NI 43-101 Technical Report

Čoka Rakita Project

Pre-Feasibility Study

Eastern Serbia

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1 EXECUTIVE SUMMARY

1.1 Property Description, Ownership, Location and Access

Dundee Precious Metals Inc. (DPM or the Company) is a Canadian-based international gold mining company with headquarters in Toronto, Ontario, Canada and operations and projects located in Bulgaria, Serbia and 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, approximately 35 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).
- 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/km² or US$ 95.5/km² 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 do not have seasonal shutdowns.

1.2 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.

1.3 Geological Setting, Mineralisation, and Deposit Types

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 which is part of the Tethyan (or Alpine-Himalayan) orogenic system. It is approximately 85 km long and extends from the town of Majdanpek in the north to the village of Bučje in the south. It hosts 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 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.

The mineralisation conforms to an oxidised gold skarn type deposit. 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 Mineral Resource Estimate (MRE) has been prepared on the portion of the Project where gold-rich skarn mineralisation occurs.

1.4 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.

1.5 Drilling

A total of 271 drillholes for 126,496 m have been drilled since 2008, with the majority drilled since 2021. The drilling has been predominantly diamond (175 for 94,923 m) and diamond tail (46 for 22,454 m) with 48 reverse circulation (RC) drill holes drilled for 7,959 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 targeting the skarn mineralisation that is the subject of this study.

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 predominantly at least 30 m x 30 m.

1.6 Sampling Preparation, Analyses and Security

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.

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, internal (preparation laboratory) duplicates and umpire laboratory duplicates were used as quality control material to monitor accuracy, precision and contamination.

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), SGS Burgas (SGS BUR), ALS Rosia Montana (ALS_BO). These laboratories are certified to ISO-standards and are independent of DPM.

Gold grades within skarn domains used in the Mineral Resource Estimate 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.

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.

1.7 Data Verification

The QP (Ms. Maria O'Connor, MAIG) visited site on October 3 and 4, 2023 for the purposes of reviewing drilling activity, logging facilities, and the independent, on-site laboratory. The site visit was supplemented by a review of data collection procedures and spot-checking locations of drill collars. Site discussions were held with key DPM personnel; and various aspects of data collection, management, chain of custody and geology and mineralisation interpretation workflow was reviewed. The QP (Ms. Maria O'Connor, MAIG) considers that the proper amount of review through reports, technical data, interviews, and physical presence has been completed to support the data verification requirements under NI 43-101.

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. The results of the program indicated that a flowsheet comprising flotation followed by leaching of the float tails can yield gold recoveries from 92.9% to 96.7%.

The Base Metallurgical Laboratories (BaseMet Labs) testwork program in 2023 was performed on three (3) composite samples, representing low, mean, 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. These are all reported in the sections below. The testing culminated in a set of locked cycle tests (LCTs) which simulated the selected flowsheet and employed optimised test conditions.

Table 1.1 summarises the gold recoveries achieved during these LCTs and compares them with a scenario where the tails from this LCT were 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. DPM did not consider a flowsheet using cyanide appropriate for this Project, hence the selected flowsheet uses gravity concentration followed by flotation.

Table 1.1 – 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 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 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

Sample ID	Test Procedure	Head Au Grade (g/t)	Au Recovery (%)	Tails Au Grade (g/t)
MetCRA23-03	Gravity -> Flotation 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

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 0.2% threshold for triggering penalties.

Settling testwork 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.

Preliminary filtration testing showed that the tailings sample can be dewatered to 23% moisture, by pressing only. However, it can be further reduced to approximately 18%, with the addition of air blowing. 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 DTSF does not require lining; however, the design includes provisions for a lined DTSF.

BaseMet Labs performed the PFS testwork program (also referred to as Phase 3) in 2024. The purpose of the PFS testwork program was, primarily, to understand variability with respect to both lithology and mineralisation types. The secondary objective was to address recommendations from the PEA. New variability samples were selected from the available drill cores for the variability testing program. The selection criteria for these samples were to represent the various lithologies, mineralisation as well as spatial distribution within the orebody

Table 1.2 summarises the comminution results recorded for the variability samples. These results are in good agreement with what was recorded during previous testwork programs and confirm that the material is of average to below average hardness and abrasiveness.

Table 1.2 – BaseMet Labs Test Program (2024) Comminution Results

Percentile	SG	A x b	BBWi (kWh/t)	Ai
15th	2.92	44.6	12.3	0.064
50th	3.10	59.8	13.9	0.103
85th	3.30	72.6	15.2	0.167

Extended Gravity Recoverable Gold (EGRG) testing of the variability samples showed that the coarse gold particles are generally within the moderately coarse (P_{80}=120 µm) to coarse (P_{80}=200 µm) range. FLSmidth used the size-by-size coarse gold data to model various gravity recovery circuits for the trade-off study. The modelling performed by FLSmidth showed that the primary gravity gold recovery step should recover approximately 33 to 40% of the gold from the Run-of-Mine (ROM) feed. This is marginally lower than what was adopted in the PEA Phase (43%).

A set of tests which mimics the gravity-flotation flowsheet was performed on the variability samples. The test procedure comprised grinding followed by two stages of gravity concentration and two stages of open-ended flotation. It was assumed that a portion of the gold in the intermediate product streams would report to the concentrate if these were recycled as would be the case in an operating plant. The results from these tests, weighted by how much each sample represents of the block model, are as follows:

- Mass pull to the gravity concentrate = 0.15%.
- Gold recovery to the primary gravity concentrate = 37.4%.
- Mass pull to the final flotation concentrate = 9.1%.
- Gold distribution from ROM feed to final flotation concentrate = 50.0%.
- Overall gold recovery = 87.4%.

These key results align well with the results from composites tested during previous phases. For the PFS phase, the design criteria was adjusted to align with these variability test recoveries. The mass pull numbers were not used in the design criteria because the test procedure does not accurately represent full-scale operations.

RMS tested a sample of tailings in 2024 to support the design of the tailings dewatering and storage facilities. Sedimentation tests showed that the material settles well with an underflow of 65 wt% achieved at a flux rate of 0.5 t/m²/h and 15 wt% feed concentration, with clear overflow (TSS <100 ppm). Vacuum disc and belt filtration tests showed that the tailings filter effectively at feed concentrations of 65 wt% or higher. UCS testing demonstrated that a GGIBFS blend (90:10) achieved greater strength than cement alone, reaching 1 MPa with 5% binder and 2 MPa with 10% binder after 21 days and that there is no strength loss after 90 days of curing.

1.9 Mineral Resource Estimates

DPM implemented an acQuire GIMS (Geological Information Management Systems) 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.

Mineralisation 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 mafic sills and internal waste zones which cut across the mineralisation. Three 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 to all domains, with gold values exceeding 144 g/t capped for the footwall domain (largest mineralisation domain) with further capping set to 37 g/t Au at distances greater than 30 m within the estimate. 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 the marl, skarn and mafic intrusion geological domains using Ordinary Kriging. Mean density was assigned to the remaining geological domains, where the variance of measured density was lower.

Gold and silver grades were estimated into mineralisation domains into 10 m x 10 m x 5 m (X x Y x Z) blocks using Ordinary Kriging and with hard boundaries imposed between all domains, both within mineralisation domains and between mineralisation and waste. Optimal block size was informed by Kriging Neighbourhood Analysis and taking mining considerations into account. A three-phase search strategy was used with increasing ranges. The MRE satisfies RPEEE by demonstrating the spatial continuity of the mineralisation based on a 2 g/t Au reporting cut-off grade and stope volumes created by Deswik's Stope Optimizer (DSO) and was classified as Indicated and Inferred Mineral Resources, informed by drill spacing supported by a drill hole spacing study ("DHSS"), QA/QC, quality of data, confidence in geological and mineralisation interpretations.

The MRE was prepared by QP, Ms. Maria O'Connor, MAIG, with an effective date of 30 August 2024. The MRE tabulation constrained within the DSO volumes is presented in Table 1.3. Summary assumptions used in RPEEE calculations are presented in the notes section of Table 1.3 and are discussed in more detail in Section 14.19.1.

Table 1.3 – Čoka Rakita MRE Using Underground Mining Scenario

Čoka Rakita Mineral Resource Estimate
Effective Date of 30 August 2024

Mineral Resource category	Tonnes (Mt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)
Indicated	1.45	3.30	154	1.30	61
Inferred	0.11	3.11	11	1.24	5

Notes:

- The cut-off grade value of 2 g/t assumes US$1,700/oz gold price, 88.8% gold recovery, 0% dilution, US$71.7/t operating cost (mining, process and G&A), US$11.19/t sustaining capital cost, as well as offsite and royalty costs.
- Mineral Resources are reported within DSO underground mining shapes generated at a 2 g/t Au cut-off grade, to ensure Mineral Resources meet RPEEE. The stope optimisation 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 metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing political, or other risk factors that might materially affect the estimate of Mineral Resources, other than those specified in Table 1.4.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Mineral Resources are reported exclusive of Mineral Reserves.
- Figures have been rounded to reflect that this is an estimate and totals may not match the sum of all components.

The QP is not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other factors that could materially impact the MRE disclosed in this Report, other than those specified below and in Table 1.4:

- 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 sensitive to the choice of top cut grades; therefore, changes to those values and how they are treated within the estimate could impact the grade and tonnage above the cut-off grade of the MRE. This means that if the top cut chosen is too high, metal content could be overestimated; if top cut chosen is too low, metal content would be underestimated. Sensitivity analysis on choice of top cuts (110, 125, 144, 175 g/t Au) shows variance in metal estimated to be in the order of 5% between lowest and highest top cut tested.
- 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 1.4. The overall risk to the Čoka Rakita MRE is reflected in the current resource classification as Indicated and Inferred Mineral Resources; and is considered moderate, which is consistent with the high-grade nature of the Project.

Table 1.4 – Qualitative Risk Assessment

Factor	Risk	Comment
Sample collection, preparation and assaying	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. The level of coarse gold has been characterised as being a 'Medium' type coarse gold mineralisation, which is likely to have 'Medium' sampling problems. This generally indicates that a specialised protocol will be required to achieve quality assay data (e.g. SFA) (Dominy, 2024). Analysis for the vast majority of samples has been completed using screen fire assay which requires larger volumes, which is considered good practice for this kind of deposit. The majority of the gold is associated with finer fractions, but coarse gold is associated with higher grades. The mineralised systems that have problematic levels of coarse gold generally lie within the "High-Med" to "Extreme" types. These generally require specialised sampling systems/protocols which may include whole core or large RC lot sampling followed by large assays such as SFA, LeachWELL or PhotonAssay. For High to Extreme types, tonnage-scale bulk sampling is likely required to validate resource/reserve estimates (Dominy, 2024).
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. A focussed duplicates program should be undertaken to monitor the transition from core shed to assay. This should include half diamond core duplicates, RC rig duplicates (where applicable), laboratory coarse split duplicates and assay/pulp duplicates (Dominy, 2024).
Geological model	Low	Uncertainty in accuracy of location of intrusives modelled is moderate based on core. However, this is mitigated by the use of lithogeochemical analysis and modelling of multi-element geochemistry which allows for differentiation of these intrusives from S2/S1 unit, which is difficult by viewing core alone. The fact that core can look very similar in terms of skarnification and intensity of alteration but have different grade character across short distances.
Mineralisation model	Moderate	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 is mitigated by the low cutoff grade of the Mineral Resource and Mineral Reserve.

Factor	Risk	Comment
Treatment of outliers (grade caps)	Moderate	The MRE is very sensitive to the choice of grade cap. A relatively conservative grade cap was applied, which cuts 0.7% of the data and c. 20% of the metal. When data is top cut (at 144 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. An additional cap and distance threshold has been imposed within the estimate to reflect the fact that higher grades are unlikely to have the same continuity as lower grades. Sensitivity work on various top cuts conducted during this MRE (110, 125, 144 and 175 g/t Au) has indicated that the chosen top cut (144 g/t Au) results in a metal estimate approximately 3% higher than the lowest tested (110 g/t Au) and 2% lower metal than the highest tested (175 g/t Au).
Grade estimate	Moderate	The coefficient of variation (CV) is on the upper end of what is considered tolerable for Ordinary Kriging, largely due to the presence of a high-grade tail. 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 can be 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. The classification as Indicated and Inferred Mineral Resources reflects this risk.

1.10 Mineral Reserve Estimates

The mine design, scheduling, and Mineral Reserve Estimate for the Čoka Rakita Project was prepared by WSP in accordance with the 2014 CIM Definition Standards and 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines.

The Čoka Rakita Deposit Mineral Reserve Estimate is based only on Indicated Mineral Resources identified in the block model provided by ERM. All mineral resource material in the block model that was classified as Inferred was assigned a zero grade. The reference point at which Mineral Reserves are defined, is the point where the ore is delivered to the process plant.

Mineral Reserves are reported at a gold price of US$1,500/oz gold, and in-situ cut-off grade (COG) of 2.5 g/t Au for stopes and 1.0 g/t Au for development.

The design parameters include an external dilution factor of 10% applied to the mining shapes with an average grade of 0.5 g/t Au and a 95% mining recovery.

For the PFS stage, the mining method adopted is longitudinal long-hole stoping and the current LOM period is estimated to be eight (8) years at an average of 850 kt annually.

The Mineral Reserve Estimates have an effective date of August 30, 2024. The QP responsible for the Mineral Reserve Estimate is Mr. Khalid Mounhir, P.Eng., a WSP employee.

The Mineral Reserve Estimates are summarised in Table 1.5

Table 1.5 – Mineral Reserve Estimate – Effective as of August 30, 2024

Classification	Tonnes (Mt)	Au (g/t)	Contained Gold (Moz)
Proven	-	-	-
Probable	6.633	6.38	1.359
Proven + Probable	**6.633**	**6.38**	**1.359**

1. At the time of this Report, there are no Proven Mineral Reserves for the Čoka Rakita Project.
2. The Mineral Reserves disclosed are classified as Probable and are based on the 2014 CIM Definition Standards and 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines.
3. The Inferred Mineral Resources are treated as waste and do not contribute to reserves estimation.
4. Mineral Reserves have an effective date of August 30, 2024.
5. The reference point at which the Mineral Reserves are defined is where the ore is delivered to the process plant and therefore not inclusive of milling recoveries or payable metal deductions.
6. Long-term metal price assumed for the evaluation of the Mineral Reserves is US$1,500/oz for gold.
7. Mineral Reserves are based on a global rounded stoping cut-off grade of 2.5 g/t and development COG of 1g/t (in-situ). Refer to Subsection 15.3.1 for COG estimation details.
8. Mineral Reserves account for 10% external mining dilution and 95% mining recovery applied to the stopes
9. Contained Metal is calculated as follows: Au Contained Metal, (oz) = Tonnage (Mt) * Grade (g/t) / 31.1035.
10. The Mineral Reserve Estimation was completed under the supervision of Mr. Khalid Mounhir, P.Eng., Lead Mining Engineer at WSP Canada Inc., who is a QP as defined under NI 43-101.
11. The QP is not aware of any metallurgical, environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that might affect the estimate of Mineral Reserves, other than those specified in Section 15.6.
12. Sum of individual table values may not equal due to rounding.

The Mineral Reserves are subject to the types of risks common to underground gold mining operations and include:

- Geological complexity, geological interpretation, and Mineral Resource block modelling.
- Changes in Market conditions, commodity prices and COG estimation assumptions.
- Changes in rock quality and geotechnical assumptions.
- Changes in hydrogeological conditions and water inflow assumptions
- Input factors used to assess dilution and mining recovery factors.
- Assumptions regarding social, permitting, and environmental conditions.

1.11 Mining Methods

1.11.1 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.

A geotechnical study carried out by WSP on positioning the infrastructure at HW or FW showed that the quality of the sedimentary host rock in the hanging wall generally ranges from fair to good, while the footwall of the deposit exhibits poorer ground conditions. Consequently, the geotechnical findings supported the decision to position the mine development and underground infrastructure on the hanging wall side of the deposit.

1.11.2 Mining Method

DPM plans to mine the deposit with sublevel long-hole open stoping. Most of the ore 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 ore.

The mine will be divided into two (2) mining horizons: the upper horizon will be from 440L to 620L, and the lower horizon from 360L to 440L. The 420L is considered as a sill pillar. The longholes will be drilled in rings angled downwards as inverted fans from the upper production drive and loaded with bulk emulsion explosives. After each stope blast, a Load Haul Dump (LHD) vehicle will muck the broken ore from the lower production drive of the stope. Mine trucks will haul the material to surface via the spiral ramp and up either the lower or upper decline depending on proximity and availability. Most stopes will be filled with paste, but cemented rock fill and uncemented rock fill will be used where the sequence permits.

1.11.3 Underground Geotechnical Assessment

The underground geotechnical assessments for the Čoka Rakita PFS were collected as part of the geotechnical and hydrogeological investigations program completed by DPM and their local contactors (Faculty of Mining and Geology at the University of Belgrade). The WSP underground geotechnical team worked closely with both DPM and the University of Belgrade staff during the collection of the data in 2024.

A total of 15 boreholes (8,751 m), primarily located in the hanging wall, footwall and ore zones, were drilled, logged geotechnically, sampled, and partially surveyed with downhole geophysics to support the various PFS underground geotechnical assessments. Some of the data was not yet captured when the rock mechanics assessments were completed; this was intentional to meet the PFS schedule.

Rock mass classification values (RMR_{76} and Q' values) were estimated for each rock mass based on the data available. The main rock masses are classified as exhibiting fair to good character and have similar classification averages. A fault model, developed by DPM, indicates there are numerous fault planes/structures in the hanging wall, footwall, and ore zones and some of the fault zones intersect some of the long-term infrastructure. The fault zones create lower rock mass quality (poor to very poor) areas approximately 0.5 m to several meters in width that can have an impact on local stability and increase stope dilution or ground support requirements. Ground support recommendations were developed for the range of conditions expected from the main rock masses and the faults zones.

Empirical stope stability assessments and 3D numerical modelling assessments were completed based on the PFS mine design to identify stability concerns regarding the stope sizing, mining sequence, temporary sill pillar, and standoff distances of the access drifts, spiral ramp, and surface ventilation raise. Ground support designs were also completed for each type of excavation.

Geotechnical data collected in Q4 2024 and Q1 2025 will need to be reviewed and applied where required to confirm and/or update the PFS mine design including the key infrastructure excavations. Recommendations for underground geotechnical work is provided in Section 26.6.

1.11.4 HYDROGEOLOGY

The hydrogeological data for the Čoka Rakita PFS were collected as part of the geotechnical and hydrogeological investigations carried out in 2024 in support of the surface and underground mine infrastructure design. The objectives of the hydrogeological investigations were to assess the groundwater conditions within the areas of the proposed mine development. Selected boreholes across the site were instrumented to allow collection of additional hydrogeological data and to establish baseline conditions prior to mine construction.

Following completion of the field program, WSP compiled the available data as part of a conceptual hydrogeological assessment of the project area. The conceptual hydrogeological model formed the bases for development of a 3D numerical groundwater flow model that was used to predict groundwater inflows into the underground development during the LOM in support of the underground water balance and design of the dewatering infrastructure. Total predicted annual groundwater inflows to the mine reached a peak of approximately 20 L/s during the Mine Year 1. After that the inflows remained relatively steady until the end of the simulation with inflows varying from about 17.5 L/s to 18.5 L/s.

The available information provides an understanding of the hydrogeological conditions within the proposed project and surrounding areas, as required for the current stage of the project. However, additional data collection is recommended to advance the understanding of the hydrogeological conditions at the site, as summarised in Section 26.6 (Recommendations).

1.11.5 Mine Design

The stopes will be 20 m wide (measured from FW to HW), 20 m high and 30 m long and mined in a retreating sequence. The stope void will be created by a conventional 2.1 x 2.1 m drop raise and the production holes 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 development declines will have a 5.5 x 6.0 m (width x height) arched cross-section, and the production drives will measure 5.0 m x 4.5 m in profile. The spiral ramp, level access headings, and crosscuts will be driven at 5.5 m x 5.5 m.

1.11.6 Mine Access and Underground Facilities

The deposit will be accessed by developing two (2) surface declines (one decline labelled as the upper decline and another decline labelled as the lower decline) concurrently. The upper and lower declines will both be utilised for haulage, material movement and personnel movement. Both the upper and lower declines will serve as fresh-air intakes for the mine. The lower decline will also act as the main dewatering path for the mine.

The development of the spiral ramp will commence from both faces of the upper and lower declines. The development of the spiral ramp will continue to be developed until they converge and breakthrough near 500L. It is not expected that the lower ramp to the final levels will be developed until Year 3 of the mine life. The spiral ramp will provide access to all levels of the mine and enable the transport of ore to surface throughout the life of mine, and waste during the years prior to the mill operation.

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 return air ventilation raises connecting the sublevels will be equipped with ladderways, providing an emergency escape alternative to the spiral ramp. A maintenance shop and auxiliary infrastructure are designed on 620L while additional supporting infrastructure are located throughout the mine such as dewatering sumps, main pumping station, electrical substations, explosive magazine, emulsion bay, fuel bay and paste backfill stations. Lastly, in terms of mine safety infrastructure, there will be three (3) portable mine rescue chambers and escapeway ladderways in each of the internal ventilation raises which act as egresses in case of emergency.

1.11.7 Mine Equipment

The following list details the maximum requirement for mobile mining equipment and their proposed OEMs; the units will all be diesel-powered:

- Three (3) ea. development two (2) boom Sandvik DD422i.
- Two (2) ea. production drill rigs, Sandvik DL421.

- One (1) ea. Production drill rig, Sandvik DL 432i.
- Two (2) ea. emulsion chargers; Normet Charmec SF.
- Six (6) ea. mine trucks, 45 t payload; Sandvik TH545i.
- Five (5) ea. LHDs, 15 t tramming capacity; Sandvik LH515i.
- One (1) ea. cable bolter; Sandvik DS421.
- One (1) ea. shotcrete sprayer; Normet Spraymec.
- Two (2) ea. transmixers, 5.6 m^3 capacity; Normet LF 600.
- Three (3) ea. portable air compressors, Sullair L160.
- One (1) ea. mobile rock breaker; MacLean RB3.
- Two (2) ea. service machines; wheel loader CAT 930H.
- One (1) ea. telehandler; manitou 1440 Easy STS.
- Two (2) ea. utility vehicles (boom, water tank, fuel tank cassettes); Paus 50-3 WS N.
- One (1) ea. mine rescue vehicle (ambulance); Paus MinCa 18 A KT.
- Fifteen (15) ea. personnel vehicles – pickup truck; Toyota Hilux.
- One (1) ea. grader; CAT 20k.
- One (1) ea. diamond drill rig; Boart Longyear LM75.

1.11.8 MINE PERSONNEL

The annual staffing roster for the underground mine for a full year of production includes provisions for absences, vacations, and temporary vacancies. The underground mine will reach its peak workforce of 328 employees in Year 2 of the LOM plan.

The mine will operate on three (3) 8-hour shifts, with the workday duration as stipulated by Serbian legislation. The workforce will be organised into four (4) rotating groups, each working three (3) shifts followed by a period of time off.

1.11.9 PRE-PRODUCTION SCHEDULE

This pre-production phase will commence in the year before initiating the development of the two declines. The principal activities will include excavating portals, procuring essential mining equipment, and recruitment of personnel. Year -2 of the schedule will focus on driving the declines and starting the spiral ramp, with additional progress achieved in developing ventilation drifts and raises, level access headings, and infrastructure excavations. In Year -1, while development will continue on 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 at the end of quarter 2 of Year 1.

1.11.10 LIFE-OF-MINE PLAN

The Life-of-Mine (LOM) plan extends over eight (8) years, with full-capacity ore production sustained for six (6) years. Stope production begins at the 440 sublevel, by the end of Year -1. The deposit will be mined in two (2) phases. The upper horizon will be mined first, delivering ore as per the schedule, with stoping progressing upward from the 440 sublevel initially to higher levels. Full production starts in Year 2, producing over 855 kt of ore annually. In Year 4, mining begins at the 360 sublevel following completion of the spiral ramp, with lower horizon production supplementing the upper horizon. Production remains at full capacity through Year 7, tapering to 830 kt in Year 8.

Mine development will progress at an average rate of approximately 3,800 m/year from Years -2 to 2, decreasing to 3,200 m in Years 3 to 6 and dropping to 850 m in Year 7. Early years focus on declines, the spiral ramp, and infrastructure development. Crosscuts and production drives become the focus in Year 1, with the spiral ramp resuming development in Year 3 and reaching the mine's lowest level in Year 4. From Years 5 to 7, the focus will be development targets, crosscuts and drives for stoping; with all development completed by the end of Year 7.

1.11.11 MINING VENTILATION

There are two (2) x 560 kW main fans installed atop the surface ventilation raise in a parallel horizontal arrangement. The operating pressure of the main fans is 2,925 Pa, with a total airflow of 277 m^3/s. Numerous internal ventilation return air raises and regulators connect the mining levels to form the mine ventilation circuit. Internal ventilation raises will be 3.0 m in diameter with a planned escapeway ladder system for secondary egress. Auxiliary ventilation is required in the early stages of development of the lower and upper declines that consist of two (2) x 150 kW fans located at each portal. To develop the production levels, 1.2 m diameter flexible duct will be used with 75 kW fans to supply enough air for an LHD. For production levels, the same 75 kW auxiliary fans will be used and installed upstream of the regulators. There will be two fans installed on each level to develop the two faces.

1.11.12 MINE BACKFILL

The paste backfill underground distribution system (UDS) design is driven by the Mine Design and the results obtained from laboratory testing on mill tailings that was conducted during this study.

This phase of the Project determined the location of the paste plant through the completion of a Location Trade-off Study (ToS #7), which concluded that the paste plant would be located on the crusher terrace of the processing facility.

This chosen location of the paste plant requires a surface pipeline approximately 380 m long with an increase in elevation of 8 m from the plant to the upper decline portal.

Although a second lower decline was added in this phase of the study, all paste backfill, required by the mine, will continue to be delivered via the upper decline.

Although no specific laboratory testing was conducted to measure the pipeline friction losses of paste flow in the UDS, WSP empirically determined the values from available laboratory data and applied the values in a hydraulic flow model assessment which provided the system design pressures thereby allowing the selection of the positive displacement paste pump as well as the materials of construction of the distribution system components, such as pipes and couplings.

The entire UDS is designed around the pressurised paste flow generated by a positive displacement pump, which is to say that there are no sections of the system dependant on gravity flow.

The model also determined that the lower friction losses associated with a higher slump paste would be required to limit the system operating pressure below a design value of 150 bar. The higher slump paste, however, will likely increase the binder addition rate required to achieve the target unconfined compressive strength required for mining adjacent to backfilled stopes.

Redundancy of system components from the paste pumps to the end of decline piping was maintained, with the one exception being to the surface pipeline component. There continues to be no redundancy in the distribution system piping in the orebody, however, all underground boreholes continue to have redundancy and continue to be cased, thereby mitigating possible poor rock quality concerns.

The cemented rock fill system utilises a modular plant on surface to mix the cement slurry that is transported underground by the fleet of transmixers, combined with development waste rock using LHDs and placed into approximately 55% of the volume available in an individual stope. This value is calculated based on dumping CRF from the stope overcut and the estimated angle of repose of the CRF in the stope. The remainder of the stope volume is filled with cemented paste backfill delivered through its underground distribution system.

1.11.13 Mine Dewatering

The mine dewatering system design has been updated from the PEA design, to reflect the introduction of the lower decline in the mine design.

All water leaving the mine will be pumped through the lower decline and report to the water treatment facility at the mill.

The main dewatering station is relocated to the 440L from its previous location at the intersection of the upper decline and the orebody.

The introduction of the main dewatering station has allowed many of the level and secondary sumps to gravity feed through boreholes to the 440L horizon, thereby reducing the need for pumps within the system.

Level sumps below the 440L horizon gravity flow to the lowest level sumps and the water is pumped through secondary sumps to a central secondary sump that pumps upwards and delivers all water from below the 440L to the main dewatering station.

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

- Groundwater seepage;
- Water required for mining operations, including drilling, dust suppression when mucking, and other ancillary water consumption;
- Paste backfill contained water release at placement and system flushing;
- Water generated from diesel fuel combustion emissions;
- Water in ore and waste leaving the mine; and
- Water in the ventilation system leaving the mine.

Hydrogeological data collected during the field investigations conducted as part of this study were used to develop a numerical groundwater flow model and to predict groundwater inflows into the planned underground development over the life of mine. The base case ground water inflow rates were used in the dewatering system design.

Water generated through mining operations is based on the revised mining plan as well as the current fleet of mobile equipment in use.

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. The water generated from distribution system cleaning and flushing was included in the estimate.

Water generated from diesel fuel combustion was estimated based on the mobile equipment fleet.

Water quantities leaving the mine in ore and waste, as well as in the ventilation system, were subtracted from overall mine water balance.

The resulting sum of all water generated and lost formed the basis of the dewatering system component design: sump volumes, pumps, pipeline, and borehole sizes.

1.11.14 Electrical and Communications

A PFS load list is developed based on the mine underground and surface ventilation, underground mine dewatering, underground mine mobile equipment, and surface paste backfill and dry tailings storage facility (DTSF) inputs.

Table 1.6 identifies the calculated peak and average load demand for both normal and emergency power.

Table 1.6 – Peak and Average Load Demand for Normal and Emergency Power

Description	Power Installed (kVA)	Normal Power		Emergency Power	
		Peak Demand (kVA)	Average Demand (kVA)	Peak Demand (kVA)	Average Demand (kVA)
DTSF – (Surface)	2,950	1,468	939	0	0
Underground Mine Mobile Equipment – (UG)	11,524	1,779	1,244	0	0
Mine Ventilation – (UG and AG)	4,131	2,425	2,385	1,016	1,016
Mine Dewatering – (UG)	7,603	1,872	1,310	351	246
Mine Non-Process Loads – (UG)	1,090	752	692	322	274

Power for the underground mine loads will be supplied from the surface substation main 10 kV switchgear 6720-SWG-002. 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. One feeder cable will be routed down from the switchgear breaker on busbar A to the upper decline and the second feeder cable will be routed down from the switchgear second breaker on busbar B to the lower decline.

These two (2) mine power feeders will provide a redundant connection to improve reliability. UG main electrical substations are connected through the ring (radial) system. This redundancy on feeder cables and UG ring power distribution design is especially important for critical loads and mine operation as well as ventilation, dewatering and mobile equipment.

Each main UG electrical substation that supplies power to the loads will house a 10 kV Ring Main Unit (RMU) switchgear that provides the ring design, a 10/1 or 0.69 kV transformer, a main distribution power centre, a 230 transformer & distribution board, and required Direct On-Line (DOL) or Variable Speed Drive (VSD) starters.

The 1000 V or 690 V power distribution output of the transformer will supply power to infrastructure and mining equipment. The final LV distribution voltage will be identified in the FS stage based on the latest mine level layouts, equipment location and cost study to reduce the cable costs and the number of the required main electrical substations to supply the mining equipment. Pumps and fans will be controlled by DOL or VSD controllers.

Two (2) redundant fibre optic cables will be routed through the upper and lower declines to the UG mine main electrical substations to provide backbone to the process, business, communications, vehicle and personnel tracking, surveillance, gas detection, and fire protection networks.

Surface main exhaust fans including the Electrical-House (E-House) and all accessories are tied to the surface main 10 kV and fibre optic rings.

1.12 Recovery Methods

Metallurgical testing showed that the mineralisation is amenable to a flowsheet featuring crushing, grinding and gravity concentration, followed by direct smelting of the gravity concentrate and bulk sulphide flotation of the gravity tailings. This process flowsheet will produce doré bars and two clean gold concentrates with an overall gold recovery around 87%. The proposed process plant will produce doré bars from the second stage gravity concentrate, a saleable gravity concentrate (the second gravity stage tailings) and a bulk flotation concentrate, all of which will be shipped off-site separately. Final plant tailings will be disposed of underground as paste backfill or above ground as filtered cake in a dedicated Dry Tailings Storage Facility (DTSF).

Preliminary testing showed that the material is of moderate competency and hardness with an average SMC test Axb value of 62 and a Bond Ball Mill Work Index of 13.7 kWh/t. Gravity and flotation recovery testwork conducted indicated an optimum grind of 80% passing 53 µm. A comminution trade-off study was conducted to evaluate the options available to DPM including the potential re-use of equipment from another operating mine that will be de-commissioned in the near future. The primary recommendation of the trade-off is to re-use the refurbished ex-Ada Tepe primary tumbling mill (Ada Tepe being DPM's Ada Tepe facility in Krumovgrad, Bulgaria), pebble crusher and vertical stirred mill. The primary mill will be operated in fully autogenous grinding mode i.e. with no grinding media added.

The gravity concentration circuit will be integrated into the grinding circuit with a portion of the secondary cyclone underflow diverted to the parallel gravity scalping screens. Screen oversize as well as centrifugal concentrator tailings will recycle to the grinding circuit. Screen undersize will report to the gravity concentrators. Two-stage gravity concentration with smelting of the final concentrate and selling the secondary gravity tails as an intermediate grade product is recommended for the PFS phase.

A trade-off study was conducted to compare flotation technologies. New Jameson cells were compared with re-using the Staged Flotation Reactors (SFRs) currently installed at Ada Tepe, augmented by a new SFR cell in each stage, as recommended by the Supplier. The Jameson cell option is recommended due to its simplicity and overall reduced costs.

The nameplate capacity of the processing plant will be 850,000 dry tonnes per annum (dmtpa). The crushing section will be staffed for 12 hours daily, during which it is expected to be utilised at approximately 80% (i.e. it will operate for 9.6 hours daily). The filtration section will operate at a reduced utilisation of 75%, while the grinding and flotation sections will operate 92% of the time. The gold room, which includes secondary gravity concentration, dewatering and smelting, will operate 5 days per week during daytime shift only.

Cement will be added at 5% to the paste backfill and will represent the most expensive reagent. Other reagents used in the process will include flocculant, PAX, MIBC, collector A3477, copper sulphate and smelting fluxes. The grinding media consumption rates are expected to be low due to the moderate hardness ROM ore combined with its below average abrasiveness.

A sitewide water balance was constructed and indicates that the site will have a positive water balance even during dry years. This is primarily due to the expected large volume of water influx to the underground workings. This water will be pumped directly to the clarifier, from which it will either be disposed of via the second stage water treatment plant or used as make-up in the processing plant. Another factor contributing to the positive water balance is that minimal water exits the site since all tailings will be disposed of as filtered cake on surface or as paste underground.

1.13 Project Infrastructure

The main Project on-site infrastructure components include:

- Roads and terraces.
- Site buildings.
- Filter cake and paste backfill system.
- Mine waste rock management.
- Dry tailings storage facility (DTSF).
- Water management.
- Power supply and distribution.
- Automation and Communication.

Project infrastructure elements were arranged to optimise the use of available space near the designated DTSF and process plant terrace. The ROM stockpile will be located close to the upper decline mine portal. Underground development waste rock will be used in the construction of the base of the DTSF. Minimal waste rock haulage is anticipated during LOM.

Mine dewatering will provide process, treated, gland seal, tap and mine make-up water. There will be three (3) ponds on-site:

- A mine dewatering pond that will be fed by dewatering of the mine and management of 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.
- A DTSF reclaim water pond that will either provide process water make-up or be treated through a water treatment plant (WTP) and returned to environment.

It is assumed that the mine dewatering contact water does not require chemical water treatment, only turbidity treatment via settling, and reverse osmosis prior to environmental discharge.

Support infrastructure is located on two (2) main terraces, namely the Administration and Process Terraces. The Administration Terrace is located at the entrance of the site and will house the main administration building as well as mine administration and mine dry. The Process Terraces will house all process plant related infrastructure, tailings filter building, paste backfill plant and mining infrastructure. An overall site layout is provided in Figure 1.1, and details are further explained and illustrated in Section 18.

