in Table 21.3.

Table 21.3 – Currency Conversion Rates

Currency*	Code Name	US Dollar Equivalent to 1.00 Currency	Currency Equivalent to 1.00 USD
US$	United States Dollar	1.0000	1.0000
CAD	Canadian Dollar	0.7484	1.3362
EUR	Euro	1.0857	0.9210
GBP	British Pound	1.2728	0.7856
AUD	Australian Dollar	0.6608	1.5134
ZAR	South African Rand	0.0540	18.5185
RSD	Serbian Dinar	0.0091	109.8901

* Not all presented currency conversions are necessarily used in the Capex
Source: OANDA Exchange Rate, 2024

21.1.5 Underground Mine Capex

21.1.5.1 Summary of Estimated Costs

Table 21.4 presents WSP's LOM Capex for the underground mine in US$. The estimate accuracy aligns with Class 5 standards, as defined by AACE International.

The QP is of the opinion that the estimated Capex for the underground mine is reasonable for a PEA level analysis.

Table 21.4 – Underground Mine Capex

Description	Year-2 (US$ M)	Year -1 (US$ M)	Year 1 (US$ M)	Total (US$ M)
Personnel	4.0	6.6	2.7	**13.3**
Equipment Operation	5.0	8.3	3.3	**16.6**
Materials and Services	6.3	10.5	4.2	**21.0**
Capital Acquisitions	7.6	12.7	5.0	**25.3**
Total Capex	**22.9**	**38.1**	**15.2**	**76.2**

21.1.5.2 *Basis For Underground Mine Capex*

The database of unit cost inputs for the estimate were derived from the following sources:

- Quotations from suppliers for new equipment;
- Costs from the Chelopech Mine;
- Cost data from other mines and projects; and
- Estimates based on experience from other projects.

Capital costs for mine development and infrastructure are categorised into Personnel, Equipment Operation, and Materials and Services.

Capital Acquisition costs primarily comprise mobile and fixed equipment purchases, including associated expenses such as shipping to the site and commissioning.

21.1.5.3 *Assumptions*

The following assumptions are made in the preparation of the underground mine Capex:

- Quotations obtained from vendors for equipment and materials are provisional and intended solely for budgetary purposes. DRA has reviewed and deemed them reasonable for the PEA.
- Tailings will be suitable for preparing paste backfill with the required quality and assumed cost.
- Personnel with the required experience and skills will be available starting from the initiation of the development of the twin declines. An Operational Readiness plan will ensure that there are sufficient costs and list the required skill to start and operate.
- Pieces of equipment reaching the end of their useful life will be replaced with new units; consequently, the cost estimate does not include overhauls.
- Decommissioned pieces of equipment have no salvage value.
- Except for activities related to raise boring and service hole drilling, DPM employees carry out all tasks within the underground mine, including horizontal development, construction of underground infrastructure, and installation of permanent services.
- DPM will purchase all equipment for the underground mine except for the raise bore machine and drill rig for service holes.
- The Capex is based on Q1 2024 pricing without allowances for inflation.

21.1.6 Process Plant and Associated Infrastructure

21.1.6.1 *Material Take-off and Unit Rates*

Quantities were factored based on the Process Mechanical Equipment List and supplemented by estimates or allowances, wherever applicable. The quantities were developed based on the following documents:

- Site Plot Plans;
- Equipment List;
- Overall Process Flow Diagram;
- High Level Layout Drawings; and
- Basic Electrical Single Line Diagram.

21.1.6.2 Civil and Bulk Earthworks

Quantities of clearing and grubbing were established in square metres from the site plot plans and design software.

Quantities for mass earthworks were developed using the benchmarks established on the topographical plans to provide the amount of cut and fill required for embankments and terracing. No ground water level is assumed to be at the plant site or at any offsite location.

A cross section of the in-plant roads and the plant layout drawing were used to develop road quantities to a PEA level.

It is assumed that there is no piling whatsoever for the Project.

21.1.6.3 Concrete and Detail Earthworks

For equipment base and piers of all mechanical equipment outside of the buildings, quantities were factored as a percentage of the mechanical equipment cost based on DRA's historical data and experience.

Building sizes and concrete measures were based on similar past projects.

21.1.6.4 Structural Steel

For equipment steel support of all mechanical equipment outside of the buildings, quantities were factored as a percentage of the mechanical equipment cost based on DRA's historical data and experience. Building sizes and structural steel measures were based on similar past projects.

21.1.6.5 Architectural

The building areas were defined from preliminary layout drawings and sizes are similar to other DPM operations with similar throughput.

21.1.6.6 Mechanical Equipment

The mechanical equipment list, indicating size/capacity, dead weights and power, along with the process flow diagrams were used to provide mechanical equipment quantities.

Generally offshore supplied equipment is quoted FOB port of embarkation. There was a separate estimate for freight, insurance, import duties and taxes.

Mechanical and electrical equipment were divided into the following categories:

- Major equipment requiring budget quotations;
- Equipment priced from in-house data; and
- Allowances.

The mechanical installation contractor's indirect costs were priced by applying a percentage in the estimate.

21.1.6.7 Piping

The cost for process plant piping was factored as a percentage of the mechanical equipment cost based on DRA's historical data and experience.

Overland piping was quantified with material take-offs and historical unit prices from suppliers and fabricators of piping systems were used.

21.1.6.8 Electrical

The electrical equipment list and the single line diagrams formed the basis for the electrical equipment estimate.

Bulk electrical quantities of raceway were taken from the plot plan, equipment layouts and single line diagrams.

Power and control cables were factored by use of $/kW historical units.

Yard lighting was calculated based on $/m.

21.1.6.9 Instrumentation

The cost for instrumentation & controls were factored as a percentage of the mechanical equipment cost based on DRA's historical data and experience.

The DCS was priced using historical data.

21.1.7 LABOUR

The Capex is based on applying unit labour hours and crew rate for mechanical, electrical and instrumentation equipment and by factoring labour costs as a percentage of the equipment cost.

21.1.7.1 Labour Pricing

A blended composite crew rate was developed for mechanical equipment, and electrical, control, and instrumentation (EC&I) commodities. Crew rates reflect the price of contracted rates including:

- Contractor temporary buildings;

- Mobilisation / Demobilisation;
- Travel Allowance;
- Training;
- Small tools and consumables;
- Safety supplies;
- Construction equipment; and
- Contractor overhead and profit.

21.1.8 Indirect Costs

21.1.8.1 Contractor Indirect Costs

Contractor indirect costs include all Contractor overheads such as contractual requirements (safety, sureties, insurance, etc.), site establishment and removal thereof, and company / head office overheads.

The Contractor field indirects also include construction temporary facilities which include:

- Temporary Services; and
- Construction Equipment.

21.1.8.2 Capital Spares and Inventory

Capital Spares and inventory include:

- Capital Spares;
- Operational Spares (excluded from initial Capex);
- Commissioning Spares; and
- Initial Fills such as chemical reagents, ball load, water consumption, fuels, oils, etc.

21.1.8.3 Project External Consultants

Project External Consultants include:

- Consultant Engineering Procurement;
- Consultant Geology;
- Consultant Geotechnical;
- Consultant Project Management;
- Consultant EP support;
- Consultant Expenses; and

- Vendor Representatives.

The requirements for vendor representatives to supervise installation of equipment or to conduct a checkout of the equipment prior to start-up of the equipment as deemed necessary for equipment performance warranties was calculated and included in the estimate.

21.1.8.4 Commissioning

Commissioning includes the cost for a commissioning team for preparing the Process plant for start up and ramp up to plant operation. Commissioning (up to hot commissioning) costs in the indirect estimate include:

- Supervision;
- List of packages;
- Preparation of turnover documentation;
- Quality revision;
- Adjustment of equipment, erection, and process specifications; and
- Corrections.

The estimate includes the cost of support crews for commissioning among others, including:

- Tools and equipment;
- Safety equipment and services;
- Radios;
- Consumables; and
- Transportation, housing and meals.

Pre-Commissioning Teams include start up and commissioning personnel for:

- Owner's Team;
- EPCM Team;
- Contractor Support; and
- Vendor Support.

21.1.8.5 Freight and Logistics

Freight and logistics costs include allowances for the following items. The allowances are estimated on a percentage of equipment and materials costs.

- Ocean freight;
- Land freight;

- Air freight;
- Brokerage and Agent Fees;
- Warehouse Services; and
- Import Duties.

21.1.9 Sustaining Capital

Sustaining costs are included in the overall Project financial evaluation as capital costs incurred during the operations period. Sustaining costs are included for the following:

- Mining;
- Filtered TSF (by SLR, and supported by high level quantities and recent regional rates); and
- Paste backfill.

21.1.10 Closure Costs

Closure costs are included in the overall Project financial evaluation as costs incurred during the later years of the LOM. These costs include the following:

- Historical database factor estimated Paste Backfill closure; and
- Process plant and infrastructure closure.

21.1.11 Contingency

Contingency is calculated on a discipline-by-discipline basis, considering items that were quoted, estimated or factored. Contingency is included to cover items which are included in the scope of work as described in this Report, but which cannot be adequately defined at this time due to lack of accurate detailed design information. Contingency also covers uncertainty in the estimated quantities and unit prices for labour, equipment and materials contained within the scope of work.

21.2 Operating Cost Estimate

This section provides information on the estimated operating costs (Opex) of the Project and covers Mining, Processing, Site Services and Administration. The sources of information used to develop the Opex include in-house databases and outside sources particularly for materials, services and consumables. All amounts are in US$, unless specified otherwise.

21.2.1 Summary Opex

The total LOM Opex and average Opex, given as dollars per tonne of material treated, are summarised in Table 21.5.

Table 21.5 – Summary of LOM Total and Average Opex

Area	LOM Total Opex (US$ M)	Average Opex (US$/t of material treated)
Mining	294.8	37.13
Processing and Tailings	175.6	22.11
G&A	99.0	12.46
Total	**569.4**	**71.70**

Numbers may not add due to rounding.

21.2.2 Summary of Personnel Requirements

Table 21.6 presents the estimated personnel requirements for the Project. This workforce comprises staff as well as hourly employees. Supervisory personnel as well as the administration employees will work on a 5 day per week basis.

The hourly workforce at the plant will work on rotation to provide 24 hour per day coverage, 7 days per week. It is assumed that all employees will come from the area.

Table 21.6 – Total Personnel Requirement (Year 2)

Area	Number
Mining	327
Processing	105
Plant Administration, Infrastructure & Tech. Serv.	20
Total Personnel	**452**

Costs for the above personnel are detailed in the following sections.

21.2.3 MINING OPEX

The mining Opex consists of Personnel, Equipment Operation, and Materials and Services. The mining Opex is based on Owner furnished equipment, materials and mine operators and based on the annual mine plans. Table 21.7 presents the unit rates that were applied to the tonnages for each period of the mine plan to arrive at the total expenditures for the mine operations.

Table 21.7 – Summary of Mining LOM Total and Average Opex

Description	LOM Total Opex (US$ M)	Average Opex (US$/t of conc.)
Labour	82.0	10.32
Equipment Operation	128.5	16.19
Materials and Services	84.3	10.62
Total	**294.8**	**37.13**

Numbers may not add due to rounding.

21.2.4 PROCESSING OPEX

For a typical year, the estimated initial material processing annual Opex for the plant production is summarised in Table 21.8 which shows the breakdown by six major components: Labour, Electrical Power, Reagents and Consumables, Laboratory, Maintenance, and Miscellaneous. These costs are derived from supplier information and / or DRA experience.

Table 21.8 – Summary of Processing LOM Total and Average Opex

Description	LOM Total Opex (US$ M)	Average Opex (US$/t of Material Treated)
Labour	19.6	2.47
Power	62.0	7.81
Reagents and Consumables	76.5	9.63
Laboratory	1.4	0.18
Maintenance	2.7	0.34
Miscellaneous	13.4	1.68
Total	**175.6**	**22.11**

Numbers may not add due to rounding.

21.2.4.1 Electrical Power Costs

Electrical power is required for Process Plant equipment (crushers, mills, conveyors, screens, pumps, agitators, screening, paste backfill, filtration, air, water, etc.). The unit cost of electricity was established at $0.143 per kWh. The annual operational electrical energy consumption is estimated at 45,563,148 kWh which results in a unit energy consumption rate of 54.78 kWh per tonne.

21.2.4.2 Grinding Media

The grinding mills will need regular addition of balls to replace worn media and exercise proper grinding action on the material. Media consumption is estimated based on steel consumption observed in similar operations and the abrasion indices and power consumption. SAG mill grinding balls are added by an automated system to reduce the grinding ball consumption. The SAG mill media consumption rate was estimated at 0.65 kg of steel per tonne treated. This is low by industry standards and is a consequence of the relatively low abrasion index and average material hardness recorded for the samples tested. Approximately 552 t of grinding balls are required for mill operation annually. The total cost for grinding media is $828,750 / a or $0.98 per tonne of material treated.

21.2.4.3 Reagent Consumption Costs

Reagents consist of flotation reagents (eg: MIBC, PAX) and flocculants (for thickeners). The total cost of the reagents for flotation is $1,851,522 or $2.178 per tonne of material treated.

Cement is required for the paste backfill operation. Total cement requirement is 16,745 tpa based on a 5% addition rate and represents $2,846,650 annually or $3.35 per tonne of material treated.

21.2.4.4 Consumables Costs

The consumptions and costs for crusher liners, screen deck panels, grinding mill liners, flotation cell wear parts, pump wear parts, filter cloths, etc. for different equipment were obtained from equipment suppliers and from experience with similar operations. The costs for diesel fuel for the plant mobile equipment and TSF dozer are also included. The costs of consumables and wear parts are estimated at $2,661,360 per year or $3.13 per tonne of material treated.

21.2.5 Administration And Technical Services Costs

The G&A costs reflect personnel related to Administration, Accounting, Purchasing, Stores and Human Resources, as well as Material and Technical Services and Community Relations. The Opex summary for G&A is estimated at $99 M per year or $12.46 per tonne of material treated.

22 ECONOMIC ANALYSIS

22.1 General

The economic analysis of the Project is preliminary in nature and is based on inferred mineral resources, which are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorised as mineral reserves. As a result, there is no certainty that the 2024 PEA will be realised. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The results of the economic analysis are forward-looking information under Canadian securities law. These results rely on inputs that are subject to known and unknown risks, uncertainties, and other factors, which may cause actual results to differ from those presented here.

The economic analysis is based on the discounted cash flow (DCF) method on a pre-tax and after-tax basis at a discount rate of 5%. For the purposes of the evaluation, it is assumed that the operations are established within a single corporate entity. The Project has been evaluated on an unlevered, all-equity basis.

The production schedule used in this analysis is based on the LOM Production Plan and the Concentrate Production Schedule outlined in Sections 16 and 17, respectively. The capital and operating costs are taken from the estimates detailed in Section 21.

All costs and pricing are in Q1 2024 US dollars. No provision is made for the effects of inflation in this analysis.

22.2 Forward Looking Information

The results of the economic analyses discussed in this section represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here. Forward-looking information includes assumptions and estimations of:

- Price of gold;
- Amount of mineralised material and material grade;
- Proposed mine production plan;
- Mining dilution and mining recovery;
- Recovery rates of gold in the processing plant;
- Ability of plant, equipment, processes to operate as anticipated;
- Sustaining and operating costs;
- Environmental, social, and licensing risks;

- Taxation policy and tax rate;
- Royalty agreements;
- Cost inflation;
- Geotechnical or hydrogeological considerations during mining;
- Ability to maintain social license to operate;
- Unrecognised environmental risks;
- Closure costs; and
- Unforeseen reclamation expenses.