1.13.1 Filter and Paste Plants

The level of design and detail aligned with that expected of a PFS-level of engineering definition.

The Filter Plant was designed to functionally dewater all the mill tailings for use in the production of Cemented Paste Backfill (CPF) or disposed of in the mine DTSF in the form of filter cake.

The Paste Backfill Plant will use a portion of the dewatered tailings, in the form of filter cake, and produce CPF to support underground mining.

When preparing CPF, all the filter cake produced by the Filter Plant will report to the Paste Plant.

Figure 1.1 – Site Layout

Source: DRA, 2024

1.13.2 Stockpiles Facilities

The Project site will contain a number of stockpiles over the operational life, including:

- Dry Stack Tailings Facility (containing both tailings and mine waste rock);
- Three Topsoil and two Overburden Stockpiles; and
- Preliminary Ore Stockpile.

A DTSF will be constructed within two valleys adjacent to the Process Terrace. Tailings will be filtered at the process plant and conveyed to the valley for construction of the DTSF. A proportion of the tailings produced on site will be stored underground as backfill and the remainder will be stored within the DTSF.

Mine waste rock produced from the underground excavation will be brought to surface and stored within the DTSF to encapsulate any potentially-acid generating (PAG) material. The preliminary ore will be stockpiled at surface adjacent to the upper portal, the stockpile was designed to store ore in the early stages of the Project. Two (2) topsoil stockpile locations were identified in the study, namely the initial topsoil stockpile and terraced topsoil stockpile. The former will be constructed on top of natural ground and the latter will consist of a series of terraces constructed from locally won overburden material.

1.13.3 Dry Tailings Storage Facility

The PFS for the Čoka Rakita Project in Serbia outlines the design and planning for a DTSF and associated water management structures. The design report details the following design elements.

The PFS optimises the previous PEA design. A Dam Break Analysis (DBA) was conducted early in the PFS, guided by the Canadian Dam Association (CDA), to determine design requirements based on potential DTSF failure impacts. Consequence Classification Assessment (CCA) was performed classifying the DTSF as "Significant" with minimal environmental impact and no effect on external infrastructure. The DTSF was designed to contain 3 Mm^3 of filtered tailings and 380,000 m^3 of mine waste rock. The DTSF design comprises PAG mine waste rock that will be stored in lined embankments with measures to prevent seepage of possible acid rock drainage (ARD) with filtered tailings placed above. The lining systems selected comprise a composite liner system of HDPE and GCL, overlain by a protective geotextile. Tailings transportation via trucks was selected over conveyors due to operational flexibility and lower costs. The project will present a continuous operation, 24/7, with tailings transported and deposited in an 8-hour daily shift. The DTSF ring-road and additional access roads are included with the PFS design. The design is presented over four (4) construction phases, spanning over ten years, focusing on material balance and site preparation. Preliminary instrumentation included in the design will be installed in all embankments and pond structures to monitor water levels and stability during construction and operations.

The Čoka Rakita DTSF will operate 8 hours a day, 336 days a year, with filtered tailings produced and transported to a storage terrace within the DTSF footprint. During operational hours, tailings are loaded into trucks and hauled to the DTSF. In non-operational hours, tailings are stockpiled adjacent to the process terrace. A wheeled front loader will feed trucks from the storage terrace, and in poor weather, tailings are moved to a covered laydown area. Trucks will end-tip tailings into the deposition zone, where a dozer will spread them into 500 mm layers, compacted to 300 mm by a soil compactor. Excavators will be used to maintain drainage ditches and channels. Initial development requires significant amounts of clean rock and overburden fill. PFS closure plans consider final slope gradients and capping options designed to limit seepage. Progressive and active closure plans include maintaining water infrastructure and monitoring until pre-mining conditions are met. Passive closure involves no additional treatment, with potential design modifications based on climate change scenarios. The project risk register identifies key hazards and mitigation actions, with top risks including dam failure, PAG, seepage, and instability from inadequate construction materials. Further ground investigation, laboratory testing, optimisation, stability and seepage assessments, and risk register development are recommended for the FS.

The Reclaim Pond, situated at the toe of the DTSF, is designed to manage precipitation and contact water during operations. The total capacity of 41,000 m^3, with 10,000 m^3 allocated for operating volume. Water will be pumped to the mine dewatering pond to maintain a low operating level. A composite lining system will inhibit seepage from the pond. To avoid structural damage from a potential DTSF dam breach, the pond will be strategically positioned 50 m from the downstream toe. A basal drain extends under the reclaim pond to convey groundwater downstream of the facility. Stability assessment is recommended for the next phase to ensure compliance with CDA and Serbian standards. The lining system selected comprises GCL and HDPE, anchored into the access road corridor with an anchor trench and backfilled with aggregate for stability.

The mine dewatering pond, with a capacity of 53,000 m^3, is designed to manage water from various sources, including the reclaim pond, upstream catchment runoff, and the ROM Pad and Terrace area. It is designed to includes a surge capacity for 5 days of WTP maintenance. An access road will be constructed around the pond for monitoring and inspection, connecting to the main haul road. Pumping and decant systems will be designed later. Internal drainage for potential seepage will be addressed in the next phase. Stability assessment is recommended for the next phase to ensure compliance with CDA and Serbian standards. The lining system selected utilises a composite lining system of GCL and HDPE, anchored with an aggregate-filled trench for stability.

The non-contact water pond, with a capacity of 63,000 m^3, is designed to capture discharge from the WTP and runoff from the southeast catchment. Key features include that the design can supply the process plant for approximately two months. An access road will be constructed around the pond for monitoring and inspection, connecting to the main haul road. Pumping and decant systems will be designed later. Internal drainage for potential seepage will be addressed in the next phase. Stability assessment is recommended for the next phase to ensure compliance with CDA and

Serbian standards. The lining system selected utilises a single layer of HDPE to collect clean water, not expected to be contaminated by seepage through filtered tailings.

The Project's Preliminary Ore Stockpile is designed to temporarily store excavated ore near the upper portal and process terrace. The design is to be built early in the LOM before processing begins. It includes a 10 m perimeter gap for access roads and drainage channels, with ramps having a maximum 10% slope. A 2 mm HDPE liner (19,000 m^2) is included in the design to prevent acidic leachate from seeping into the terrace. Stability assessment conducted using Morgenstern-Price and Fellenius/Ordinary methods, along with seismic analysis, confirming stability under various conditions and meeting Serbian standards. Contact water from the stockpile is managed by perimeter channels, discharging into the terrace water management systems and ultimately flowing to the mine dewatering pond.

The Project's topsoil stockpiles are designed to store all site-generated topsoil and overburden excavated during the LOM. The design includes topsoil thickness of 0.3 m, maximum height of 10 m, slope of 1:3 (V:H), and density of 1.6 t/m^3. Seven locations were assessed, with the "Terrace Stockpile" and "Initial Topsoil Stockpile" selected. The Terrace Stockpile will store both overburden and topsoil, while the Initial Topsoil Stockpile requires no earthwork preparation. Stability assessment conducted using CDA (2013) guidelines and the Morgenstern-Price method. Factors of Safety (FoS) of 2.2 and 1.8 were calculated for the Terrace and Initial Topsoil Stockpiles, respectively, under drained static conditions. Further assessments, including undrained and seismic stability, are recommended for the next phase. Runoff from both stockpiles is classified as non-contact water.

1.13.4 Site Water Management and Treatment

1.13.4.1 Surface Water Management

As part of the PFS, WSP developed the Surface Water Management design of the following areas:

- DTSF Water Management, comprising:
  - Non-contact Water Channels;
  - Contact Water Channels;
  - Reclaim Pond;
  - DTSF Spillways; and
  - Reclaim Pond Spillways.
- Stockpile Water Management, comprising:
  - Terrace Topsoil Stockpile Non-Contact Perimeter Channels;
  - Preliminary Ore Stockpile Contact Perimeter Channel; and
  - Initial Topsoil Stockpile Non-Contact Diversion Channel.

- Mine Dewatering Pond Water Management, comprising:
  - Mine Dewatering Pond;
  - Mine Dewatering Pond Diversion Channel; and
  - Mine Dewatering Pond spillway.
- Non-Contact Water Pond Water Management, comprising:
  - Non-Contact Water Pond; and
  - Non-Contact Water Pond Spillway.

1.13.4.2 Sitewide Water Balance

WSP has developed a sitewide salt and water balance model (SWBM) as part of the PFS using GoldSim modelling software, covering three (3) phases of the LOM:

- Start of decline construction (Year -2 to Year -1);
- Process plant commissioning (Year 1); and
- Operations (Year 1 to Year 10).

The elements included in the SWBM comprise:

- Contact water management infrastructure, namely Reclaim Pond, Mine Dewatering Pond and Non-Contact Water Pond;
- Process Plant;
- WTP;
- Underground mine water management (total outflows taken as a single input to model); and,
- DTSF.

The SWBM includes features to estimate:

- Anticipated inflows (including runoff, direct rainfall, and groundwater ingress) into mine site infrastructure.
- Consumption of water by the mining process.
- Water quality within the mine dewatering pond, reclaim pond and non-contact water pond.
- Release of water from the operation into the natural environment.
- Residual volumes of water to be managed safely on site to ensure operational integrity.

To simulate the site's proposed water management systems, the SWBM was set-up stochastically and run for 100 realisations, thereby introducing variability into the results and allowing the model to assess site water management under periods of both flood and drought conditions. The SWBM also encompasses a contaminant transport component to assess site water qualities.

The SWBM was set-up to simulate both historical observed climate data and climate change projections to stress-test the site resilience to future climate conditions. Additional analysis was conducted on the sensitivity of WTP treatment rate and failures of the underground dewatering system.

1.14 Environmental Studies, Permitting and Social or Community Impact

1.14.1 Introduction

As the Project is located in Serbia, it will be permitted to operate and be 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), Institute for the Preservation of Cultural Heritage and the Institute for Nature Conservation of Serbia.

Serbia was granted EU 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.

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.

1.14.2 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.

1.14.3 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.

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. While a decision by the Serbian government to initiate the development of the Special Purpose Spatial Plan is currently pending, the Company's approach includes having all preparatory work completed and ready for submission while continuing to proactively engage with relevant stakeholders to mitigate the risk of administrative delays.

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 and adjacent tributary valley, 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 August 2024, and periodically reviewed throughout the PFS. 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.

1.14.3.1 Watercourses and Groundwater

A network of rivers and streams runs through the Project area (Dumitrov Stream, Ogašu Lu Gjori Stream, Valja Saka River and the Lipa River – Figure 20.2). 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 DTSF will be situated over the Dumitrov Stream and adjacent tributary valley, while the process plant, UG mine portals, 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 DTSF, the DTSF reclaim pond and the south of the process plant terrace where the ROM pad is located.

1.14.3.2 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.

Following baseline surveys undertaken in 2024, the presence of IUCN Red List Vulnerable species, and species listed in Annex II and Annex IV of the Habitat Directive meet the current criteria for 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.

A number of nationally protected habitats and species were identified in the project area; and these will require detail mitigation plans to avoid, reduce and restore impacts in line with the mitigation hierarchy approach and best practice guidelines.

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 to be 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.

1.14.3.3 *Noise, Vibration, Nuisance Dust and Greenhouse Gas Emissions*

Baseline noise surveys began in 2022, two (2) seasonal campaigns are planned, of which the summer survey was completed in 2022. Both sets of 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 early in 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, 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 locations, monitoring of NO_2 and SO_2 will continue at five (5) locations throughout 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.

1.14.3.4 *Soil and Geology*

An assessment of the soil in the context of the baseline physical parameters and chemical quality was completed in October 2024, with results of the survey program reported in November 2024. Consistent with the PFS planned layout, where topsoils are to be disturbed, the principle will be to conserve these in stockpiles for ultimate reuse as a rehabilitation medium. Soils were predominantly silty loams with naturally elevated metal concentrations recorded across the 500 ha survey area. These results will be considered further in the Feasibility Study (FS).

Geotechnical studies have commenced; however, no results are yet available from these studies. It is, however, noted that many of the Project infrastructure locations, especially access roads and the DTSF, are on very steep terrain. Risks from landslides and other ground instability will need to be assessed by detailed geotechnical investigations in the future.

1.14.3.5 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 DTSF, 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 located at various locations along the route of the line, and at Bor. The railway and the location of the labour camps 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 Project is also located close to the site of two (2) of the labour camps (one to the north called Westfalen-altabor, and one to the south called Tirol-altabor), whose exact location have yet to be determined.

The environmental impact assessment will address potential impacts on both tangible and intangible heritage. Any rural and railway structures likely to be disturbed will be recorded archaeologically.

1.14.3.6 Social Setting, Land Acquisition and Livelihoods

Land acquisition is required for the development of the Project, access to the site and 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, and 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 development of a more detailed Land Acquisition Plan, once the Project design is complete.

1.14.4 SOCIAL AND COMMUNITY ENGAGEMENT

DPM has worked to establish good relationships with the local community since 2007 and communications are managed through 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 groups which will require targeted engagement.

1.14.5 MINERAL AND NON-MINERAL WASTE

1.14.5.1 Mineral Waste

Prior to 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. Waste rock, conservatively assumed to be PAG, will be stored within the DTSF for the first two years of operation. After the first two years, the waste rock will remain underground; and be stored underground with paste backfill. There will be no other waste rock facilities required by the operation or in construction.

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, eight (8) classify as PAG, thirteen (13) as uncertain, and eight (8) as non-PAG.

A representative set of samples of all rock and tailings materials will be subject to long-term kinetic testing. Excavated rock, to be 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 to spatially model and predict the ML/ARD characteristics of rocks and tailings across the deposit.

1.14.5.2 *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.

1.14.6 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.

1.15 Capital and Operating Costs

The capital cost estimate (Capex) and operating cost estimate (Opex) were compiled by DRA; and are based on the scope of work presented in sections of 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. All contributors to the Capex and Opex are outlined Table 1.7.

Table 1.7 – Capex/Opex Responsibilities

Consultant	Scope
DRA	Overall responsibility for Capex/Opex, Process Plant, Infrastructure, On-site infrastructure.
WSP	Prepared underground mining and mining infrastructure direct costs for initial and sustaining Capex and Opex. Prepared initial direct Capex for Filter and Paste Plant Equipment; where DRA estimated bulk earthworks, concrete, structural steel, architectural, and electrical requirements. Prepared construction quantities for DTSF, Water Management and topsoil / over-burden / waste rock stockpiles; and DRA applied rates to establish direct capital costs.
DPM	Provided Owner’s General and Administration Costs, Taxes, Duties; and Closure Costs.

1.15.1 Capital Cost Estimate

The Capex was developed to deliver an overall accuracy range of -20% to +30%. Ranges could exceed those shown, if there are unusual risks.

In this particular case, the Capex reflects an EPCM-type execution model. Although some individual elements of the Capex may not achieve the target level of accuracy, the overall Capex should fall within the parameters of the intended accuracy.

All Capex and Opex costs are expressed in United States Dollars (US$ or $) and are based on Q4 2024 pricing.

In the case of the mine portion of the Capex, the Capex reflects an Owner's crew supervised and managed by Owner's personnel type execution. DPM expects that there is a possibility of expatriate assistance from its Bulgarian supervisors and senior operators, for the decline construction.

The Capex consists of direct and indirect costs, as well as contingency. Provisions for sustaining capital are also included.

Table 1.8 presents a summary of the Initial Capex, and Sustaining Capex distributed over the LOM indicated separately. Owner's costs, contingencies and risk amounts are included in this Capex.

Table 1.8 – Capital Cost Summary

Area	Description	Initial Capex (US$ M)	Sustaining Capex (US$ M)	Total (US$ M)
2000	Underground Mine	85.01	21.43	106.44
3000	Ore Handling	13.57	0.00	13.57
4000	Processing Plant	51.95	0.00	51.95
5000	Filtered Tailings / Water Treatment Facilities	34.49	7.60	42.09
6000	On-Site Infrastructure, Site-Wide General	54.66	0.00	54.66
7000	Incoming Power	1.42	0.00	1.42
8000	Operational Readiness	28.07	0.00	28.07
9000	Indirect Costs	46.38	0.00	46.38
9100	Owner's Costs	13.65	0.00	13.65
9900	Project Contingency	49.83	0.00	49.83
9900	Project Closure and Rehabilitation[1]	0.00	27.30	27.30
	Total Major Area Capex	**379.03**	**56.33**	**435.36**

Numbers may not sum precisely due to rounding.
1. Closure costs include the non-recoverable VAT of approximately $2.3 M.

1.15.2 Operating Cost Estimate

A summary of the overall LOM project Opex is presented in Table 1.9. The Opex costs presented in the table exclude pre-production Opex allowances for mining, process and G&A. These are covered in the Capex. Labour rates for this Report were provided by DPM. Direct employment during operations will total approximately 532 people including mine, concentrator, tailings, maintenance, management, and infrastructure. The 6,632,883 tonnes milled provide the unit costs shown in Table 1.9.

Table 1.9 – Opex by Major Area

Area	**LOM Total Opex** (US$ M)	**Average Opex** (US$/t milled)
Mining	251.1	37.85
Processing and Tailings	165.7	24.98
G&A	98.3	14.82
Royalties	107.1	16.15
Offsite Costs	69.2	10.43
Total Opex	**691.3**	**104.23**
Numbers may not add due to rounding.		

1.16 Economic Analysis

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

The economic analysis is based on the discounted cash flow (DCF) method on a pre-tax and after-tax basis. The key metrics determined in the analysis are the Net Present Value (NPV) at a discount rate of 5%, the Internal Rate of Return (IRR), and the Payback Period. 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, supporting a 10-year life of the Project with 8-year of processing period. All prices and costs are in US dollars and accurate as of Q4 2024. No provisions have been made for the effects of inflation.

Table 1.10 provides a summary of the key technical assumptions and inputs. At an assumed long-term gold price of $1,900 per ounce, the financial results indicate a positive pre-tax NPV of $735.0 M, at a discount rate of 5%, a pre-tax (and after-tax) IRR of 41.4% and a payback period of 1.7 years.

Table 1.10 – Economic Analysis Parameters

Description	Unit	Value
Macroeconomic Parameters		
Gold Price	US$ per oz	1,900
Discount Rate	%	5.0
Project Parameters		
Mine Life	years	8
Mineable Mineral Resource (LOM)	Mt	6.6
Grade Mined (LOM average)	g/t	6.38
Annual Mill Throughput	tpa	850,000
Gold Recovery (LOM average)	%	86.8
Gold Payability (LOM average)	%	98.5
Total Gold Produced (LOM)	Moz	1.2
Average Annual Gold Production (LOM)	oz	147,000
Average Annual Gold Production (first five years)	oz	170,000
Government Royalty (NSR)	%	5.0
Capital Cost Estimate		
Initial Capital	US$ M	379
Sustaining Capital (LOM)	US$ M	29
Closure Costs[2]	US$ M	27
LOM Operating Unit Costs		
Mining	US$ per tonne processed	38
Processing	US$ per tonne processed	25
General & Administrative	US$ per tonne processed	15
Royalties	US$ per tonne processed	16
Offsite Cost	US$ per tonne processed	10
Total Opex	*US$ per tonne processed*	***104***
LOM Average All-in Sustaining Cost [1]	US$ per oz gold	644

1. Numbers may not add due to rounding.
2. Closure costs include the non-recoverable VAT of approximately $2.3 M.

[1] All-in sustaining costs are non-GAAP financial measure or ratio and has 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 be reconciled to the nearest comparable measures under IFRS. Refer to the "Non-GAAP Financial Measures" in Section 2.7 of this Report for more information, including a detailed description of each of these measures.

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 a 10-year life of the Project with 8-year of processing period. Owing to this tax relief, the after-tax economic metrics are the same as the pre-tax metrics, as shown in Table 1.11.

Table 1.11 – Economic Results Summary

Description	Unit	Pre-Tax	After-Tax
NPV @ 5%	US$ M	735.0	735.0
IRR	%	41.4	41.4
Payback Period	Years	1.7	1.7

Numbers may not add due to rounding.

A sensitivity analysis was carried out to assess the impact of variations in gold price, Capex, and Opex on the NPV and IRR. The after-tax results of the sensitivity analysis are shown in Table 1.12 to Table 1.14 and Figures 1.2 and 1.3. The NPV and IRR of the Project are most sensitive to variations in the gold price followed by Opex and Capex. The Project retains a positive NPV at the lower limit of the price interval tested. The IRR is more sensitive to variations in Capex than Opex, as evidenced by the steeper slope of the Capex curve in Figure 1.3. The NPV appears to be equally sensitive to variations in Capex and Opex. Overall, within the limits of accuracy of the cost estimates in this study, the Project's potential after-tax viability does not seem significantly vulnerable to the under-estimation of capital and operating costs up to 20%, when taken individually.

Table 1.12 – Economic Metrics Sensitivity to Variations in the Gold Price

Au Price	Units	+20%	+10%	Base	-10%	-20%
NPV @ 5.0%	US$ M	1,043.3	889.2	735.0	580.9	426.7
IRR	%	52.0%	46.8%	41.4%	35.5%	29.0%
Payback	Years	1.4	1.5	1.7	1.9	2.2

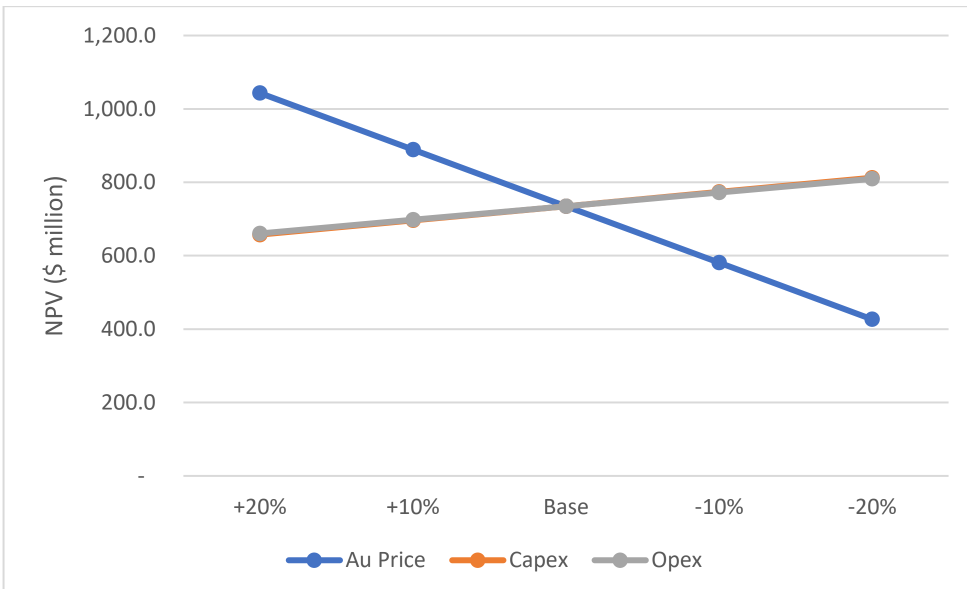
Table 1.13 – Economic Metrics Sensitivity to Variations in the Capex

Capex	Units	+20%	+10%	Base	-10%	-20%
NPV @ 5.0%	US$ M	657.6	696.3	735.0	773.7	812.5
IRR	%	33.9%	37.4%	41.4%	46.0%	51.4%
Payback	Years	2.0	1.8	1.7	1.5	1.4

Table 1.14 – Economic Metrics Sensitivity to Variations in the Opex

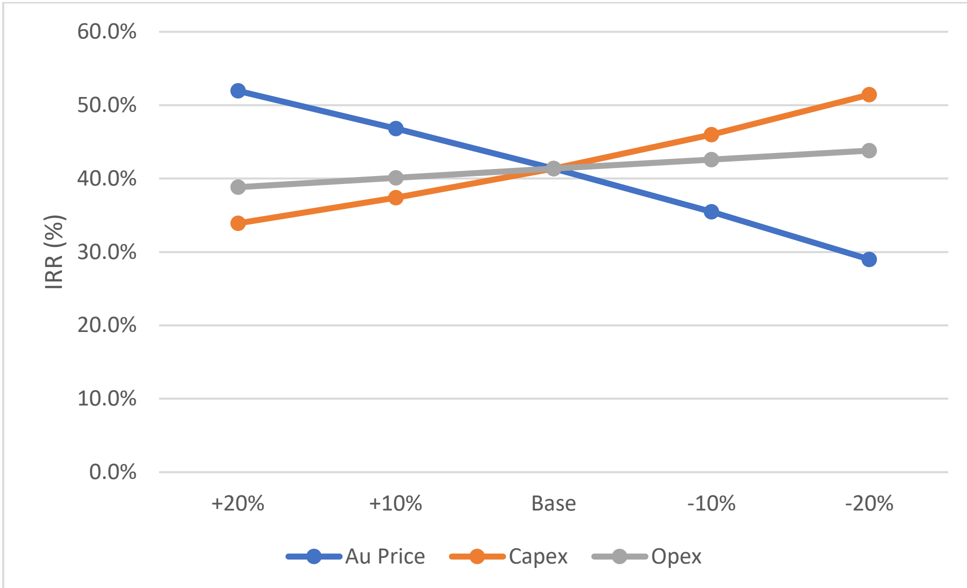
Opex	Units	+20%	+10%	Base	-10%	-20%
NPV @ 5.0%	US$ M	660.6	697.8	735.0	772.2	809.4
IRR	%	38.8%	40.1%	41.4%	42.6%	43.8%
Payback	Years	1.8	1.7	1.7	1.6	1.6

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 1.15 with the base case highlighted.

Table 1.15 – Gold Price NPV and IRR Sensitivity After-Tax

Gold Price	Unit	US$ 1,500/oz	US$ 1,700/oz	US$ 1,900/oz	US$ 2,300/oz
IRR	%	28.3%	35.1%	41.4%	52.5%
NPV at 5%	US$ M	$410.5	$572.7	$735.0	$1,059.6

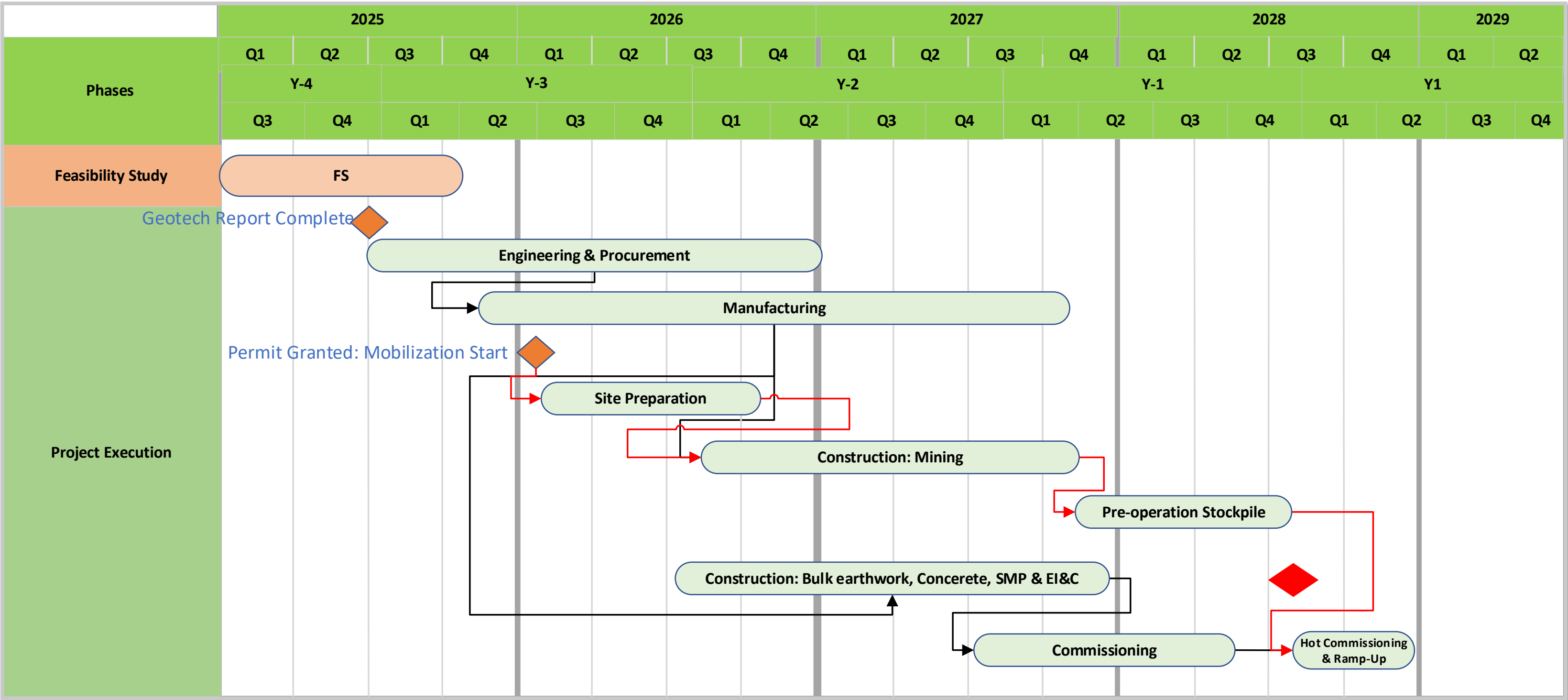
1.17 Other Relevant Information

1.17.1 Project Implementation Schedule

The project implementation schedule, adopted for the PFS, was prepared and includes all work leading up to the commissioning and startup of the Project. The schedule assumes continuous work commencing with the completion of the PFS, preparation of the Feasibility Study (FS) to be completed by mid-2025; and then basic engineering, leading to the project implementation, scheduled for production in Q3 2028.

A high-level Project Development and Implementation Schedule is provided in Figure 24.1, based upon the Project information available to date.

Figure 1.4 – Overall Key Date Schedule



Site Preparation: Trees Removal, Temporary Access Road, Terrace and Box Cut

Source: DRA, 2024

1.17.2 Project Execution Plan

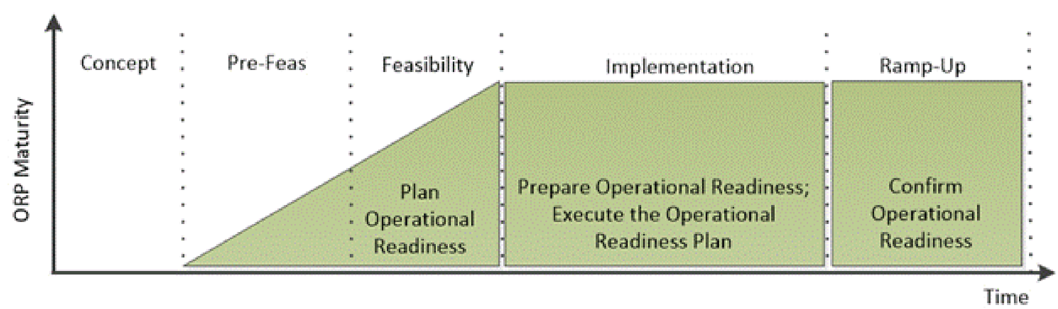
The Project Execution Plan (PEP) defines the methods and project management elements that will be used to manage the execution of the Project. The PEP establishes the execution philosophy and defines the organisation, work processes and systems necessary for management of the Project. The PEP will be further developed by the DPM project team during the FS and basic design phases.

1.17.3 Operation Readiness Development

The Operation Readiness Plan (ORP), developed by the DPM Operations Readiness (OR) team, will be focussed on readying the operational business, to effectively accept handover from the construction contractors; and prepare the operations and maintenance teams for day-to-day operations of the entire facility. The OR team will be directly involved in the commissioning and ramp-up activities, monitoring the contractor completion activities and checklists, while preparing the operators and maintenance personnel for day-to-day operations.

The preliminary ORP was developed by DPM during the PFS stage. The plan and scope will be matured, by the DPM OR team, during the FS phase; with further enhancements established during the basic and detailed engineering phases. The ORP, also, incorporates the handover period, the ramp-up period and steady state operations of the Čoka Rakita facility.

Figure 1.5 – ORP Process



Source: DPM, 2024

Prior to addressing areas identified in the ORP framework, an overview of operational phase strategy and assumptions will be developed by DPM.

1.18 Interpretations and Conclusions

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

1.18.1 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 30 August 2024. A total of 271 drillholes totalling 126,495 m were included in the estimation of the MRE. The current drillhole spacing within the mineralised domains is predominantly at least 30 m x 30 m. 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 20% in the main mineralisation domain. A further strategy of capping within the estimate at distances greater than 30 m was also applied.

Mineral resource domains were created within volumes of moderate to intense skarn alteration and guided by grade composites generated at an approximate1 g/t Au cut-off value. Detailed lithology and structural models were developed and used to constrain domain extents, as well as to incorporate mafic sills which can either cut across the mineralisation or are capping structures to trap the mineralisation as well as continuous internal waste units. 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 5 mZ parent block size with sub-celling to honour domain volumes.

The Mineral Resource was reported exclusive of Mineral Reserves. A breakeven cut-off value of 2 g/t Au and a minimum width constraint of 3 m was used to define optimised mineable shapes using Deswik's Stope Optimizer (DSO) process to support Reasonable Prospects for Eventual Economic Extraction (RPEEE).

Material within the reporting constraints was classified as Indicated and Inferred Mineral Resources, according to Mineral Resource confidence definitions 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. A Drill Hole Spacing Study (DHSS) was completed in 2024 which supported drill spacing of 30 m x 30 m for the classification of Indicated Mineral Resources, based on achieving a ±15% tolerance (with 90% confidence) on annual production volumes (Parker et al, 2014)

1.18.2 Mineral Processing and Metallurgical Testing

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 PFS.

In the QP's (Mr. Niel Morrison, P. Eng.) opinion:

- Metallurgical testwork completed to date has been appropriate to establish appropriate 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.

The QP is not aware of any processing factors or deleterious elements, other than arsenic, that could have a significant impact on potential economic extraction. Blending of the final concentrates will be required to minimise penalties for exceeding the arsenic grade threshold.

1.18.3 Mineral Reserve Estimate

Under the assumptions and parameters discussed in this Report, the Čoka Rakita Project shows positive economics that supports the recommendation to advance the Project to the FS and later to the decision to approve construction.

1.18.4 Mining Methods

Conclusions on mining methods are contained in Section 25.4, for the following aspects:

- Underground geotechnical;

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

1.18.5 Recovery Methods

The main conclusions established during the studies are:

- For the treatment of gravity concentrate the logical solution to the analytical challenges posed by the presence of coarse gold is to concentrate it into a high-grade secondary gravity concentrate which can then be smelted and sold as doré bars.
- The most economical and most flexible flowsheet is to re-use the equipment at Ada Tepe with the tumbling mill operated in autogenous grinding mode, which eliminates the operating cost associated with grinding media. The Ada Tepe pebble crusher will be re-used to avoid build-up of pebbles in the primary circuit. One of the vertical stirred mills from Ada Tepe will perform secondary grinding duties. Both circuits will be closed out by hydrocyclones.
- Jameson cells will be used instead of the refurbished SFR cells ex-Ada Tepe. The Jameson circuit is significantly simpler and less expensive; and its adoption avoids the complexity and expense of de-commissioning and dismantling the larger number of SFR cells at Ada Tepe.

1.18.6 Filter and Paste Backfill Plant

The PFS design of the Filter and Paste Plants is based on information provided by DPM and laboratory testing by RMS Corp. and BaseMet Laboratories. The level of engineering and design is commensurate with that expected for a PFS design.

The design and capacity of the Paste Plant is solely focused on the use of cemented paste fill (CPF) for the mine since although cemented rock fill (CRF) will offset the quantity of CPF required in certain years, there will be years near the end of mine life which will solely rely on CPF, since cemented rock fill will no longer be available.

1.18.7 Environmental Studies, Permitting and Social or Community Impact

Conclusions reached for the Project are contained in Section 25.7.