22.3 Economic Criteria

22.3.1 Metal Price

Project revenues consist of gold concentrate sales only. No other product is considered as part of the economic analysis. A gold price of US$1,700 per oz of gold is used for the economic analysis throughout the LOM based on the price projection adopted by DPM.

22.3.2 Commercial Terms and Freight

The commercial terms for the sale of gold concentrates and estimates for concentrate freight costs used in this analysis were provided by DPM, as discussed in Section 19. A summary of the key terms for the gravity and flotation concentrates is provided in Table 22.1. Estimates of the cost of concentrate freight are also provided in this table.

Transport losses of 0.2% on a dry metric tonne basis are applied to each concentrate.

Table 22.1 – Commercial Terms for the Sale of Gold Concentrates

Description	Unit	Gravity Gold Concentrate	Flotation Gold Concentrate
Treatment Charge	$/dmt	170	150
Refining Cost (Gold)	$/oz	6.0	7.5
Penalty	$/dmt	0	0
Gold Payable	%	99.8	97.5[1]
Freight, Inland	$/dmt	132	51.3
Freight, Ocean	$/wmt	75	0

Note:
1. Gold payability in flotation concentrate is subject to a -1 g/dmt minimum deduction if gold grade is less than 40 g/t.

22.3.3 MINERAL ROYALTIES

The Government of Serbia imposes a mineral royalty fee of 5% on the net revenue value.

22.3.4 TAXES

22.3.4.1 Income Tax

The standard corporate tax rate in Serbia is 15%.

Current legislation allows for relief from income tax for large investments for a maximum, uninterrupted period of 10 years subject to the fulfilment of certain conditions, including:

- Investment of more than 1 billion Serbian Dinar (RSD) or US$9.1 M (at an RSD:US$ exchange rate of 0.0091) in fixed assets for use in registered business activities.
- Employment of 100 new employees for an indefinite time period during period of the tax relief.

DRA has been advised by DPM that the Project is eligible for this tax relief and that the tax relief will be declared by DPM at the start of production. The effective income tax rate applied is 0% over the 10-year life of the Project.

22.3.4.2 Value Added Tax

Value Added Tax (VAT) is levied at a rate of 20% on the supplies of goods/services in Serbia and on the importation of goods. Certain items are exempt from VAT based on the provisions of the Law on VAT.

Input VAT can be claimed or used as a credit against future VAT liabilities within a period of five (5) years provided that certain conditions are met. It is assumed that VAT and VAT returns will be reconciled on a monthly basis, except in the case of Closure Cost VAT, which is not claimed.

22.4 Base Case Cash Flow Analysis and Economic Results

At the assumed gold price of $1,700 per ounce, the results of the economic analysis indicate a positive pre-tax NPV of $588.2 M at a discount rate of 5% using a mid-year adjustment for cash flows. The pre-tax Internal Rate of Return (IRR) is 33.0% and the payback period is 2.4 years. Owing to the income tax relief applicable to the Project, the after-tax metrics are the same as the pre-tax metrics, as shown in Table 22.2.

Table 22.2 – Economic Results Summary

Description	Unit	Pre-Tax	After-Tax
Free Cash Flow (LOM) [2]	US$ M	891.2	891.2
NPV @ 5%	US$ M	588.2	588.2
IRR	%	33.0	33.0
Payback Period	Years	2.4	2.4
Numbers may not add due to rounding.			

These results are based on the assumptions described in Section 22.3. The key technical and cost inputs are summarised in Table 22.3. An overview of the production schedule and estimated cash flows of the Project is presented in Table 22.4.

Table 22.3 – Key Technical Assumptions and Cost Inputs

Description	Unit	Value
Gold Price	US$ per oz	1,700
Government Royalty (NSR)	%	5.0
Mineable Mineral Resource	Mt	7.9
Average Grade Mined (LOM)	g/t	5.68
Annual Throughput	tpa	850,000
Average Grade Processed (LOM)	g/t	5.68
Average Metallurgical Recovery	%	88.8
Mine Life	years	10
Total Gold Produced (LOM)	M oz	1.3
LOM Gold Payable	%	98.4
Average Annual Gold Production (LOM)	oz	129,000

2 All-in sustaining costs and free cash flow are non-GAAP financial measures or ratios and have no standardised meaning under IFRS Accounting Standards ("IFRS") and may not be comparable to similar measures used by other issuers. As the Project is not in production, DPM does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS.

Description	Unit	Value
Average Annual Gold Production (first five years)	oz	164,000
Capital Cost Estimate		
Initial Capital	US$ M	381
Sustaining Capital (LOM)	US$ M	83
Closure Costs[1]	US$ M	28
LOM Operating Unit Costs		
Mining	US$ per tonne processed	37
Processing	US$ per tonne processed	17
Filtered Tailings and Paste Fill	US$ per tonne processed	5
General & Administrative	US$ per tonne processed	13
Total Opex	*US$ per tonne processed*	*72*
LOM Average All-in Sustaining Cost [3]	US$ per oz gold	715

Numbers may not add due to rounding.
1. Closure costs do not include the non-recoverable VAT of approximately $3 M.

[3] All-in sustaining costs and free cash flow are non-GAAP financial measures or ratios and have no standardised meaning under IFRS Accounting Standards ("IFRS") and may not be comparable to similar measures used by other issuers. As the Project is not in production, DPM does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS.

Table 22.4 – Summary of Production Schedule and Cash Flows

Period #			-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Period			FY-2	FY-1	FY1	FY2	FY3	FY4	FY5	FY6	FY7	FY8	FY9	FY10	FY11	FY12
PRODUCTION SCHEDULE		*(Tot. / Avg.)*														
Feed																
Tonnes Milled	*(ktonnes)*	**7941**	-	-	615	860	853	853	853	853	850	857	857	491	-	-
Feed Grade - Gold	*(g/t)*	**5.68**	-	-	6.67	6.49	7.55	7.36	6.31	5.77	4.36	3.63	4.34	3.74	-	-
Contained Metal - Gold	*(koz)*	**1449.1**	-	-	132	179	207	202	173	158	119	100	120	59	-	-
Recovery Schedule																
Gold Recovery to Gravity Concentrate	*(%)*	**43.3**	-	-	44.4	44.2	45.2	45.0	44.0	43.5	41.8	40.8	41.8	40.9	-	-
Gold Recovery to Flotation Concentrate	*(%)*	**45.5**	-	-	44.5	44.7	43.9	44.0	44.8	45.2	46.6	47.4	46.6	47.3	-	-
Gravity Concentrate Production																
Concentrate Grade - Gold	*(g/t)*	**859**	-	-	1,000	973	1,133	1,104	947	866	653	544	651	561	-	-
Contained Metal - Gold	*(koz)*	**632**	-	-	59	79	94	91	76	69	50	41	50	24	-	-
Concentrate Tonnage Produced	*(ktonnes)*	**22.9**	-	-	1.8	2.5	2.6	2.6	2.5	2.5	2.4	2.3	2.4	1.3	-	-
Flotation Concentrate Production																
Concentrate Grade - Gold	*(g/t)*	**55**	-	-	66	64	76	74	62	56	41	34	41	35	-	-
Contained Metal - Gold	*(koz)*	**654**	-	-	59	80	91	89	78	72	55	47	56	28	-	-
Concentrate Tonnage Produced	*(ktonnes)*	**372.7**	-	-	27.8	39.1	37.3	37.5	39.1	39.9	41.8	43.3	42.2	24.7	-	-
Total Gold Production																
Contained Metal - Gold (Total)	*(koz)*	**1286**	-	-	117	159	184	180	154	140	105	88	106	52	-	-
Payable Metal - Gold (Total)	*(koz)*	**1266**	-	-	115	157	182	177	151	138	104	86	104	51	-	-
PRE-TAX CASH FLOW		*(Tot. / Avg.)*														
Net Revenue	*($'000s)*	**$2,059,519**	-	-	$189,250	$256,925	$299,062	$291,205	$247,586	$225,032	$165,977	$136,812	$166,622	$81,048	-	-
Less: Total Operating Costs	*($'000s)*	**($569,428)**	-	-	($50,333)	($65,603)	($63,310)	($60,491)	($59,815)	($60,161)	($57,675)	($58,942)	($56,642)	($36,454)	-	-
Less: Property Taxes	*($'000s)*	**($598)**	($46)	($46)	($46)	($46)	($46)	($46)	($46)	($46)	($46)	($46)	($46)	($46)	($46)	-
Less: Government Royalties	*($'000s)*	**($102,976)**	-	-	($9,463)	($12,846)	($14,953)	($14,560)	($12,379)	($11,252)	($8,299)	($6,841)	($8,331)	($4,052)	-	-
Less: VAT Refund Account		-	($4,509)	($3,006)	$2,370	$2,636	$130	($83)	$222	($8)	$91	$44	$82	$2,030	-	-
Operating Earnings	*($'000s)*	**$1,386,516**	($4,555)	($3,052)	$131,779	$181,065	$220,882	$216,025	$175,567	$153,565	$100,047	$71,027	$101,685	$42,526	($46)	-
Capital Expenditures																
Development Capital exc. Capitalized Amounts	*($'000s)*	**($380,939)**	($114,282)	($190,470)	($76,188)	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	*($'000s)*	**($83,270)**	-	-	($17,897)	($8,355)	($6,491)	($16,014)	($8,399)	($7,673)	($8,760)	($3,475)	($3,319)	($2,885)	-	-
Closure Capital	*($'000s)*	**($28,385)**	-	-	-	-	-	-	-	-	-	-	-	-	($28,385)	-
Less: VAT Payable after Credit Applied	*($'000s)*	**($2,725)**	-	-	-	-	-	-	-	-	-	-	-	-	($2,725)	-
Changes in Working Capital	*($'000s)*	**($0)**	-	-	($27,587)	($10,154)	($7,084)	$1,098	$7,224	$3,631	$9,538	$4,881	($4,994)	$23,446	-	-
Pre-Tax Cash Flow	*($'000s)*	**$891,197**	**($118,836)**	**($193,521)**	**$10,107**	**$162,556**	**$207,307**	**$201,109**	**$174,392**	**$149,523**	**$100,825**	**$72,433**	**$93,372**	**$63,087**	**($31,156)**	-
Discounted Pre-Tax Cash Flow	*($'000s)*	**$588,236**	**($115,972)**	**($179,864)**	**$8,946**	**$137,038**	**$166,442**	**$153,777**	**$126,998**	**$103,702**	**$66,597**	**$45,566**	**$55,941**	**$35,997**	**($16,931)**	-
Pre-Tax IRR	*(%)*	**33.0%**														
Payback Period	*(years)*	**2.39**														
AFTER-TAX CASH FLOW		*(Tot. / Avg.)*														
Pre-Tax Cash Flow	*($'000s)*	**$891,197**	($118,836)	($193,521)	$10,107	$162,556	$207,307	$201,109	$174,392	$149,523	$100,825	$72,433	$93,372	$63,087	($31,156)	-
Less: Income Tax Paid	*($'000s)*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
After-Tax Cash Flow	*($'000s)*	**$891,197**	**($118,836)**	**($193,521)**	**$10,107**	**$162,556**	**$207,307**	**$201,109**	**$174,392**	**$149,523**	**$100,825**	**$72,433**	**$93,372**	**$63,087**	**($31,156)**	-
Discounted After-Tax Cash Flow	*($'000s)*	**$588,236**	**($115,972)**	**($179,864)**	**$8,946**	**$137,038**	**$166,442**	**$153,777**	**$126,998**	**$103,702**	**$66,597**	**$45,566**	**$55,941**	**$35,997**	**($16,931)**	-
After-Tax IRR	*(%)*	**33.0%**														
Payback Period	*(years)*	**2.39**														

Source: DRA, 2024

Closure costs referenced in this table do not include the non-recoverable VAT of approximately $3 M.

22.5 Sensitivity Analysis

A sensitivity analysis was carried out, with the base case described above as a starting point, to assess the impact of changes in the price of gold, total pre-production Capex and Opex on the Project NPV @ 5% and IRR. The impact of each variable is examined individually with an interval of ±20% and increments of 10% applied.

The after-tax results of the sensitivity analysis are shown in Table 22.5 to Table 22.7 and Figures 22.1 and 22.2. The NPV and IRR of the Project are most sensitive to variations in the gold price followed by Capex and Opex. The Project retains a positive NPV at the lower limit of the price interval tested.

Table 22.5 – Economic Metric Sensitivity to Variations in the Gold Price

Au Price	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	888.2	738.2	588.2	438.3	288.3
IRR	%	43.5	38.4	33.0	27.1	20.7
Payback	Years	1.94	2.14	2.39	2.72	3.19

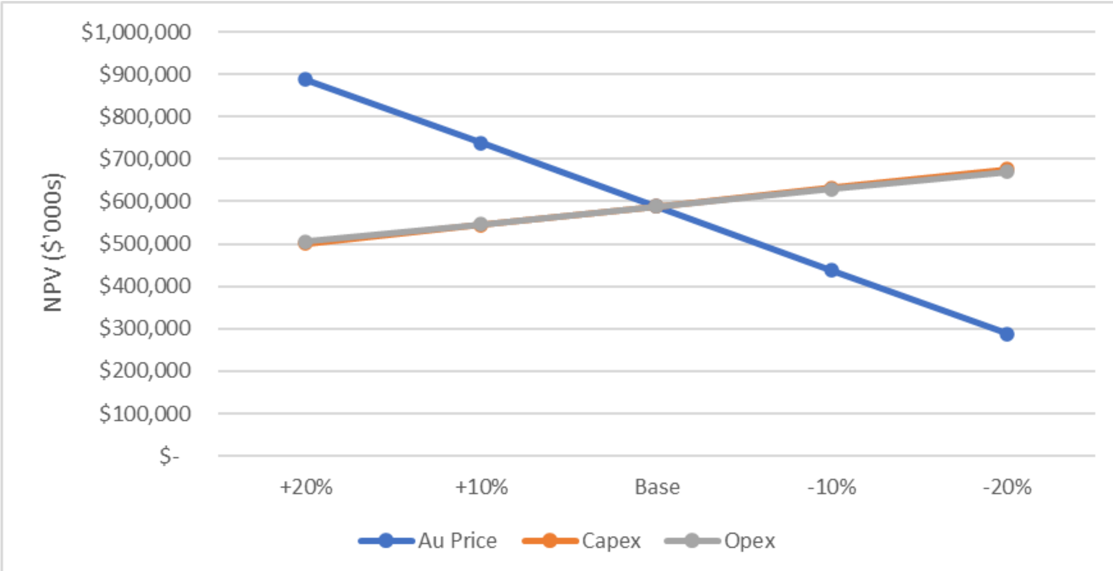
Table 22.6 – Economic Metric Sensitivity to Variations in the Capex

Capex	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	500.8	544.5	588.2	631.9	675.6
IRR	%	26.2	29.3	33.0	37.2	42.2
Payback	Years	2.79	2.59	2.39	2.19	1.99

Table 22.7 – Economic Metric Sensitivity to Variations in the Opex

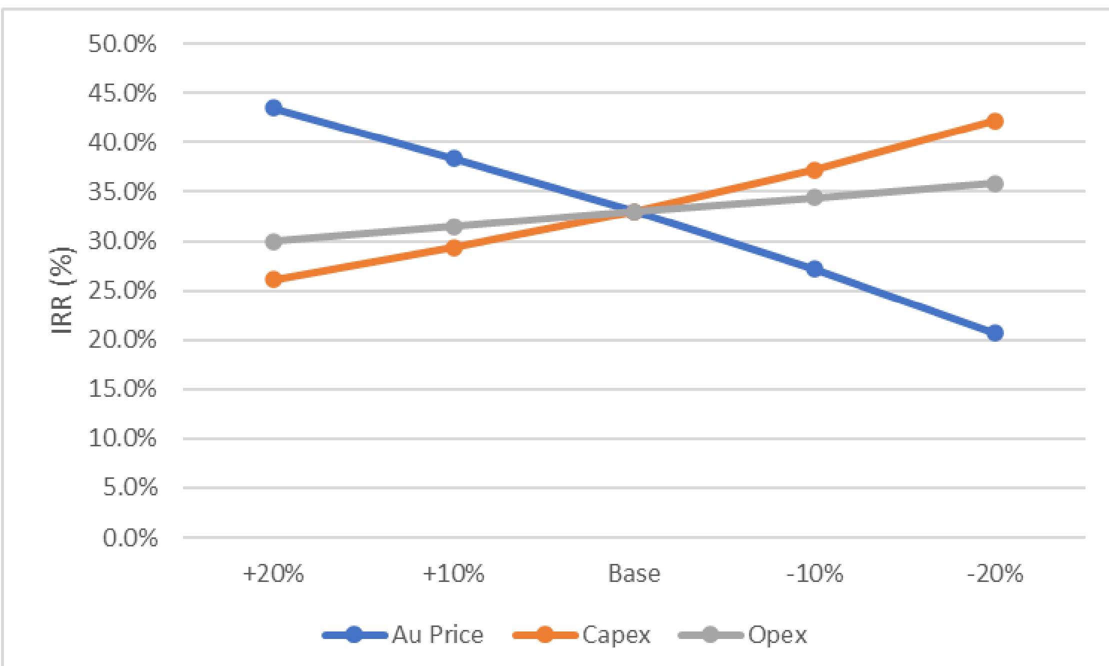
Opex	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	505.9	547.1	588.2	629.4	670.5
IRR	%	30.0	31.5	33.0	34.4	35.9
Payback	Years	2.55	2.46	2.39	2.31	2.24

Figure 22.1 – After Tax NPV 5%: Sensitivity to Capex, Opex and Price



Source: DRA, 2024

Figure 22.2 – After-Tax IRR: Sensitivity to Capex, Opex and Price



Source: DRA, 2024

23 ADJACENT PROPERTIES

23.1 Timok Gold Project (DPM)

The Timok Gold Project, owned by DPM, is a sediment-hosted gold deposit located in the central-eastern region of Serbia and located approximately 3 km northwest of the Project. The Timok Gold Project property includes the Bigar Hill, Korkan, Korkan West, Chocolate and Chocolate South prospects which are hosted on the adjacent Potaj Čuka license, that covers an area of 63.5 km^2. Figure 23.1 shows the location of the Timok Gold Project deposits in relation to the Čoka Rakita Project.

Intensive exploration at Timok commenced in July 2010 following the acquisition of the projects by Avala and subsequently by DPM. A systematic exploration approach has been undertaken with the assembly of the following datasets over the whole area: topography, geological mapping, rock-chip sampling, trenching, channelling, and stream sediment geochemistry. Stream sediment sampling was previously completed over the entire Project area, at a nominal density of one sample per square kilometre. A total of 1,277 drillholes (257,884 m) have been completed at Timok as of May 2020 and include RC and diamond drilling, geotechnical/hydrogeological drilling, and metallurgical test drilling (DPM, 2021).

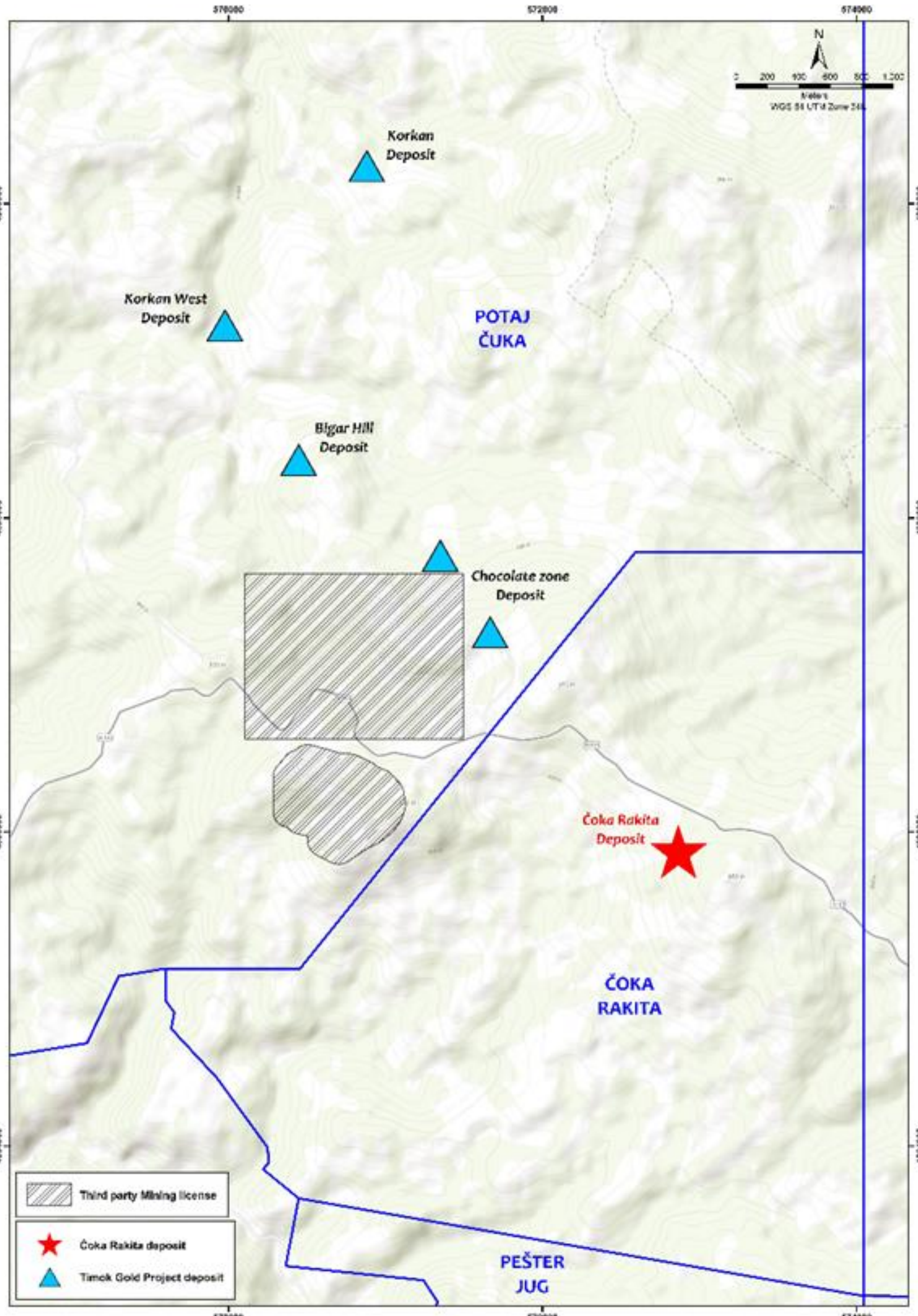
DPM completed a Prefeasibility Mining Study for the Timok Project in 2021 (De Weedt et al., 2021) which is available on SEDAR+ at www.sedarplus.ca. The MRE used as a basis for the study (effective date of 29 May 2020) includes 32.3 Mt of Indicated Mineral Resource with an average grade of 1.27 g/t Au and 1,319 koz of contained gold and 0.9 Mt of Inferred Mineral Resources with an average grade of 1.5 g/t Au and 45 koz of contained gold (DPM, 2021). Mineral Resources were estimated based on conceptual US$1,400/oz gold price pit shells to support RPEEE. Mineral Resources were reported in accordance with CIM definition standards (May 2014).

Probable Mineral Reserves of 19.2 Mt were reported, with an average grade of 1.07 g/t Au and 662 koz of contained gold (effective date of 29 May 2020). The reported Mineral Reserves assumed a conventional open-pit mining scenario (DPM, 2021) and were estimated at a gold price of US$1,250/oz and included modifying factors related to mining cost, and dilution and recovery, process recoveries and costs, G&A, royalties, and rehabilitation costs. A marginal cut-off of 0.21 g/t Au was used for the Oxide material and 0.24 g/t for the Transitional material for all deposits. Mineral Reserves were also reported in accordance with CIM definition standards.

Given the high-grade gold potential of the Čoka Rakita Project, DPM has paused further FS work on the Timok Gold Project to focus on Čoka Rakita.

The QP has been unable to verify the scientific and technical information disclosed above on the Timok Gold Project and this information is not necessarily indicative of the mineralisation and resource potential of the Čoka Rakita Project that is subject of this Report.

Figure 23.1 – Schematic Map Showing Timok Gold Project Deposits in Relation to Čoka Rakita Project



Source: DPM, 2023

24 OTHER RELEVANT INFORMATION

There is no other relevant information on the Čoka Rakita Project known to the QP that would make this Report more understandable or if undisclosed would make this Report misleading.

25 INTERPRETATION AND CONCLUSIONS

25.1 Geology and Mineral Resource Estimate

Gold-rich skarn mineralisation is hosted within carbonate-rich sandstones and conglomerates, located on the hanging wall of a sill-like body and abutting a monzonite intrusive body to the west. The mineralisation forms a shallow-dipping tabular mineralised body located between 250 m and 450 m below surface, measuring 650 m long, up to 350 m wide, and with variable thickness from less than 20 m in the margins to more than 100 m in the core of the mineralised zone. Coarse gold is often observed in areas of intense retrograde skarn alternation and is found mainly in proximity to syn-mineral diorites within the higher-grade core of the deposit. The current MRE has been conducted on the portion of the Project where gold-rich skarn mineralisation occurs.

The QP (Ms. Maria O'Connor, MAIG) conducted a personal inspection of the Project on October 3 and 4, 2023 and is of the opinion that the data used and described in this Report is adequate for the purposes of mineral resource estimation of the Project. The QP reviewed the policies and procedures for sample methods, analyses, and transportation, as supplied by DPM and they were found to be in line with CIM exploration best practice guidelines and industry best practice.

The QP is satisfied that the relevant procedures have been followed consistently, all laboratories used for analyses are adequately certified, and are independent of DPM, and that the standards used as part of the QA/QC routine adequately reflect the characteristics of the mineralisation.

The drillhole database was handed over as of 16 November 2023. A total of 173 drillholes totalling 80,723 m were included in the estimation of the MRE. The current drillhole spacing within the mineralised domains is approximately 30 m x 30 m in the core of the system, with an up to 60 m x 60 m grid on the periphery. Gold grades within skarn domains have been determined systematically using a screen fire assaying technique, which is preferred for mineralisation with coarse gold. Grade capping was applied to composites to limit the influence of anomalously high-grade values, resulting in a cut of metal of approximately 30%.

Mineral resource domains were created within volumes of moderate to intense skarn alteration and guided by grade composites generated at a 1 g/t Au cut-off value. Detailed lithology and structural models were developed and used to constrain domain extents, as well as to incorporate post-mineralisation diorite sills which cut across the mineralisation. Block grade estimates have been undertaken for gold, silver, (which are reported here) and copper, sulphur and arsenic (which are used for geometallurgical characterisation) using Ordinary Kriging at a 10 mE x 10 mN x 10 mZ parent block size with sub-celling to honour domain volumes.

A breakeven cut-off value of 2 g/t Au and a minimum width constraint of 5.0 m x 5.0 m x 2.5 m was used to define optimised mineable shapes using Datamine's MSO. These shapes were subsequently smoothed and used to constrain continuous zones of mineralisation for reporting the final Mineral Resource statement.

The application of MSO shapes at the MRE stage provides a robust estimate for the purposes of a future PFS for the Project, and a higher confidence in the potential for the conversion of Mineral Resources into mineable tonnes and grades for the purposes of a mine plan for any future PFS in the next phase of work. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. The MSO shapes have been used to ensure the Mineral Resources demonstrate RPEEE.

Material within the reporting MSO constraints (smoothed) was classified as Inferred Mineral Resources according to Mineral Resource confidence categories defined in the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). Data quality and quantity, geological and grade continuity, and confidence in the grade, density and RPEEE criteria were considered when classifying the MRE. Given the relatively continuous and stratified mineralisation style at Čoka Rakita, the QP has reason to expect that the majority of Inferred Mineral Resources could be upgraded to Indicated Mineral Resources with additional infill drilling.

The qualitative risk assessment is presented in Table 14.24. The overall risk to the Čoka Rakita MRE is reflected in the current resource classification as Inferred Mineral Resources and is considered moderate, which is consistent with the early-stage nature of the Project.

25.1 Mineral Processing and Metallurgical Testwork

Testing demonstrated the amenability of the samples to treatment through the grind-float flowsheet and provided adequate data for the sizing of all major equipment for a PEA.

In DRA's opinion:

- Metallurgical testwork completed to date is appropriate to establish suitable processing methods for the Čoka Rakita resource;
- Metallurgical testing data supports the metal recovery estimates contained in the LOM plans and metal recovery schedules;
- Samples used to generate the metallurgical data have been representative, and support the estimates of future performance;
- Testwork has been done by reputable laboratories to typical industry standards; and
- DRA is not aware of any processing factors or deleterious elements that could have a significant impact on potential economic extraction.

25.2 Mining Methods

25.2.1 Underground Geotechnical

The underground geotechnical data available for the Project was limited. The rock mass qualities that were used in the stope stability and dilution assessments, ground support design and temporary pillar stability review were based on DPM's knowledge of the property, reviews of limited geotechnical drilling and information from the Timok property, where appropriate. From these reviews the rock mass qualities were generally defined as Good quality (Q'>20) to Fair, with limited sections of Poor (Q'<4) quality rock mass. An empirical stope stability assessment was completed and a longitudinal stope configuration of 20 m high x 20 m wide x 15 m strike length was considered stable for Good and Fair rock mass quality. In areas where Poor quality rock masses are encountered, additional ground support will be required in the back and hanging wall. The support systems recommended were based on DPM's support systems at the Chelopech Mine, except for the addition of resin grouted-rebar in permanent excavations. All main mine infrastructure (spiral ramp and access drifts) is designed in the hangingwall and it was flagged that in the widest areas of the mineralised material zones (80 to 105 m wide), the spiral ramp had the potential to be impacted by mining induces stresses due to the close proximity to the mineralised material zones.

Based on the underground assessments completed the following concepts should be reviewed in the next stage of the Project:

- Define the stable distances of spiral ramp in proximity to the zone of mineralised material to minimise impacts of mining induced stresses;
- Review placement of spiral ramp and other mine related infrastructure in the footwall versus the hanging wall; and
- Review different mining methods in the widest part of the zones of mineralised material.