1.18.8 Capital and Operating Cost Estimates

The estimated capital and operating costs are reasonable for a PFS level analysis.

1.18.9 ECONOMIC ANALYSIS

Based on the available information, the project has an after-tax NPV of $735.0M at a discount rate of 5% and an IRR of 41.4%. The sensitivity analysis indicates that the Project economics are most sensitive to the gold price. Even with a gold price 20% below the base case of $1,900/oz, the Project maintains a positive after-tax NPV.

1.18.10 OTHER RELEVANT INFORMATION

The Project is considered a greenfield site which involves construction of a 70 kt per month concentrator plant. The equipment and infrastructure to be procured for the Project constitute a blend of new equipment and equipment to be relocated from DPM's Ada Tepe facility in Krumovgrad, Bulgaria.

The Project Implementation Schedule assumes a continuous flow of work from completion of PFS engineering activities through to commissioning and startup of the Project, in 2028. Various consultants will participate in the future work phases.

DPM will finalise the ORP during the FS stage.

The Project Execution Plan and Schedule will continue to evolve in the FS and basic engineering phases of the Project; with commissioning and startup of the Project currently planned for 2028.

1.18.11 RISKS

The Project team (DPM, DRA, ERM, WSP) conducted a multi-day project risk review workshop during the week of August 12, 2024, with the expectation of identifying and evaluating potential risks that could impact the overall success of the Project.

During this workshop, potential risks were identified, with each item's cause, likelihood, and potential impact evaluated across different inherent risk categories. The PFS risk review workshop utilised the risk register established during the PEA as the basis for the PFS workshop.

At completion of the PFS risk workshop, additional activities were conducted. The first activity grouped similar risks together, as per DPM corporate risk procedure and then applied the DPM corporate 5x5 risk matrix to establish the individual risk ratings. After the DPM ratings were established, a second activity determined the potential impact reductions which could be assumed, if effective mitigation measures are implemented into the design and/or procedures.

The total number of risks identified using the DPM corporate methodologies was 90 risks; which comprised: 36 "High" risks, 48 "Moderate" risks, and 6 "Low" risks.

Once the mitigation activity was completed, it was identified that several of the risks could be potentially re-rated to comprise: 11 "High" risks, 60 "Moderate" risks, and 19 "Low" risks.

During the FS, significant work will be undertaken to try and eliminate all E Extreme- and High-risk items.

For Additional details related to the Project Risk Review, refer to Section 25.12.

1.19 Recommendations

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

1.19.1 Proposed Work Program

The following activities should be undertaken in the next phase of the Project, leading up to the completion of the FS. These activities as well as their estimated costs are presented in Table 1.16.

Table 1.16 – Estimated Budget for Next Phase

Activities	Estimated Budget (US$ M)
Geology and Exploration	0.2
Geotechnical / Hydrological	1.6
Mining	1.3
Metallurgical Processing	0.4
Environmental Studies	0.7
Permitting	1.6
Feasibility Study	11.1
Engineering	3.4
Subtotal	*20.3*
Contingency	2.3
Total	**22.6**

Excluding operational readiness, land acquisition, and Owner’s cost.

1.19.2 Geology

The work programs set out below are part of the next phase of the Project, unless otherwise stated.

1.19.2.1 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.

1.19.2.2 *Drilling*

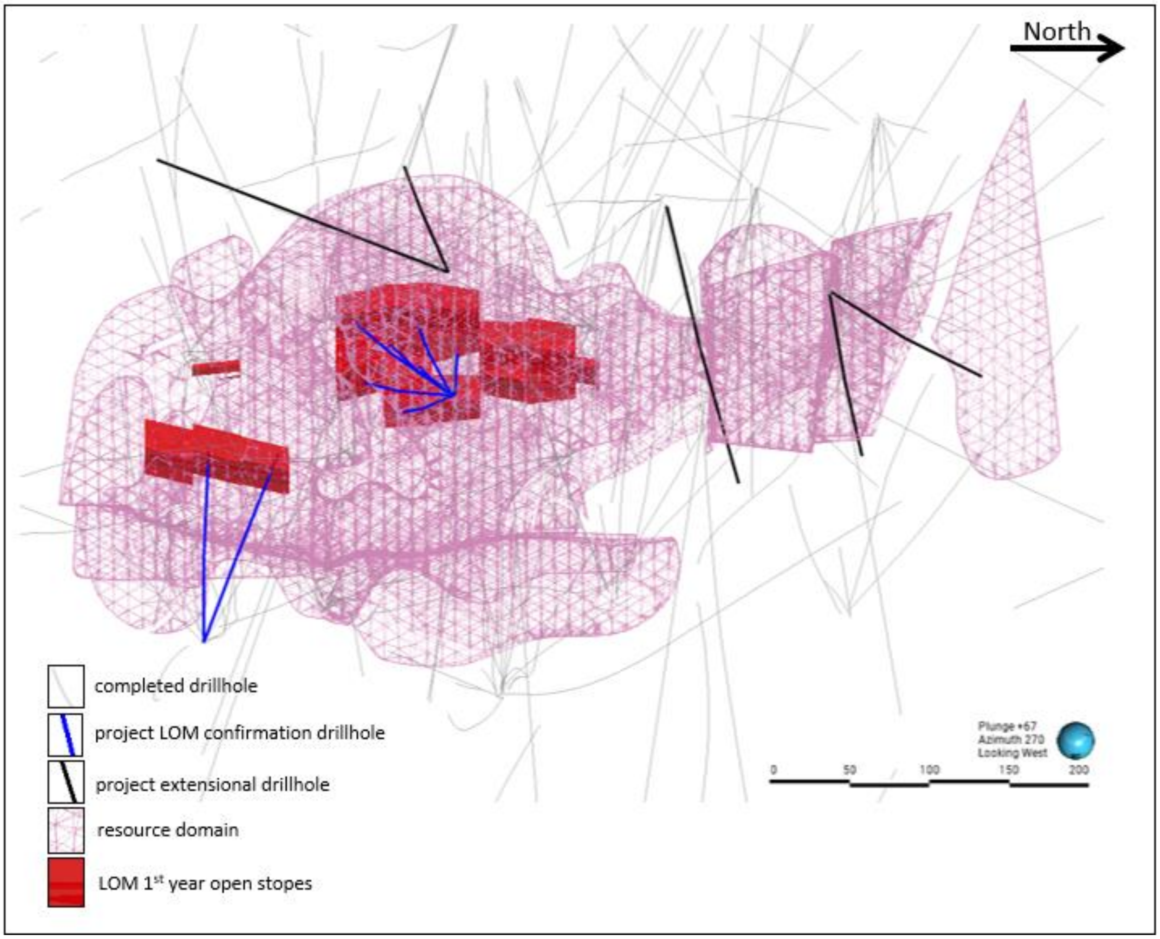
DPM is planning a drilling program by the end of 2025 to support further technical studies and ensure alignment between production and the FS model (Table 1.17). Current drilling plans include:

- Approximately 5,000 m of drilling will be conducted as part of the FS study for the Čoka Rakita Project. This phase includes close-spaced infill drilling (15-20 m drill grid), with a primary goal of better constraining mineralisation and minimising grade uncertainty within the volumes planned for mining during the first year of the PFS LOM. In addition, approximately 4,000 m of drilling are planned in the peripheral parts of the mineralised domains. These drillholes are designed to test for extensions to the current mineralisation domains (Figure 1.6). Additionally, about 3,600 m of near-surface geotechnical and hydrogeology drilling is planned for site infrastructure investigations.
- Additionally, DPM has plans to complete approximately 40,000 m of additional exploration drilling at existing skarn targets and to test for manto-like copper-gold skarn identified across the Čoka Rakita licence and neighbouring licences held by DPM, including Potaj Čuka and Pešter Jug.

Table 1.17 – Čoka Rakita Licence – FS Planned Drilling Metres and Budget

Phase	Drilling category	Planned Metres	Budget (US$)
FS	LOM 1st year confirmation drilling	5,000	900,000
FS	Extensional drilling	4,000	700,000
FS	Geotechnical/hydrogeology drilling	3,600	600,000
2025 Exploration	Exploration drilling	44,000	7,900,000
Total	**All**	**56,600**	**10,100,000**

Figure 1.6 – Confirmation and Extensional Drilling Plan for FS at Čoka Rakita



Source: DPM, 2024

1.19.2.3 Database

DPM is using a reliable and well-known solution to capture and manage the data (acQuire). However, the database and data management practice are still evolving. To ensure that 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 sulphide and vein table, meaning, e.g. missing Lithology 1, Sulphide 1, Vein 1 respectively. Generally, this would lead to a validation error in relational databases, meaning a non-standard approach may have been used and this should be modified.
- Hole size information is present in the Geotech table for the diamond holes but no information was present for other hole types. Best practise suggests that this be stored in a table for all holes and should also contain information such as start/end dates for each particular hole size, drill contractor and rig as a minimum.

- There are nine diamond holes that are missing recovery and RQD information with 0.03% of data having a percentage value outside the recommended tolerance of 110%, this data should be reviewed.
- Structure data should be presented in two separate tables, one for interval data and another for point data.
- 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 exported with the CRM data.
- 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.

1.19.2.4 Assay QA/QC

No gold 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 123 umpire samples were selected and assayed during 2024 at ALS_BO laboratory, however, these were not analysed for gold. The silver, arsenic, copper and sulphide results reviewed demonstrate overall strong consistency and reliability between the original and umpire assay data, suggesting confidence in the data quality. Minor outliers and discrepancies should be reviewed, especially for lower concentration samples.

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 were obtained and inserted for this purpose in 2024 e.g. OREAS 993.

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.

For the QC data (Duplicates, Repeats and Splits) the overall quality of the data appears to be good, but attention should be paid to the outliers at higher concentrations, particularly for gold and silver.

1.19.3 MINERAL PROCESSING AND METALLURGICAL TESTING

Proposed future testwork programs are recommended in Section 26.3. Piloting of the Jameson cells is recommended, as this will resolve most of the remaining uncertainties regarding the behaviour of the material.

1.19.4 MINERAL RESOURCE ESTIMATE

It is recommended that the MRE be updated, based on new drilling and screen fire assays, where possible. This will form the basis of the FS, due to commence in 2025.

1.19.5 MINERAL RESERVE ESTIMATE

DPM should prioritise infill drilling within the deposit areas defined in the PFS, to rapidly convert Inferred Mineral Resources to the Indicated category. Simultaneously, an exploration expansion drilling program should be conducted to grow the mineral resource base and enhance the Project's overall value.

1.19.6 MINING METHODS

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

- Underground mining;
- Geotechnical;
- Hydrogeology;
- Mine ventilation;
- Mine backfill; and
- Mine dewatering.

1.19.7 FILTER AND PASTE BACKFILL PLANT

Refinements in the LOM Plan will ultimately determining the maximum annual CPF production requirements and therefore the overall paste backfill plant design capacity.

- The paste backfill plant design criteria and equipment sizing should be re-examined, and a determination made if revisions are required based on the FS LOM Plan.
- Upon selection of the major equipment vendors, equipment dimensions should be incorporated into the Project design and based on the equipment, the plant functionality refined, as required.

1.19.8 RECOVERY METHODS

Additional strategies for ensuring that the flotation concentrate product avoids attracting penalties for arsenic content should be explored in future phases of the Project.

1.19.9 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, although nationally the Spatial Planning procedure is pending. 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 have been presented in Table 1.18. It is essential that these are completed so that impacts are properly identified; and 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.

Table 1.18 – Interactions Between Project Activities and Social/Environment

Item	Aspect									Receptor									
	Emissions to Air	Noise and Vibration	GHG Emissions	Water Usage	Discharges	Land Take	Light	Hazardous Materials		Air Quality	Soils and Geology	Water	Flora and Fauna	Land Use	Health and Safety	Fishing and Hunting	Livelihoods	Cultural Heritage	Landscape
Construction																			
Change in land use						✓				✓	✓	✓	✓	✓	✓	✓	✓	✓	✓
Decline construction	✓	✓	✓	✓	✓		✓	✓		✓	✓	✓	✓		✓	✓		✓	✓
Construction of infrastructure	✓	✓	✓	✓	✓		✓	✓		✓	✓	✓	✓		✓	✓		✓	✓
Road and Powerline Construction	✓	✓	✓		✓	✓		✓		✓	✓	✓	✓	✓	✓			✓	✓
Operation																			
Underground		✓		✓	✓			✓			✓	✓			✓				
Water supply				✓								✓	✓		✓	✓	✓		
Dewatering				✓	✓							✓	✓						
Discharges					✓						✓	✓	✓		✓	✓	✓		
Chemical / fuel storage								✓		✓	✓	✓	✓		✓				
Waste management	✓		✓		✓					✓	✓	✓	✓		✓		✓		✓
Employment															✓		✓		
Major accidents	✓	✓	✓		✓			✓		✓	✓	✓	✓		✓	✓	✓		
Closure																			
Removal of structures	✓	✓								✓	✓	✓			✓				
Rehabilitation	✓		✓	✓	✓						✓	✓	✓	✓	✓	✓	✓		

Source: ERM, 2024

1.19.10 Capital Cost Estimates

- 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 measure would improve the accuracy of cost estimates related to mobilisation, site setup, project logistics, equipment procurement, personnel recruitment, and training.
- For equipment procurement, establish details beyond the basic machine acquisition such as, but not limited to, OEM to provide remote radio control for operation of LHDs and for operation of Production Drills, rims for tire exchanges, drill steel and drill string, assembly supervision, operational and maintenance training specific to the equipment.
- Incorporate known growth factors related to ore outlines moving with definition diamond drilling (elevating ore reserves from possible to proven), excavation beyond "neat" outlines as part of drilling practices to avoid bootlegs, added excavations / slashes for operator positioning as part of radio remote operations, floor slashes for CRF mixing, etc.
- Establish relationships with local contractors and service providers; and
- Prepare organisational charts to support construction management and operating costs.

1.19.11 Other Relevant Information

All efforts should be placed on obtaining the permits required to develop the Project as permitting is one of the critical areas that could affect the overall delivery of the Project on the proposed schedule.

2 INTRODUCTION

Dundee Precious Metals Inc. (DPM or the Company) is a Canadian-based international gold mining company with headquarters in Toronto, Ontario, Canada and operations and projects located in Bulgaria, Serbia, and Ecuador. DPM is listed on the TSX (TSX:DPM), with headquarters at 150 King Street West, Suite 902, Toronto, Ontario M5H 1J9.

The Pre-Feasibility Study (PFS) report and the associated Technical Report (the Report) have been prepared on behalf of DPM, by the Qualified Persons (QPs) indicated in Section 2.2. The purpose of this Report is to support disclosure of the results of developmental work performed for the Čoka Rakita Project (Project).

Following the completion of the Mineral Resource Estimate (MRE), DPM commissioned DRA Americas Inc. (DRA) to lead a team of specialists to develop a PFS 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 ore.

The purpose of the PFS was to build upon the effort of the previous PEA and develop an optimal solution for the configuration for the mine and processing facilities, based on the latest available testwork, as described in Section 13 and the MRE in Section 14. The PFS 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 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 Study Participants

DPM retained various Consultants to participate in and prepare this Report. More specifically, the Consultants listed below provided Qualified Persons (QPs) (as identified in Section 2.2) 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, as well as to compile this Report.
- **Environmental Resources Management Ltd.** (ERM): develop Mineral Resource Estimate (MRE) as well as environmental studies, permitting, and social impact.
- **WSP Canada Inc.** (WSP): design the underground mine and related mine infrastructure; design tailings management facilities, topsoil storage facilities, overburden storage facilities, and waste rock storage facilities; develop a site wide water balance for the property; and provide a dry tailings and paste backfill design.

2.2 List of Qualified Persons

As described above, DRA and other specialised consultants prepared the work in a collaborative manner. The QPs are responsible for preparation of this Report, and their roles and responsibilities, by Section, are listed in Table 2.1.

Table 2.1 – Qualified Persons and their Respective Sections

Section	Title of Section	Qualified Person(s)	Company
1	Summary	All QPs	All Parties
2	Introduction	Ian Major	DRA
3	Reliance on Other Experts	Ian Major	DRA
4	Property Description and Location	Maria O'Connor	ERM
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Maria O'Connor	ERM
6	History	Maria O'Connor	ERM
7	Geological Setting and Mineralization	Maria O'Connor	ERM
8	Deposit Types	Maria O'Connor	ERM
9	Exploration	Maria O'Connor	ERM
10	Drilling	Maria O'Connor	ERM
11	Sample Preparation, Analyses and Security	Maria O'Connor	ERM
12	Data Verification	Maria O'Connor	ERM
13	Mineral Processing and Metallurgical Testing	Daniel (Niel) Morrison	DRA
14	Mineral Resource Estimates	Maria O'Connor	ERM
15	Mineral Reserve Estimates	Khalid Mounhir	WSP
16	Mining Methods		
	Sections 16.1, 16.4 to 16.11, and 16.14	Khalid Mounhir	WSP
	Section 16.2	Paul Palmer	WSP
	Section 16.3	Michal Dobr	WSP
	Sections 16.12, 16.13, 16.15, and 16.16	Isaac Ahmed	WSP
17	Recovery Methods	Daniel (Niel) Morrison	DRA
18	Project Infrastructure		
	Sections 18 to 18.5 and 18.10, and 18.11	Ian Major	DRA
	Section 18.6	Bruno Mandl	WSP
	Sections 18.7 and 18.8	Isaac Ahmed	WSP
	Section 18.9	Michal Dobr	WSP
19	Market Studies and Contracts	Daniel Gagnon	DRA

Section	Title of Section	Qualified Person(s)	Company
20	Environmental Studies, Permitting and Social or Community Impact	Kevin Leahy	ERM
21	Capital and Operating Costs		All Parties
	Capital Costs Section 21.1 except for 21.1.7	Ian Major	DRA
	Sections 21.1.7 (Capex) and 21.2.3 (Opex)	William Richard McBride	WSP
	Operating Costs Section 21.2 except for 21.2.3	Daniel (Niel) Morrison	DRA
22	Economic Analysis	Daniel Gagnon	DRA
23	Adjacent Properties	Maria O'Connor	ERM
24	Other Relevant Data and Information	Ian Major	DRA
25	Interpretation and Conclusions	All QPs	All Parties
26	Recommendations	All QPs	All Parties
27	References	All QPs	All Parties
28	Abbreviations	All QPs	All Parties

2.3 Site Visits

Visits to the Čoka Rakita site, Serbia, and to the Ada Tepe site, in Krumovgrad, Bulgaria, were conducted at different times, by the QPs from the different consultant companies. Table 2.2 provides details of the personal inspections of the Property by each of the QPs.

Table 2.2 – Site Visit

Qualified Person	Company	Date of Site Visit
Kevin Leahy	ERM	October 9, 2023
Maria O'Connor	ERM	October 3 to 4, 2023
Ian Major	DRA	May 13 to 17, 2024 December 4 to 5, 2024
Niel Morrison	DRA	May 13 to 17, 2024
Isaac Ahmed	WSP	June 11 to 13, 2024
Michal Dobr	WSP	June 10 to 13, 2024
Khalid Mounhir	WSP	June 10 to 13, 2024
Paul Palmer	WSP	June 17 to 20, 2024

2.4 Effective Date

The effective date of the Report is December 18, 2024. As of the effective date of this Report, the QPs are not aware of any material fact or material change with respect to the subject matter of this Technical Report that is not presented herein, or which the omission to disclose could make this report misleading.

2.5 Sources of Information

QP Ian Major, P. Eng. from DRA was the lead author for the PFS and for supervision and compilation of this Report, in collaboration with the other QPs responsible for preparing various portions of this Report. This Technical Report is inclusive of the work and deliverables by and/or under the supervision of all QPs.

This Report relies on QPs from 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 27.

2.6 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 28.

2.7 Non-GAAP Financial Measures

Certain financial measures referred to in this Report are not measures recognised under International Financial Reporting Standards (IFRS) and are referred to as non-Generally Accepted Accounting Principles (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.

The non-GAAP financial measures used in this Technical Report and common to the gold mining industry are defined below:

- All-in sustaining cost and all-in sustaining cost per ounce of gold sold: All-in sustaining cost consists of all production related expenses including mining, processing, services, filtered tailings and paste fill, royalties, and general and administrative, plus treatment charges, penalties, transportation and other selling costs, cash outlays for sustaining capital expenditures and leases, and rehabilitation-related accretion and amortization expenses. All-in sustaining cost per ounce of gold sold is calculated as all-in sustaining cost divided by payable gold ounces. All-in sustaining cost and all-in sustaining cost per ounce of gold sold capture the important components of the Company’s production and related costs and are used by the Company and investors to monitor cost performance at the Company’s operations.

As the Project is not in production, the QPs do not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios presented may not be reconciled to the nearest comparable measure under IFRS.

3 RELIANCE ON OTHER EXPERTS

The QPs (Qualified Persons) have reviewed and analyzed data and reports provided by DPM, together with publicly available data, drawing its own conclusions augmented by direct field examination.

The QPs who prepared this Report relied on information provided by experts who are not QPs. The QPs believes 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 technical report.

For Section 4, QP, Maria O'Connor (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.
- DPM's management and legal counsel with regards to the legal status of each exploration license and any royalty agreements as discussed in Section 4.4.

QP, Maria O'Connor (ERM), has not independently verified legal ownership of surface title and exploration licenses comprising 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.

For Section 19, QP Daniel Gagnon (DRA) fully relied upon:

- Commercial terms Excel Spreadsheet provided by DPM dated October 30, 2024.

For Section 22, QP Daniel Gagnon (DRA) fully relied upon:

- Royalty memo provided by DPM via email dated November 17, 2023; and
- Taxation memo provided by DPM via emails dated April 12 and April 30, 2024.

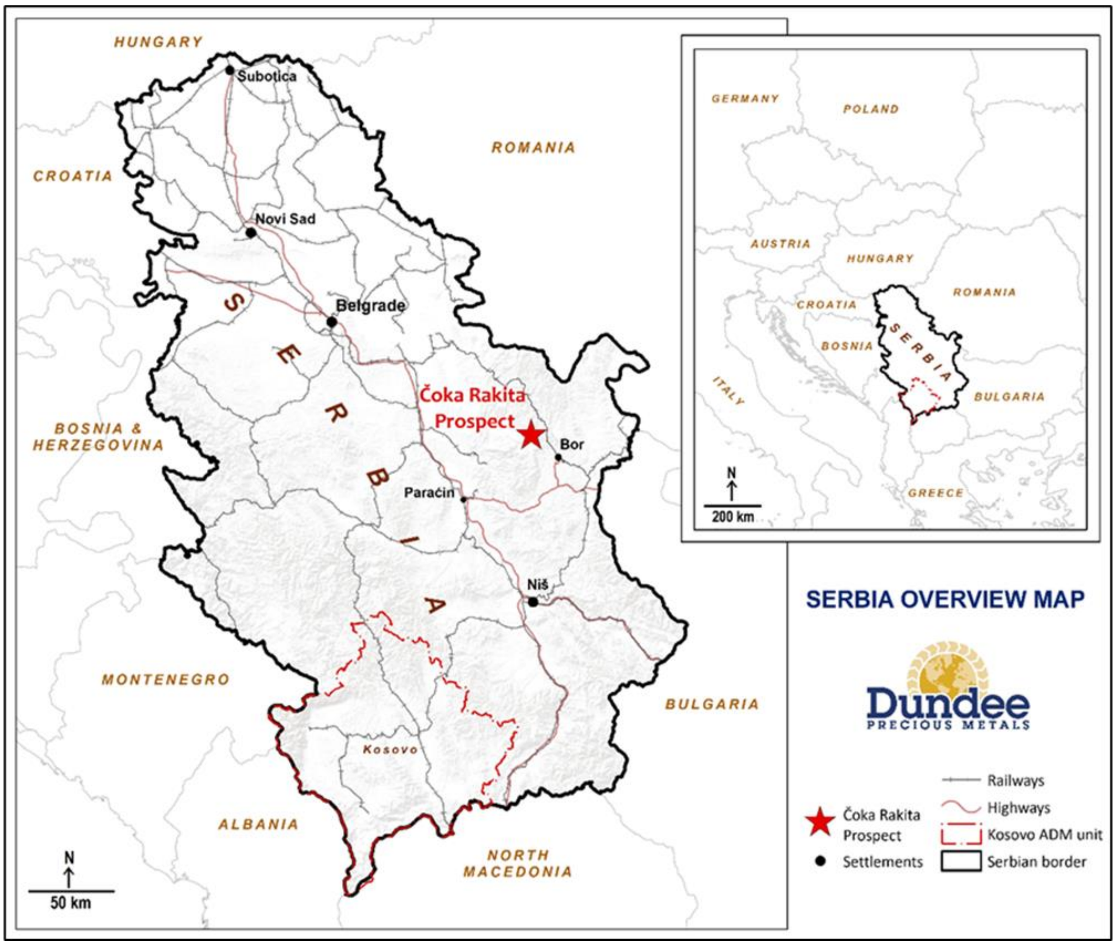
The QPs have assumed, and relied on the fact, that all the information and existing technical documents listed in the References Section 27 of this Report are accurate and complete in all material aspects, but have taken all appropriate steps in their professional judgement to confirm same and are not disclaiming any responsibility for this Report except as permitted under applicable securities laws. The QPs reserve the right, but will not be obligated, to revise the Report and conclusions, if additional information becomes known subsequent to the date of this Report.

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

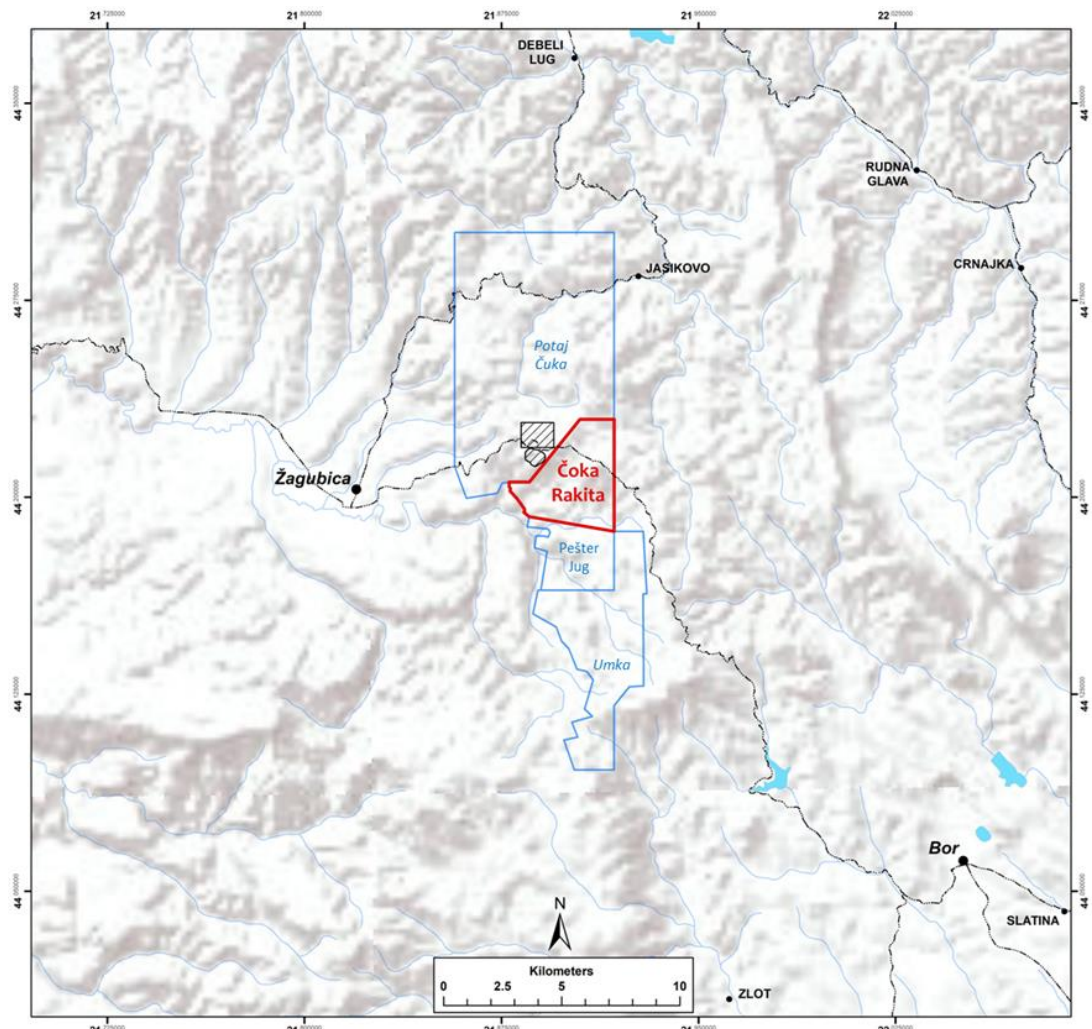
4.2 Mineral Tenure and Surface Rights

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 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, Potaj Čuka, and Pešter Jug exploration licenses are 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. The renewal process for the Umka exploration license – which expired on October 19, 2024 - is ongoing and DPM fully expects the license extension to be granted. 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	Renewal 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

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 the Republic of 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 within 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, together with the Bor smelter, all located proximal to 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 licence.

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 ~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 and Mineral Reserve Estimates

No historical Mineral Resource or Mineral Reserve estimates 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.

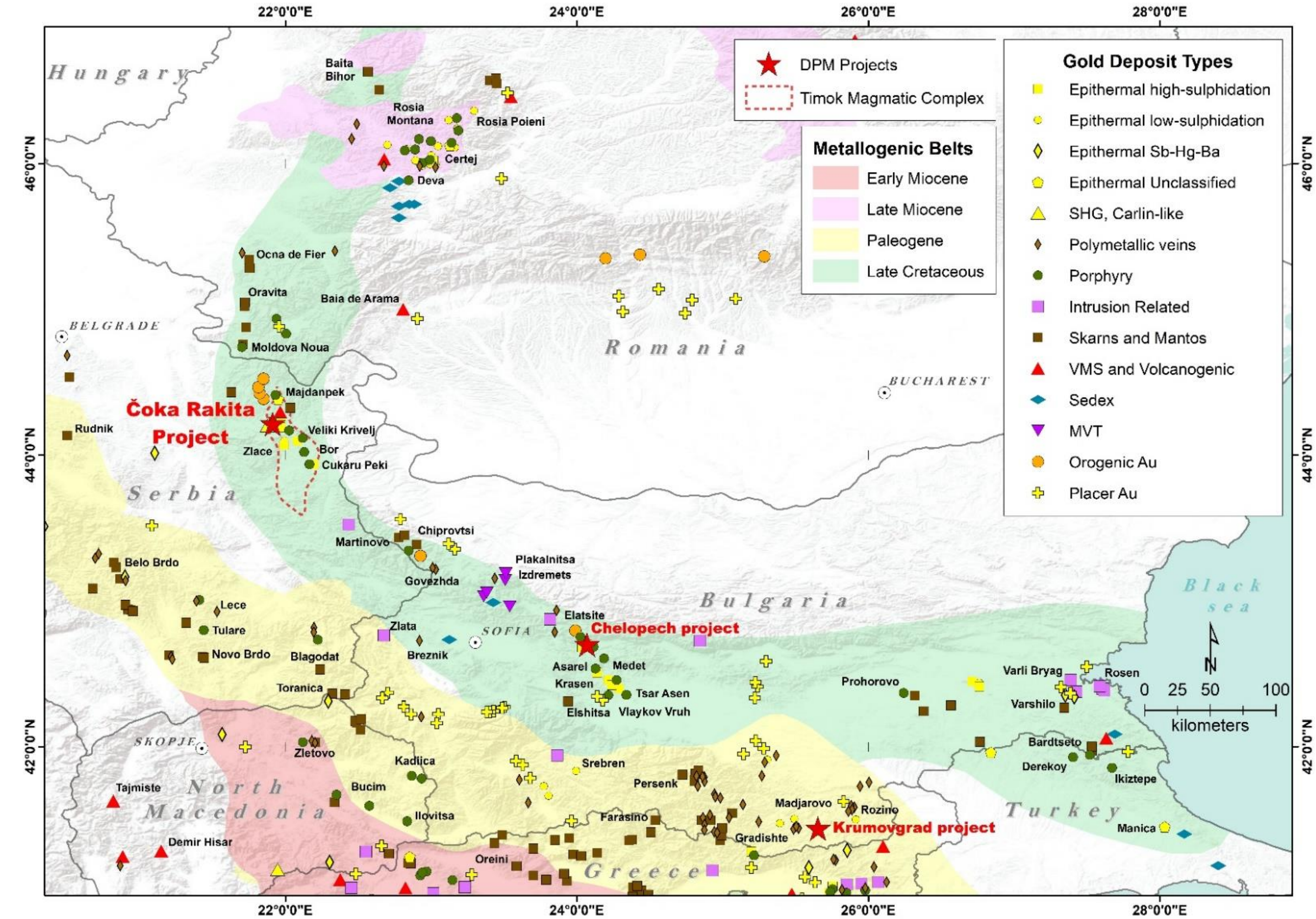
The complex arcuate geometry of the collision interface, and the presence of several micro-plates within the orogenic collage, resulted in a variety of collision products (Gallhofer et al., 2015). Some segments are characterised by extensive regional metamorphism, whereas others by calc-alkaline igneous 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.

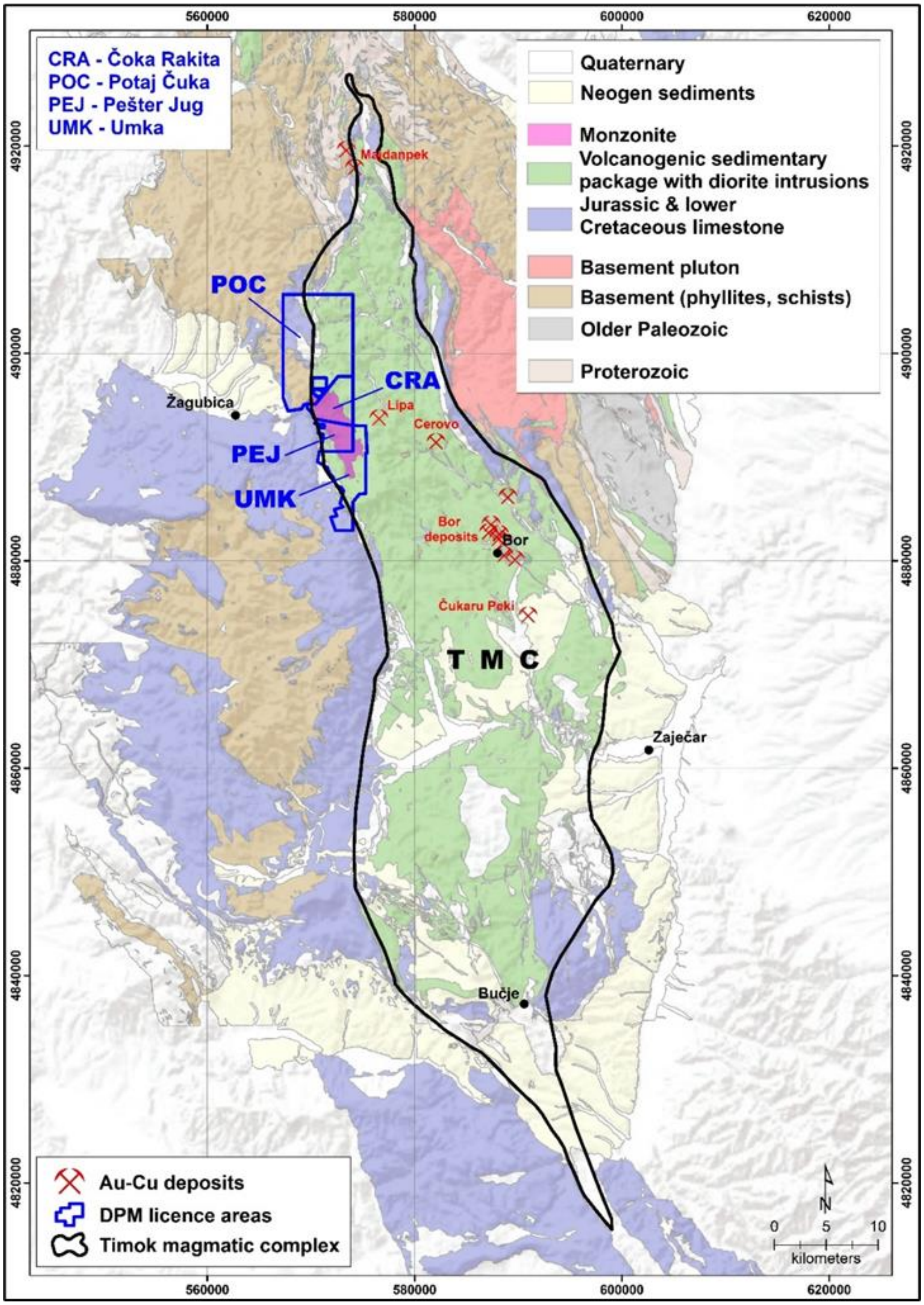
7.3 Local Geology

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.

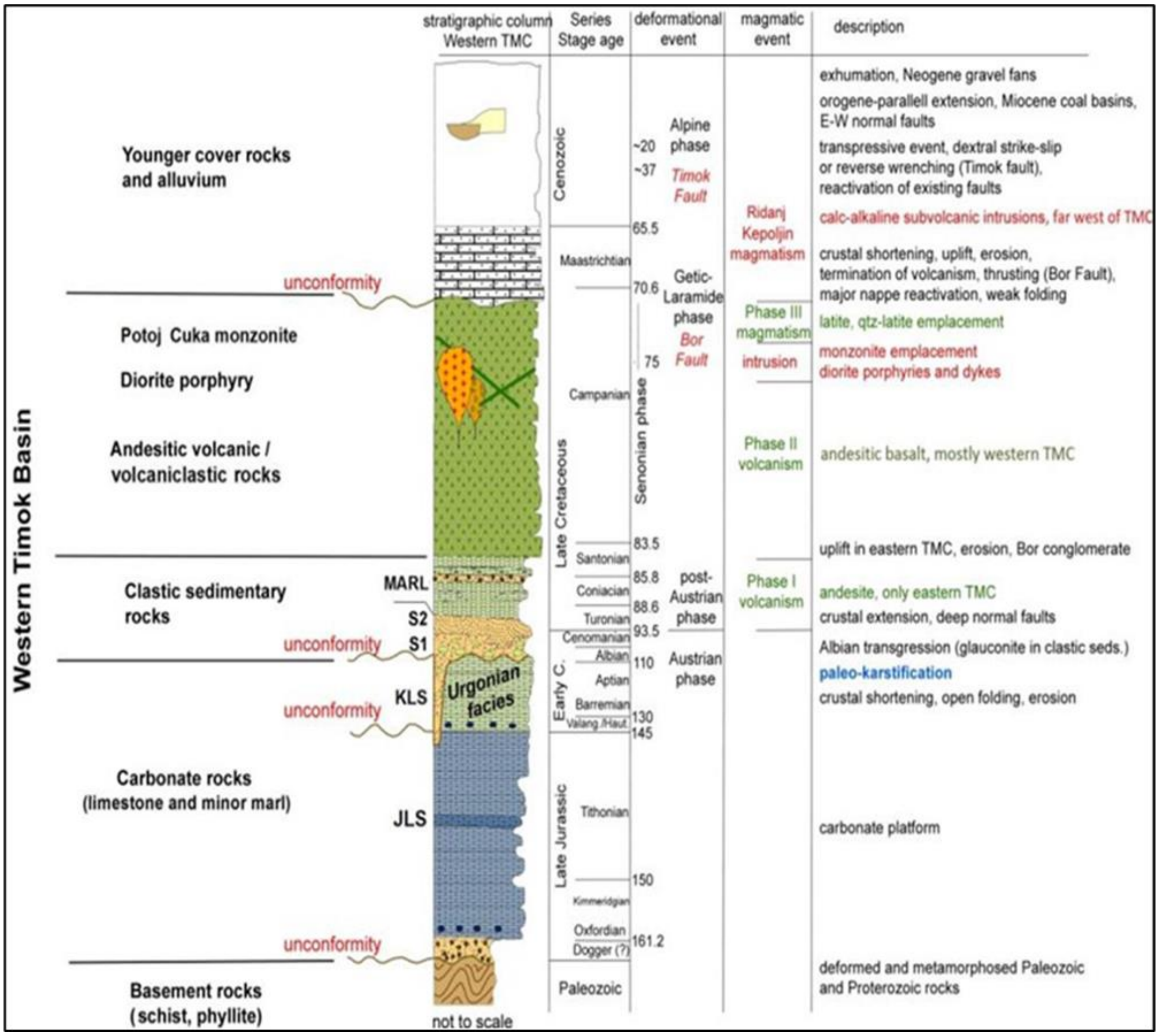
Subsequent phases of magmatism occurred from the Santonian to lower Campanian and comprised pyroxene basaltic andesite (Osnić basaltic andesite) and amphiboles andesite (Ježevica andesite). This suite is mostly found on the central and western portions of the TMC.

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.

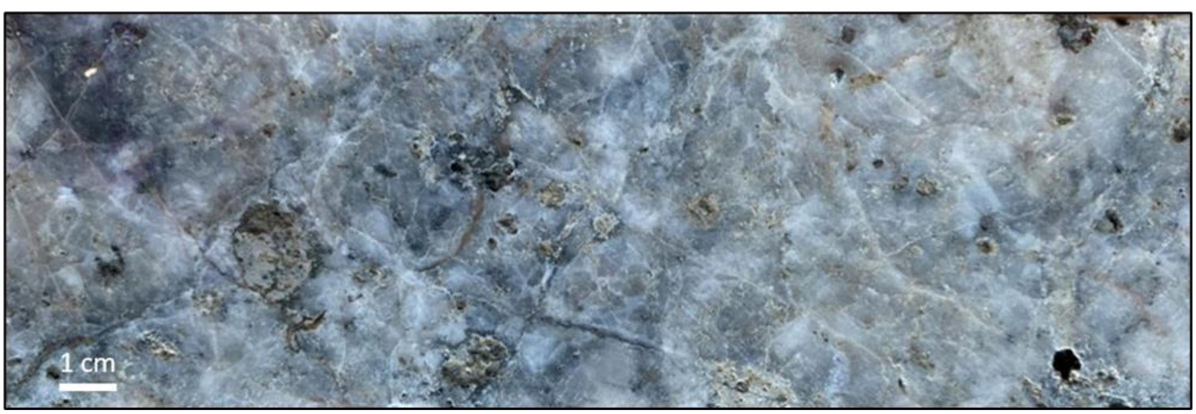
Figure 7.4 – Core Specimens of the Marble and Garnet-Hematite-Quartz Skarn

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



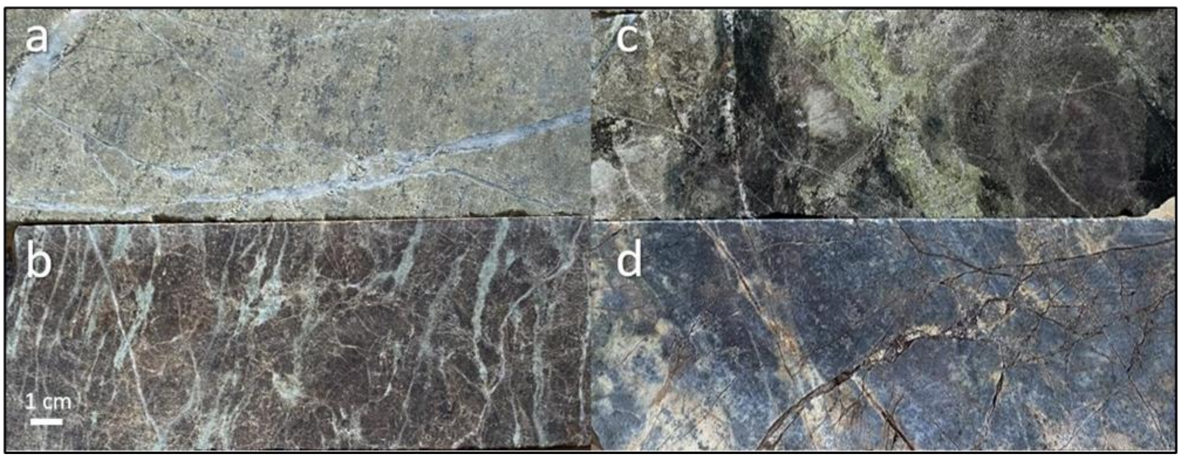
Source: DPM, 2023

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

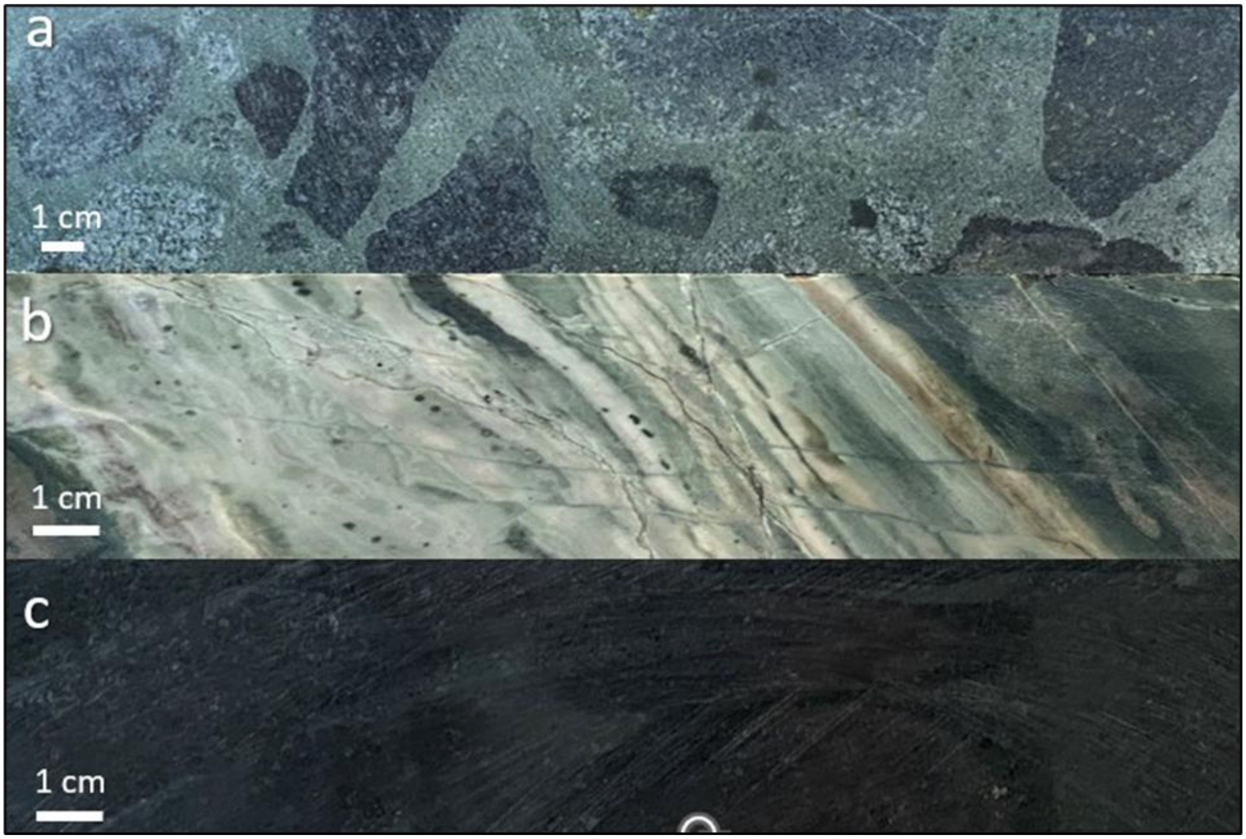
The Late Cretaceous epiclastic (SFD) unit is andesitic in composition and characterised by rapid facies changes throughout the sequence. The lower part of the epiclastic unit is characterised by polymictic basaltic andesite conglomerate and breccia, whereas the upper part is dominated by monomictic basaltic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

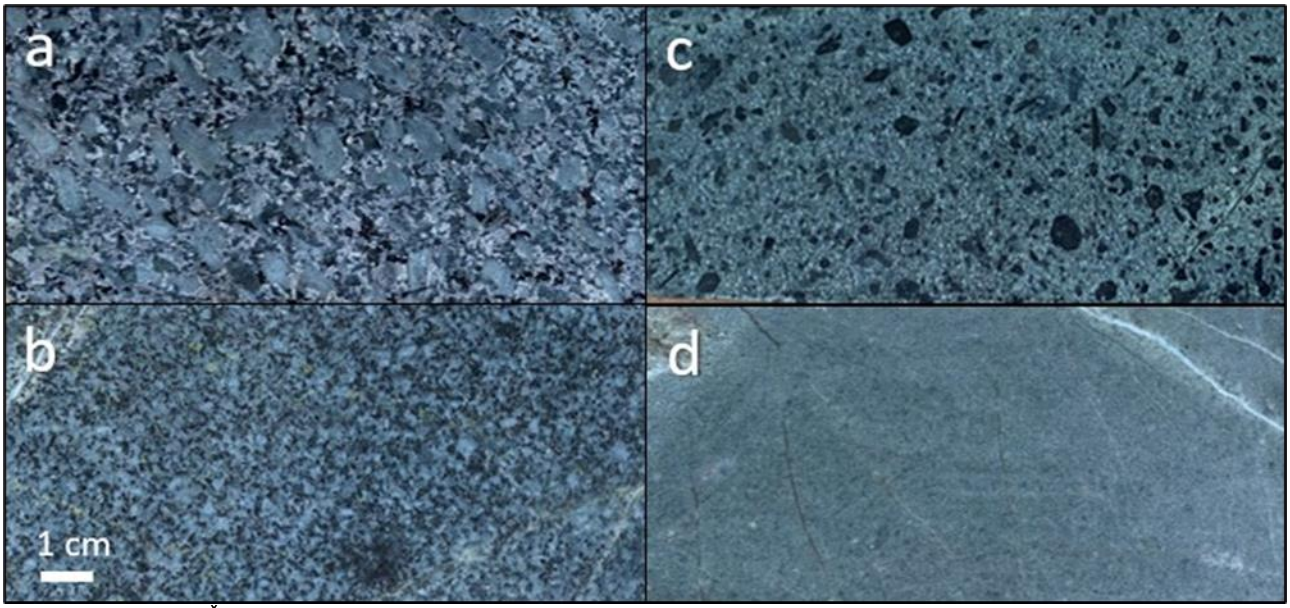
Numerous dioritic stocks, dykes, and sill-like intrusions are observed at Čoka Rakita. These bodies dominantly trend in a north-westerly orientation, which most likely represents a structural fabric in the subsurface that controlled their emplacement. The main intrusive phases identified on Čoka Rakita 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.

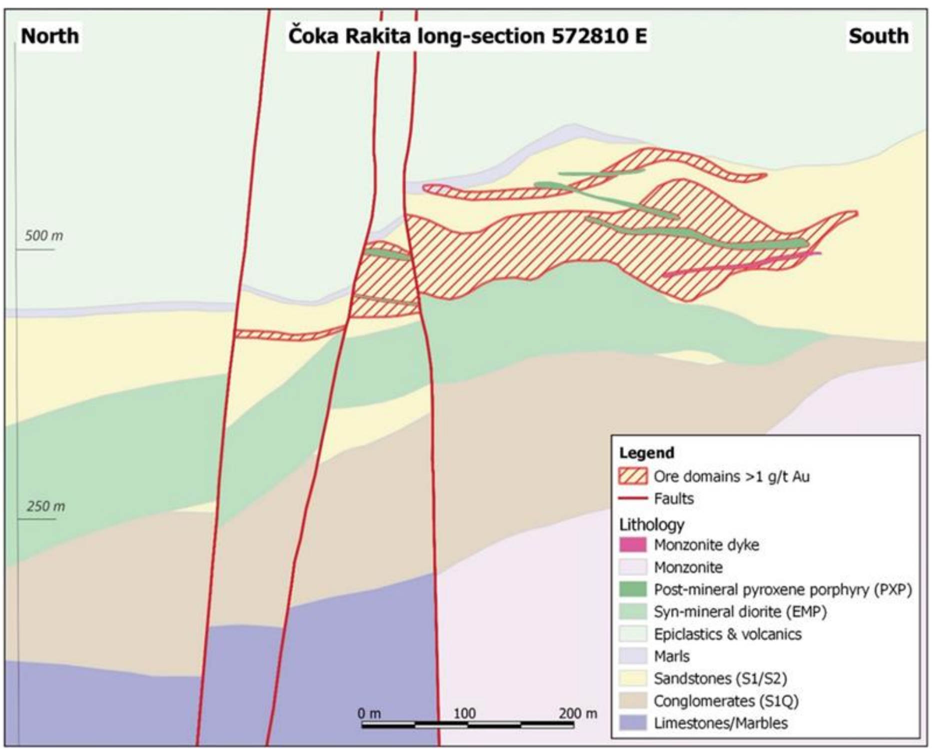
The Cretaceous clastic and epiclastic units are crosscut mainly by shallow angle faults, subparallel to the main stratigraphic bedding – dipping to the east, southeast and northeast at 10° to 40°. Such low angle faulting is localised to the contacts between the sedimentary units, such as epiclastics-marls, 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.

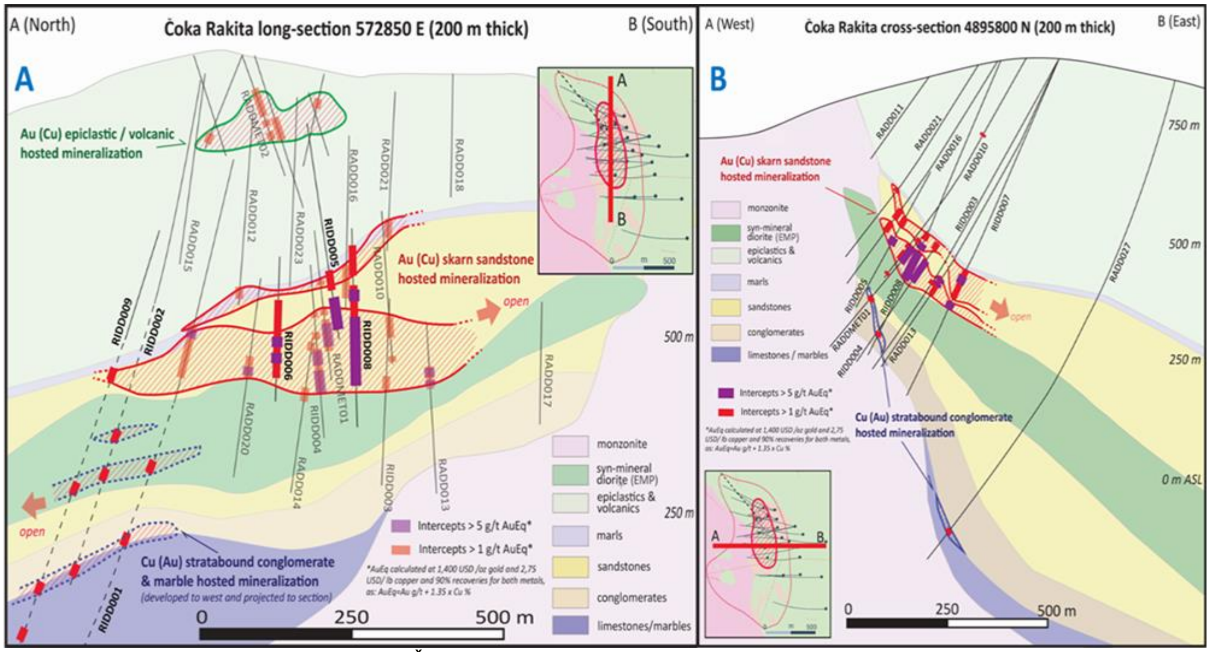
The location of the main mineralisation styles relative to the skarn-hosted gold mineralisation is shown in Figure 7.11.

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 of the Main Mineralisation



(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.

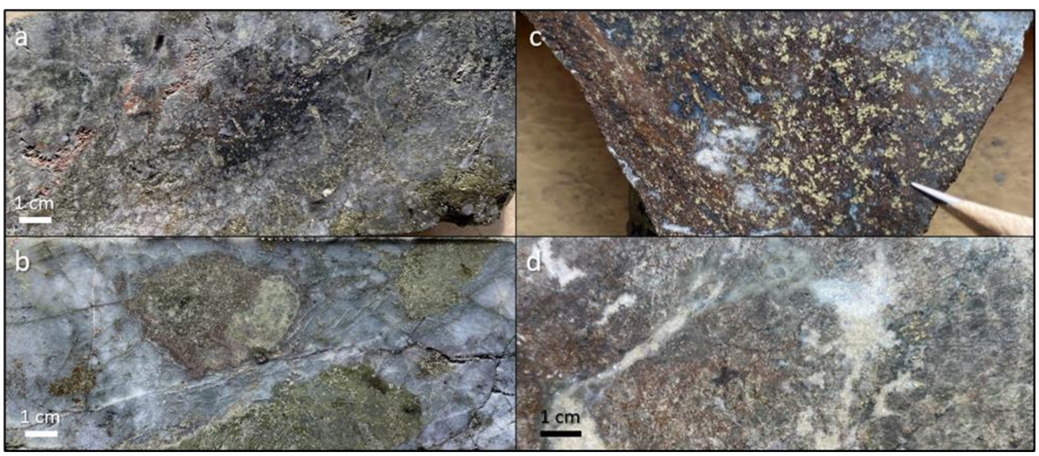
A pervasive secondary biotite±magnetite-K-feldspar assemblage formed in the EMP intrusion is associated with the low-grade disseminated and porphyry quartz-veinlet hosted copper±gold-molybdenum mineralisation. The mineralisation is represented by chalcopyrite, magnetite, pyrite and molybdenite. Although, 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

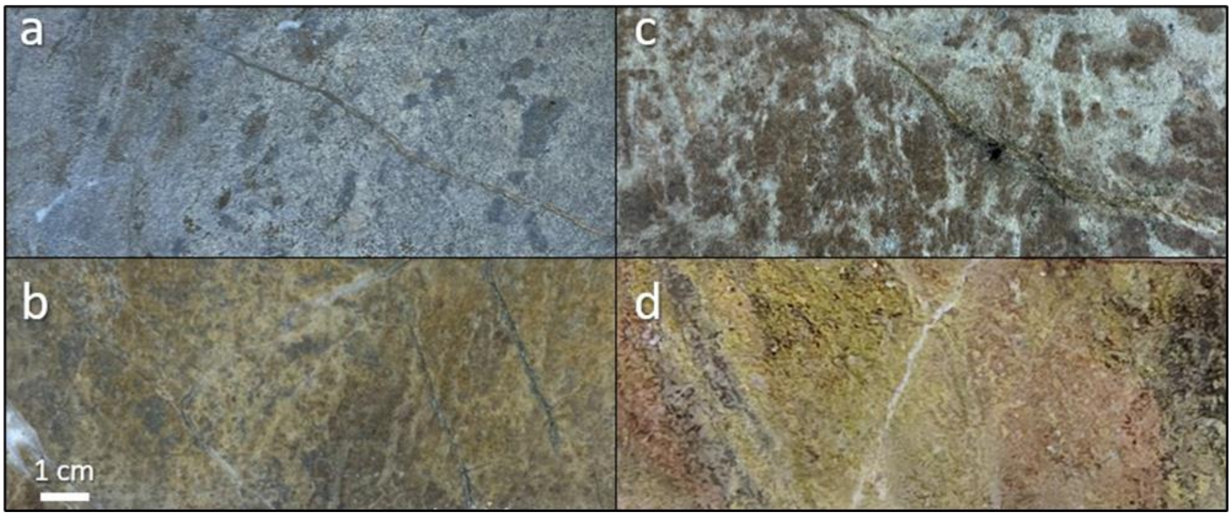
The Čoka Rakita calcareous sedimentary rocks are exposed to intensive levels of skarnification. Both prograde and retrograde phases of alteration are developed. Mineral, chemical, and textural variability is common in both phases. Both the sulphide and gold mineralisation events are mainly associated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

A brownish-beige-yellow pyrrhotite-magnetite skarn was identified in several drill holes within the S1/S2 unit. This altered rock type is associated with gold (Au) and pyrite – pyrrhotite - magnetite mineralisation with minor marcasite – sphalerite – galena – chalcopyrite – tetrahedrite – tennantite – Pb (-Sb) sulfosalts. Pyroxene and grossular member of andradite-grossular series are major rock constituents. Arsenic (As), antimony (Sb), sulphur (S), magnesium (Mg) and manganese (Mn) exhibit elevated concentrations while there is a decrease in sodium (Na) and potassium (K) compared to the surrounding clastic S1/S2 unit. The north-south orientation of the pyrrhotite-magnetite skarn suggests a structural control, with subsequent lateral spreading along the upper parts of the clastic sequence.

Figure 7.14 – Exoskarn Alteration



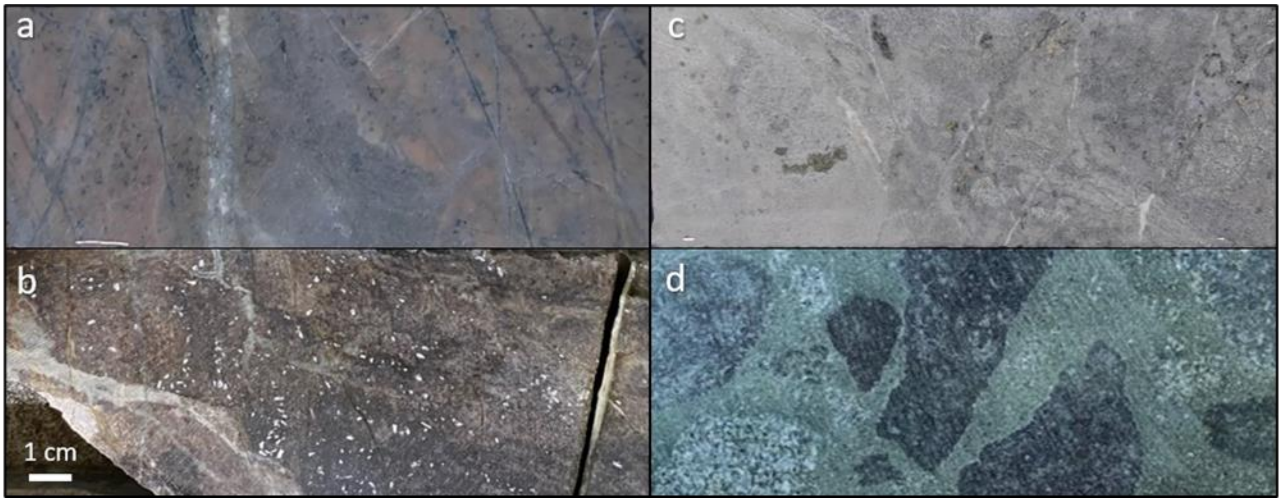
(a) Prograde, pale-yellow garnet-pyroxene skarn. (b) Prograde, yellow ochre garnet skarn. (c) Prograde, dark red garnet, wollastonite, pyroxene skarn with "leopard" texture. (d) Retrograde, epidote, actinolite, calcite skarn overprinting prograde skarn.
Source: DPM, 2023

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 mineral 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 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.

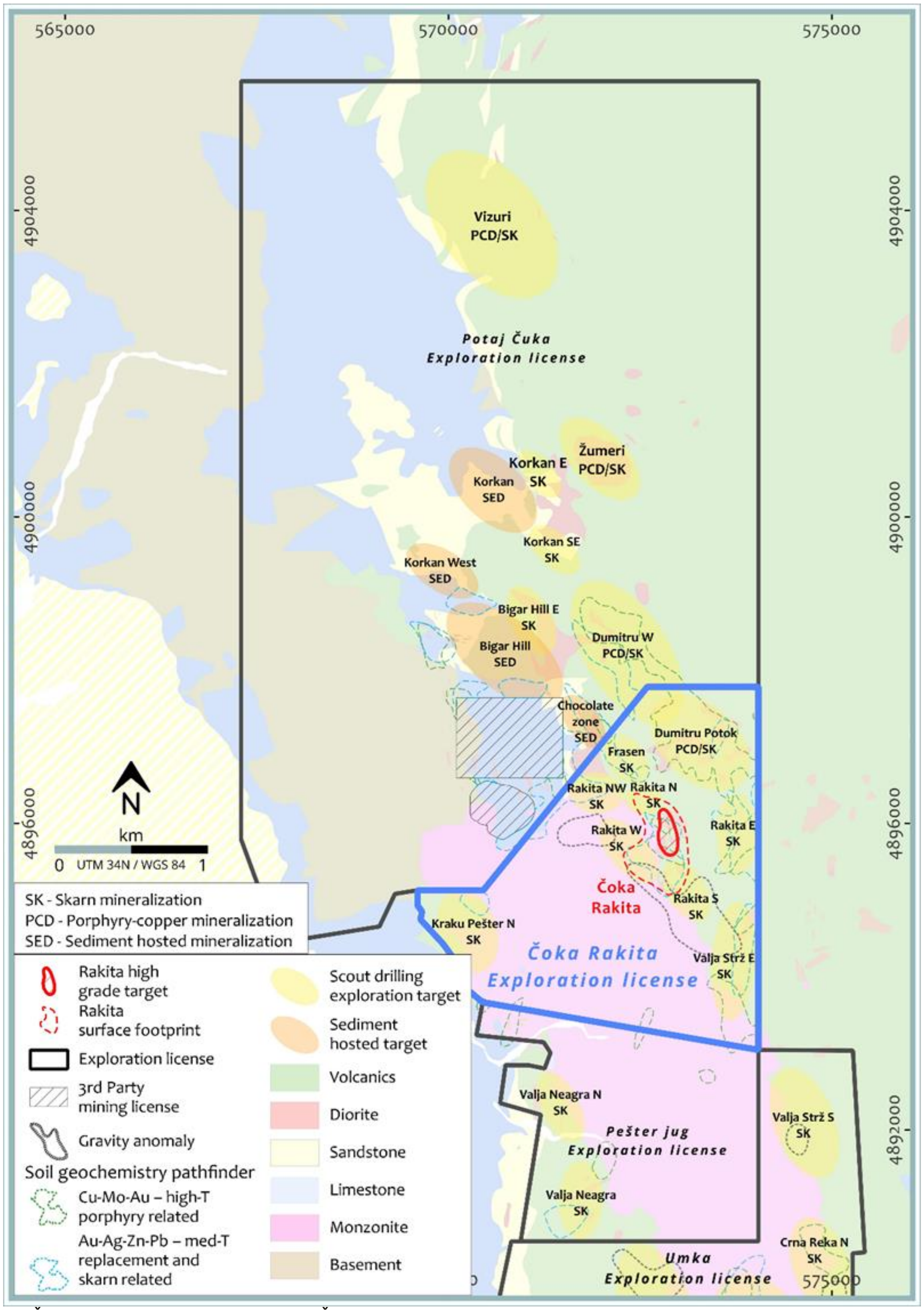
When exploration efforts were resumed, a new phase of drilling in 2016 was completed. A single deeper drillhole, that aimed to understand the potential for porphyry copper-gold mineralisation and search for potential skarn formation beneath the known extents of epiclastic-hosted gold mineralisation was 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 August 2024, 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.

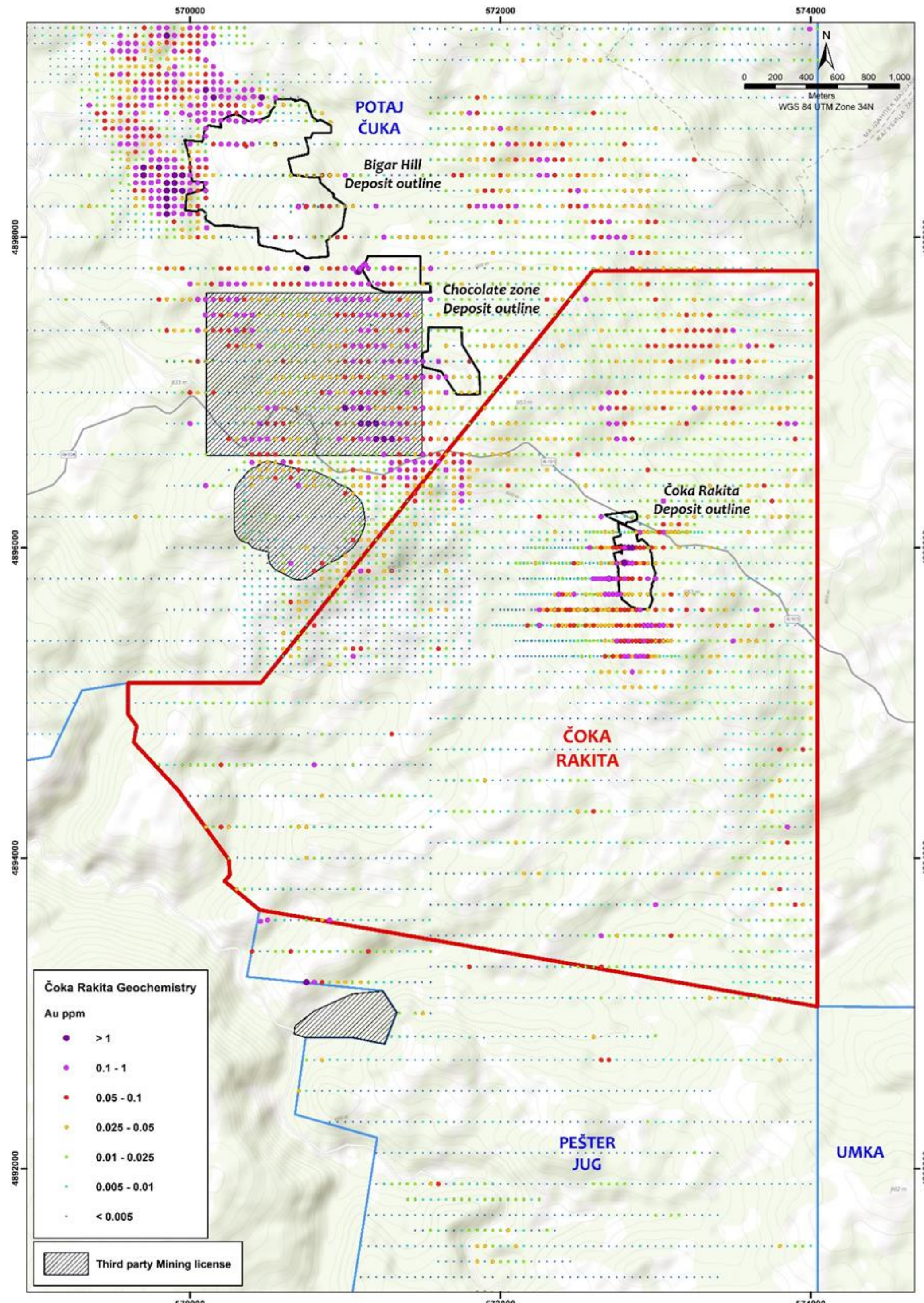
Within the Čoka Rakita exploration license, gold-in-soil anomalies to the northeast of the license are associated with porphyry mineralisation that broadly follows the trace of the Dimitru Potok porphyry system. Anomalous gold in soil results located above the Čoka Rakita deposit are related to epiclastic 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 August 2024, 10,358 m of surface trenching and 1,576 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.