25.2.2 Hydrogeology

No site-specific hydrogeological information from the Čoka Rakita Project is currently available. The summary of groundwater conditions prepared in support of the PEA is based on the review of hydrogeological studies obtained from the Timok project located 3 km northwest of the Čoka Rakita Project area. This information provides a basic understanding of the groundwater conditions expected at the Čoka Rakita Project site. However, hydrogeological information specific to the Čoka Rakita Project site will need to be confirmed during the planned geotechnical and hydrogeological field investigations.

25.2.3 Mining Methods

- Developing the mine from the hanging wall side of the deposit is the correct approach, given the current understanding of the ground conditions.

- Sublevel stoping is an appropriate method for mining the Čoka Rakita deposit.
- The underground infrastructure, mine services, and fixed equipment planned for the Čoka Rakita Mine are suitably designed and scaled for the mining operations.
- The number of units in the planned equipment fleet and their types, makes, and models are appropriate for the projected production rate, mining method, and development requirements at Čoka Rakita.
- The proposed personnel structure is appropriate for the requirements of an underground mining operation of Čoka Rakita's scale.
- DPM is investigating solutions to adapt the proposed make and model of development jumbo for installing ground support. The unit's fixed feed for 4,305 mm drifter rods is too long for this purpose, and the hole drilled with the optional telescopic feed is too short for the Split Sets and resin rebar specified in the mine design parameters.

25.2.4 ELECTRICAL AND COMMUNICATIONS

The scope of the PEA was limited to underground loads and power distribution. The surface power connections and capacity will be evaluated separately. The 10 kV distribution voltage proposed for underground power distribution is appropriate for the total expected demand and distances. Reasonably sized cables for the ring main can be used to deliver the maximum power demand.

The utilisation voltage will be further evaluated in the next engineering phase to optimise cable sizes with equipment cost and availability. Standard nominal voltages like 400 V and 690 V make it easier to find equipment suppliers that can offer their standard products at minimal lead times and costs. The 1000 V level that was chosen for the study is ideal for large mining equipment that operates at great distances from the level substations. All the fixed loads, like pump stations and vent fans, are located near the level substations, so they would not benefit from the reduced cable sizes that 1000 V allows.

The communications network will be a ring topology with fibre optic cable routing along the same path as the power. Every level will have a network connection that suits the needs of the personnel, equipment and monitoring.

25.2.5 MINE PASTE BACKFILL

The paste backfill underground distribution system design is typically driven by the results obtained from laboratory testing on the Mill tailings, which determines the paste characteristics. In particular, the testing determines the expected pipeline friction losses in the distribution system, establishing the anticipated pipeline pressures and construction materials. The pipeline friction losses also serve in the design of the paste pumps required at the paste plant.

In this PEA assessment, laboratory testing on paste characteristics has not yet been done; therefore, WSP's database of results on tailings with similar characteristics was used to determine the distribution system design. Once laboratory testing determining paste characteristics is completed, the information provided will be used to refine the design in subsequent phases of the Project.

The paste backfill assessment conducted in this PEA portion of the Project provides a relative indication of the system requirements based on the expected level of accuracy.

25.2.6 Mine Dewatering

The water considered in the dewatering assessment originates from the following sources:

- Ground water seepage;
- Paste backfill contained water release and system flushing; and
- Water required to conduct mining operations, including drilling, dust suppression when mucking, and other ancillary water consumption.

Since geotechnical drilling has not yet been conducted to determine the ground water seepage more accurately, the hydrogeological team provided estimates based on data from the DPM Timok Mine. Once the geotechnical drilling is completed for the Project, the water seepage estimates will be revised for the dewatering portion, in subsequent phases of the Project, as required.

Paste backfill contained water release characteristics were estimated based on WSP's database of results on tailings of similar characteristics to determine the contained water release of the deposited paste. Once laboratory testing to determine paste backfill characteristics is completed for the Project, the backfill contained water release estimate will be revised for the dewatering portion in subsequent phases of the Project, as required.

Water leaving the mine in mineralised material and waste, as well as the ventilation system or quantities that are generated from diesel fuel combustion emissions, were not considered in this phase of the Project. If required, these elements will be discussed and included in the dewatering assessment in future phases of the Project.

The dewatering assessment conducted in this PEA provides a relative indication of the system requirements based on the level of accuracy expected.

25.2.7 Paste Backfill

The backfill supply to underground and tailings disposal strategy as well as TSF type and location selection were identified as a primary goal for the PEA. The initially recommended lowest cost Project, using the dry stacking disposal and paste backfill strategy are Options 4a and 4b (Base Case 1). This option showed that utilising Site 1 and Site 3 (Figure 18.6) or equivalent Site 5 (Figure 18.7) appears to be the most cost-effective solution. Concurrently, Option 4b offers "zero" waste rock disposal to surface and potential strength improvements of the paste backfill, which was identified as a benefit to the overall Project.

Applied pressure filtration technology for dewatering Čoka Rakita's tailings, increases the operating and design complexity. An alternative solution for simplifying the paste backfill plant design is to use vacuum filtration technology with or without a filtration aid. Also, depositing thickened tailings with non-segregating properties with thin layers method following by desiccation, drying and excavation would simplify the paste preparation process and reduce capital and operating costs. Transiting from non-segregating thickened tailings to cellular (paddocking) disposal by adding the pressure filtration circuit to partially filter the full plant tailings over the LOM would concurrently reduce the operating cost allowing the entire Project to be built on the sustainable capital cost (time investment basis).

The paste backfill scenario involves mixing a trucked filter cake with process water and binder to produce a paste. Operating costs using the blend FPT/CWR=85/15 could be lower in this scenario as required paste strength could be achieved with lower binder content, but more capital is required to establish the process. Options 4a and 4b (Base Case 1) call for filter cake trucking which will have associated dust suppression requirements. The environmental impact would be mitigated by optimising trucking cycle time and filter cake containment area, which will be investigated in more depth in the next Project phase. The ultimate truck fleet will be established for the known process and paste backfill plant locations as well as for the known road details (i.e. distance, profile etc.).

25.3 Processing and Recovery Methods

The main conclusions arrived at during the ToSs are:

- The flotation circuit will use Jameson flotation cell technology which provides the benefit of reduced Capex through the smaller footprint.
- Three (3) options were identified for further investigation concerning the beneficiation of gravity concentrate.
- The comminution ToS revealed a refurbished SSAG (Option 4) to be the least expensive option to implement, and it is recommended for adoption during future phases of the Project. The inherent assumption is that the construction and decommissioning schedules will align as expected. Option 3 (new SAG mill followed by two new vertical stirred mills) is not recommended, as this option will be significantly more costly to procure and construct.
- Both the flotation and grinding ToSs will be updated during the PFS with updated testwork results and an assessment of the viability of refurbishing and re-using the Ada Tepe equipment.

25.4 Environmental Studies, Permitting and Social or Community Impact

DPM has been actively working in the Čoka Rakita area and adjacent Timok area since 2007, and during this time they have undertaken permitting efforts, environmental studies and community engagement. Environmental and social baseline work is planned including water, noise and air quality monitoring, biodiversity, cultural heritage and social aspects.

Environmental and social factors have been considered in the assessment of the various alternatives, trade-off and options studies in this PEA design. Key environmental and social mitigation features include:

- A relatively small footprint, which comprises TSF, process plant, portal and ROM pad site and paste backfill plant.
- Concurrently using acid generated waste rock for underground backfilling and non-acid generated tailings to surface disposal and paste backfill to mitigate environmental impact.
- Use of liners where appropriate to manage potential acid and metal leaching risks to groundwater and surface water.
- Active water management with contact / non-contact water separation, underground mine dewatering separation, testing and treatment to the required permit limits prior to discharge and managed discharge flow rates.
- Active management of health and safety.
- Full design compliance with the Serbian regulatory requirements, within the context of the EIA permitting framework.
- Critical habitat assessment, if required, and the development of biodiversity management and action plans, as well as ecosystems services assessment.

25.5 Capital and Operating Cost Estimates

The estimated capital and operating costs are reasonable for a PEA level analysis.

25.6 Opportunities

25.6.1 Mineral Processing and Metallurgical Testwork

An opportunity exists to improve overall recoveries if cyanidation is included in the flowsheet. While a decision has been made to avoid cyanide the opportunity remains that could be explored in future.

The grind size of 80% passing 53 µm represents an opportunity to improve costs. Further testing has been proposed to allow for an accurate grind optimisation exercise aimed at evaluating whether the grind could be coarsened to, say, a P_{80} of 75 µm without sacrificing much recovery.

Alternatively, coarse particle flotation could be considered as an initial rougher step to allow rejection of barren non-sulphide tails at a coarse primary grind (>150 µm). Only the concentrate from this step will then need to be ground to 75 µm ahead of a cleaner flotation circuit.

25.6.2 Mining Methods

Implementing tele-remote and automated technology with certain equipment could extend their utilisation during what would otherwise be non-productive time, such as the time between shifts and post-blasting smoke clearance.

Implementing battery electric mining equipment could improve the quality of the underground work environment, reduce the ventilation airflow requirement, and enhance compliance with increasingly stringent exhaust emission regulations.

Evaluating the Mineral Resource with marginal and incremental cut-off grades in addition to the full-cost break-even cut-off grade should increase the size of the potentially viable mineral resource and could contribute positively to the Project cash flows.

Applying the drift and fill method to specific portions of the deposit could be advantageous for mining the more complex or irregular zones and managing challenging ground conditions in the mineralisation or adjacent to the hangingwall.

25.6.3 Recovery Methods

If pebble production is expected to be consistent and low, the SAG discharge screen could be omitted in favour of a trommel screen only. This will reduce the height of the SAG mill and consequently the height of the building.

Additional testing should be conducted to determine conclusively whether Jameson cells will increase gold recovery in the finer particle size range. For the PEA, the recoveries and revenue used are based on conventional cell test results only.

25.7 Risks

The Project team undertook two (2) risk assessment workshops aimed at evaluating potential risks that could affect its success. The initial workshop occurred on November 23, 2023, followed by a second session on December 18, 2023.

The Čoka Rakita Project is subject to a range of risks that must be managed during the Project life. This section identifies and evaluates the key risks associated with the Project, categorising them by their potential consequences and likelihood. It also outlines the proposed control measures to mitigate these risks and presents the residual risk levels after the implementation of these measures.

25.7.1 Risk Categories and Descriptions

The risks associated with the Project are categorised into several key areas, each with specific implications and potential consequences. These include:

- **Geological Risks:**
  - Inaccuracies in mineral resource estimates;
  - Grade variability due to coarse gold presence and geological uncertainty;
  - Structural complexity impacting mineralised material body geometry;
  - Geotechnical conditions may be underestimated.
- **Environmental Risks:**
  - Acid rock drainage (ARD) from waste rock dumps;
  - Unplanned water discharges;
  - Groundwater and surface water contamination;
  - Seepage of process chemicals, hydrocarbons, or leached metals into soil.
- **Permitting Risks:**
  - Not identifying all permitting requirements;
  - Delays in applying for permits;
  - New permits coming into force before production start;
  - Regulator delay;
  - Lack of clarity in regulations, especially around water.
- **Infrastructure Risks:**
  - Power supply reliability;
  - Site road construction and operation;
  - Water management and contact water ponds.
- **Tailings Management Risks:**
  - Dam stability and potential for failure;
  - Environmental contamination from TSF.
- **Operational Risks:**
  - Mining equipment performance and availability;
  - Ground support and stability;
  - Dewatering and water management.

25.7.2 Proposed Control Measures

To address the identified risks, a series of control measures are proposed, which are designed to reduce the likelihood and consequence of each risk to an acceptable level. These measures include:

- **Environmental:**

  Implementing groundwater protection plans, conducting full EIAs, and lining waste rock dumps.

- **Geological:**

  Additional core drilling and geotechnical investigations to enhance the understanding of the deposit and host rock conditions.

- **Operational:**

  Enhancing ground support, improving equipment fleet planning, and implementing fire prevention and response systems.

- **Infrastructure:**

  Advance equipment orders, geotechnical design considerations, and the installation of electrical heat tracing cables to prevent water pipe freezing.

- **Tailings Management:**

  Characterising tailings and designing for closure, including dam stability and spillway sizing.

These control measures are integral to the Project's risk management plan and are subject to ongoing review and refinement as the Project progresses.

25.7.3 Residual Risk Analysis

After the implementation of the proposed control measures, the risk levels were reassessed. Most risks were reduced to moderate or low levels, but some major risks remain, particularly in environmental and operational areas.

25.7.4 Conclusion

The risk assessment workshops identified and assessed significant risks associated with the Project. Through the implementation of a series of control measures, the Project team aims to mitigate these risks to an acceptable level. Ongoing monitoring and further detailed design work will be essential to manage residual risk effectively. The Project team remains committed to ensuring the Project's environmental, social, and economic sustainability.

26 RECOMMENDATIONS

26.1 Proposed Work Program

To ensure the potential viability of the mineral resources, the following activities should be undertaken in the next phase of the Project leading up to the completion of the PFS. These activities as well as their estimated costs are presented in Table 26.1.

Table 26.1 – Estimated Budget for Next Phase

Activities	Estimated Budget (US$ M)
Resource Drilling and Assays	14.6
Geotechnical / Hydrological Drilling and Testwork	2.1
Metallurgical Testwork	0.5
Trade-Off Studies	Included in PFS
Environmental Studies	0.5
Permitting	0.7
PFS Report	4.6
Subtotal	23.0
Contingency	1.3
Total	**24.3**

26.2 Geology

The work programs set out below are part of the next phase of the Project, unless otherwise stated.

26.2.1 Exploration

Much of the focus of modern-day exploration strategies have focused on Cu-Au bearing mineralisation styles, in particular porphyry, high sulphidation as well as sediment-hosted gold type deposits. Skarn type mineralisation has been relatively underexplored for to date. Exploration teams are recommended to focus on re-evaluation of known targets to determine if potential skarn targets have been overlooked.

26.2.2 Drilling

DPM is planning an aggressive drilling program in 2024 to support further technical studies (Table 26.2) and current plans include:

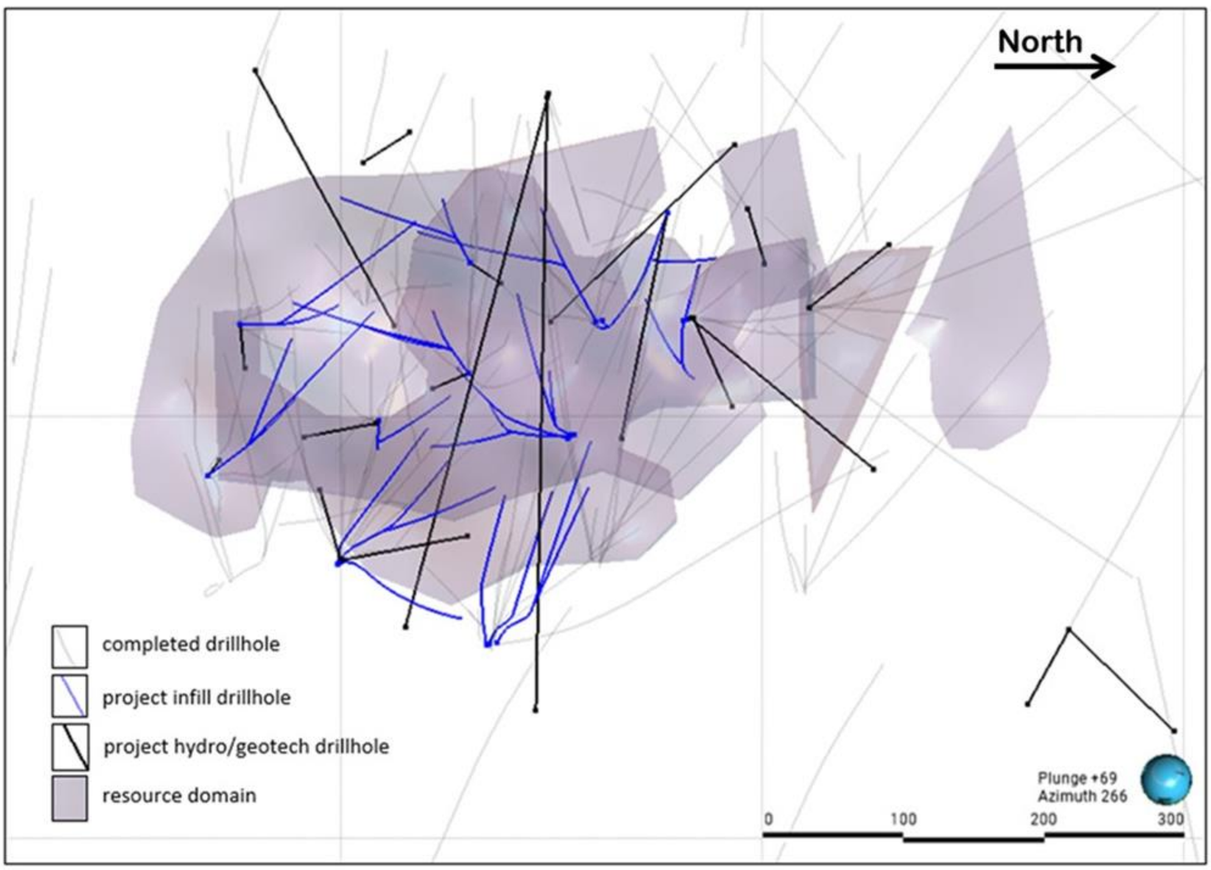
- Approximately 26,000 m of infill drilling to allow conversion of inferred mineral resources into higher resource categories.
- Additionally, 9,000 m of hydrogeological and geotechnical drilling has been budgeted.

- Both drilling campaigns are planned to be complete before embarking on a PFS for the Project.
- Additionally, DPM has plans to complete 55,000 m of additional exploration drilling at existing skarn targets and to test for manto-like copper-gold skarn identified across the Čoka Rakita.

Table 26.2 – Čoka Rakita License – Planned Drilling Metres and Budget

Description	Drilling Category	Planned Metres (m)	Budget (US$ M)
PFS	Infill drilling	26,000	4.7
PFS	Geotechnical/hydrological drilling	9,000	2.1
Exploration	Exploration drilling – 2024 Field Season	55,000	9.9
	Total	**90,000**	**16.7**

Figure 26.1 – Infill and Hydrology/Geotechnical Drilling Plan for 2024 at Čoka Rakita



Source: DPM, 2024

26.2.3 Database

DPM uses a reliable and well-known solution to capture and manage the data (acQuire). However, the database and data management practices are still evolving. To ensure CIM Exploration Best Practice Guidelines and industry best practice are followed, the QP recommends the following:

- There are entries with no main code associated with data logged in the lithology, alteration and mineralisation table, meaning, e.g., missing Lithology 1, Alteration 1, Mineral 1, respectively. Generally, this would lead to a validation error in relational databases, meaning a non-standard approach may have been used.
- The laboratory method (Analysis Suite) should not be combined as compound entries, the method should be captured separately.
- The expected values and standard deviation values should be captured in the database.
- The density measurement method should be included in the database.

A database health check/audit is recommended to provide more in-depth and targeted recommendations.

26.2.4 Assay QA/QC

No analytical umpire samples are available for the mineral resource drilling programs at the Project. DPM procedure is for approximately 5% of all samples exhibiting a gold grade greater than 0.1 g/t Au are sent for umpire analysis to a third-party laboratory to assess the reliability of primary analytical data. The QP understands that umpire samples have been selected and will be assayed during 2024, however, the results were not available at the time of reporting.

DPM should strive to ensure a suite of CRMs are available that match the grade character of the skarn mineralisation. The current suite of CRMs is generally suitable for lower-grade porphyry and sediment-hosted gold type grade tenors. However, higher grades, like as seen within gold-rich skarn deposits, are underrepresented. The QP understands CRMs have been ordered and will be inserted in 2024.

A review of the QA/QC in the context of domaining and focussing on different mineralised grade zones is recommended.

The failed CRMs should be investigated as best practice dictates, although they are not indicative of fatal flaws.

Continued vigilance is required considering the extremely high gold grade values that have been encountered while drilling.

26.2.5 LITHOGEOCHEMISTRY

Based on the review of multi-element data and initial lithogeochemical assessment, the QP recommends the following:

- Continue collecting four-acid ICP-MS multi-element data and SWIR spectral data in tandem as those data are valuable for exploration and geological modelling and will help to model metallurgical and environmental characteristics in the future.
- Consider requesting over-range analysis for Ca >15% at SGS in the future or use pXRF on the assay pulps. Alternatively, DPM could re-analyse only a selection of all assay pulps with over-range Ca data and generate a predictive imputation for the remainder of the dataset.
- Consider routinely analysing all future samples by XRF to not only analyse Ca but also Si, K, Zr, and Ti, to support mineralogical and geometallurgical modelling efforts. The latter can also be supported by routine analysis of total C content (could be combined with LECO S analysis), which in this context, can be useful for modelling of ARD/ML behaviour of waste rock and tailings, but is also useful for general lithogeochemical characterisation.
- Use the initial lithogeochemical classification generated under the supervision of the QP to cross-validate geological logging data and inform 3D geological modelling and resource domaining, as well as the basis for representative sample selection for metallurgical and environmental variability testwork, including mineralogical characterisation, to assess processing response and environmental characteristics of the key units and whether the identified classes can be combined/simplified or need to be refined further.

26.3 Mineral Processing and Metallurgical Testwork

26.3.1 PROPOSED PROGRAM OF TESTING FOR FURTHER PROCESS DEVELOPMENT

- Preparation of bulk samples from which nuggets have been removed to make it easier to compare subsequent test results: Bulk composite samples, which represent each domain within the entire mineral body, should be prepared by first grinding to a coarse grind of 80% passing 150 µm. Afterwards, the entire sample should be processed through a Knelson concentrator, rougher concentrates, and then through a shaking table. The final shaking table gravity concentrate can be used in subsequent tests aimed at supporting the gravity concentrate ToS.

  Upon completion of the gravity test at 150 µm all tailings should be combined, blended, homogenised and then split into sub-samples, which will be used for the subsequent tests described below. This tails sample will be referred to as the "De-nuggeted bulk sample" below. The expectation is that this preparatory step will remove all the very coarse gold nuggets which causes the occasional spikes in gold head grade in smaller sub-samples. The de-nuggeted bulk sub-samples should thus all have a very similar gold grade which allows for more precise and direct comparison of recoveries from downstream tests performed with these sub-samples.

- Grind optimisation series using de-nuggeted bulk sub-samples: A series of five (5) tests involving different P_{80} grinds of say 150 µm (as is), 102 µm, 75 µm, 53 µm and 38 µm to determine the optimum grind. The test procedure should comprise a rougher and 2-stage cleaner flotation. The wide range of grinds and five (5) datapoints should allow for a reliable curve to be fitted to the overall recovery versus grind dataset. From this a decision would be made on which grind size to adopt for all future testing. The gravity gold recovered during de-nugetting should be added when calculating the overall gold recovery at each grind.
- Flotation reagent optimisation tests should be performed using de-nuggeted sub-samples. These tests should be performed with combinations of a range of PAX and A3477 additions.
- A series of tests should be performed, again using the de-nuggeted bulk sub-samples, to evaluate whether regrinding of rougher concentrate is beneficial. For this series the rougher test can be performed at a coarser than optimum grind followed by regrinding of the rougher concentrate ahead of the cleaner steps to three finer grinds.
- The de-nuggeted sample should also be used to repeat the set of tests comparing Jameson cells versus conventional cells.
- An LCT should be performed at the optimum conditions to provide design criteria regarding the dynamic mass balance and recoveries for the full-scale plant.
- Jar mill tests should be performed to accurately estimate grinding costs of vertical stirred mills.
- Sedimentation and filtration testing of the final concentrate including rheology.
- Additionally, sedimentation and filtration testing of a tailings sample may also be considered.

26.3.2 Proposed Program of Testing to Support Gravity Concentrate Treatment ToS

Further processing of concentrates on site should be explored. A sufficiently large bulk sample of gravity concentrate should be produced, blended, homogenised, and split into sub-samples for this program. This program should consist of both intensive cyanidation testing as well as additional gravity separation testing:

- Intensive cyanidation testing was completed during the PEA program at conditions designed to demonstrate its feasibility. The testing during the next phase should focus on optimising conditions by exploring lower cyanide additions and lower liquid to solids ratios. Two series of tests each consisting of three tests should be conducted to test three different cyanide concentrations and three lower liquids to solids ratios (3:1, 2:1 and 1:1).
- Additional gravity separation testing is proposed to generate sufficient data to allow modelling of gravity circuits with multiple cleaning stages aimed at generating a very high-grade product. This testing should employ multiple cleaning steps to generate a complete grade versus mass pull curve. Additionally, EGRG testing of the single composite sample should be done to generate data per size fraction that the vendors can use to model various circuits which can accurately predict product grades and recoveries for different configurations of gravity

concentrators. This information can then be used to conduct a trade-off of the cost of more complex circuits against the benefits gained in terms of the quality of the product produced.

26.3.3 Proposed Program of Variability Testing

A minimum of at least 15 distinct samples representing a good distribution of various locations, depths and lithologies should be processed through an identical series of tests which mimic the proposed flowsheet. As such it would make sense to delay this and only perform it once all the process development testing has been completed so that optimised durations, grind sizes and reagent addition rates can be used in the variability program. The variability program should consist of the following for each of the variability samples:

- Comminution testing: JKTech/SMC parameters, Bond Ball Mill test as well as abrasion index.
- Calibration tests to determine how long each sample should ground to get to the same P_{80} size.
- Grind a sample to the targeted P_{80} then perform a gravity concentration test on each consisting of both a centrifugal as well as a secondary shaking table procedure.
- Rougher flotation test followed by two stages of cleaning and scavenging to mimic the selected flowsheet using the exact same set of optimised conditions. Budget permitting, the cleaning tests could be LCT, otherwise just batch.
- Static sedimentation test on each tailings sample. Dynamic tests would be better but they are more expensive and will require larger sample sizes than a simple batch beaker test.
- Basic filtration tests would also be beneficial, if the tests can be done with small samples.

26.3.4 Additional Recommendations Regarding Testwork

- Given the high gravity gold content and how it affected initial testing all comparative tests should be done on samples that have been de-nuggeted.
- Flotation tailings samples should be stored so that diagnostic leaching can be done, if deemed worthwhile, to determine gold association within the tails samples. This will provide insight into why gold was lost and may help explain some outlier results.
- The presence of talc in the head samples indicates that it may be worthwhile to examine if it can be depressed (using CMC) which could possibly improve flotation concentrate grades.
- The paste backfill test program is described in Section 18.7.

26.4 Mineral Resource Estimate

A close spaced drilling program is recommended to better understand the variability of the mineralisation at short distances. Due to the depth of the mineralisation, drilling from surface would be costly and time consuming, but directional drilling using a mother hole may be effective in gaining drill coverage over a small area of the high-grade core of the deposit. This would be used to inform a variability study to be conducted prior to the next MRE update.

Following the close spaced drilling program, the drill spacing density should be reviewed where the highest risk to the resource is with a view to tightening the drill spacing in that area; this is considered Phase 2 work contingent on the close spaced drill program being completed.

Improved modelling of the late-stage intrusions continues, involving re-logging in association with lithogeochemical analysis, with this work being undertaken to reduce the uncertainty of the precise location and volume of late-stage intrusions, currently representing approximately 4% of the mineralisation volume.

Sensitivity to grade estimation methodology is recommended to assess methods for modelling the higher grades of the mineralisation.

Drilling is continuing at the Project and the MRE will be updated based on new drilling, with the current mineralisation model being tested and interpretations revised as required, at a suitable point.

The qualitative risk assessment is presented in Table 26.3. The overall risk to the Čoka Rakita MRE is reflected in the current resource classification as Inferred Mineral Resources and is considered moderate, which is consistent with the early-stage nature of the Project.

Table 26.3 – Qualitative Risk Assessment

Factor	Risk	Comment
Sample collection, preparation and assaying	Low to Moderate	There are written procedures and data management practices in place. The nature of coarse gold means there is an inherent higher risk relating to and risk associated with sample preparation and analysis, but this is mitigated by the analysis for the vast majority of samples being screen fire assay which requires larger volumes. The majority of the gold is associated with finer fractions, but coarse gold is associated with higher grades.
QA/QC	Moderate	While screen metallics testing is the preferred method for analysing high gold grades in coarse gold environments, the nature of SFA means that direct quality control is less possible than it is for other methodologies. Quality control review has been performed on FA and has indicated no material issues of concern. Insertion of blind standards, duplicates and blanks is recommended.
Geological model	Moderate	Uncertainty in accuracy of location of late-stage intrusives modelled. The fact that core can look very similar in terms of skarnification and intensity of alteration but have different grade character across short distances, is notable.

Factor	Risk	Comment
Mineralisation model	Moderate to High	The nature of coarse gold means there is an inherent uncertainty in its location and grade since it can be missed in half-core sampling, and variability at close ranges can be high. The mineralisation has been constrained within moderately to intensely skarnified S1/S2 material and guided by grade composites generated at 1 g/t Au. It is important to retain the geological basis of the interpretation and not be guided only by grade since level of selectivity can be low in this kind of environment. This risk can be mitigated by using larger diameter core barrels such as HQ or PQ to collect more sample for assay analyses and a better representative sample. This can also be mitigated through a bulk sample using closely spaced PQ cores.
Treatment of outliers (grade caps)	Moderate	The MRE is very sensitive to the choice of grade cap. Given the early stage of the Project and broad drill spacing, a relatively conservative grade cap was applied, which cuts 2% of the data and and 30% of the metal. When data is top cut (at 70 g/t Au for the largest domain), variograms indicate nuggets that are moderate and not extreme, indicating grade continuity is not extremely low and grade variability is not extremely high.
Location of post-mineralisation intrusives	Moderate	The location of post-mineralisation intrusives represents a low volume but precise location is uncertain based on current broad drill spacing.
Grade estimate	Moderate	The grade estimate has been intentionally smoothed to reflect the uncertainty of the location of coarse gold. Sensitivity to grade estimation methodology is recommended to assess methodology for improved modelling of the high-grade tail.
Tonnage estimate	Low	The density estimate is considered low risk. The volume estimate is moderate risk, associated with uncertainty in the mineralisation model but not unreasonably so, considering the stage of resource development and level of classification.
Permitting risk	Low to Moderate	A potential risk to the Project is associated with permitting delays. Such delays caused by potential changes to Serbian regulations to align with EU Law, regulator delay, public challenge to the Spatial Plan or EIA and administrative appeals. Similar risks have been experienced by other private sector mining projects permitted in Serbia.
Overall rating	Moderate	The current MRE carries a moderate level of uncertainty and risk which is reflected in its classification as Inferred Mineral Resources.