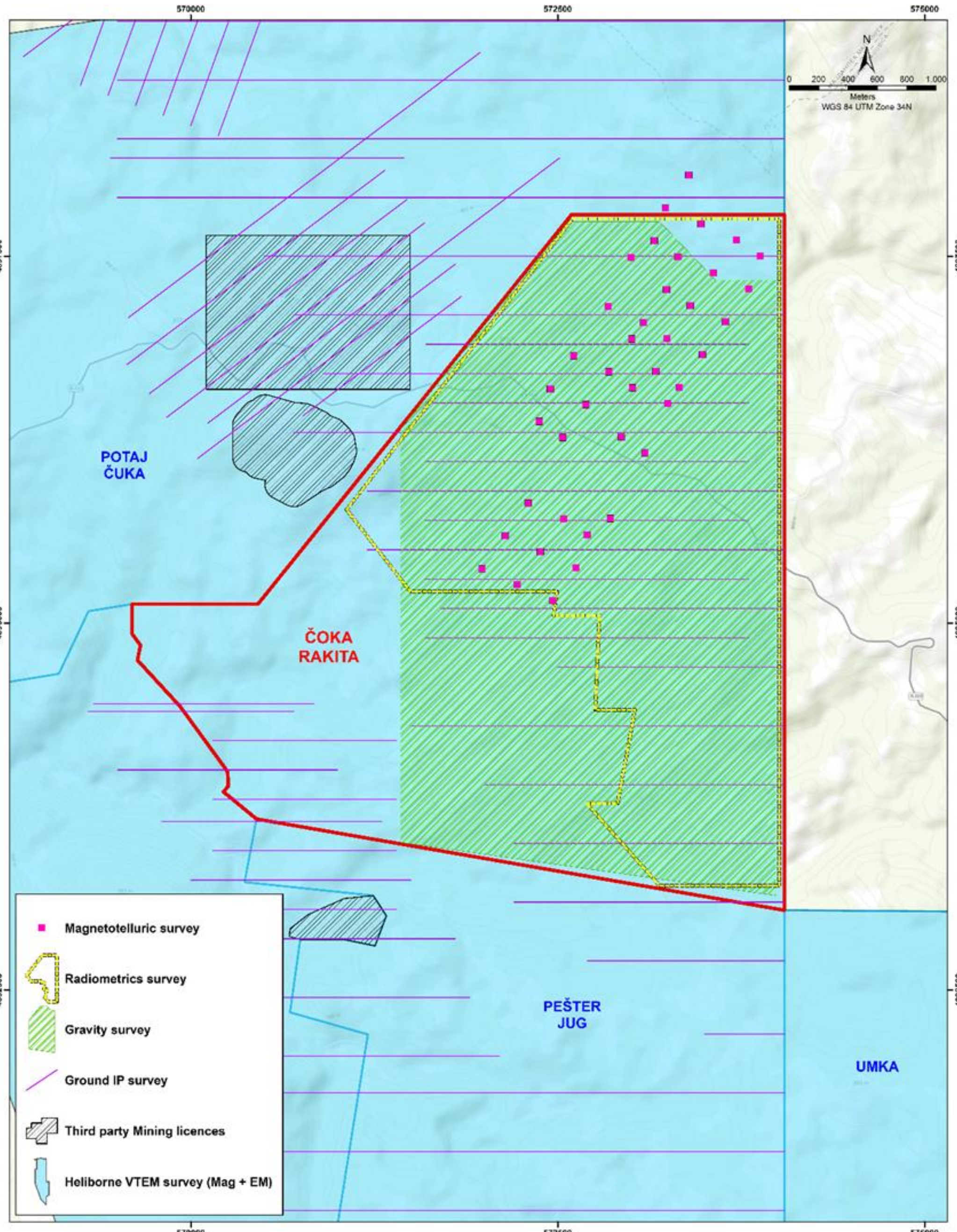
9.6 Topographic Surveys

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

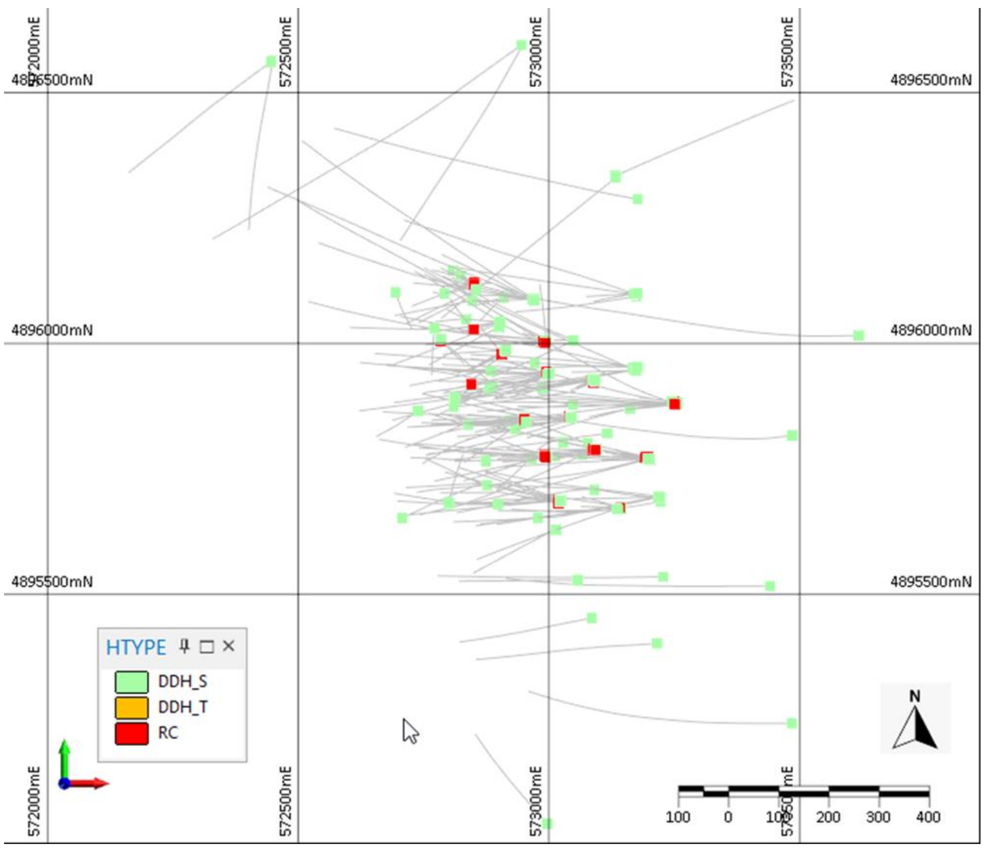
Drilling operations are summarised by area and year completed in Table 10.1. All the drilling activities on the Project have been completed during the tenure of DPM. Figures 10.2 and 10.3 present the drillholes completed at each deposit.

Table 10.1 – Summary of Drilling by Type and Year at Čoka Rakita, up to August 2024

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	64	40,390	44	7,485	30	16,520	-	-
	2024	72	34,073	-	-	16	5,934	-	-
Total		**175**	**94,923**	**48**	**7,959**	**46**	**22,454**	**2**	**1,160**

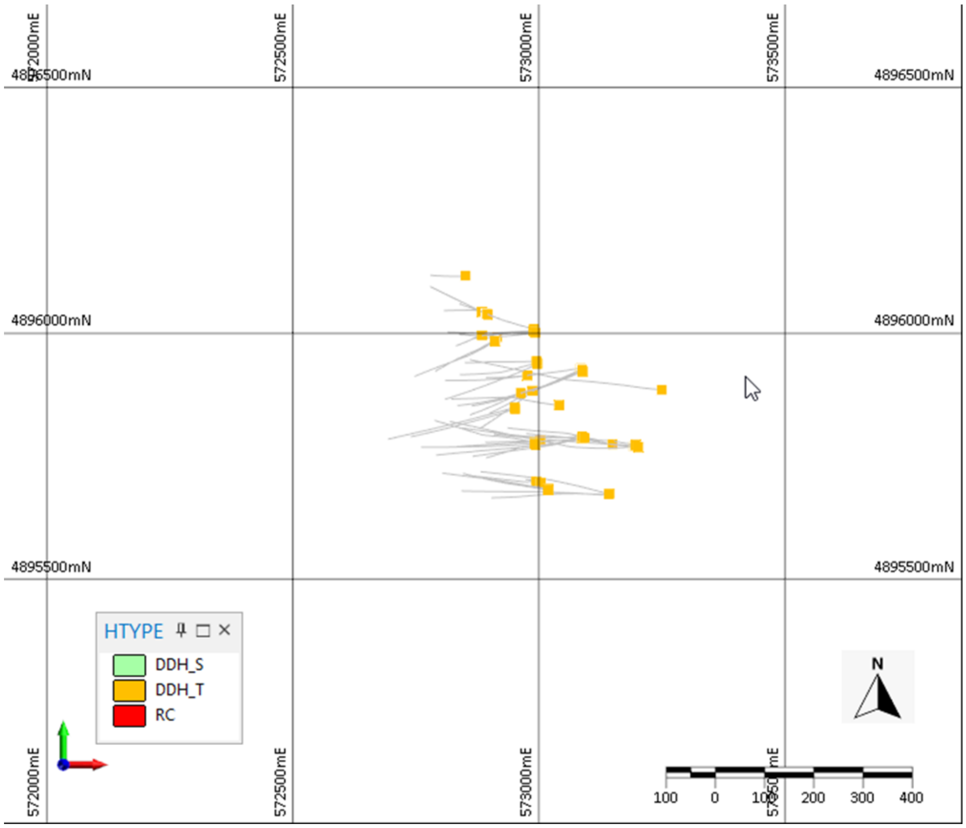
Source: ERM, 2024

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



Source: ERM, 2024

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



Source: ERM, 2024

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 97.64% 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

No RC pre-collar drilling was completed in 2024. 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 since 2023.

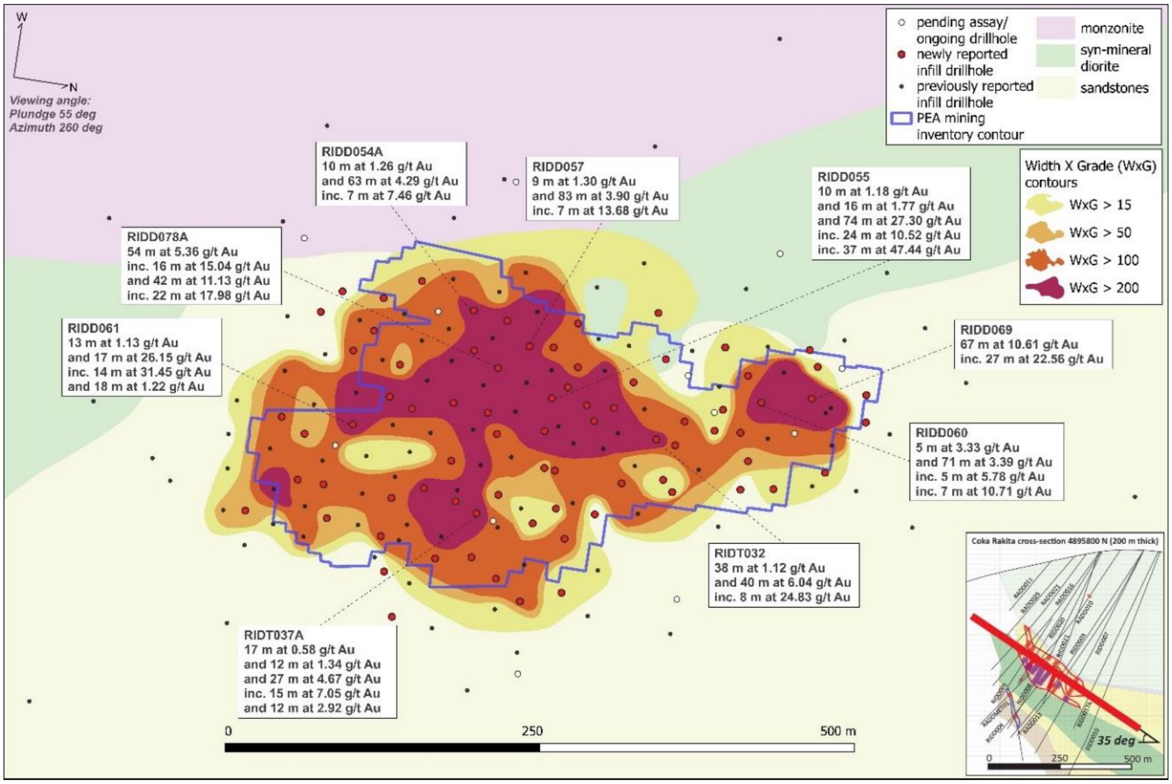
10.8 Metallurgical Drillholes

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

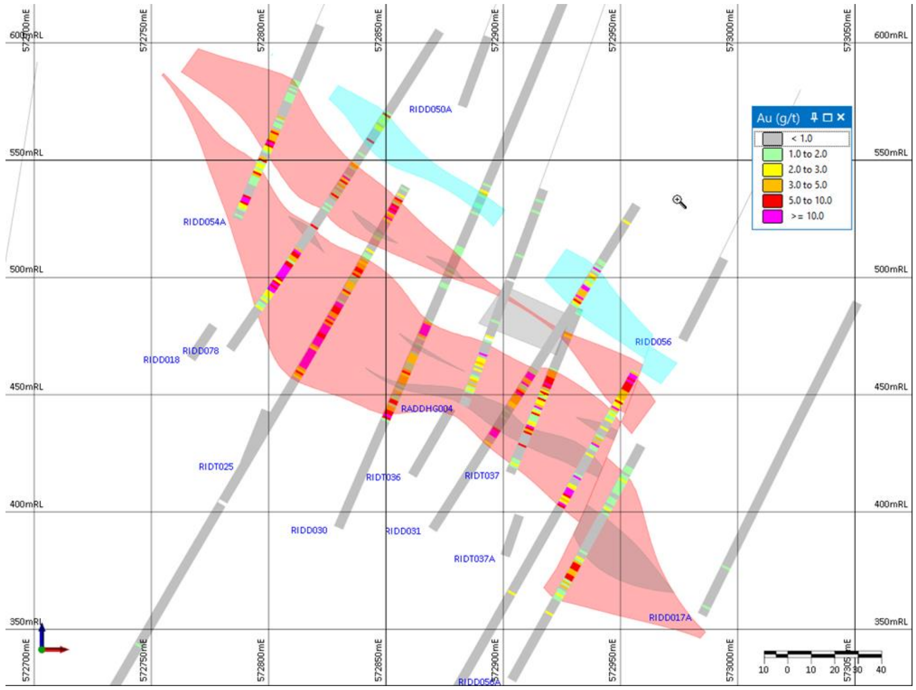
Drilling results received as of 16 November 2023, defined a wide zone of skarn-hosted gold mineralisation over a footprint of 650 m long, up to 350 m wide, and with variable thickness from less than 20 m in the margins to more than 100 m in the core of the mineralised zone. A tilted slice through the 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 Figures 10.5 and 10.6. The QP (Ms. Maria O'Connor, MAIG) is not aware of any drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the results shown.

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



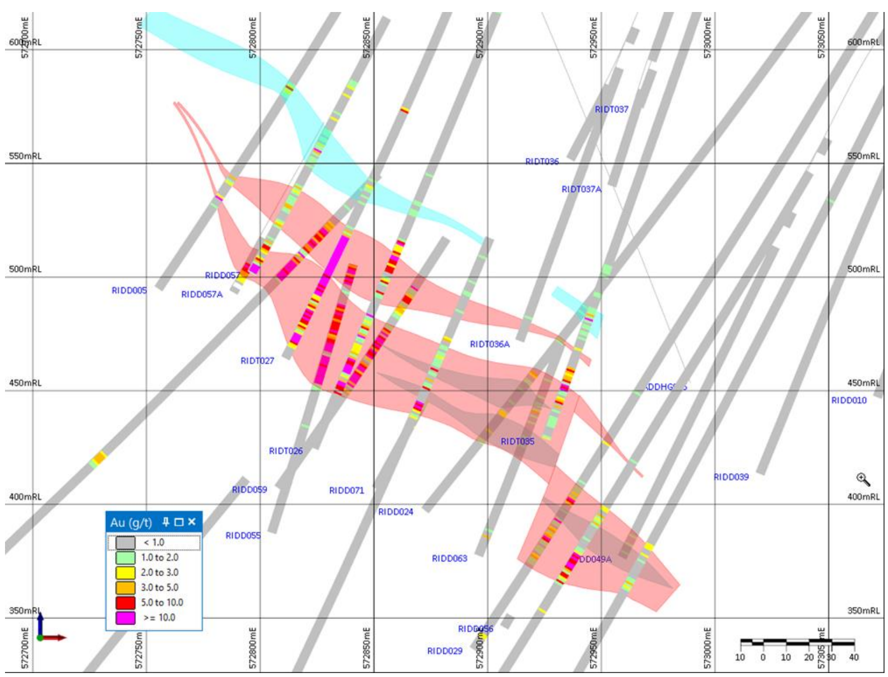
Source: DPM, 2024

Figure 10.5 – Cross Section 4895830 mE Showing Drilling, Interpreted Mineralisation (cyan, red) and Internal Waste (grey) at Čoka Rakita



Source: ERM, 2024

Figure 10.6 – Cross Section 4895880 mE Showing Drilling, Interpreted Mineralisation (cyan, red) and Internal Waste (grey) at Čoka Rakita



Source: ERM, 2024

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Introduction

The QP (Ms. Maria O'Connor, MAIG) 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 Sample Preparation and Quality Control Methods

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. Sample preparation and quality control methods inserted prior to sample dispatch are discussed below.

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 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.

11.4 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 fully independent of DPM and all are ISO certified apart from SGS Bor.

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.

Name and Location	Dates (Primary Assaying)	ISO Certification	Testwork Performed
SGS, Bor, Serbia	2010 to 2024	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 2024	ISO9001:2008 and ISO/IEC 17025:2005	Crushing and pulverising of soil, trench, channel, RC, diamond core samples and umpire QC.
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.
SGS, Burgas, Bulgaria	2024	ISO/IEC 17025:2018	Gold and sulphur analysis for some diamond core samples.

Source: ERM, 2024

11.4.1 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/DPM 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.

11.4.2 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; and
- A minimum of 10% of submitted samples are subject to repeat analysis.

Second splits generated by the SGS Comlabs Computerised Laboratory Automation System (CCLAS) 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.

11.4.3 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 Romania.

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 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 at the ALS laboratory 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). The fraction 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. The entire coarse fraction is assayed to extinction, 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). 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.

DPM engaged a coarse gold expert to review sampling and the appropriateness of SFA for Čoka Rakita. The following conclusions/recommendations are summarised from that report (Dominy, 2024):

- Continued use of SFA: Given the undeniable presence of some coarse gold and the observed bias between SFA and FA, the current use of SFA is justified and should continue.

- FSE Variability: The Fundamental Sampling Error (FSE) for both FA and SFA protocols is generally less than ±30%, except when dealing with very coarse gold. FSE is influenced by the coarse crush split at P85 4 mm, where the P95 exceeds 4.5 mm.
- Potential Improvements to the existing protocols: For SFA, using a P85 2 mm crush, pulverising the entire reject, and splitting 1,000 g for assay would improve accuracy. It is also essential to use a riffle splitter for pulp splitting, as scooping is inappropriate. While these changes could enhance the protocols, the improvements may not justify the effort unless the cost implications are reviewed by DPM.
- Duplicates Program: A focused duplicates program is recommended to track errors throughout the assay process. This should include diamond core duplicates, RC rig duplicates, laboratory coarse split duplicates, and assay/pulp duplicates to ensure consistent and reliable results. DPM currently implement all of this except for diamond core duplicates.

11.4.4 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.

11.4.5 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.

11.5 Quality Assurance and Quality Control

11.5.1 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.

11.5.2 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 standard deviations, or any two consecutive standards differing more than two standard deviations constitutes a failure and the project geologist is required to submit the affected batches for re-assaying.

11.5.3 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.

11.5.4 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.

11.5.5 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.

11.5.6 QP QA/QC Report: 30 September 2023 to 30 August 2024

DPM supplied the QP with an export of the data from the acQuire database. The data were investigated per laboratory for diamond drilling samples. QA/QC data for the following laboratories were reviewed: SGS Bor (SGS_BO), SGS Burgas (SGS_BUR) where Au and S were assayed, and ALS Bor (ALS_BO). This supplements the QAQC reported previously (1 March 2007 to 31 October 2023) where the conclusions were that the QA/QC procedures implemented were 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 bias). Precision for diamond drill samples were acceptable. Sampling procedures are appropriate and adequate security exists to minimise the risk of contamination or inappropriate mixing of samples (O'Connor, 2024).

The quality control data under revied for the period 30 September 2023 to 30 August 2024 for all elements also 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.

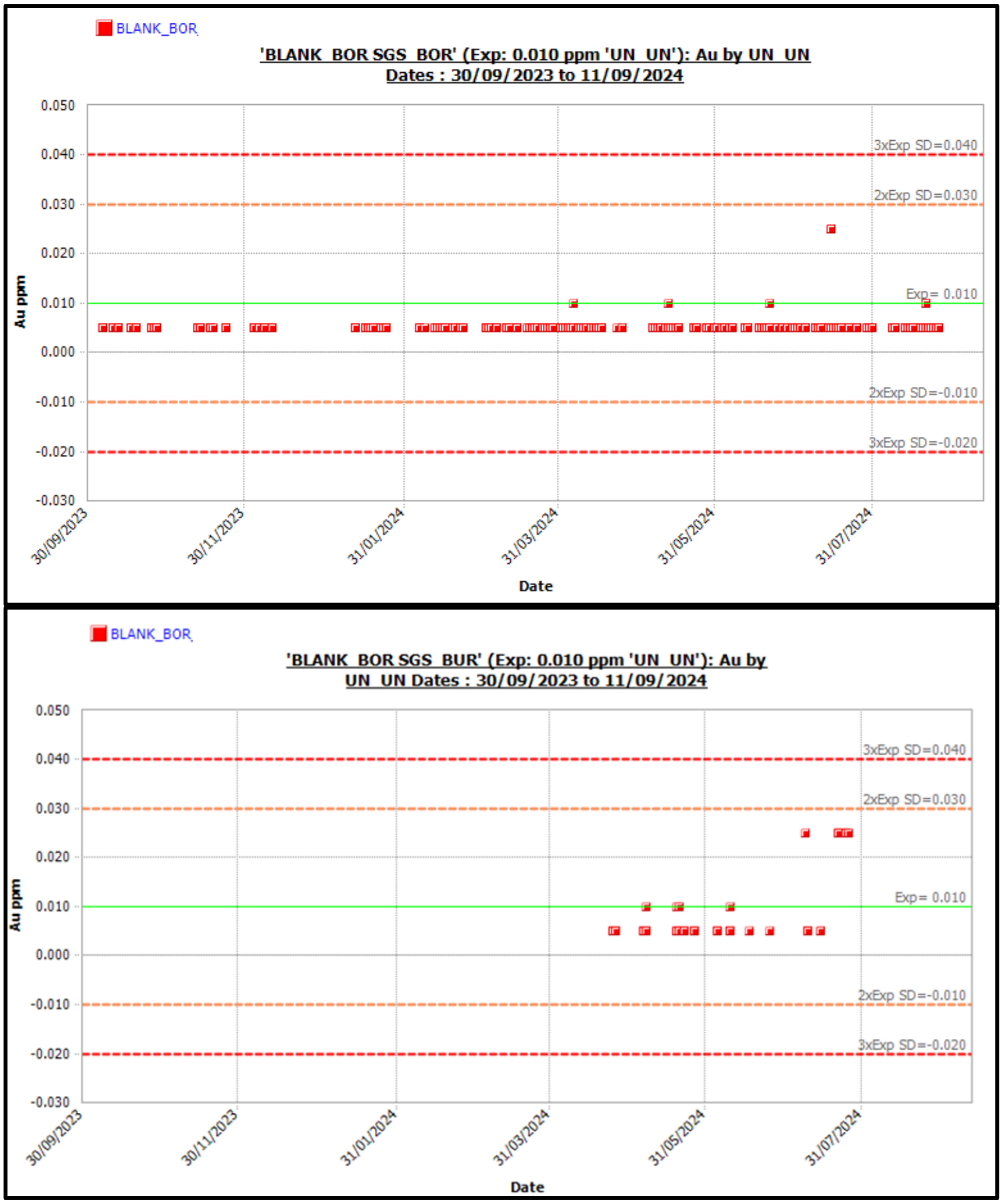
All incoming assay results are emailed as digital files from the analytical laboratory. The database export does not contain expected reference values for specific assay methods for all the relevant CRMs used during the period under review. This does not conform to industry best practice and CRM samples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

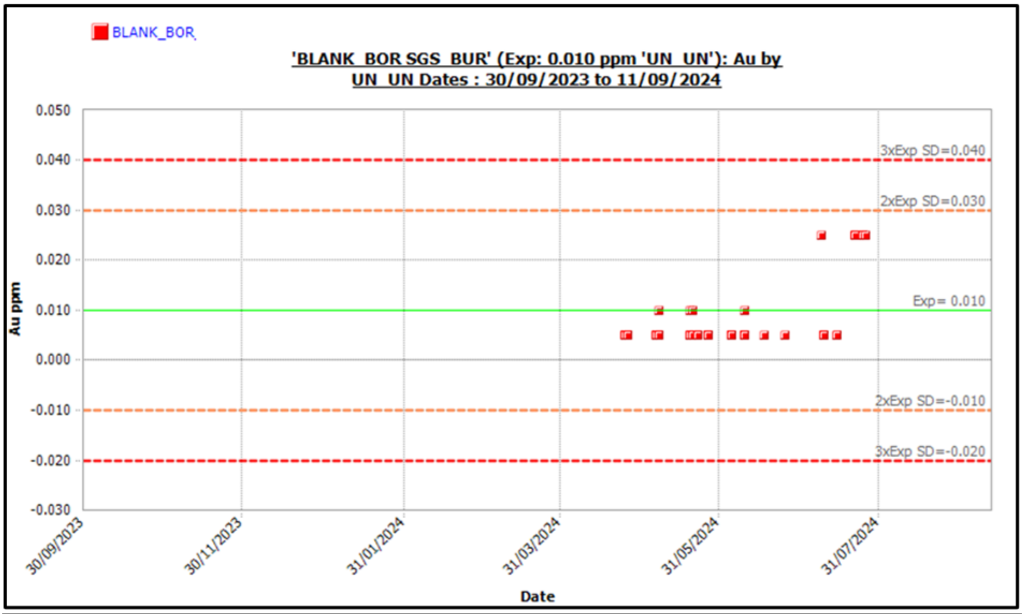
The CRMs used as part of the quality control program were not consistently captured in the database. The expected values and allowed standard deviations per element, per analysis method were not captured consistently in the database. The implication of this is that a CRM cannot fail analysis, as there 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, the database setup can be improved. 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.

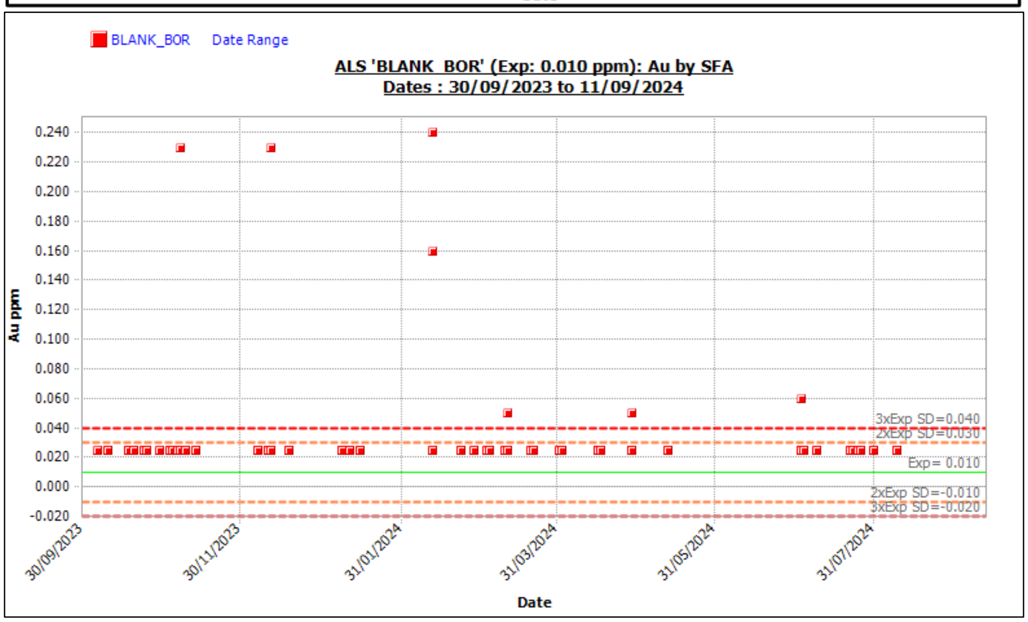
11.5.6.1 Contamination

The data for the blank, BLANK_BOR, did not show any fatal flaws. Two thousand five hundred and fourteen (2,514) BLANK_BOR samples were tested. Two thousand and seventy-four (2,074) were analysed at SGS_Bor, one hundred and forty-nine (149) at SGS_Bur and ALS with eight (8) samples falling outside three (3) standard deviations. The eight (8) samples were from the three hundred and five (305) samples sent to ALS_BO (Rosia Montana) laboratory for analysis by SFA and represented 2.5% of the population for that laboratory. One (1) sample showed a significant deviation by reporting 3.93 ppm Au. These samples should be reviewed to see if this result may possibly be attributed to sample swaps or incorrect labelling of CRMs.

Figure 11.1 – Blank Performance for SGS Bor





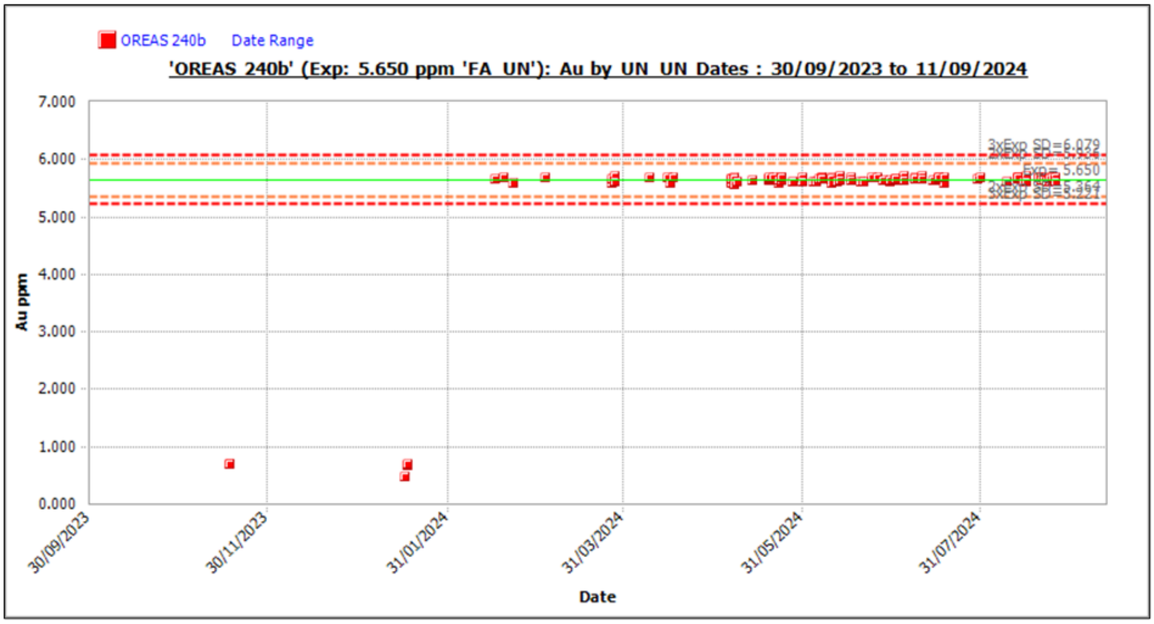


Excluding outlier returning 3.93ppm Au
Source: ERM, 2024

11.5.6.2 Accuracy

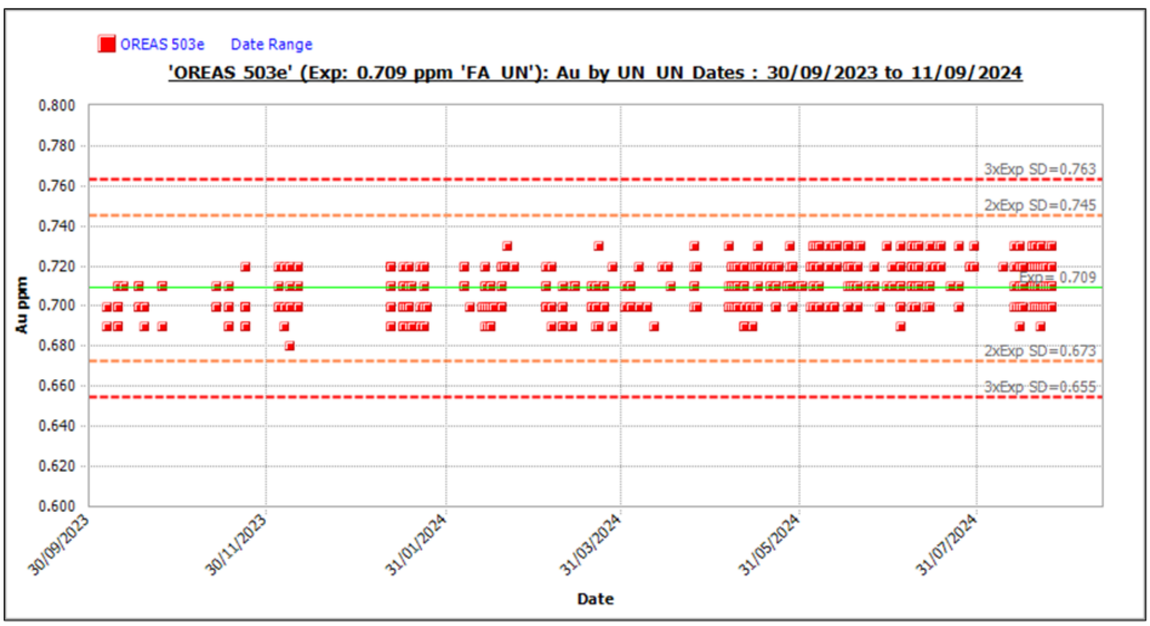
The CRM results were investigated per individual CRM for all laboratories combined, as well as per CRM per laboratory. There were some biases noted but are not indicative of any systemic biases. The results do not highlight any fatal flaws, and samples that failed may be attributed to sample swaps or 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 240b Performance for Au, All Laboratories Combined



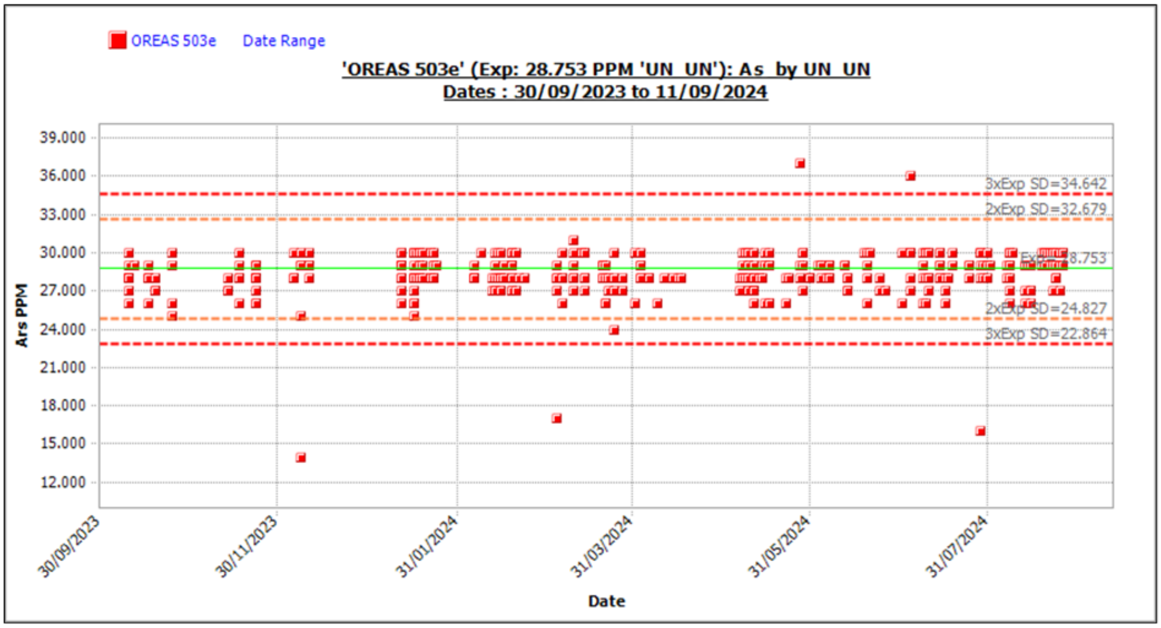
Outliers most likely attributed to sample swaps or incorrect labelling of CRMs as failure was across all elements
SGS_BO=103; SGS_BUR=30
Source: ERM, 2024

Figure 11.3 – OREAS 503e Performance for Au, All Laboratories Combined



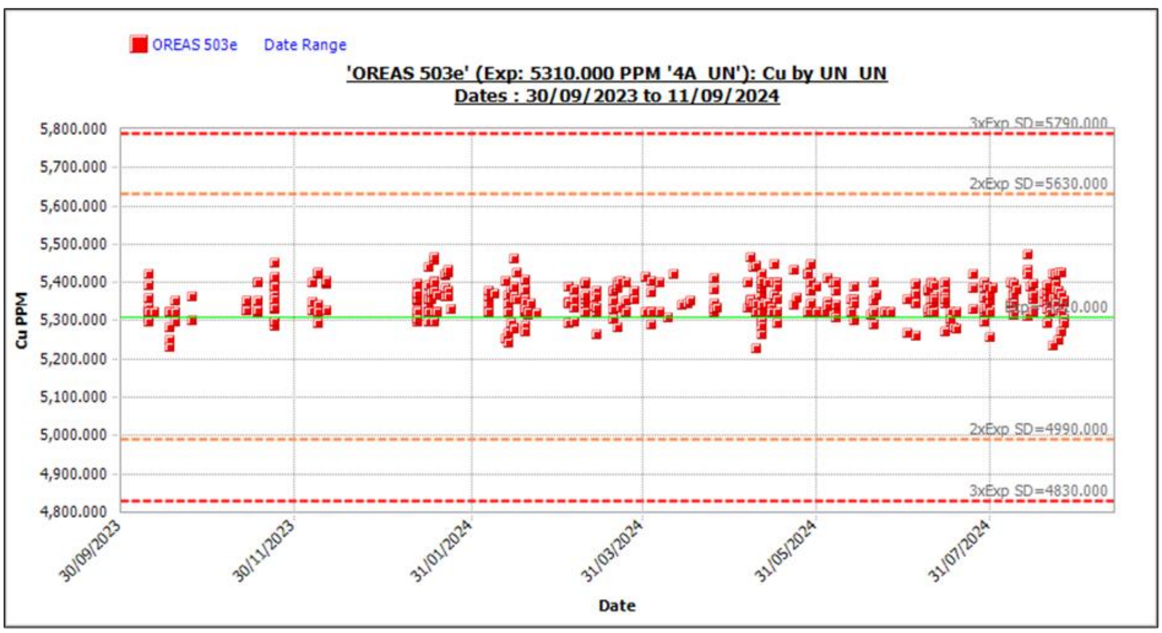
SGS_BO=440; SGS_BUR=68
Source: ERM, 2024

Figure 11.4 – OREAS 503e Performance for As, All Laboratories Combined



Source: ERM, 2024

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



Source: ERM, 2024

11.5.6.3 Precision

Preparation duplicates as well as external check (SFAs as Umpire) results were compared for diamond drill samples. There were no analysis dates in the database for the SFA results, so the hole completion date was assigned for the purpose of this review and the date filter adjusted so that holes in August and September were also included. The duplicate results were compared for primary samples submitted to SGS Bor, SGS Bur, and ALS Bor. The precision data for gold for sample type half diamond core fall within acceptable practice limits (Table 11.2). While there is a positive bias for SFA versus FA, this is an expected result since SFA produces more reliable results for coarse gold deposits because of:

1. improved representativity (separately analysing the coarse fraction meaning larger gold particles can be accounted for);
2. reduced nugget effect which reduces bias and improves accuracy especially in samples with high variability in gold particle size when compared to fire assay, and;
3. better repeatability because results are generally more consistent when dealing with coarse gold.