26.5 Mining Methods

26.5.1 Mining

1. Conduct further geotechnical investigations to confirm that the footwall of the deposit is unsuitable for locating development and infrastructure. Due to the anticipated unfavourable ground conditions in the footwall, DPM plans to situate the development and underground infrastructure in the hangingwall. This strategy departs from standard mining practice, favouring developing deposits from the footwall side.

2. Conduct a ToS to evaluate alternative methods to handle mineralised material and waste compared to truck haulage. One such possibility is installing a conveyor system in the ventilation decline.
3. Conduct a ToS to compare the planned longitudinal method with transverse sublevel stoping, a more typical approach for mining thick deposits like Čoka Rakita. Mining the mineralisation with transverse stopes would require developing waste drives adjacent to the hangingwall of each sublevel and, from these headings, extending regularly spaced crosscuts across the width of the deposit to the footwall.
4. For future resource evaluations, investigate whether drift and fill would be advantageous for mining specific portions of the deposit. This method provides greater flexibility and selectivity than sublevel stoping for mining the deposit's more complex or irregular zones. Furthermore, it is better suited to managing challenging ground conditions in the mineralisation or adjacent to the hangingwall, as it limits the size of the openings to the width of a drift.
5. For future studies, refine the estimates of the unplanned dilution and mining recovery factors based on the mining method, geotechnical conditions, and backfill properties.
6. For future studies, incorporate passing bays and safety cuddies into the spiral ramp design to enhance operational efficiency and safety.
7. Include a permanent mine rescue chamber in the infrastructure design. This facility would have greater capacity than the portable units and serve as a central underground base for mine rescue operations during an emergency.
8. Consider implementing dedicated escapeway raises in the mine rather than installing ladderways in ventilation raises. A ladderway in a ventilation raise reduces its efficiency for airflow by increasing friction loss. Additionally, the high velocity and pressure of the airflow in a ventilation raise render it less practical and safe as a travel way. A ladderway can be installed in a smaller diameter raisebore than is required for ventilation.
9. Conduct a ToS to evaluate implementation of battery-electric mining equipment as an alternative to diesel-powered units. The ToS would consider various factors such as cost, efficiency, environmental impact, and operational capabilities to determine the most suitable option for mining operations.
10. Investigate opportunities for implementing automated and tele-remote operation of mining equipment at the mine. This technology could be beneficial for extending equipment utilisation during what would otherwise be non-productive periods, such as shift changes and post-blasting smoke clearance. Potential applications for this technology include:
    - Automated and tele-remote mucking of longhole stopes;
    - Automated longhole drilling; and
    - Tele-remote mucking of development rounds between shifts.

11. Explore alternatives to traditional underground mining trucks as a strategy to reduce capital and operating costs for haulage equipment. These alternatives, which have been proven effective in underground mines, include:
    - Articulated haulers of the type utilised at construction sites;
    - Highway-type dump trucks or tippers; and
    - Tippers retrofitted for mining.
12. For future resource evaluations, assess the potential benefits of engaging contractors for specific mine development and infrastructure construction tasks, considering the varying intensity of these activities throughout the mine development program.
13. For future studies, verify that the portal entrances of the twin declines, currently planned with a 30 m centre-line spacing, are sufficiently separated from each other such that the intake of the ventilation decline does not draw return air exhausted from the haulage decline.
14. In future revisions of the Pre-Production Schedule, WSP recommends enhancing the clarity and detail of the activities required before the initiation of development of the twin declines, including the following aspects:
    - Timelines for ordering and delivery of critical equipment;
    - Strategies for procurement and logistics;
    - Plans for the mobilisation of resources and installation at the Project site; and
    - Recruitment and training of personnel.
15. For future resource evaluations, conduct a ToS to assess whether the benefits of utilising a development jumbo for multiple functions outweigh the efficiency gained through using specialised equipment dedicated to scaling and installing ground support. This ToS should consider factors such as operational costs, time efficiency, equipment versatility, operator safety, and overall productivity to determine the most effective approach for the Project.
16. In future studies, redesign the ventilation system, making the haulage decline and spiral ramp the intake airways. This approach enhances safety by ensuring personnel remain in fresh air when exiting the mine during an underground fire. The downcasting airflow may also enhance engine cooling efficiency for mine trucks hauling up-ramp fully loaded.
17. Evaluate potential advantages of engaging a mining contractor to drive the twin declines. The contractor could bring in experienced personnel, whereas, with the self-perform approach, DPM would need to hire, train, organise, and manage new crews who may have less experience. Additionally, if the contractor supplies their own equipment, it could allow extra time for the delivery of DPM's equipment, providing a buffer in the Project schedule against potential delays.

26.5.2 Ventilation

- To prevent the formation of dust, WSP recommends regularly spraying water at the tapping points and installing water spray systems at the twin-decline drift.
- A ToS should be carried out in the PFS phase to determine the feasibility of installing regulators instead of secondary fans at the production levels, which could reduce Capex and Opex.
- During the PFS, a simulation of the distribution of equipment between the various active production levels should be carried out, to provide a better estimate of secondary ventilation.
- A ToS should be carried out during the PFS to reduce the ventilations raises from 3 to 2.
- It would also be necessary to examine whether air cooling and heating are required.
- Further refinement will be carried out using local standards and regulations, which could lead to a significant Capex reduction.
- Based on experience, diesel particulate is the contaminant expected to be the limiting factor for ventilation. WSP suggests that diesel concentrations generated by equipment should be assessed prior to purchase. This will ensure that health and safety standards are met. As a reference, the standards for the different jurisdictions are shown below:
  - European Union (EC) TWA: 50 µg/m³ (European Union, 2019);
  - United States (EC) TWA: 160 µg/m³ (DieselNet, n.d.); and
  - Ontario, Canada (TC) TWA: 120 µg/m³ (Ontario Government, 2023).
- To reduce air demand and therefore electricity costs, WSP suggests a trade-off between diesel and electric equipment.

26.5.3 Mine Backfill

Since the decline is the only access for personnel, mobile equipment, supplies, and mineralised material and waste haulage for the mine, it is recommended that the paste backfill pipeline running from the bottom of the surface borehole to the top of the deposit be located in the exhaust decline to lessen the possibility of impact by mobile equipment.

As well, should there be a blockage or leak in the backfill pipeline, or to conduct the required pipeline maintenance, it will lessen the impact on mining operation.

Since the exhaust decline will be less travelled, it is also recommended that the pipeline be monitored along its length to identify possible blockages or leaks through the use of pressure transducers or cameras in critical areas.

Since paste causes a degree of wear on pipelines, and the pipeline in the decline is used throughout the mine life, a subjective assessment on the anticipated longevity of carbon steel pipe versus more wear resistant materials of construction should be conducted in the PFS phase.

A hydraulic flow model of the entire distribution system should be completed in future Project phases to more clearly define design parameters associated with the underground distribution system.

26.5.4 Mine Dewatering

The recommendations for the mine dewatering pipeline, which runs from the top of the deposit to the decline portal, are virtually the same as those presented for the paste backfill pipeline and are also applicable to the dewatering pipeline. The one exception would be that a standard schedule 40 carbon steel pipeline would be applicable for use rather than more elaborate material of construction.

In discussions late in the assessment, it was brought forward that an HDPE dewatering pipeline in the decline would be preferred to a steel pipeline. Due to the pressure limitations associated with HDPE pipe, this would require a number of sumps equipped with suitable pumps at intervals up the decline. These could be the cuddies already present in the decline. It is recommended to investigate the approach for the use of an HDPE dewatering pipeline in future phases of the Project.

26.5.5 Paste Backfill

An unconfined strength (UCS) testing campaign would be required to identify the ratio between full plant tailings and crushed waste rock as well as the paste backfill recipe that will be used for the detailed paste backfill plant. The strategy as such would aim at to define the operating envelope of the paste backfill plant and to define the cost-effective paste backfill strategy.

Vacuum filtration (disc and belt) testing with and without the use of a filtration aid and determination of the non-segregating thickened tailings properties for Čoka Rakita's tailings are needed. This testing aims to identify the best filtration technology and non-segregating properties of the tailings stream that could simplify the paste backfill process and overall tailings surface disposal strategy.

Lastly, Option 4a/4b (Base Case 1) has the qualities to meet the needs and standards of the PEA study. Concurrently with above-described testing campaign, Option4a/4b (Base Case 1), Option 5, Option 6 and the Base Case 2 (if permitted) have the merits to be studied at the prefeasibility level of engineering. In order to reduce the paste plant footprint area on the surface and to reduce the paste backfill pumping requirements, an underground paste backfill plant installation is also to be studied during the PFS phase.

26.6 Recovery Methods

During the PFS phase, it is recommended to update the process mass and water balances with new testwork data as it becomes available. Any new information with regard to mining plan, and other new data should also be incorporated along with up-to-date data provided by equipment suppliers.

26.7 Project Infrastructure

During the PFS phase, the following areas are recommended to be investigated further:

- Additional exploration and testing to complete detailed designs in the site infrastructure areas where no direct exploration was done.
- Conduct a detailed geotechnical field investigation study to support the Project facilities locations and infrastructure design, with drilling completed specifically in the proposed building areas to support foundation recommendations and analysis.
- Perform ToSs to determine optimised systems for buildings and equipment foundations.
- Optimise layout by reviewing options to move the mine portal closer to SAG Mill Feed Coarse Material Bin and potentially move the paste backfill facility closer to portal or move underground, to eliminate a terrace and reduce the Project footprint.
- Discuss closure plan and use of infrastructure after mine closure with the local communities.
- Develop a plan on development and permitting required for power utility infrastructure.
- Complete the geohydrological studies to determine stability of the water supply.

26.7.1 Tailings Storage Facility and Water Management

During the PFS phase, the follow areas are recommended to be investigated further:

- Conduct geotechnical investigations to:
  - Characterise dam foundations;
  - Identify borrow areas; and
  - Confirm that bedrock is not soluble/karstic.
- Conduct hydrogeological investigations of the TSF and surrounding areas to characterise the depth to groundwater and groundwater conditions that could impact the design.
- Perform a geohazard assessment to understand the risk of landslides.
- Conduct consolidated-drained and consolidated-undrained triaxial shear testing of tailings samples to support design of structural zone and develop strength parameters for stability modelling.
- Conduct a liquefaction potential assessment of tailings to support a dam break analysis.
- Develop management plans for off-spec tailings including inclement weather and filter upset conditions.
- Perform a liner ToS to consider different liner and underdrainage materials.
- Optimise height of TSF considering construction costs, slope stability, and material availability.
- Perform additional geochemical testing and characterisation of tailings and waste rock to:
  - Confirm tailings are non-PAG;

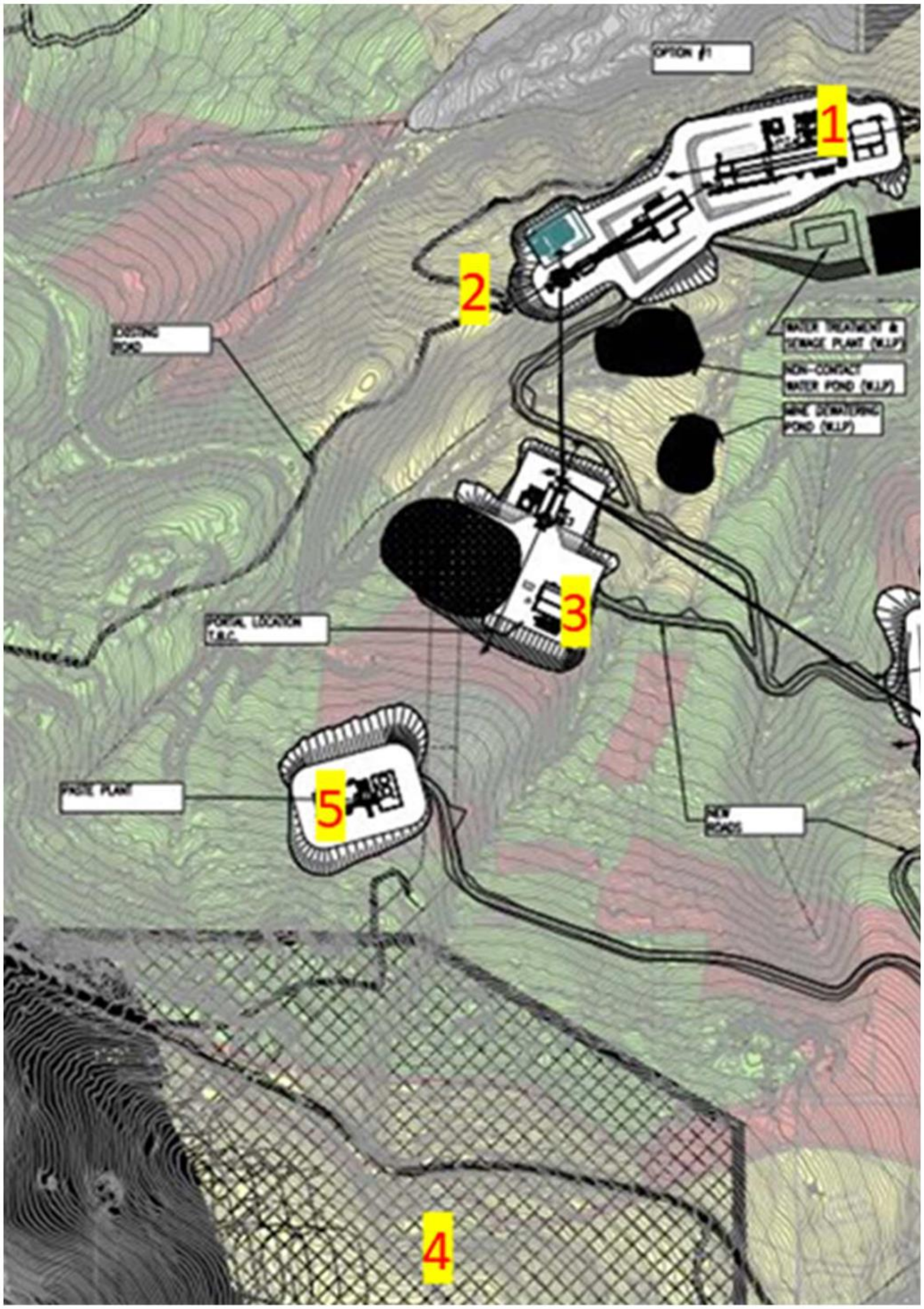
  - Delineate areas and quantities of PAG development rock;
  - Revise or confirm the proposed site water management strategy; and
  - Evaluate potential environmental issues.
- Conduct water quality modelling to identify parameters of concern in contact water streams to be managed within the Project site.
- Conduct preliminary (or concept level) assimilative capacity study for the Lipa River to estimate the maximum analyte concentrations that can be discharged to the receiver without exceeding applicable water quality objectives.
- Identify specific effluent water treatment requirements based on the geochemical characterisation, the water quality modelling and the preliminary assimilative capacity study.
- Implement collection of local meteorological data at an on-site station. Regular maintenance activities should be implemented. Regular checking of collection data should be performed to register reliable and complete records, and implement timely corrective actions in case of equipment malfunctions.
- Confirm precipitation data to be used for design (long-term precipitation records and rainfall storm events).
- Develop streamflow data representative of the existing watercourses across the Project site. The data would support a characterisation of surface runoff more representative of the Project site conditions compared to the current PEA level work that is largely based on regional data or assumptions.
- Gather data from climate change models that predict changes to discrete storm events for the Project area. This would inform the PFS design of water management infrastructure and allow evaluation of climate change vulnerability (especially as it pertains to the sizing and design of water management structures, and closure concepts).
- Refine the numerical groundwater model of the underground mine to obtain better estimates of mine dewatering throughout the projected mine life.
- Refine the water balance model to simulate the water management strategy with greater detail (e.g. continuous simulation through the projected mine life and updating model inputs as new information becomes available).
- Refine sizing of ditches and ponds as new information / data become available (precipitation data, streamflow data, mine dewatering rates from groundwater numerical modelling).
- Perform risk assessment and develop scope of work to address data gaps and intolerable risks.

26.7.2 Paste Backfill Plant Locations

In order to facilitate decision-making process in the PFS, RMS recommends an additional ToS to be carried out, at level of accuracy of ±40%. The ToS will be developed with testing results to better understand alternative dewatering (thickening and filtration) technologies that might be cost-effective for the Čoka Rakita tailings. The ToS outcome will indicate a single option that will be studied at the PFS. Once the ToS is completed, the final option selection and decision-making process will be executed jointly by DPM team and all consultants. The options outlined below were identified and recommended in the PEA and will be screened during the ToS (Figure 26.2 identifies the various locations referenced below):

- Base Case (PEA – Base Case 1) – dewatering circuit (thickening and pressure filters) installed at the mill perimeter (location 1) and paste plant installed at location 5;
- Option 1 (PEA – Base Case 2) – if permitted and approved by DPM. The Base Case 2 offers gravity supply of the paste backfill from location 4 without using paste pump, while dewatering arrangement would be at location 1;
- Option 2 (PEA – Scenario 4) – due to extensive civil earthwork ToS executed during the PEA, this option could be a viable solution for DPM. The entire process would rely on the thickening plant at location 1 and installing the pressure filtration arrangement along with the paste backfill plant at location 3. The same conveyor structure that bridges the ROM stockpile at location 3 and the process plant at location 1 could be utilised for installing an additional filter cake conveyor for delivering the filter cake to TSF. Additionally, instead of the filter cake to be conveyed to the TSF, the paste tailings could be also pumped via the same route to the TSF. The paste backfill would be pumped through the portal area to underground stopes;
- Option 3 (PEA – Option 5) – vacuum disc application with utilising locations 1 and 3;
- Option 4 (PEA – Option 6) – thickened or paste tailings and paddock tailings disposal from location 1 or location 3 and possibility to use excavated tailings for paste backfill preparation at location 3; and
- Option 5 (PEA – Option 7) underground paste backfill plant – that would be located in appropriate location underground with good ground conditions. The same truck fleet would be used for delivering mineralised material to surface and for trucking the filter cake back to the underground paste backfill plant. Concurrently, this option would exclude a borehole from surface to underground and would require less pumping.

Figure 26.2 – Dewatering Arrangement and Paste Backfill Plant Locations (PEA Study)



Source: RMS, 2024

26.8 Market Studies and Contracts

DPM will explore opportunities to enhance the commercial terms of concentrates by engaging with the market sector.

26.9 Environmental Studies, Permitting and Social or Community Impact

The impacts of the Project will be identified definitively during the EIA, and plans to mitigate, manage or offset will be formulated in the ESMP and permitting process. Ultimately, many impacts will be addressed in the Closure period where the larger spatial impacts will be rehabilitated.

It is critical that both technical design of plant, and safety management procedures are implemented in the Construction phase and maintained during the Operations phase. Careful stakeholder communications planning will be required to maintain good community relations and reduce the risk of objections during the permitting process.

Front-loading and initiating permitting processes that have a long lead time, such as the Spatial Plan will be important to maintaining progress on the overall permitting timeline. Ongoing engagement with regulators will be critical, particularly given the limited precedent for permitting private sector mines in Serbia. The Project has already made a commitment to alignment with EU and international requirements.

Maintaining the Project's social license to operate will be of key importance, building upon the existing good relationship with the local community. This will be managed through the Stakeholder Engagement Plan, Communications Strategy and Grievance Mechanism.

Outline scopes for ongoing and future baseline work required for the EIA were presented in Section 20. It is essential that these are completed so that impacts are properly identified, their significance assessed so that appropriate mitigation and management solutions can be identified, such as treated water discharge flow velocity and seasonality. The EIA process will develop an ESMP to map out these mitigating procedures and designs to be implemented as the Project goes forward to Construction.

A commitment to a Project-specific ESMP, with identified plans, procedures and appropriate resources, will encourage the implementation of the ESMP and compliance with permit conditions. Ongoing revision of the Closure Plan throughout FS, Construction, Commissioning and Operation phases, with appropriate and active stakeholder (community and regulator) participation will give the best chance for long-term closure goals to be met, and a positive legacy made from the Čoka Rakita gold mine.

26.10 Capital Cost Estimate

- Compile a comprehensive database of costs relevant to the Project to enhance the accuracy of future cost estimates.
- For future resource evaluations, develop a detailed pre-production schedule. This would improve the accuracy of cost estimates related to mobilisation, site setup, project logistics, equipment procurement, personnel recruitment, and training.
- Establish relationships with local contractors and service providers.
- Prepare organisational charts to support construction management and operating costs.

27 ABBREVIATIONS

Abbreviation	Unit or Term
%	Percent
$	Dollar
% w/w	Percent Weight per Total Weight
±	Plus-Minus
€	Euro
°	Degree
°C	Degree Celsius
µm	Micron or Micrometre, or 0.000001 Metre
2D, 3D	Two-Dimensional, Three-Dimensional
a	Annum (year)
AAS	Atomic Absorption Spectrometry
ABTSB	Apuseni-Banat-Timok-Srednogorie Belt
Ag	Silver
AG	Acid Generating
Ai	Abrasion Index
ALS_BO	ALS Bor
ARD	Acid Rock Drainage
As	Arsenic
ALS_VA	ALS Vancouver
Au	Gold
BD	Bulk Density
BHP	Big Hornblende Porphyry
BaseMet Labs	Base Metal Laboratories
BWi	Bond Ball Mill Work Index
Ca	Calcium
$CaCO_3$	Calcium Carbonate
$CaCO_3$/t	Calcium Carbonate per Tonne
Capex	Capital Expenditure, Capital Cost Estimate
CCTV	Closed-Circuit Television
CDA	Canadian Dam Association
CIM	Canadian Institute of Mining, Metallurgy and Petroleum

Abbreviation	Unit or Term
CLR	Centred-Log-Ratio
cm	Centimetre
CO	Carbon Monoxide
COG	Cut-Off Grade
CPF	Cementer Paste Fill
CRIRSCO	Committee for Mineral Reserves International Reporting Standards
CRM	Certified Reference Material
CS	Carbon Steel
csv	Comma-Separated Values
Cu	Copper
$CuSO_4$	Copper (II) Sulphate
CV	Coefficient of Variation (in statistics, the normalised variation value in a sample population)
CWi	Crush Work Index
CWR	Crushed Waste Rock
DCF	Discounted Cash Flow
DEM	Digital Elevation Model
DGPS	Differential Global Positioning System
dmt	Dry Metric Tonne
DPM	Dundee Precious Metals Inc.
DTM	Digital Terrain Model
DWi	Drop-Weight index
E (X)	Easting. Coordinate axis (X) for metre-based projection, typically UTM; refers specifically to metres east of a reference point (0,0)
EBRD	European Bank of Reconstruction and Development
EDF	Environmental Design Flood
EGL	Effective Grinding Length
EGM	Engineering Geology Model
EGRG	Extended Gravity Recoverable Gold
EIA	Environmental Impact Assessment
EMP	Early Mineral Porphyry
E-Room	Electrical Room
ESIA	Environmental and Social Impact Assessment
ETP	Effluent Treatment Plant

Abbreviation	Unit or Term
EU	European Union
FA	Fire Assay
FEL	Front-End-Loader
FPT	Full Plant Tailings
FS	Feasibility Study
ft	Foot
G&A	General and Administrative
g/L	Gram per Litre
g/t	Gram per Tonne
GAAP	Generally Accepted Accounting Principles
GEN_PE	Genalysis Perth
GIMS	Geological Information Management System
GPS	Global Positioning System
GRG	Gravity Recoverable Gold
H	Hour(s)
H&S	Health and Safety
Ha	Hectare
HDPE	High-Density Polyethylene
HPGR	High-Pressure Grinding Roll
HQ2	Size of Diamond Drill Rod/Bit/Core
HQ3	HQ Triple Tube
HR	Hydraulic Radius
HVAC	Heating, Ventilation, and Air Conditioning
ICP-AES	Inductively Coupled Plasma-Atomic Emission Spectrometry
ICP-MS	Inductively Coupled Plasma-Mass Spectrometry
ICP-OES	Inductively Coupled Plasma-Optical Emission Spectrometry
IDF	Inflow Design Flood
IDW^2	Inverse Distance Weighting Squared
IFRS	International Financial Reporting Standards
IP	Induced Polarisation
ISO	International Organization for Standardisation
ITH	In-the-hole

Abbreviation	Unit or Term
IUCN	International Union for Conservation of Nature
JLS	Jurassic Limestone
K	Potassium
K/Th	Potassium / Thorium
kA	Kiloampere
kg	Kilogram
kg/m, kg/m^2	Kilogram per Metre, Kilogram per Square Metre
kg/t	Kilogram per Tonne
KLS	Cretaceous Limestone
km, km^2	Kilometre, Square Kilometre
KNA	Kriging Neighbourhood Analysis
k oz	Thousand Ounces
kP	Kilopascal
kt	Kilotonnes (or Thousand Tonnes)
kV	Kilovolt
kW	Kilowatt
kWh	Kilowatt Hour
kWh/m^3	Kilowatt Hour per Cubic Metre
kWh/t	Kilowatt Hour per Tonne
L	Litre
L/s	Litre per Second
lbs	Pounds
LCT	Locked Cycle Test
LDL	Lower Detection Limit
LiDAR	Light Detection and Ranging
LOM	Life-of-Mine
M	Million
m(E)	Metres East
m(N)	Metres North
m(RL)	Metres (Relative Level)
m/day	Metre per Day
m/s	Metre per Second

Abbreviation	Unit or Term
m^3/s	Cubic Metre per Second
$m^3/s/kW$	Cubic Metre per Second per Kilowatt
m/year	Metre per Year
m, m^2, m^3	Metre, Square Metre, Cubic Metre
Ma	Million Years
MAFWM	Ministry of Agriculture, Forestry and Water Management
masl	Metre Above Sea Level
MCC	Motor Control Centre
MCTI	Ministry of Construction, Transport and Infrastructure
MEP	Ministry of Environmental Protection
MDE	Maximum Design Earthquake
mg	Milligram
MI	Ministry of Interior
MIBC	Methyl Isobutyl Carbinol
ML/ARD	Metal Leaching / Acid Rock Drainage
MLC	Mine Load Centre
mm	Millimetre
Mm^3	Million Cubic Metre
Mm^3/y	Million Cubic Metre per Year
MME	Ministry of Mining and Energy
MoM&E	Serbian Ministry of Mining and Energy
Moz	Million Ounces
MRE	Mineral Resource Estimate
MSO	Mineable Shape Optimiser
Mt	Million Tonnes
Mtpa	Million Tonnes per Annum
MVA	Megavolt Ampere
MW	Megawatt
N (Y)	Northing. Coordinate axis (Y) for metre-based projection, typically UTM; refers specifically to metres north of a reference point (0,0)
NAG	Non-Acid Generating
NI 43-101	National Instrument 43-101 Standards of Disclosure for Mineral Projects
NNP	Net Neutralising Potential
NO_2	Nitrogen Dioxide

Abbreviation	Unit or Term
NP	Neutralising Potential
NPC	Net Present Cost
NPR	Neutralisation Potential Ratio
NPV	Net Present Value
NQ	A diamond drill core diameter of 75.7 mm (outside of bit) and 47.6 mm (inside of bit)
NSR	Net Smelter Return
OK	Ordinary Kriging
Opex	Operating Expenditure, Operating Cost Estimate
OREAS	Ore Research & Exploration
OHSMP	Occupational Health and Safety Management Plan
oz	Troy Ounce (31.1034768 Grams)
P_{80} -75 µm	Measure of Pulverisation (80% Passing 75 µm)
Pa	Pascal
PaCMAP	Pairwise Controlled Manifold Approximation Projection
PAG	Potentially Acid Generating
PAX	Potassium Amyl Xanthate
PEA	Preliminary Economic Assessment
PDC	Process Design Criteria
PFS	Prefeasibility Study
PLC	Programmable Logic Controller
PMF	Probable Maximum Flood
POF	Probability of Failure
ppm	Parts per Million
PSD	Particle Size Distribution
PVC	Polyvinyl Chloride
PXP	Pyroxene Porphyry
Q2, Q3	Quarter 2, Quarter 3
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
RC	Reverse Circulation
RL (Z)	Reduced Level; elevation of the collar of a drillhole, a trench or a pit bench above the sea level

Abbreviation	Unit or Term
ROM	Run-of-Mine
RPEEE	Reasonable Prospects for Eventual Economic Extraction
RQD	Rock Quality Designation
RMU	Ring Main Unit
RVSS	Reduced Voltage Soft Starter
S	Sulphur
SAB	SAG and Ball (mills)
SAG	Semi-Autogenous Grinding (mill)
SAV	SAG and Vertical (mills)
SEA	Strategic Environmental Assessment
SFA	Screen Fire Assay
SFD	Sequential Felsic Debris Flow (Epiclastic Unit)
SFR	Staged Flotation Reactor
SGS	Société Générale de Surveillance International Laboratory Group
SGS_BO	SGS Bor
SGS_CH	SGS Chelopech
SiO_2	Silicon Dioxide, Silica
SM	Screened Metallic
SMC	Steve Morrell Comminution
SO_2	Sulphur Dioxide
SOR	Slope of Regression
SP	Spatial Plan
SPI	SAG Power Index
SQL	Structured Query Language
SSAG	Single Stage SAG
SWIR	Shortwave Infrared Spectroscopy
t	Tonne
t/m^3	Tonne per Cubic Metre
TEM	Time Domain Electromagnetics
Ti	Titanium
TMC	Timok Magmatic Complex
TMI	Total Magnetic Intensity
ToS	Trade-Off Study

Abbreviation	Unit or Term
tpa	Tonne per Annum
tpd	Tonne per Day
tph	Tonne per Hour
tph/m^2	Tonne per Hour per Square Metre
TSF	Tailings Storage Facility
UCS	Unconfined Strength
UDS	Underground Distribution System
UG	Underground
UPS	Uninterruptible Power Supply
US$	United States of America Dollars
US$/dmt	United States of America Dollars per Dry Metric Tonne
US$/g	United States of America Dollars per Gram
US$/oz	United States of America Dollars per Ounce
US$/t	United States of America Dollars per Tonne
UTM	Universal Transverse Mercator
V	Volt
VOC	Volatile Organic Compound
VSD	Variable Speed Drive
VTEM	Versatile Time Domain Electromagnetic
VU	Vulnerable
WAI	Wardell Armstrong International
WGS	World Geodetic System
wt%	Percentage by Weight
XRD	X-Ray Diffraction
Zr	Zirconium

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29 CERTIFICATES

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Preliminary Economic Assessment – Čoka Rakita Project, Eastern Serbia*" dated June 11, 2024 with an effective date of November 16, 2023 (the "Technical Report"), prepared for Dundee Precious Metals Inc. (the "Company").