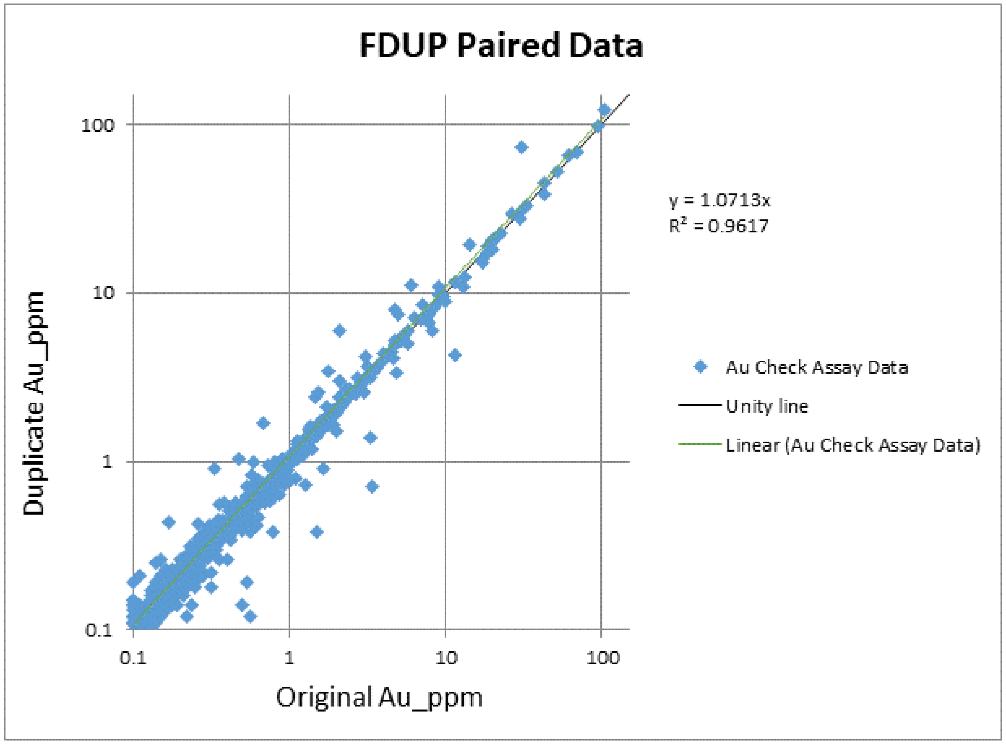
In conclusion, when coarse gold is expected, SFA is a more reliable method for obtaining more accurate and representative gold grades.

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	2,588	902	17	1.7	1.78	4%
SFA UMPIRE	DDH ½	10	20	5,589	3,432	26	5.50	6.53	19%

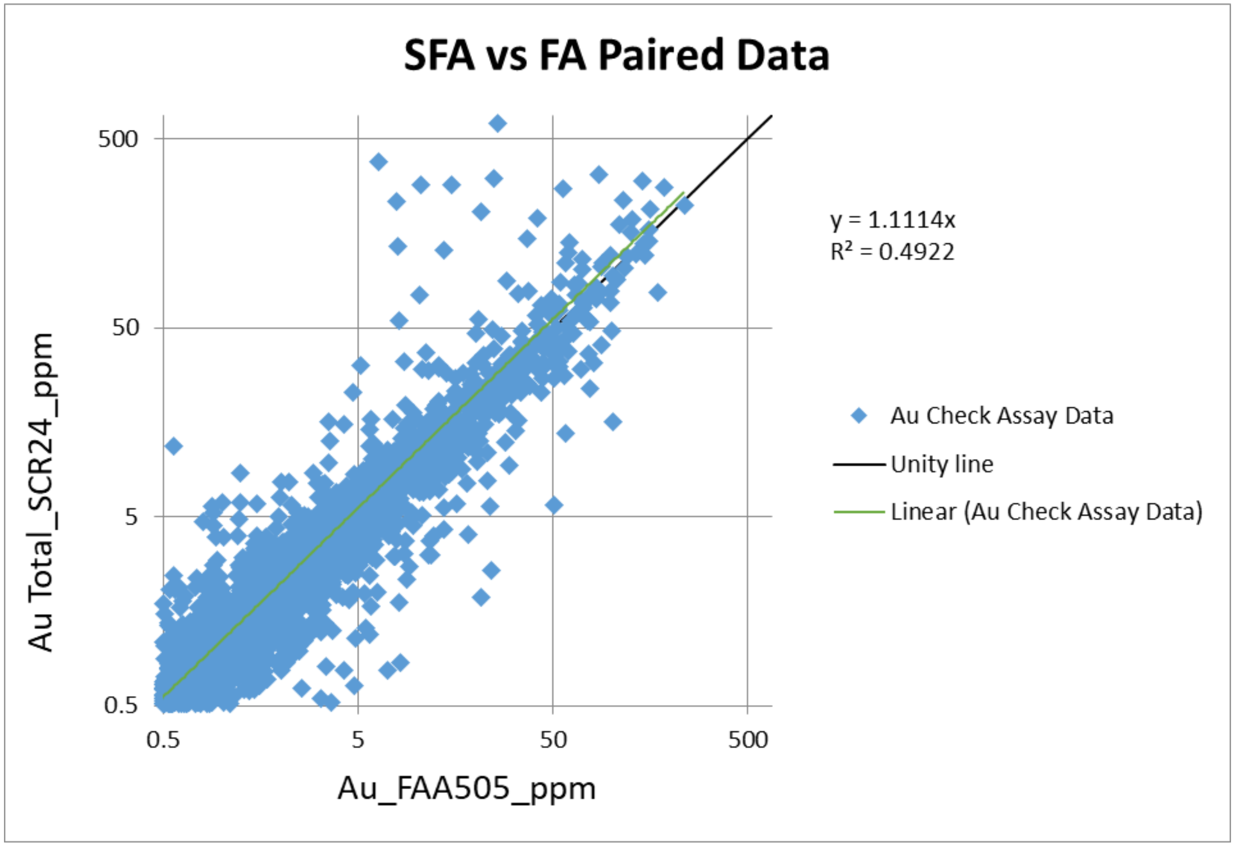
Source: ERM 2024

Figure 11.6 – Field Duplicate Paired Data



Source: ERM, 2024

Figure 11.7 – SFA vs. FA Paired Data



Source: ERM, 2024

11.6 Summary Opinion of Qualified Person

The QP (Ms. Maria O'Connor, MAIG) concludes that 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 suitable to monitor assay contamination, accuracy, and precision. The QP makes the following conclusions and recommendations.

- Approximately 82% of samples in mineralisation (>4,500 samples) are analysed by SFA (samples >1 g/t Au), which is a high percentage of samples and considered good practice given the presence of coarse gold. All SFA samples are analysed by ALS Romania. While it would be optimal to send a selection (5–10%) of coarse reject samples to another laboratory for SFA, it is understood that this is not possible since there is no reject material. It is possible to select half core and send that for duplicate analysis using the SFA workflow.
- To monitor contamination, barren washes are inserted after every SFA sample where FA analysis has returned a grade greater than 5 g/t Au.
- 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.
- One hundred twenty (120) pulp samples (including 5 CRMs and 5 Blanks), initially assayed in SGS BOR lab, were sent to ALS Ireland lab for umpire analysis in 2024. DPM selected samples that were randomly generated, representing 5% of the mineralised samples included in the mineral resource (samples exhibiting a gold grade greater than 0.1 g/t Au). Samples were assayed on 48 elements (excluding Au) by ME-MS61 (multi-element ultra trace method combining a 4-acid digestion with ICP-MS instrumentation), and in the case of over grade of Ag, Cu and Zn, by OG62 (ore grade elements by 4-acid digestion, and ICP-AES); sulphur is assayed by S-IR08 (total sulphur by oxidation, induction furnace and infrared spectroscopy). Gold was not included since the screen fire assay analysis is considered to be 'umpire' for gold. The silver, arsenic, copper and sulphide results reviewed demonstrate overall strong consistency and reliability between the original and umpire assay data, suggesting confidence in the data quality. Minor outliers and discrepancies should be reviewed, especially for lower concentration samples.
- 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 were obtained and inserted for this purpose in 2024 e.g. OREAS 993.
- Continual vigilance is required considering the extremely high gold grade values that have been encountered whilst drilling.
- The QP finds that the QA/QC procedures are adequate and that there are no fatal flaws evident.

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:

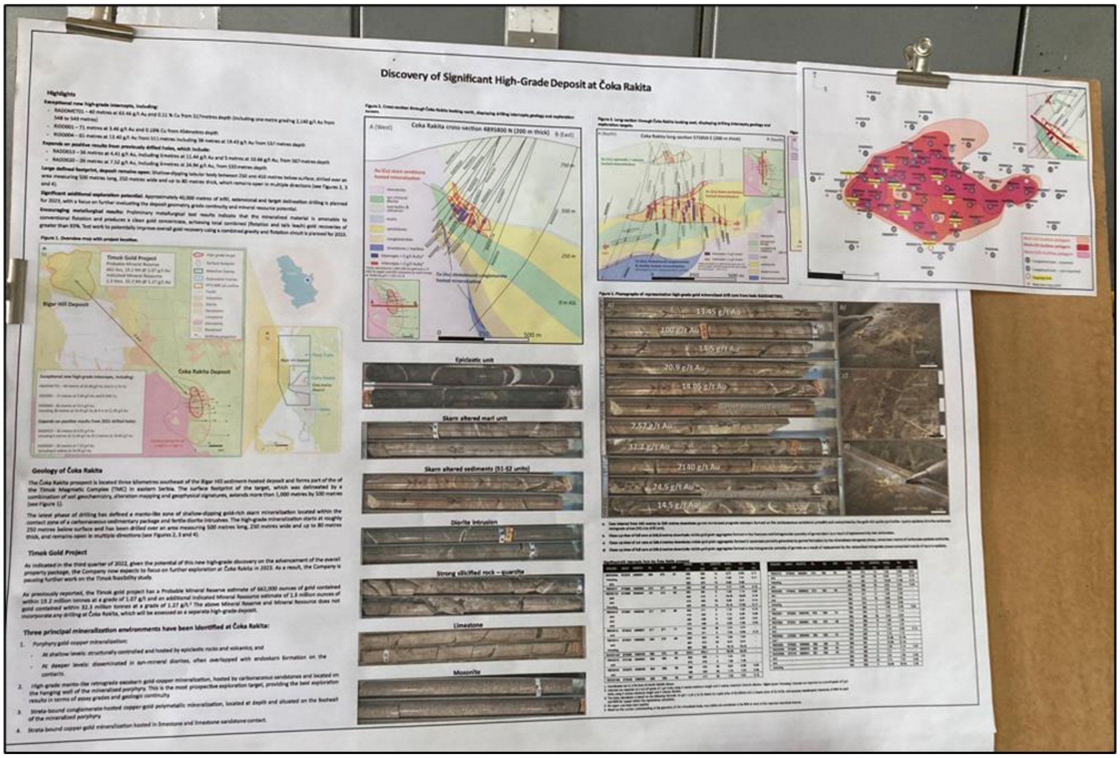
- Assay certificates for eight (8) drillholes – RADD013, RADD020, RIDD004, RIDD007, RIDD008, RIDD017A, RIDD020, RIDT004 – were requested by the QP due to their materiality to the MRE. These were reviewed for both FA (SGS) and SFA (ALS) results to enable spot checks to be completed against the database. 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 (Figure 12.5).

- Representative core from five 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 relationship between recovery and grade apparent.
- QA/QC was reviewed as described in Section 11.5.6 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 an example of good practice. 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

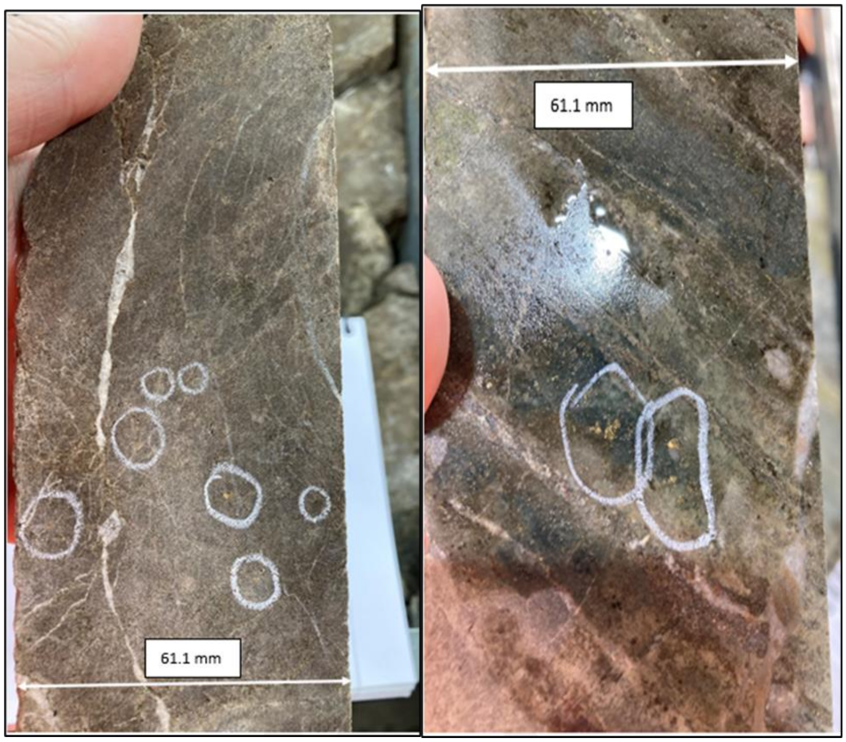


Source: CSA Global, 2023

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 (Maria O'Connor, MAIG) 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 generally relate to drillholes not yet logged, or outside the skarn, 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:

- Five (5) drillhole had zero depth recorded in the database – RIDD007B, RIDT012, RIDT014, RIDT039, RADD043A. These were terminated due to technical drilling reasons.
- Four (4) drillholes were in progress – RADDHG008, RIDD089, RIDD090, RIDD091.
- For "missing" assays, there were three (3) categories:
  - Seven (7) holes had no samples/assays recorded.
  - Eleven (11) holes had no samples/assays recorded for any part of the drill hole– RADD043A, RIDD007B, RIDD021A, RIDD068A, RIDT012, RIDT014, RIDT030C, RIDT039, RIRC013, RIRC014, RIRC031.
  - Samples in the assay file with no assay results – these were holes that had been sampled but were awaiting assay results for recently completed holes.
  - Samples where there were voids (OID/OIS - 3,009 m), or where the interpreted unit was SFD. Notably, there were no missing assays (beyond holes awaiting sampled but awaiting assays) within the skarn mineralisation which is the focus of this MRE.
- All drillholes had downhole survey records
- Four (4) diamond drillholes were missing geotechnical logging – one of which was recently drilled and the others had shallow depths < 60 m.
- All completed drillholes had logged geology and alteration except for 22. Of these, 2 are RC and 16 were drilled in 2024, meaning they are likely to be awaiting data.

- There are 125 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 density can be reliably estimated based on recorded measurements.

Overall, the drillhole database is clean and as complete as possible given 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 of mineral resource estimation of the Project.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

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

The 2023 Base Metallurgical Laboratories (BaseMet Labs) test program was performed on three (3) composite samples, representing low, medium, and high gold grades This test program was more comprehensive than the exploratory Wardell Armstrong program and covered many processing aspects including sample mineralogy, gravity concentration, flotation, cyanidation, sedimentation and filtration. Extensive testing was conducted to explore and optimise test conditions.

BaseMet Labs performed the PFS testwork program in 2024, primarily to understand the variability of the deposit with respect to both lithology and mineralisation types.

RMS tested a sample of tailings 2024 to support the design of the tailings dewatering and storage facilities. Sedimentation tests showed that the material settles well with an underflow of 65 wt% achieved at a flux rate of 0.5 t/m²/h and 15 wt% feed concentration, with clear overflow (TSS <100 ppm). Vacuum disc and belt filtration tests showed that the tailings filter effectively at feed concentrations of 65 wt% or higher. UCS testing demonstrated that a GGIBFS blend (90:10) achieved greater strength than cement alone, reaching 1 MPa with 5% binder and 2 MPa with 10% binder after 21 days and that there is no strength loss after 90 days of curing.

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)

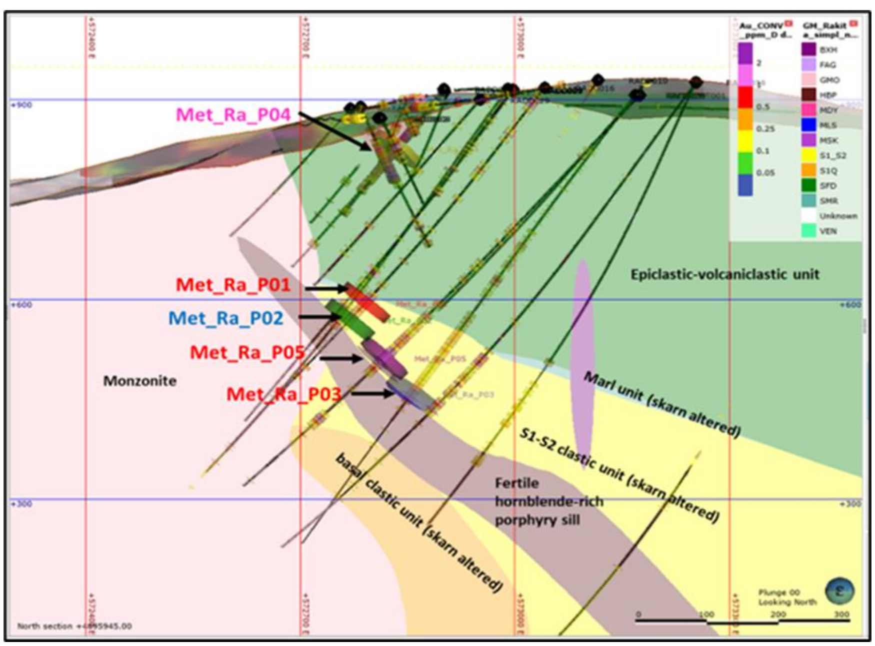
13.3.1 Sample Selection

The aim of this initial metallurgical testwork program was to understand variability related to various ore zones and paragenesis, including Au-rich retrograde skarn, moderate Au-Cu endoskarn, and Au-rich phyllic overprint. 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. The samples were as follows:

- Met_Ra_P01 taken from drillhole RADD016, interval 401 to 416 m below surface: retrograde exoskarn with high-grade Au mineralisation (2.96 g/t Au), low Cu, elevated Mn, Sn, Ca;
- Met_Ra_P02 taken from drillhole RADD016, interval 434 to 452 m below surface: retrograde endoskarn and potassic porphyry with moderate Au and Cu mineralisation (0.56 g/t Au and 0.42 % Cu);
- Met_Ra_P03 taken from RADD020, interval 540 to 559m below surface: phyllic overprinted exoskarn with high-grade Au (9.89 g/t Au), low Cu and elevated Mo and Mn;
- Met_Ra_P04 taken from drillhole RADDMET002, interval depth 107 to 118 m below surface: epiclastic-volcaniclastic (sequence felsic debris flow deposit unit), moderate potassic altered (mt-bt-ep-si) with moderate-grade Au (2.2 g/t Au); and
- Met_Ra_P05 taken from drillhole RADDMET001, interval depth 530 to 543 m below surface: retrograde exoskarn with high-grade Au (13.10 g/t Au).

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 (5) Wardell Armstrong Samples



Source: Wardell Armstrong, 2021

13.3.2 HEAD ASSAYS

Table 13.1 summarises the head assays determined for these samples.

Table 13.1 – Head Assay for Wardell Armstrong Test Program

Element	Unit	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.

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

13.3.3 MINERALOGY

Mineralogical examination showed that garnet and quartz are the most abundant minerals in these samples. Minor amounts of smectite were observed for samples P02/P03 and P04 (up to 9.8 wt%). The implication is that the ore could be sticky when wet and difficult to handle. P02 and P04 also contained some Illite/Mica (up to 7.5 wt%). 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.

The results show that approximately half of the gold present in these samples can be recovered using gravity concentration. 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 amenability of the samples to leaching. The tests were conducted at a lengthy duration of 48 hours and a cyanide concentration maintained at 1 g/l to ensure complete dissolution of all leachable gold. Gold recoveries from whole-ore cyanidations ranged from 82 to 93% with the average around 87%.

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 13.2 summarises the results of the scouting rougher flotation tests performed in 2021 at four (4) different grinds. The slurry samples were conditioned with $CuSO_4$ prior to adding PAX and MIBC at the start of the flotation stage. Flotation was performed over a total duration of 15 minutes. It is noted that the results for sample P02 are omitted.

Table 13.2 – 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 achieved 51.7% only. 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 variation in recoveries at all grind sizes.

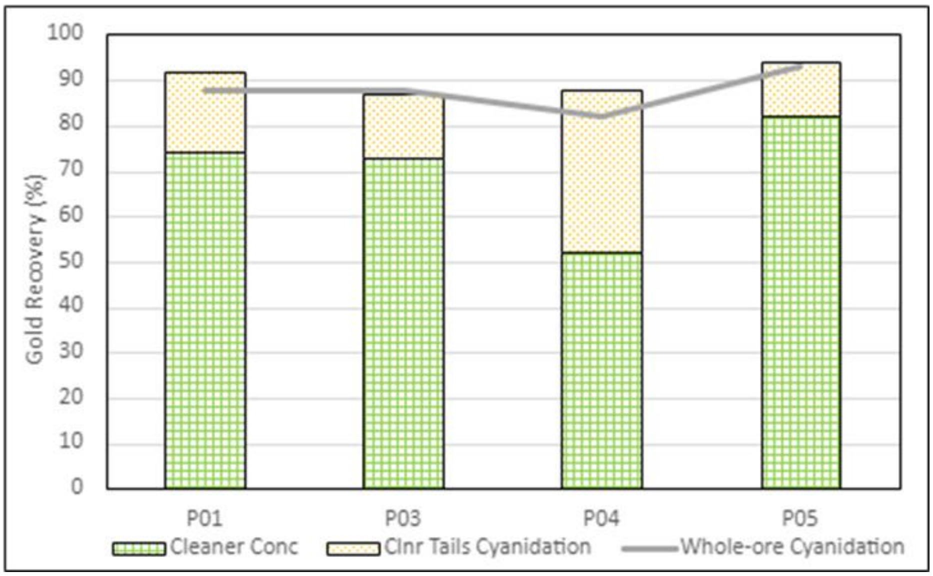
A further set of tests were performed at grinds of 75 µm and 45 µm, where a cleaner step was added after rougher flotation to assess how much the grades could be improved through cleaning stages.

The results show that the addition of a cleaner step will improve the final concentrate grade substantially 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 the 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



Source: Wardell Armstrong, 2021

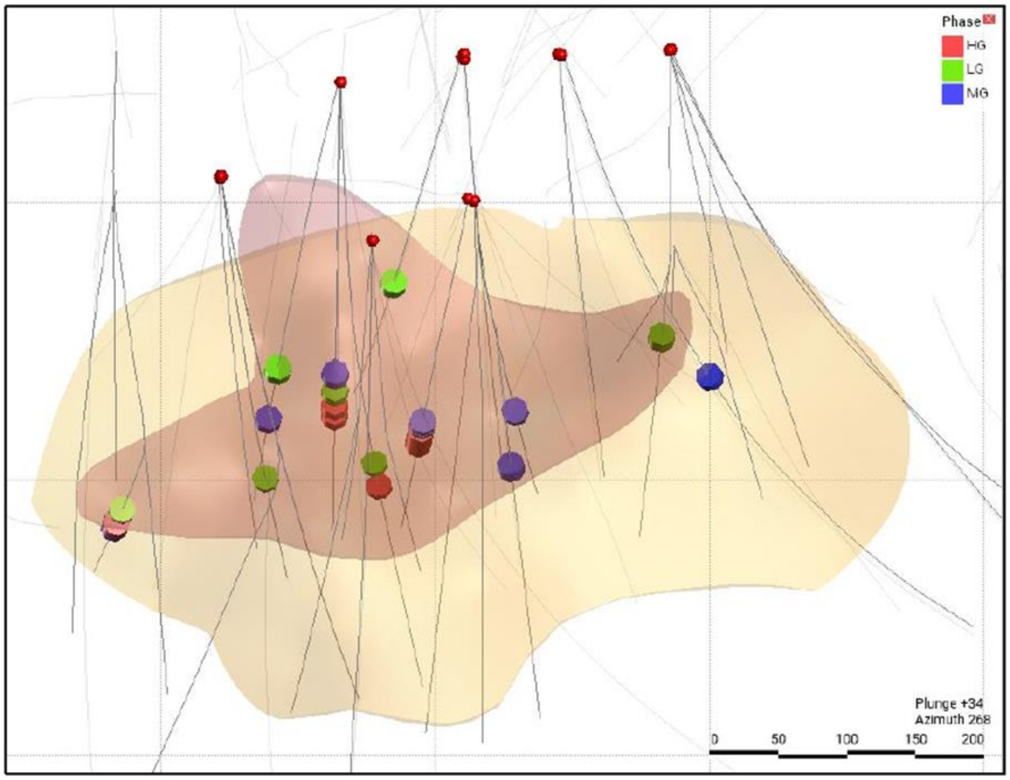
13.4 BaseMet Labs Testwork Program 2023

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. A representative weighting of lithology types was also specified. The selection targeted 10 m intervals, but a few shorter ones were also included.

Figure 13.3 – Sample Locations for the Drill Cores Selected for the Three (3) 2023 Composite Samples



Source: BaseMet Labs, 2023

13.4.2 HEAD ANALYSIS

Table 13.3 summarises the head analysis performed on the three (3) composites tested in 2023 and 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 13.3 – 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

Other major elements include silver which ranged between 0.7 and 1.1 g/t and arsenic which ranged between 48 and 115 ppm. Both are low and should not have a major impact on the final process design selected.

13.4.3 Mineralogy

AGAT Geology Dept (directed by BaseMet Labs) performed XRD analysis on powdered samples of the three (3) 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 13.4 presents the results of the laser ablation analysis of the pyrite showing the distribution of metals.

Table 13.4 – Metal Concentrations Within Pyrite

Sample ID	Average Element 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 indicates that there is very little refractory gold locked in pyrite.

13.4.5 Comminution

Table 13.5 summarises the comminution results produced for the PEA phase of the Project. Three (3) metallurgical composite samples were tested individually.

Table 13.5 – Summary of Comminution Test Results (2023 Test Program)

Sample ID	Dwi (kW/m³)	Axb	SG	SCSE (kW/t)	Ai	BBWi (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
Average	5.83	54.2	3.15	9.34	0.138	13.3
75th percentile	5.97	52.5	3.18	9.46	0.146	13.4

Source: BaseMet Labs, 2023

For the Axb parameter, a smaller number indicates a more competent ore 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.

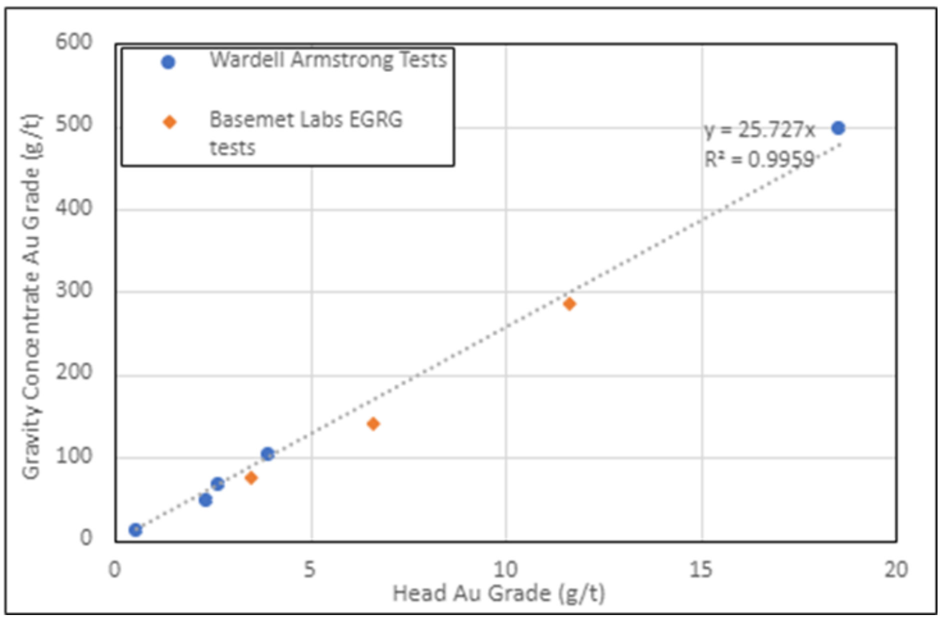
The hardness parameters measured for the samples place the Čoka Rakita material in the average hardness category. Similarly, at an abrasion index value of around 0.14 the samples can be described as only mildly abrasive.

13.4.6 Gravity Concentration

BaseMet Labs subjected their three (3) composite samples to an Extended Gravity Recoverable Gold (EGRG) procedure comprising staged recovery of gravity gold at increasingly finer grinds.

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 Labs 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 and operated.

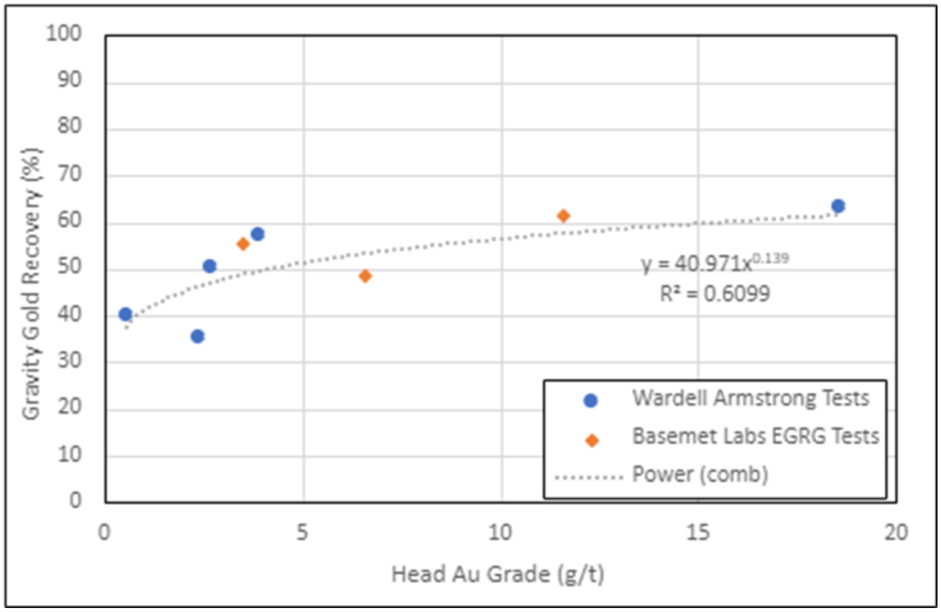
Figure 13.4 – Gravity Concentrate Gold Grade versus Head Grade



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 (2) testwork programs into one (1) graph.