I, *Stephan Blaho, P. Eng.,* of Thornhill, Ontario, Canada, do hereby certify:

1. I am Senior Principal Mining Engineer with WSP Canada Inc. with an office at 150 Commerce Valley Dr W, Thornhill, Ontario, L3T 7Z3, Canada.
2. I graduated from Queen's University, Kingston, Ontario, Canada with a Bachelor of Science in Mining Engineering in 1980.
3. I am registered member of Professional Engineers Ontario (Reg. 90252719).
4. My relevant experience includes 40 continuous years as a Mining Engineer. I have worked on similar projects to the Čoka Rakita Project; my experience for the purpose of the Technical Report includes:
    - Six years of experience in managing underground mining operations at base metal mines.
    - 30 years of experience in underground mine development, mainly managing projects.
    - Five years of experience as a principal mining engineer in mining consulting.
    - Participant and author for various NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 16, and for related portions of Sections 1, 21, and 25 to 28 of the Technical Report.
8. I visited the Property that is the subject of the Technical Report on October 3 to 5, 2023.
9. I have not had prior involvement with the property that is the subject of the Technical Report.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 11th day of June 2024, Thornhill, Ontario.

“Original Signed and Sealed on file”
Stephan Blaho, P. Eng.
Senior Principal Mining Engineer
WSP Canada Inc.

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I, *Daniel M. Gagnon, P. Eng.* of Montreal, Quebec, Canada, do hereby certify:

1. I am Senior Vice President Mining, Geology and Met-Chem Operations, with DRA Americas Inc. at 555 René Lévesque West, 6th Floor, Montreal, Quebec Canada H2Z 1B1.
2. I am a graduate of École Polytechnique de Montréal, Montreal, Quebec, Canada in 1995 with a bachelor degree in Mining Engineering.
3. I am a registered Professional Engineer in the Province of Quebec (Reg. #118521).
4. My relevant experience includes a total of 28 continuous years since my graduation as a Mining Engineer. I have worked on similar projects to the Čoka Rakita Project in Serbia; my experience for the purpose of the Technical Report includes:
   - Management of numerous studies and projects of varying complexity, involving multi-disciplinary engineering teams for projects in gold, base metals, and other commodities.
   - Design, scheduling, cost estimation and Mineral Reserve estimation for several open pit studies in Canada, the USA, South America, West Africa, and Morocco.
   - Technical assistance in mine design and scheduling for mine operations in Canada, USA, and Morocco.
   - Participant and author for various NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.

7. I am responsible for the preparation of Sections 19 and 22, and for related portions of Sections 1 and 25 to 28 of the Technical Report.
8. I did not visit the Property that is the subject of the Technical Report.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 11th day of June 2024, Montréal, Quebec.

"Original Signed and Sealed on file"

Daniel M. Gagnon, P. Eng.
Senior VP Mining Geology &
Met-Chem Operations
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I, *Kevin Leahy, B.Sc. (Hons), PhD, C.Geol., SiLC,* of London, United Kingdom, do hereby certify:

1. I am a Geologist and Technical Director with Environmental Resources Management Ltd with an office at 2nd Floor, Exchequer Court, 33 St Mary Axe, London, UK, EC3A 8AA.
2. I graduated from the University of Leeds, UK with a bachelor in Geological Sciences in 1992 and a PhD on diamond exploration in 1996.
3. I am a registered Fellow of the Geological Society of London, Burlington House, Piccadilly, London, UK and have been a Chartered Geologist there since 2005. I am also a Registered Suitably Qualified Person and Specialist in Land Condition in the UK Land Forum National Quality Mark Scheme since 2017.
4. My relevant experience includes over 25 years in geological exploration, environmental impact assessment, mine audit and land contamination remediation and closure planning on numerous mines, process plants and smelters. I have worked on similar projects to the Čoka Rakita Project in Serbia; my experience for the purpose of the Technical Report includes:
   - Geological exploration for minerals in South Africa, Equatorial Guinea, Sweden and Canada, as well as a structural geologist for numerous hydrocarbon exploration projects. In the last five years jointly developed a hydro-geochemical exploration tool, including several projects in the Timok Belt, Serbia as well in the US and Australia.
   - Environmental Impact Assessment on dozens of EIA projects in Europe, Africa and Asia, including three in Serbia, on a variety of mineral targets and deposit types. My role in EIA projects is usually as topic lead on soils and geology, also contributing to surface and groundwater and early closure planning.
   - Mine audits on several sites in Europe, Asia and South America both for transaction due diligence and for compliance with environmental standards: internal, national and international.
   - Closure planning and land contamination projects on several mine, smelter and processing sites in Europe.
   - Participant and author for various NI 43-101 Technical Reports.

5. I have read the definition of “qualified person” set out in the NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.

6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.

7. I am responsible for the preparation of Section 20, and for related portions of Sections 1 and 25 to 28 of the Technical Report.

8. I visited the Property that is the subject of the Technical Report on October 9, 2023.

9. I have not had prior involvement with the property that is the subject of the Technical Report.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 11th day of June 2024, London, UK.

“Original Signed and Sealed on file”

Kevin Leahy, B.Sc. (Hons), PhD, C.Geol., SiLC
Technical Director
Environmental Resources Management Ltd

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I, *Marcello Locatelli, P. Eng., PMP* of Toronto, Ontario, Canada, do hereby certify:

1. I am Project Manager with DRA Americas Inc. with an office at 20 Queen Street West, 29th Floor, Toronto, Ontario, Canada M5H 3R3.
2. I graduated from the University of Johannesburg, South Africa with a Bachelor in Mechanical Engineering in 2006.
3. I am registered member of Professional Engineers Ontario (Reg. 100196154).
4. My relevant experience for mining and mineral processing projects, and projects similar to the Čoka Rakita Project; for the purpose of the Technical Report includes:
   - Over 15 years of mining and mineral processing project development, for numerous clients, in capacities of Mechanical Engineer, Project Engineer, and Project Manager.
   - Management of various studies and projects involving multi-disciplinary engineering teams for a variety of commodities.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 2, 3, 15, 18 (except 18.7 to 18.10), and 24, and for related portions of Sections 1, 21, and 25 to 28 of the Technical Report, as well as for overall report compilation.
8. I visited the Property that is the subject of the Technical Report on October 3 to 5, 2023.

9. I have not had prior involvement with the property that is the subject of the Technical Report.

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 11th day of June 2024, Toronto, Ontario.

"Original Signed and Sealed on file"

Marcello Locatelli, P. Eng., PMP
Project Manager
DRA Americas Inc.

// DRA Americas Inc.

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I, *Daniel (Niel) Morrison, P. Eng.* of Toronto, Ontario, Canada, do hereby certify:

1. I am Principal Process Engineer with DRA Americas Inc. with an office at 20 Queen Street West, 29th Floor, Toronto, Ontario, Canada M5H 3R3.
2. I graduated from the University of Stellenbosch, South Africa with a Bachelor of Chemical Engineering in 1990.
3. I am registered member of Professional Engineers Ontario (Reg. 100134360).
4. My relevant experience includes 32 years of experience since my graduation as a Metallurgist and Process Engineer. I have worked on similar projects to the Čoka Rakita Project; my experience for the purpose of the Technical Report includes:
    - More than 30 years of experience, 6 years in research and development, 10 years in process plant operations and the remainder in process plant flowsheet design and engineering;
    - Polymetallic flotation testwork interpretation and flowsheet design for several studies and projects;
    - Participant and author for various NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 13 and 17, and for related portions of Sections 1, 21, and 25 to 28 of the Technical Report.
8. I did not visit the Property that is the subject of the Technical Report.

9. I have had prior involvement with the property that is the subject of the Technical Report.
    - *Served as QP for - "Technical Report on the Maiden Mineral Resource Estimate for the Čoka Rakita Gold Project, Serbia"* prepared by Environmental Resources Management Limited (trading as CSA Global) with an effective date of November 16, 2023 (issued Report Date January 24, 2024, prepared for Dundee Precious Metals Inc.).
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 11th day of June 2024, Toronto, Ontario.

"Original Signed and Sealed on file"
Niel Morrison, P. Eng.
Principal Process Engineer
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I, *Maria O'Connor, B.Sc. (Hons), MAIG.* of Horsham, United Kingdom, do hereby certify:

1. I am a Technical Director Mineral Resources for Environmental Resources Management Limited (trading as CSA Global) located at Springfield House, Suite 2 First Floor, Horsham, West Sussex, RH12 2RG, United Kingdom.
2. I graduated from the University College Dublin with a Bachelor in Environmental Geochemistry in 2004.
3. I am registered member of the Australian Institute of Geoscientists (AIG Membership # 5931).
4. My relevant experience includes over 19 years in geology since I graduated. I have worked on similar projects to the Čoka Rakita Project; my experience for the purpose of the Technical Report includes:
    - Exploring, evaluating, and completing mineral resource estimates for skarn systems and other mineral deposits in Europe, Australia, central Asia, USA, and Africa.
    - Participant and author for several NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 4 to 12, 14, and 23, and for related portions of Sections 1 and 25 to 28 of the Technical Report.
8. I visited the Property that is the subject of the Technical Report on October 3 and 4, 2023.
9. I have had prior involvement with the property that is the subject of the Technical Report.
    - *Served as QP for - "Technical Report on the Maiden Mineral Resource Estimate for the Čoka Rakita Gold Project, Serbia"* prepared by Environmental Resources Management

Limited (trading as CSA Global) with an effective date of November 16, 2023 (issued Report Date January 24, 2024, prepared for DPM).

10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 11th day of June 2024, Horsham, United Kingdom.

"Original Signed and Sealed on file"

Maria O'Connor, B.Sc. (Hons), MAIG
Technical Director Mineral Resources
Environmental Resources Management Ltd.

20 Queen Street West / 29th floor / Toronto / Ontario / Canada / M5H 3R3
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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Preliminary Economic Assessment – Čoka Rakita Project, Eastern Serbia*" dated June 11, 2024 with an effective date of November 16, 2023 (the "Technical Report"), prepared for Dundee Precious Metals Inc. (the "Company").

I, *Ninoslav Pavlovic, P. Eng.,* of Sudbury, Ontario, Canada, do hereby certify:

1. I am Senior Process Engineer for Mine Backfill and Tailings & Waste Rock Management with Responsible Mining Solutions (RMS) Corp. with an office at 531 Notre Dame Ave, Greater Sudbury, Ontario, P3C 2X1, Canada.
2. I graduated from the University of Belgrade (Serbia) in 2004 with a Bachelor of Science in Engineering, and in 2008 with a Master of Science in Engineering.
3. I am a registered member of Professional Engineers Ontario (Reg. 100514597).
4. My relevant experience includes 14 years as a Mineral Processing Specialist and Engineer. I have worked on similar projects to the Čoka Rakita Project, and on integrated tailings management system projects and designs in various mining jurisdictions; my experience for the purpose of the Technical Report includes:
   - Trade-off studies on integrated tailings management systems (process design, plant locations) to meet underground backfill and tailings surface disposal requirements;
   - Reviewing mining methods and life-of -mine plans, mine production (ore, waste rock) and schedules;
   - Determining paste backfill densities and tailings synergy between underground and surface disposal, to facilitate the tailings storage facility design;
   - Designing process flow diagrams, with process mass and water balance, as well as equipment sizing and guidance on backfill plant arrangement drawings;
   - Capital and operating cost estimating;
   - Previous Technical Report preparation (with recommendations and paths forward).
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.

6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 18.7, and for related portions of Sections 1, 21, and 25 to 28 of the Technical Report.
8. I did not visit the Property that is the subject of the Technical Report.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 11th day of June 2024, Sudbury, Ontario.

"Original Signed and Sealed on file"
Ninoslav Pavlovic, P. Eng.
Senior Process Engineer
Responsible Mining Solutions (RMS) Corp.

// DRA Americas Inc.

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I, *Eric Sellars, P. Eng.,* of Toronto, Ontario, Canada, do hereby certify:

1. I am a Geotechnical Engineer with SLR Consulting (Canada) Ltd. with an office at 55 University Ave., Suite 501, Toronto, Ontario, M5J 2H7, Canada.
2. I graduated from the University of Waterloo, Waterloo, Ontario, Canada with a Bachelor of Applied Science in Geological Engineering in 2009.
3. I am registered member of Professional Engineers Ontario (Reg. 100199889).
4. My relevant experience includes 14 years as a Geotechnical Engineer. I have worked on similar projects to the Čoka Rakita Project; my experience for the purpose of the Technical Report includes:
    - Site selection and best technology studies for tailings and waste rock facilities.
    - Design, construction, and operational support for tailings containment facilities.
    - Experience in dam safety governance and current best practices.
    - Participant and author for various NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Section 18.9, and for related portions of Sections 1, 21, and 25 to 28 of the Technical Report.
8. I did not visit the Property that is the subject of the Technical Report.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible

have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 11th day of June 2024, Toronto, Ontario.

"Original Signed and Sealed on file"

Eric Sellars, P. Eng.
Geotechnical Engineer
SLR Consulting (Canada) Ltd.

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CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*NI 43-101 Technical Report – Preliminary Economic Assessment – Čoka Rakita Project, Eastern Serbia*" dated June 11, 2024 with an effective date of November 16, 2023 (the "Technical Report"), prepared for Dundee Precious Metals Inc. (the "Company").

I, *Luis Vasquez, M.Sc., P. Eng.,* of Toronto, Ontario, Canada, do hereby certify:

1. I am a Principal Hydrotechnical Engineer with SLR Consulting (Canada) Ltd. with an office at 55 University Ave., Suite 501, Toronto, Ontario, M5J 2H7, Canada.
2. I graduated from the Universidad de Los Andes, Bogotá, Colombia with a Bachelor in Civil Engineering in 1998 and a Master of Science in Water Resources Engineering in 1999.
3. I am registered member of Professional Engineers Ontario (Reg. 100210789).
4. My relevant experience includes 23 years in the field of Water Resources Engineering. I have worked on similar projects to the Čoka Rakita Project; my experience for the purpose of the Technical Report includes:
   - Experience in the field of water management.
   - Design and operational support experience on various projects and mine sites.
   - Participant and author for various NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 18.8 and 18.10, and for related portions of Sections 1, 21, and 25 to 28 of the Technical Report.
8. I did not visit the Property that is the subject of the Technical Report.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.

11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 11th day of June 2024, Toronto, Ontario.

"Original Signed and Sealed on file"

Luis Vasquez, M.Sc., P. Eng.
Principal Hydrotechnical Engineer
SLR Consulting (Canada) Ltd.