Figure 13.5 – Gravity Gold Recovery versus Head Grade



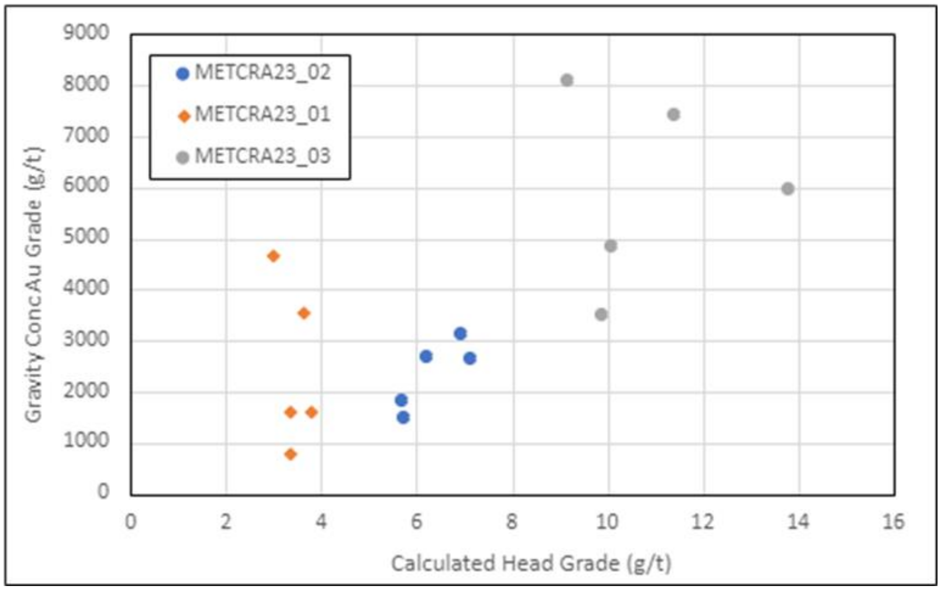
Source: DRA, 2023

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 (2) 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 (It appears the settings on the shaking table and/or Knelson must have been modified to produce a higher-grade gravity concentrate. As a result 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). Note that 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 (2) grades. The significant increase in variability is partly due to the different grinds, but likely mostly due to the change in methodology which targeted a much smaller mass pull.

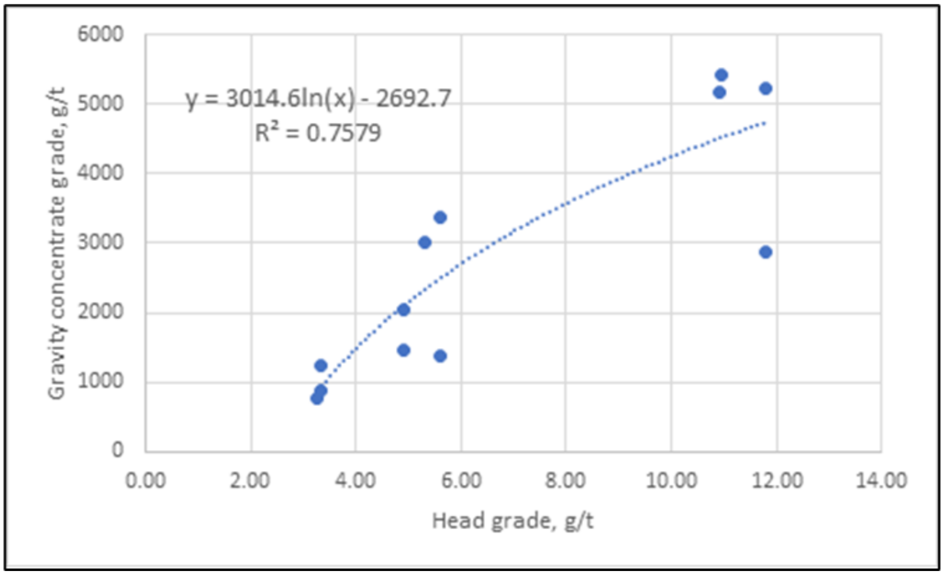
Figure 13.6 – Gravity Concentrate Gold Grade versus Sample Head Grade



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.

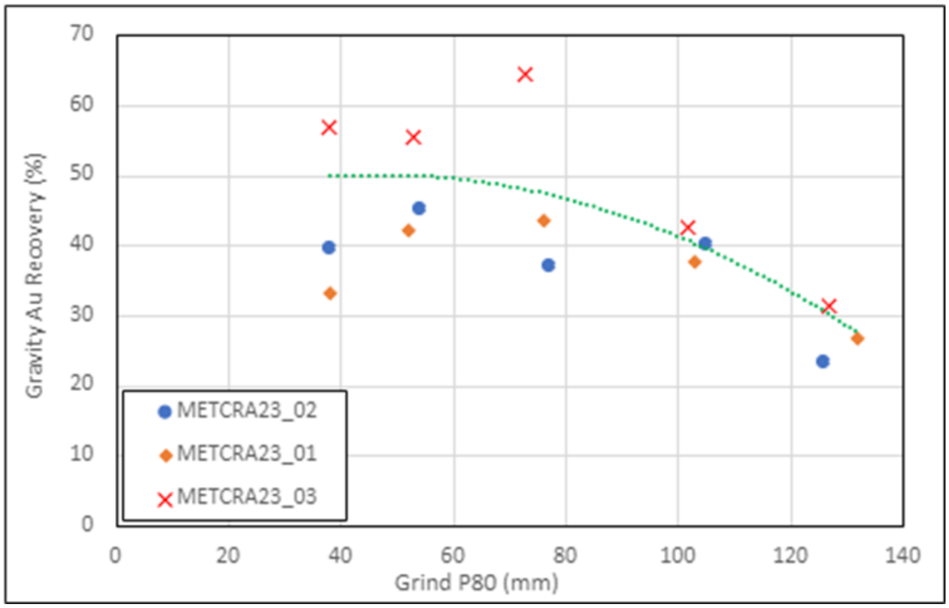
Figure 13.7 – Gravity Concentrate Grade versus Head Grade (53 µm grind)



Source: DRA, 2023

Figure 13.8 illustrates 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

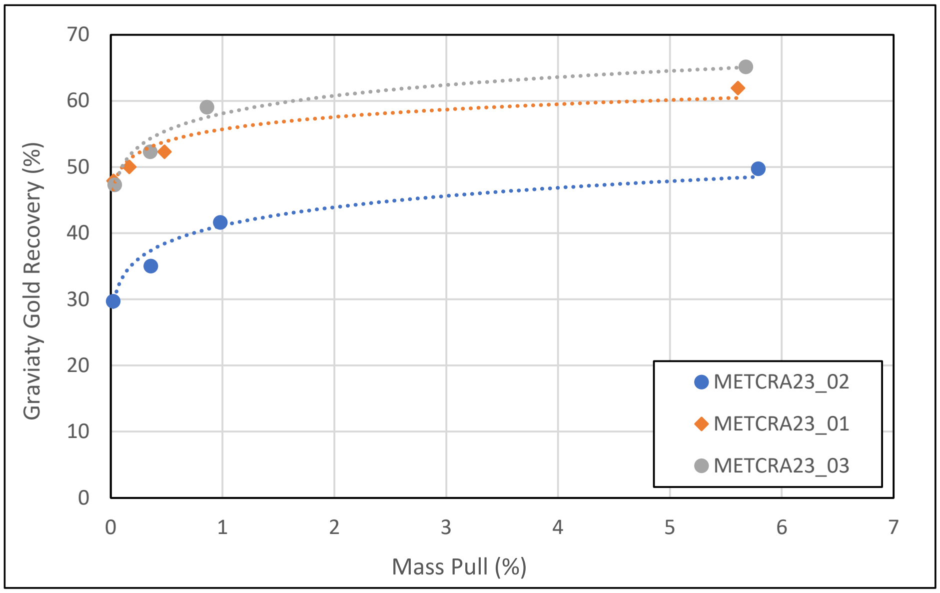
From Figure 13.8, it can be concluded that gravity gold recovery decreases as the grind coarsens above 53 µm. Below 53 µm, the recovery of coarse gold appears to be unaffected by the grind size. It appears all gravity recoverable gold is fully liberated at a grind of 80% passing 53 µm.

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 the 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 (3) 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 (refer to the BaseMet Labs report) shows that a leach duration of 24 hours is sufficient.

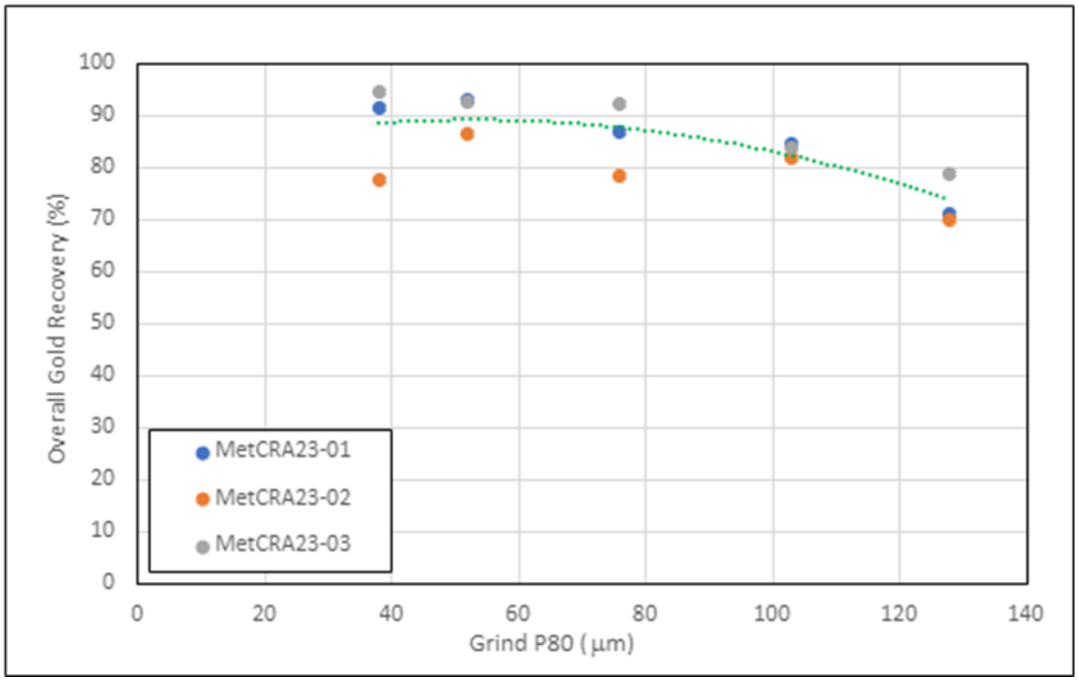
13.4.8 GRAVITY AND ROUGHER FLOTATION TEST PROGRAM

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 comprises 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. 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, 2023

The green dotted line is the best fit to all the data points and it shows an optimum grind around a P_{80} of 53 µm. The curve flattens out below 53 µm with no significant further increase in overall gold recovery. The additional marginal revenue would likely not exceed the cost increase required to grind finer than 53 µm (especially not if a specialised grinding mill is required to achieve the fine primary grind).

13.4.9 CLEANER FLOTATION TEST PROGRAM

BaseMet Labs subjected the three (3) 2023 samples to cleaner flotation testwork to investigate whether this could be used to improve concentrate quality. The effects of regrinding and primary grind size were also explored during these tests. Table 13.6 summarises the results of this set of tests.

Table 13.6 – 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

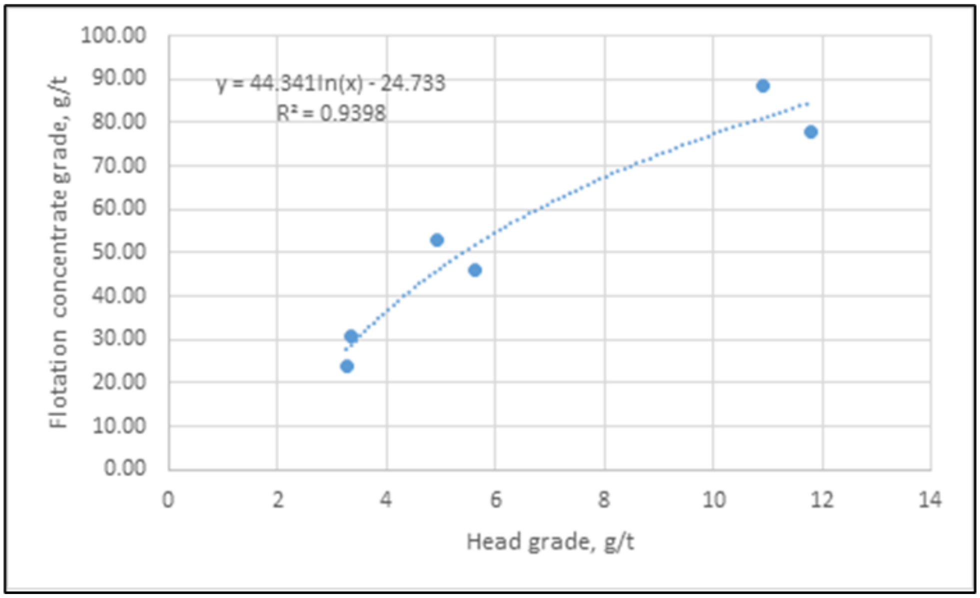
Source: BaseMet Labs, 2023

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.

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%.

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

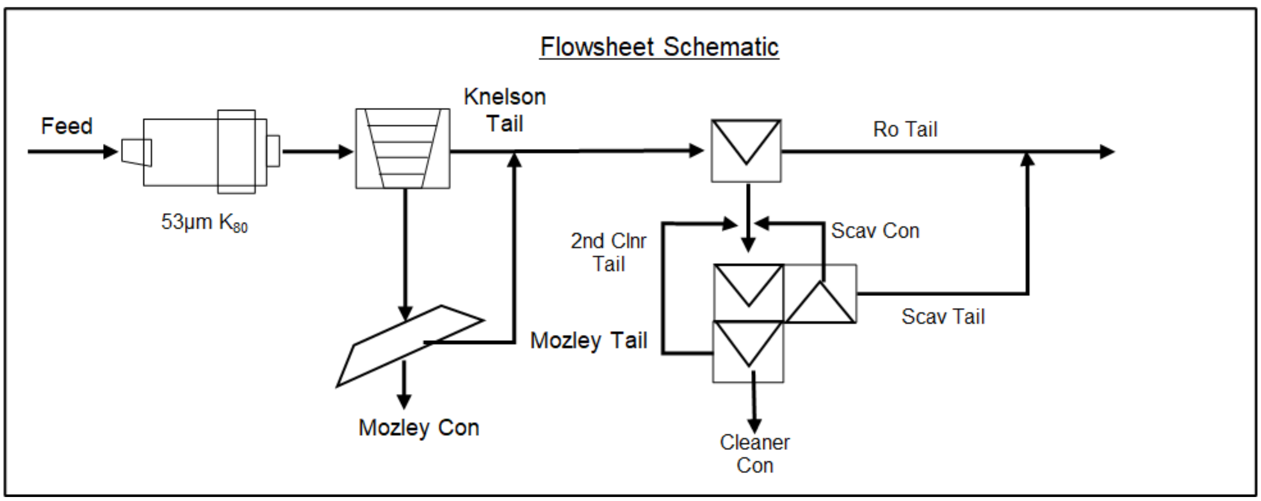


Source: DRA, 2023

13.4.10 Locked Cycle Flotation 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 (2) 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 min 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 min;
- Cleaner 1 duration = 15 min, Cleaner 2 duration = 10 min, Cleaner scavenger = 6 min.
- Add 20 g/t PAX and 3477 to Cleaner 1.
- Add 10 g/t PAX and 3477 to Cleaner 2 and cleaner scavengers.
- MIBC addition varied depending on visual assessment of the froth.

Table 13.7 summarises the mass and pertinent gold data obtained from the LCTs. It is noted that the data for some of the streams were calculated.

Table 13.7 – 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 Distrib %	Wt%	Au (g/t)	Au Distrib %	Wt%	Au (g/t)	Au Distrib %
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
2nd Clnr Tail	0.9	8.97	2.7	1.2	11.2	2.6	0.3	53.1	1.8
2nd Clnr Feed	6.3	21	42.4	6.0	39.1	44.3	4.2	96.6	38.5
Clnr Scav Conc.	0.7	7.09	1.6	0.8	12.2	1.8	0.8	24.6	1.8
Clnr Scav Tail	6.1	2.03	4	5.9	3.8	4.3	4.8	7.7	3.5
Clnr Scav Feed	6.8	2.56	5.6	6.6	4.8	6.0	5.5	10.1	5.3
1st Clnr Conc.	6.3	21	42.4	6.0	39.1	44.3	4.2	96.6	38.5
1st Clnr Tail	6.8	2.56	5.6	6.6	4.8	6.0	5.5	10.1	5.3
1st Clnr Feed	13.1	11.4	48	12.6	21.0	50.3	9.7	47.2	43.9
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 the cleaner scavenger tails contributed only 4.3%.

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.

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 13.8 summarises the most significant elements.

Table 13.8 – Analysis of Concentrate Samples

Element	Units	Met_CRA23_01	Met_CRA23_02	Met_CRA23_03	Gravity Con
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

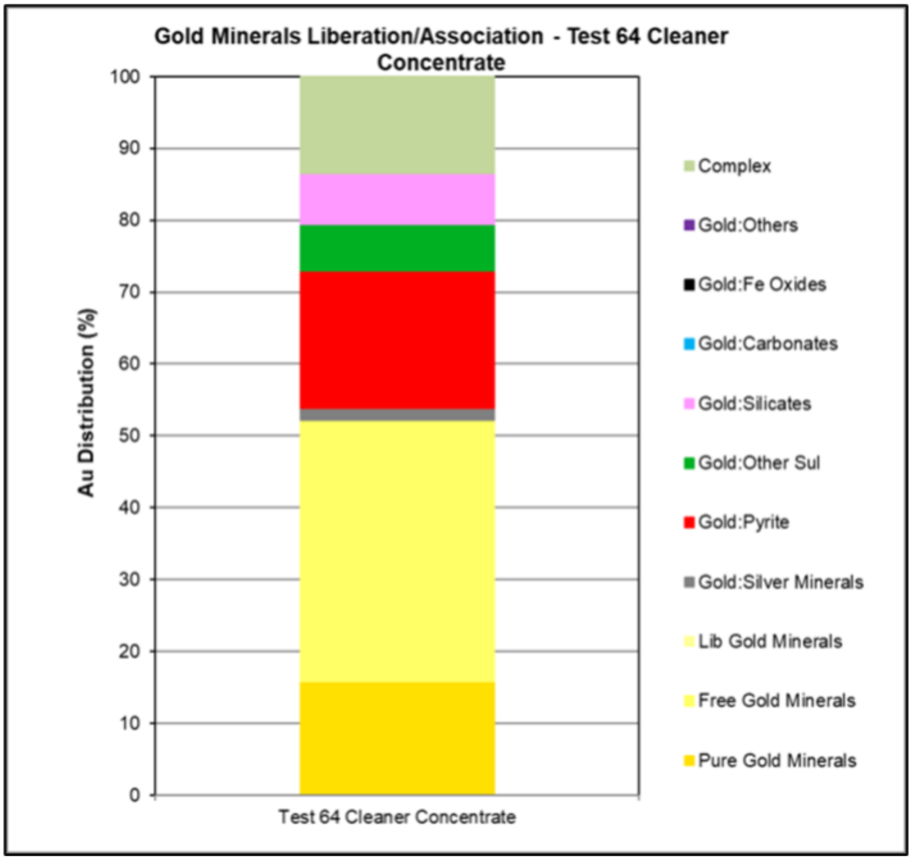
Source: BaseMet Labs, 2023

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 is reproduced here from the BaseMet Labs report. 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

13.4.10.2 Gold Distribution by Size in Feed and Tails Samples

The three (3) feed samples and three (3) LCT samples were screened into size fractions and then each size was analysed for gold grade.

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 size distributions for the tailings shows 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 (3) LCTs were combined and subjected to a 48 h cyanidation. Table 13.9 summarises the results.

Table 13.9 – 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 LCTs an overall recoveries exceeding 96% for all three (3) samples is achieved. While this step would increase gold recovery significantly, it would also introduce cyanide to the site which is currently not included in the permitting. A decision has been made to avoid the use of cyanide on site.

Comparing the 96% recovery against the 88% recovered through whole ore 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 (2) flowsheets as it can be assumed all gravity gold will also have leached in the 48 h 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.

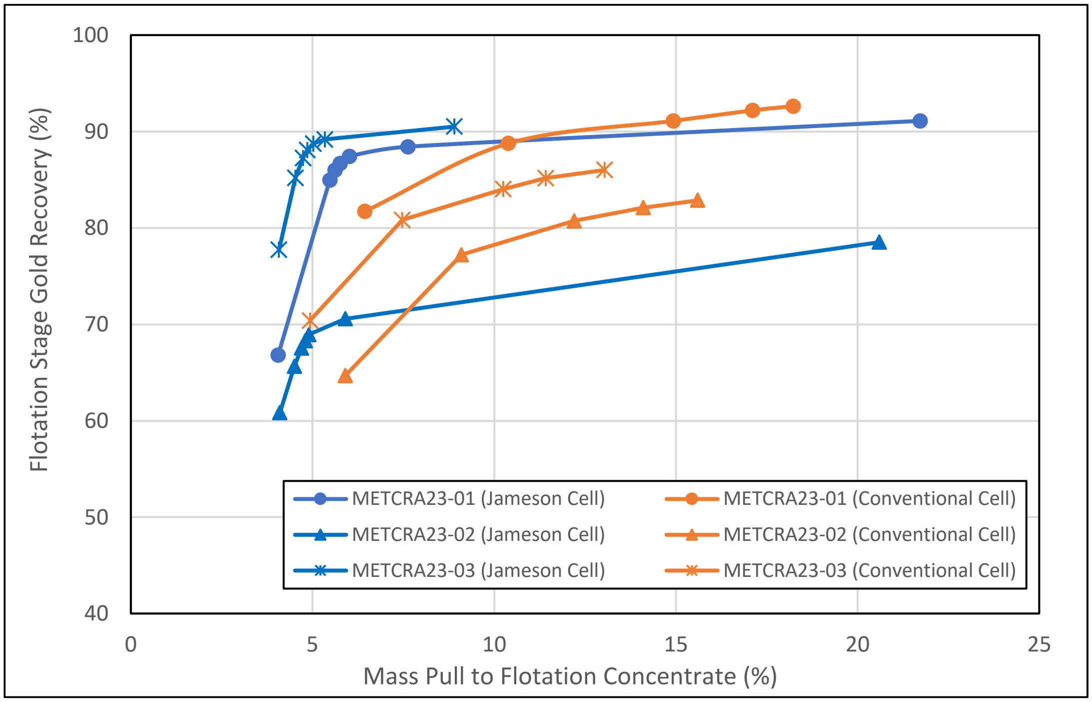
The results showed that near complete leaching is achievable (99.8% to 99.9%) after 72 h of cyanidation. Leach kinetics was slow with about 7% of the leaching occurring in the final interval from 48 h to 72 h. Cyanide consumptions ranged from 30 kg/t to 80 kg/t, but this is possibly due to the oxidation 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 (dilution amenability test) 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 13.14 presents the stage recovery of gold from the gravity tails to the concentrate versus mass pull to the flotation concentrate for the three (3) samples tested.

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.

Figure 13.14 – Gold Stage Recovery to Flotation Concentrate versus Mass Pull



Source: DRA, 2023

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 (2) 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 ore. Static tests were conducted at pH 8 and using five (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 tails and conc samples using different dosages of the selected flocculant. For the tailing 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 u/flow 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 13.10. Due to the small sample size only one (1) 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/m² can be used for the sizing of thickeners as both 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 13.10 – Dynamic Settling Tests

Sample	Loading Rate (tph/m²)	Flocculant Addition (g/t)	Rise Rate (m/h)	Turbidity (ppm)	Underflow %solids	Viscosity (Pa)
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 13.11. The starting densities of the slurries were around 60 to 70% w/w solids in both cases. It is noted that 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 13.11 – Filtration Tests Summary

Sample	Parameter	Formed	Pressed	Air Blow (1 min)	Air Blow (3 to 6 min)
Tails	Filtration rate (kg/h/m²)	1,649	1,064	449	349
	Moisture Content (%)	28.8	22.7	18.5	16.3
Conc	Filtration rate (kg/h/m²)	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 BaseMet Labs Testwork Program 2024

13.5.1 Sample Selection

The purpose of the PFS testwork program performed by BaseMet Labs in 2024 was, primarily, to understand variability with respect to both lithology and mineralisation types. Secondary objectives relate to addressing recommendations from the PEA. New variability samples were selected from the available drill cores for the variability testing program. The rest of the testwork was conducted using the remnants of the three (3) composite samples from the PEA phase. The leftover PEA samples were stored in a controlled environment to ensure they do not oxidise (given that similar recoveries were recorded it appears the sample integrity was indeed maintained properly).

The selection of intervals for the variability samples was limited to quarter core only. This is because a significant amount of the half cores was already sampled during previous testwork programs. Only RIDD and RIDT holes were selected as the old RADD holes were logistically difficult to be processed hence were deemed unsuitable.

The primary selection criterion was to cover the full range of lithologies and mineralisation within the dominant (>86%) S1/S2 lithology. Secondary criteria were to cover a wide range of grades and for the samples to also be spatially representative of the entire orebody. The defined mineralisation types and their corresponding sample IDs are as follows:

- Endoskarn (diorite) mineralisation [i.e., HBP_1, HBP_2, and HBP_3];
- Monzonite mineralisation [i.e., GMO_1];
- Marls mineralisation [i.e., SMR_1 and SMR_2]; and
- Pyrrhotite-magnetite mineralisation [i.e., POMT_1, POMT_2, POMT_3, and POMT_4], where the intervals were logged as Po-Mt skarn.

- Copper mineralisation, which were further classified into three (3) grade groups based on Cu-assays:
  - High Cu [i.e., HCU_1]: higher than 5,000 ppm;
  - Medium Cu [i.e., HCU_2]: between 2,000 ppm and 5,000 ppm; and
  - Low Cu [i.e., HCU_3]: between 500 ppm and 2,000 ppm.
- Sulphur mineralisation, which were further classified into three (3) grade groups based on S-assays:
  - High S [i.e., SUL_1]: higher than 8%;
  - Medium S [i.e., SUL_2]: between 3% and 8%; and
  - Low S [i.e., SUL_3]: lower than 3%.
- Coarse-Au Mineralisation [i.e. VG_1 and VG_2] from 4 intervals: 3 m, 11 m, 10 m, and 4 m.

It should be noted that both QCU_1 (quartzite-hosted Cu mineralisation) and MCU_1 (marble-hosted Cu mineralisation) were excluded from the PFS testwork program since they are not within the boundaries of the Čoka Rakita orebody. The remaining 18 samples were composited from 41 intervals. Table 13.12 summarises pertinent details pertaining to the variability samples.

Table 13.12 – PFS (Phase 3) Samples Summary

Lithology	Sample ID	Selection Criteria	Category	Block Model Ore Proportion
Diorite (early mineralised porphyry)	HBP_1	Diorite Lithology	Diorite	4.60%
	HBP_2			
	HBP_3			
S1/S2	HCU_1	S1/S2 Lithology, Cu Mineralisation	S1/S2	86.76%
	HCU_2			
	HCU_3			
	SUL_1	S1/S2 Lithology, Sulphur Mineralisation		
	SUL_2			
	SUL_3			
	VG_1	S1/S2 Lithology, Coarse Au Mineralisation		
	VG_2			
	POMT_1	S1/S2 Lithology, Pyrrhotite-Magnetite (Fe) Mineralisation	PoMt	9.09%
	POMT_2			
	POMT_3			
	POMT_4			
Monzonite	GMO_1	Monzonite Lithology	Monzonite	0.09%

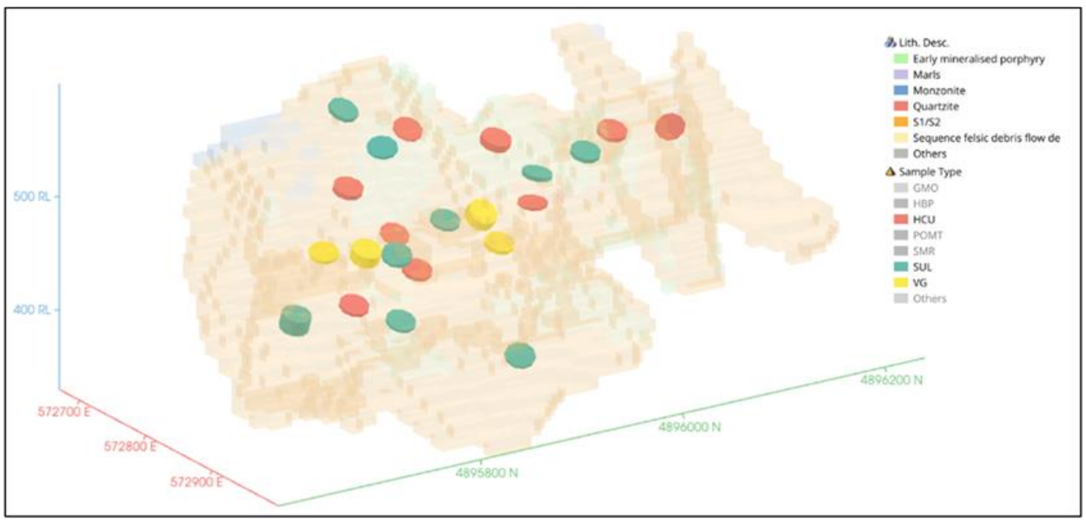
Lithology	Sample ID	Selection Criteria	Category	Block Model Ore Proportion
Marls	SMR_1	Marls Lithology	Marls	0.03%
	SMR_2			
Other	N/A	Not part of Phase 3 testing	N/A	0.15%

Source: DRA, 2024

The Monzonite and two (2) Marls lithologies samples were tested but their results were excluded from the subsequent analyses as they each represent less than 0.2% of the block model. The remaining 15 samples included in the analyses were produced from 36 intervals and represent 99.7% of the block model.

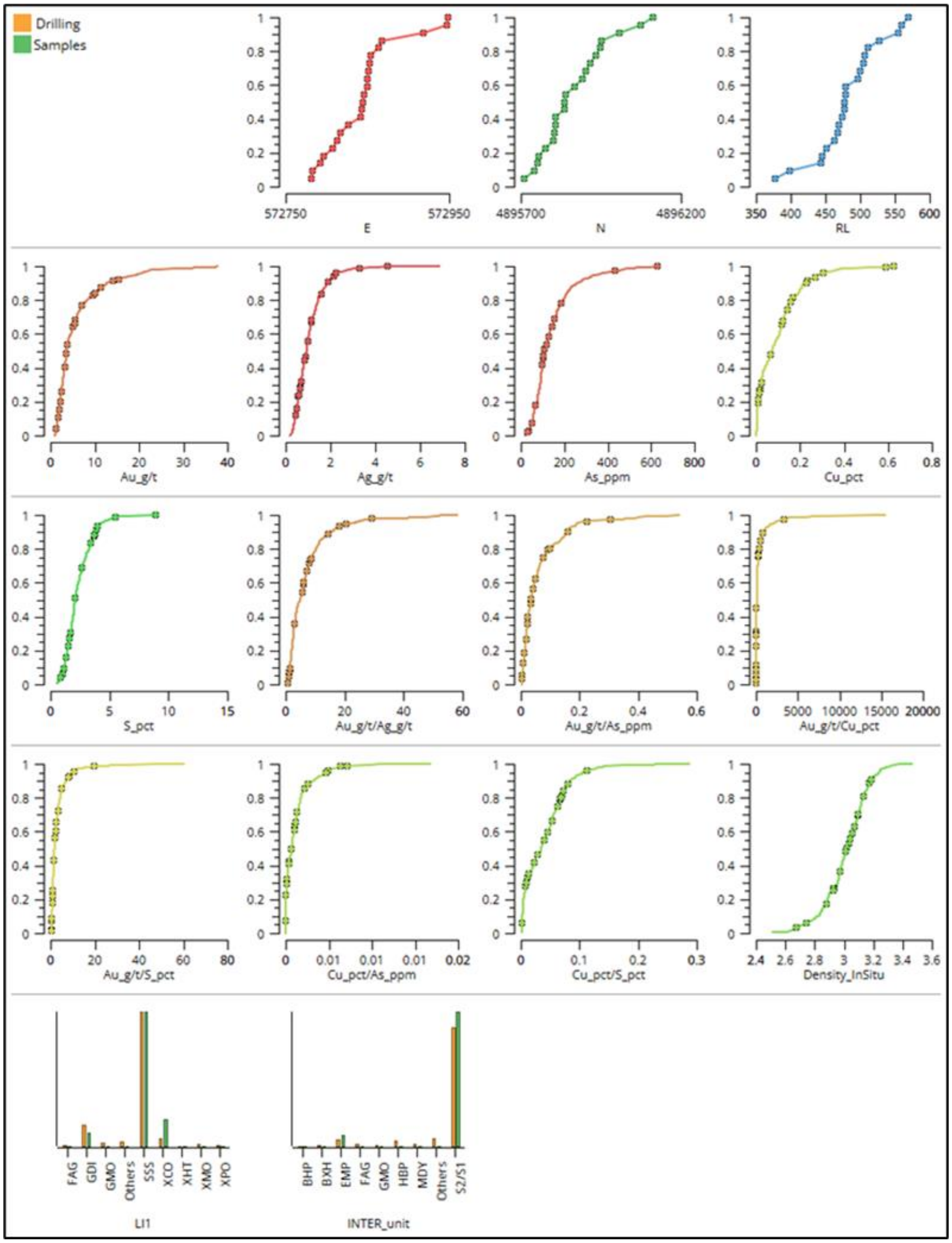
The Cancha software package was used to confirm sample representativity. Figure 13.15 presents the spatial distribution of the intervals selected for the variability samples as an overlay on the block model. Figure 13.16 plots the Cancha distribution graphs for various elements and spatial distribution. In the QP's opinion, the graphs show that the samples represent the block model well and that the full range of element grades were adequately covered.

Figure 13.15 – Spatial Distribution of Intervals Selected for Variability Samples



Source: DPM, 2024

Figure 13.16 – Cancha Distribution Graphs for Various Elements and Spatial Distribution



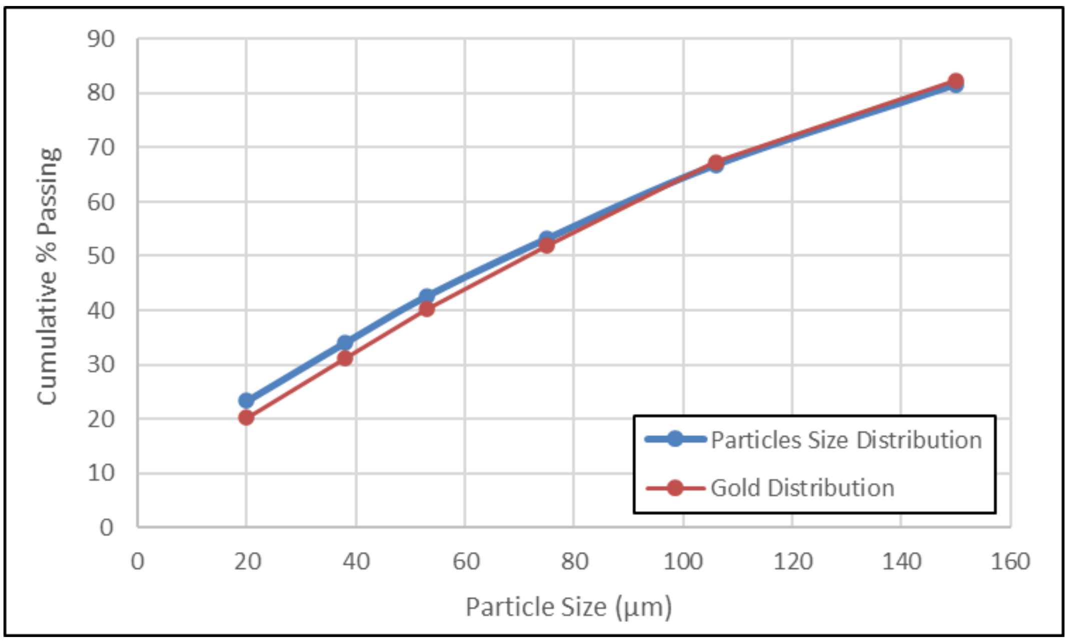
Source: DPM, 2024

13.5.2 Particle Size Distribution

A sub-sample of each variability composite was ground to approximately 80% passing 150 µm and then screened and analysed for gold per size fraction.

Figure 13.17 shows the averaged profiles for both the particle sizes and gold deportment by size. The gold content is evenly distributed across the size ranges as the profiles are nearly identical. Towards the finer sizes, there was some divergence with less gold in the finer fractions e.g. the bottom pan (-20 µm) contained 23% of the particles but only 20% of the gold. This boded well for a flotation flowsheet as it showed that that the gold present does not preferentially occur in the finer fractions during grinding, but, instead, remained within the 20 to 150 µm range where flotation recoveries are optimal.

Figure 13.17 – Size by Size Gold Deportment Profiles



Source : DRA, 2024

13.5.3 Head Analysis

Table 13.13 shows the head analyses performed on all 18 samples. Three different methods were used, namely flame atomic absorption spectrometry (FAAS) for gold and iron contents, inductively coupled plasma (ICP) for arsenic and silver contents, and LECO for sulphur content.

For the 15 samples that were considered significant for testwork analysis, the average head gold grade was approximately 4.5 g/t, which was sufficiently close to the LOM grade of around 5.5 g/t. Silver grades were generally low. Arsenic grades varied significantly and were appreciably more abundant in the POMT samples. This indicates the potential need to blend the POMT with the rest of the ore to avoid penalties for exceeding the arsenic limit specified for the flotation concentrate.

With the POMT mineralisation constituting approximately 10% of the mineral resource volume and a LOM of 8 years the implication is that there is approximately 10 months worth of POMT in the 8 year LOM. While the degree to which this is clustered/distributed is not known at this time it stands to reason that there could be prolonged periods of days or even weeks where a high proportion of POMT will be mined. It would be prudent for the mine plan developers to be cognizant of the need to limit the POMT in the ROM blend to a maximum of 50% of plant feed as much as is practical.

Table 13.13 – Head Analysis for PFS Testwork (2024)

Element		Au	S	As	Fe	Ag
Method		FAAS	LECO	ICP	FAAS	ICP
Unit		g/t	%	g/t	%	g/t
Sample ID	HBP-1	6.20	5.62	40	5.55	
	HBP-2	2.26	2.14	26	1.98	
	HBP-3	4.98	9.33	164	7.55	
	GMO-1	8.42	3.43	114	3.42	
	SMR-1	0.98	3.07	81	8.35	
	SMR-2	1.23	2.41	102	10.1	0.4
	POMT-1	6.69	7.53	786	10.9	2.4
	POMT-2	4.64	5.50	433	11.4	1.6
	POMT-3	1.37	7.67	442	7.85	1.5
	POMT-4	1.42	2.71	368	6.90	0.7
	HCU-1	5.63	8.40	77	9.25	4.0
	HCU-2	5.43	4.45	78	6.40	1.2
	HCU-3	5.76	4.83	131	9.30	
	SUL-1	2.33	11.8	250	12.1	4.5
	SUL-2	4.73	5.49	377	8.50	1.5
	SUL-3	5.18	2.07	41	8.40	0.4
	VG-1	5.34	1.00	66	7.20	0.3
	VG-2	7.17	0.90	55	6.10	0.5

Source: BaseMet Labs, 2024

13.5.4 COMMINUTION

Table 13.14 summarises the comminution results produced for the PFS phase of the Project. It should be noted that for the A×b parameter a smaller number indicated a more competent ore hence the 15th percentile was used as the most onerous result for design purposes.

Comminution results indicate only minor variations in hardness and other characteristics between the samples tested. For example, the difference between the average Bond Ball Mill Work Index (BBWi) value of 13.9 kWh/t and the 85th percentile at 15.2 kWh/t was only 9%.

The hardness parameters measured for the samples placed the Čoka Rakita material in the "average hardness" category. Similarly, at an abrasion index value of around 0.10 the samples could be described as only "mildly abrasive".

By comparing the variability results against the PEA testwork results for the three (3) metallurgical composites, it is evident that the composites tested were representative of the orebody as the hardness/competency parameters are similar. The average BBWi for the PEA was 13.3 kWh/t versus 13.9 kWh/t for the variability samples. Similarly, the A×b parameter averaged 54.2 in the PEA, as compared to 59.8 for the three (3) PEA samples.

Table 13.14 – Summary of Comminution Results (PFS Testwork)

Parameter		wt%	SG	SMC	DWi	SPI	BBWi	Ai
				A×b	kWh/m³	min	kWh/t	g
Statistics	Average		3.10	59.8	5.17	73	13.9	0.103
	15th Percentile		**2.92**	**44.6**	**4.22**	**60**	**12.3**	**0.064**
	75th Percentile		3.17	65.8	6.92	103	15.0	0.160
	85th Percentile		3.30	72.6	6.96	103	15.2	0.167

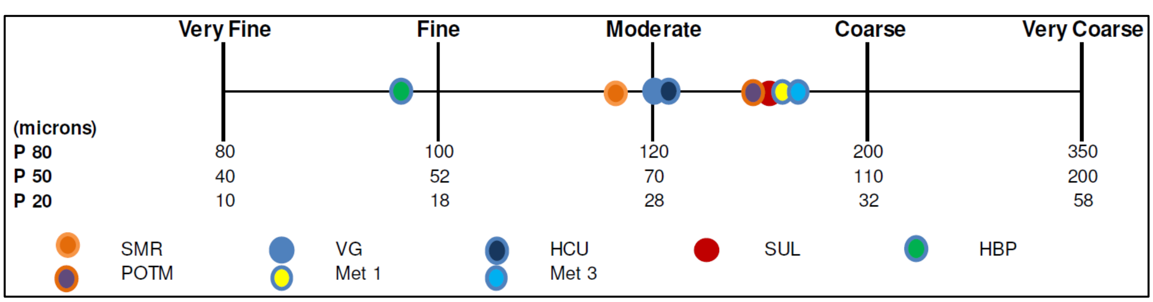
Source: BaseMet Labs, 2024

13.5.5 GRAVITY CONCENTRATION

All 18 variability samples underwent an Extended Gravity Recoverable Gold (EGRG) procedure comprising staged recovery of gravity recoverable gold at increasingly finer grinds.

The full set of laboratory results was provided to FLSmidth for modelling expected products from operating various Knelson concentrator models. Figure 13.18 shows classification, by FLSmidth, of the variability samples and the two composites tested on the Amira scale. For most samples tested the GRG was classified as "moderate" to "coarse" with only the HBP sample classified as "fine".

Figure 13.18 – GRG Fineness on the Amira Scale



Source: FLSmidth, 2024

FLSmidth used the data to model two flowsheet scenarios as follows:

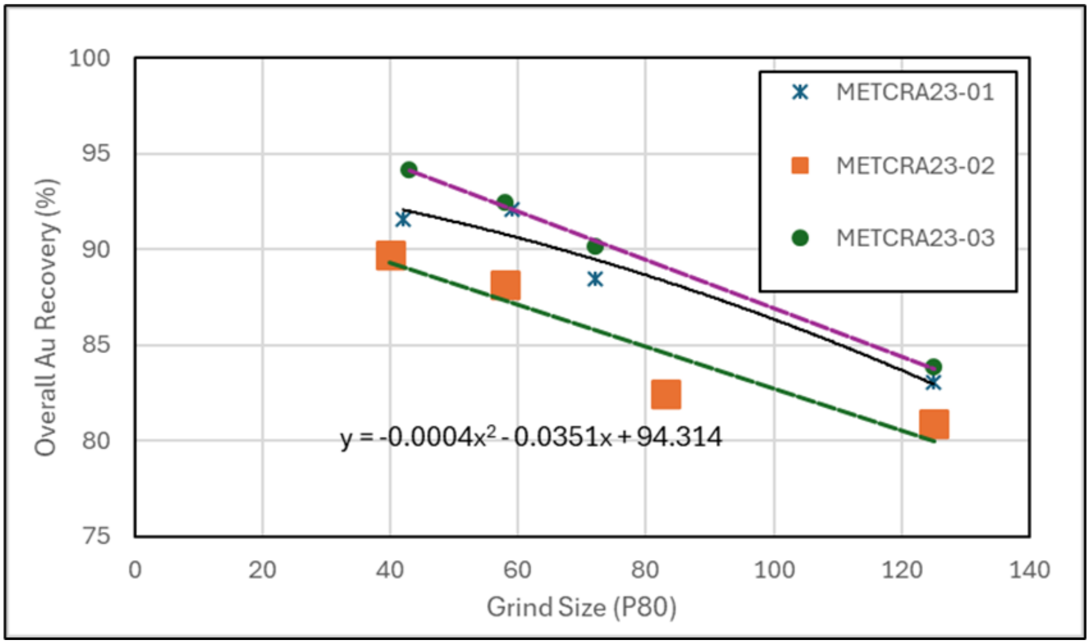
- Two (2) stages of gravity concentration with a primary centrifugal concentrator followed by a shaking table operating at average efficiency according to the FLSmidth benchmark.
- A single stage of gravity concentration using centrifugal concentrator(s) followed by intensive cyanidation with near complete (99.9%) leaching of the gold into solution. This scenario was explored as one of the options in the trade-off study and is shown below under the "To ICU Pregnant Solution" heading.

The modelled results predicted approximately mid 30's gravity recovery for a single stage gravity concentrator, which was marginally lower than the 43% that was adopted in the PEA and carried forward to the trade-off studies performed in the PFS.

13.5.6 GRINDING OPTIMISATION

One of the recommendations from the PEA was to revisit the grind optimisation testwork to validate the optimum grind. The testwork procedure was modified by adding a "de-nuggetting" step as the first step in the sequence. A bulk sample of each of the three PEA composite samples was ground to a coarse grind of approximately 80% passing 130 µm. This bulk sample was then processed through a gravity concentrator to remove all very coarse gold "nuggets". The tailings from this initial step were then homogenised and split into four sub-samples (for each composite). One was kept as is and the other three were then ground to three finer grinds. Rougher flotation tests were then conducted using these sub-samples at the four grinds. It is noted that the gravity gold removed during the de-nuggetting step was added to the flotation recovery for each test when calculating the overall gold recovery (i.e., gravity plus flotation recovery).Accordingly, Figure 13.19 illustrates a plot of the overall gold recovery versus the grind P_{80} size, whereas Figure 13.20 depicts a plot of the rougher tailings gold grade versus the grind P_{80} size.

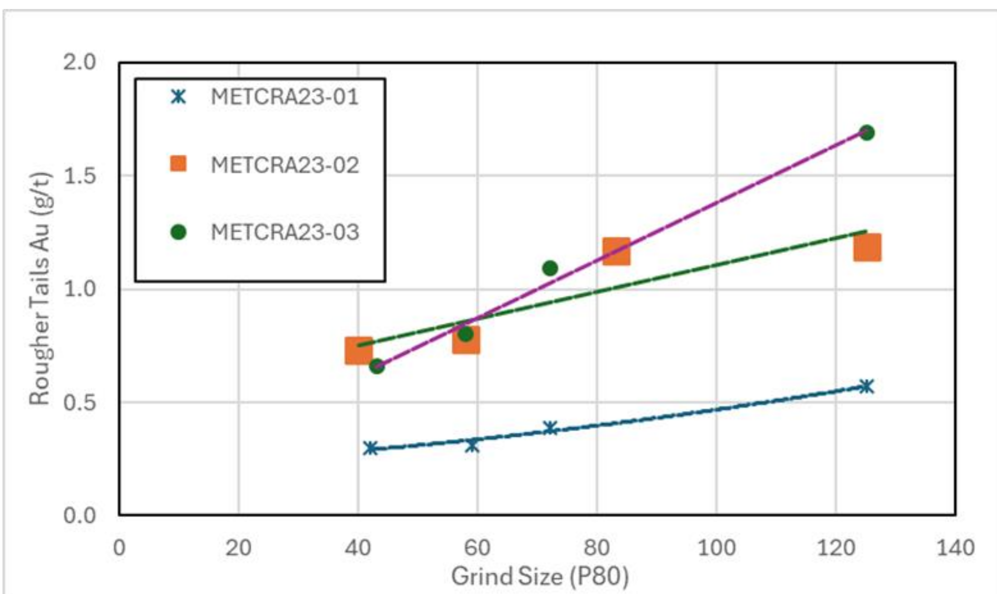
Figure 13.19 – Overall Gold Recovery versus Grind P_{80}



Source: DRA, 2024

Both sets of graphs indicate a strong correlation between overall recovery and grind size over the range of grind sizes tested (approximate P_{80} range from 40 µm to 130 µm).

Figure 13.20 – Rougher Tailings Gold Grade versus Grind P_{80}



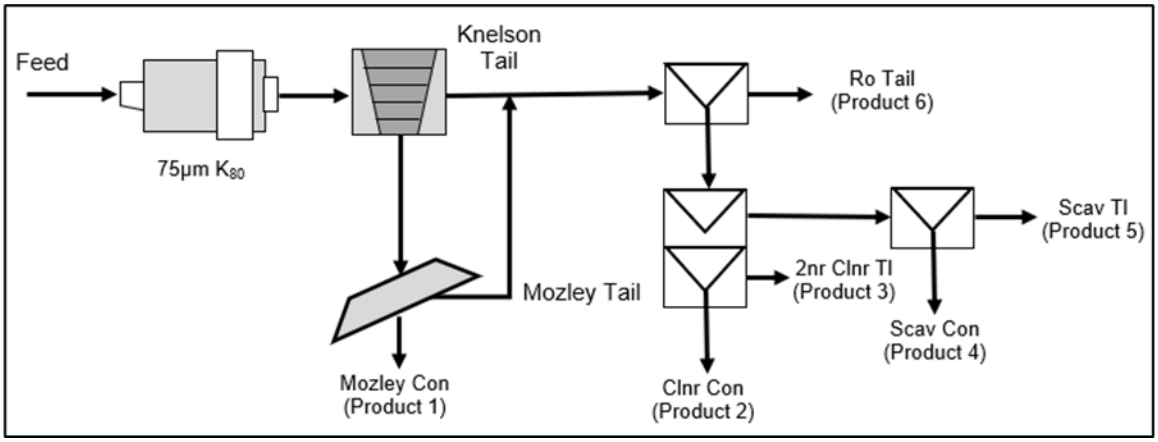
Source: DRA, 2024

DRA performed a high-level profitability analysis to determine the optimum grind size. The analysis confirmed that a grind size of 80% passing 53 µm is the optimum size.

13.5.7 FLOTATION

All variability testwork samples were subjected to a series of tests comprising gravity separation followed by both rougher and cleaner flotation (as shown in Figure 13.21). The purpose of these tests was to observe both the mass pull and gold recovery of the variability samples.

Figure 13.21 – Schematic of Flotation Variability Testwork



Source: BaseMet Labs, 2024

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.

Table 13.15 summarises the mass pulls and gold recoveries recorded after gravity concentration, rougher flotation and cleaner flotation. With this being open circuit tests (i.e. no recycling of the intermediate products as would be the case in an operating plant) some of the gold in these intermediate streams was added to the overall recovery. Based on DRA's experience it was assumed that roughly 80% of the gold contained in these intermediate streams would eventually report to the final product stream if it were recycled.

Applying the weightings per lithology sample to each variability sample results in the following averages being calculated (testwork results without any scale-up to expected full-scale operation):

- Mass pull to gravity concentrate = 0.15%.
- Gold recovery to gravity concentrate = 37.4%.
- Mass pull to the final flotation concentrate = 9.1%.
- Gold distribution to flotation concentrate = 50.0%.
- Overall recovery of gold = 87.4%.

Table 13.15 – Summary of Gravity and Flotation Tests at k_{80} of both 53 and 75 µm

Sample ID	Grind k_{80} (µm)	Gravity Concentrate			Rougher Concentrate			Cleaner Concentrate			Overall Recovery (%)
		wt %	Au (g/t)	Recovery (%)	wt %	Au (g/t)	Recovery (%)	wt %	Au (g/t)	Recovery (%)	
HBP-1	53	0.11	1,240	24.0	16.7	22.7	66.5	8.87	40.2	62.6	88.8
	75	0.14	508	11.6	20.5	23.1	78.4	9.58	47.0	74.6	88.2
HBP-2	53	0.12	381	15.9	10.8	19.0	71.9	4.55	42.7	67.9	86.5
HBP-3	53	0.12	483	13.0	22.8	14.8	73.4	14.85	20.8	67.3	84.0
GMO-1	53	0.07	6,895	64.0	13.8	18.5	32.9	6.29	39.0	31.7	96.5
	75	0.20	3,788	68.2	16.7	18.8	28.9	6.55	44.7	26.9	96.5
SMR-1	53	0.12	497	11.6	12.2	35.3	85.9	6.72	62.3	83.8	96.8
	75	0.24	324	23.7	11.0	20.9	71.3	6.54	34.1	68.9	94.2
SMR-2	53	0.07	1,033	38.7	15.2	6.6	52.8	6.99	12.6	46.6	88.7
SUL-1	53	0.21	332	25.4	28.9	6.1	65.9	20.25	8.0	60.0	88.5
SUL-2	53	0.14	673	28.3	21.3	9.3	60.6	10.93	16.6	55.6	86.6
SUL-3	53	0.11	1,439	36.2	9.7	23.3	49.9	3.64	55.4	44.3	84.0
	75	0.27	328	20.3	7.6	36.5	63.1	3.78	69.5	59.7	81.7
HCU-1	53	0.16	963	45.3	24.9	6.7	48.6	16.34	9.6	45.9	92.6
HCU-2	53	0.20	565	32.3	17.7	11.1	57.4	7.98	21.8	50.6	86.8
HCU-3	53	0.15	2,991	60.8	15.9	16.5	35.2	9.25	26.6	33.0	95.1
	75	0.19	2,026	55.9	19.7	13.7	39.2	11.94	20.3	35.3	93.8
POMT-1	53	0.17	1,101	31.3	21.8	14.2	52.6	13.61	19.9	45.8	80.2
POMT-2	53	0.16	1,101	38.3	18.7	12.5	49.6	10.10	20.3	43.7	85.1
	75	0.26	1,130	53.9	19.8	9.6	35.5	9.53	16.5	29.3	86.6

Sample ID	Grind k_{80} (µm)	Gravity Concentrate			Rougher Concentrate			Cleaner Concentrate			Overall Recovery (%)
		wt %	Au (g/t)	Recovery (%)	wt %	Au (g/t)	Recovery (%)	wt %	Au (g/t)	Recovery (%)	
POMT-3	53	0.10	661	26.4	33.1	4.9	61.8	14.84	9.2	52.2	83.4
POMT-4	53	0.14	586	34.2	12.8	9.8	52.9	5.53	20.0	46.8	84.5
VG-1	53	0.13	1,806	51.3	7.0	25.1	40.1	1.34	109.0	33.1	93.6
	75	0.15	538	23.9	9.2	23.4	62.0	1.91	95.8	53.0	94.3
VG-2	53	0.14	1,135	30.9	8.4	33.3	53.0	1.71	134.7	43.6	89.0

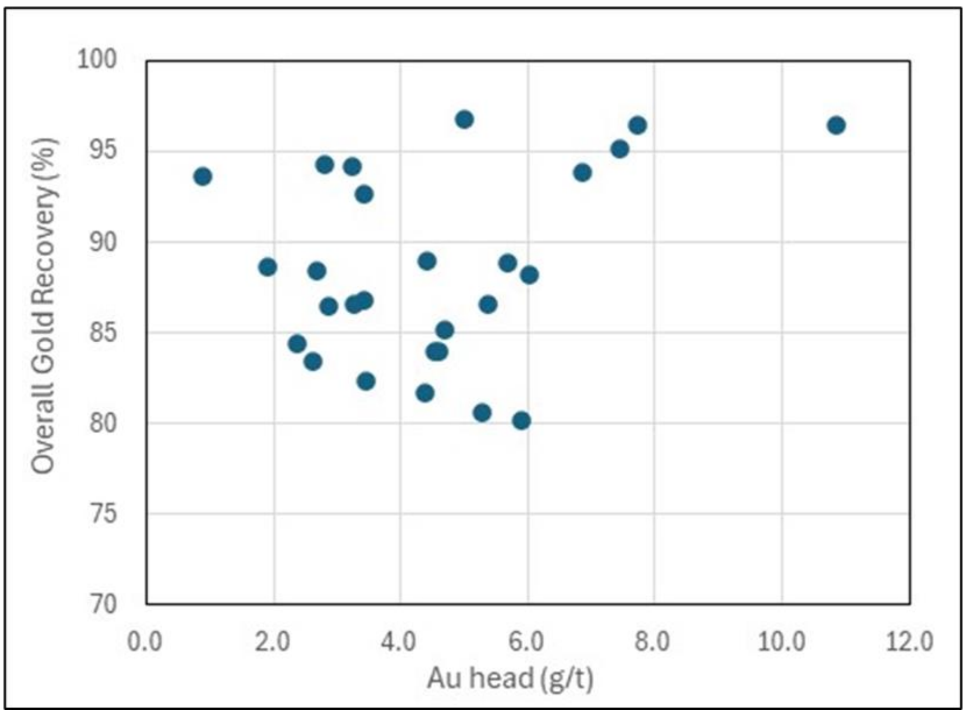
Source: BaseMet Labs, 2024

The gravity gold recovery of 37% is marginally lower than what was adopted in the PEA, but it is also aligned with what FLSmidth predictions based on their modelling of the EGRG tests. Based on this a lower gravity recovery value was adopted for the PDC of the PFS.

The overall gold recovery of 87.4% is also marginally lower than what was used in the PEA. A minor adjustment is suggested to align both sets of test results.

In terms of modelling the gravity and overall recoveries the full datasets were plotted on a variety of graphs to determine if there are any strong correlations that can be used for modelling purposes. Figure 13.22 shows the population of recoveries versus head grade for all the tests conducted on the variability samples. It is evident from this that there is no reliable correlation between grade and recovery. The implication is again that it is not feasible to define a recovery model, as a function of head grade, for this orebody and that the PDC will use the weighted average recovery for design purposes.

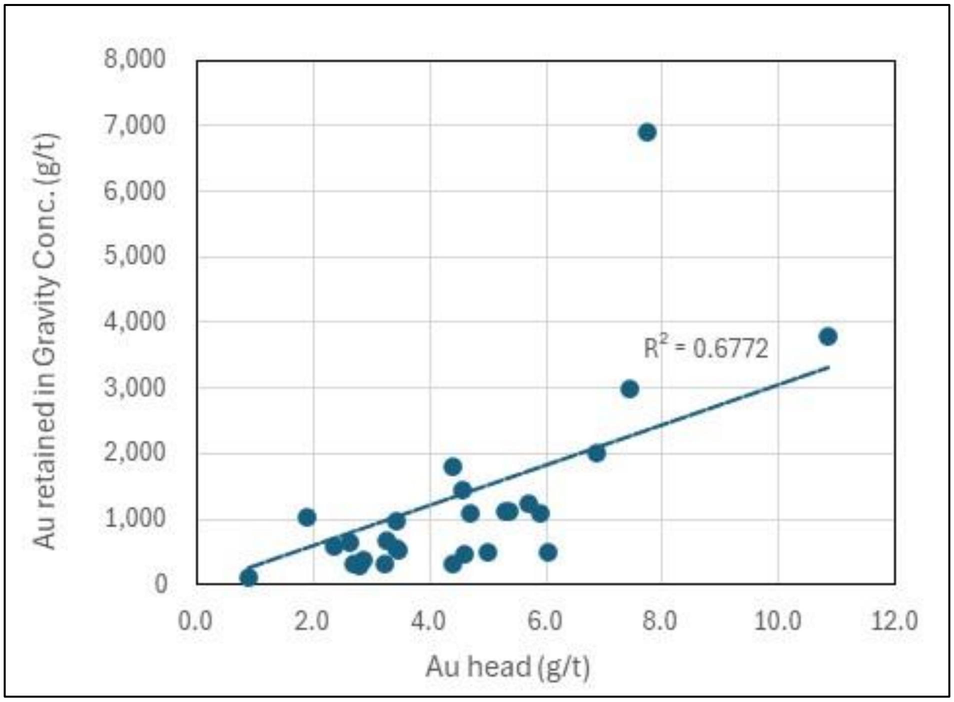
Figure 13.22 – Overall Gold Recovery vs. Head Grade



Source: DRA, 2024

Figure 13.23 shows the gravity concentrate grade for each variability sample against its head grade. The correlation coefficient at 0.68 is reasonable which validates the assumption of a constant upgrade ratio made during the PEA. It should also be noted that the outlier at a concentrate grade of 6,895 g/t is for sample GMO_1 which represents only 0.1% of the ore body. If this outlier is taken out of the population the R^2 value improves to 0.84.

Figure 13.23 – Overall Gold Recovery vs. Head Grade



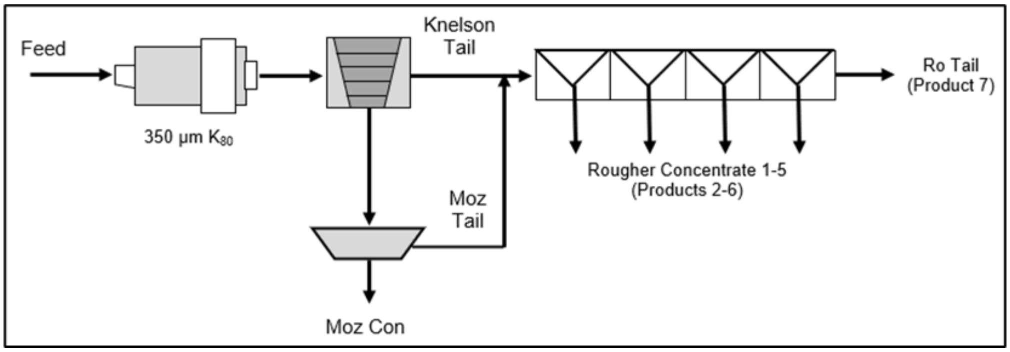
Source: DRA, 2024

13.5.8 Coarse Particle Flotation (Hydrofloat) Testwork

The MetCRA-01 sample from the previous testwork phase was subjected to a series of exploratory Coarse Particle Flotation tests to investigate the viability of including this technology in the flowsheet. If most of the gold can be floated at a coarser grind of at 350 µm it may be possible to reject the tails from this step and only regrind the concentrate to 53 µm. This would provide significant cost savings in terms of grinding costs as the ROM ore would be ground to 350 µm only as opposed to 53 µm.

Figure 13.24 provides the schematic of the rougher test procedure that served as a comparative test and resembled a traditional flotation circuit with 5 stages of roughing. Approximately 20 kg of the MetCRA_01 sample was ground to 80% passing 350 µm before passing it through the Knelson concentrator. The Knelson and Mozley table tails were then split into sub samples. A 1 kg sub-sample was then used for the baseline standard bulk sulphide rougher flotation test.

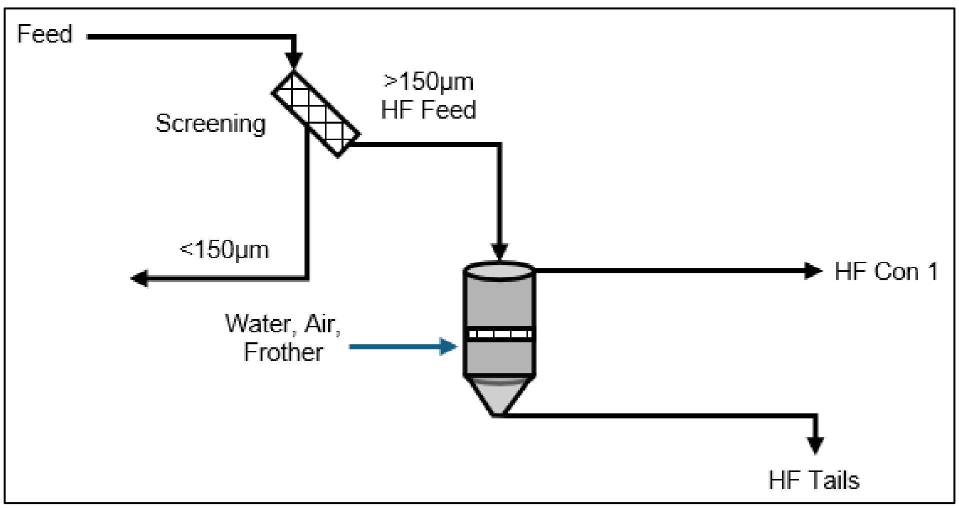
Figure 13.24 – Schematic of Rougher Test



Source: DRA, 2024

This baseline standard flotation test yielded an overall gold recovery of 54.5% of which 14% was into the gravity concentrate product and the remaining 40.5% of the recovered gold reported to the five rougher concentrates. The remaining 19 kg gravity tails sample was then used to perform the hydrofloat test. Figure 13.25 gives the diagram of the hydrofloat test procedure that would mimic the behaviour of gold in a typical hydrofloat circuit. The feed material (gravity tails) was de-slimed over a 150 µm screen. Screen undersize was collected and subsequently also subjected to a standard bulk sulphide float. Screen oversize was processed through the laboratory hydrofloat device.

Figure 13.25 – Schematic of Exploratory Hydrofloat Test



Source: BaseMet Lab, 2024

Table 13.16 summarises the hydrofloat test results showing all final products. It is noted that the gold distribution to gravity tails at 21.9% is higher than the 14% for the baseline test as this number is based on a calculated head grade (the same gold grade was used throughout).

Table 13.16 – Rougher and Preliminary Hydrofloat Test Results

Product	wt %	Au (g/t)	Au Distribution (%)
Gravity Concentrate	0.1	929	21.9
Hydrofloat Concentrate	6.1	5.59	11.4
-150 µm Float Concentrate	10.5	6.48	22.7
-150 µm Float Tails	34.0	0.48	5.5
Hydrofloat Tails	49.3	2.34	38.5

Source: BaseMet Lab, 2024

The hydrofloat test was not successful as the distribution of gold to the hydrofloat tails (38.5%) is unacceptably high. The stage recovery of the hydrofloat stage itself was 23%, which is also unacceptably low. Flotation of the screen undersize sample yielded a stage recovery of 80.5% which is significantly lower than the typical recovery of around 87% which was achieved with ROM samples ground to 53 µm.

It is concluded that Coarse Particle Flotation is not a viable flowsheet option for Čoka Rakita.

13.6 Responsible Mining Solutions (RMS) Testwork 2024

RMS performed a testwork program in 2024 aimed at evaluating the dewatering and rheological properties of the Full Plant Tailings (FPT), as well as conducting Unconfined Compressive Strength (UCS) tests to determine FPT suitability for paste backfill use.

The scope of the test work campaign included several key evaluations. Material characterisation was performed to establish baseline properties for future comparisons. Rheological testing assessed paste flow properties. Static thickening trials focused on determining optimal flocculant type and dosage. Dynamic thickening was conducted at various flux rates to determine solids loading for sizing purposes. Filtration tests examined tailings filtration properties at different slurry densities, while desiccation testing assessed tailings behaviour under seasonal conditions. Also, UCS tests evaluated backfill strength using varying binder contents over different curing times.

13.6.1 Material Characterisation

The material characterisation tests focused on establishing the basic physical and chemical properties of the FPT sample. The specific gravity of 3.2 t/m³ indicates that the tailings are dense, which is a desirable characteristic for paste backfill material, ensuring good mechanical properties. The pH, which increased slightly from 6.5 to 7.5 during testing, suggests that the tailings are chemically stable and are unlikely to pose significant acid generation risks. The particle size distribution yielded about 29% passing the 20 µm size. This indicates that the FPT has a range of particle sizes, which is important for compaction and strength development in paste backfill. The mineralogical analysis shows a high concentration of grossular (48%), a type of garnet, along with smaller amounts of calcite, quartz, and other minerals. This composition suggests the material is fairly hard and stable, which will contribute to the mechanical strength of the backfill.

13.6.2 RHEOLOGY

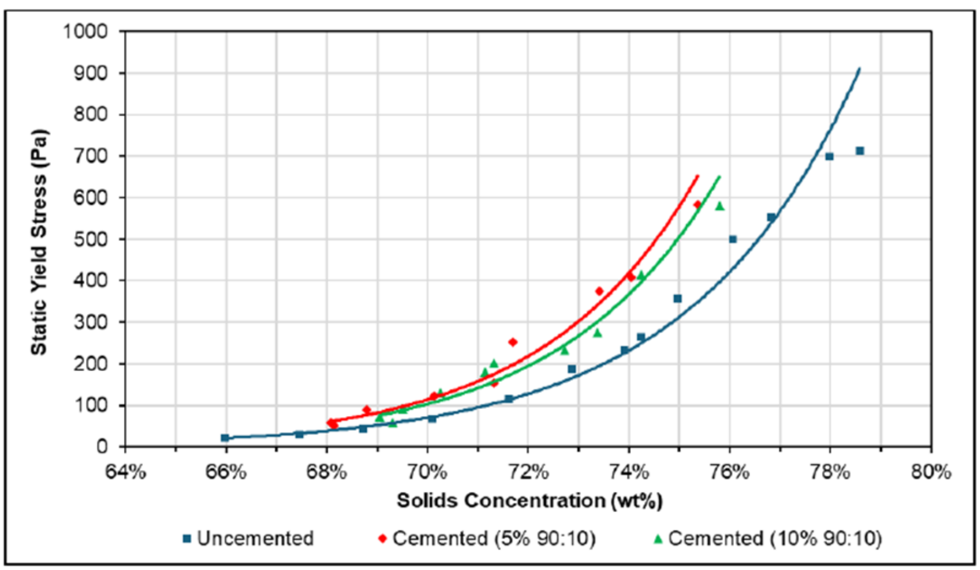
The rheological tests evaluated the flow properties of the FPT, which are critical for pumping and handling during backfill operations. The results showed that the uncemented FPT has a yield stress of 405 Pa at a 175 mm slump and 234 Pa at a 250 mm slump, indicating that it requires significant force to flow in its uncemented state. When cement is added, the yield stress decreases, making the paste easier to handle. For example, a 5% cement addition reduced the yield stress to 345 Pa at 175 mm slump. The tests also revealed that the FPT has low to medium sensitivity to water content, meaning that small changes in water content do not drastically affect the paste flow properties. This suggests that the FPT can be pumped effectively with the right water content and binder ratio, making it suitable for paste backfill applications. Table 13.17 and Figure 13.26 show the summary of the static yield stress testing.

Table 13.17 – Static Yield Stress Summary

Slump (mm)	**Uncemented FPT**		**5% Cemented FPT**		**10% Cemented FPT**	
	Solids Conc. (wt%)	**Yield Stress** (Pa)	**Solids Conc.** (wt%)	**Yield Stress** (Pa)	**Solids Conc.** (wt%)	**Yield Stress** (Pa)
175	75.3	405	73.0	345	73.5	295
250	73.9	234	71.3	154	71.1	181
Difference	1.4	171	1.7	191	2.4	114

Source: RMS, 2024

Figure 13.26 – Static Yield Stress Curves vs. Solids Concentration

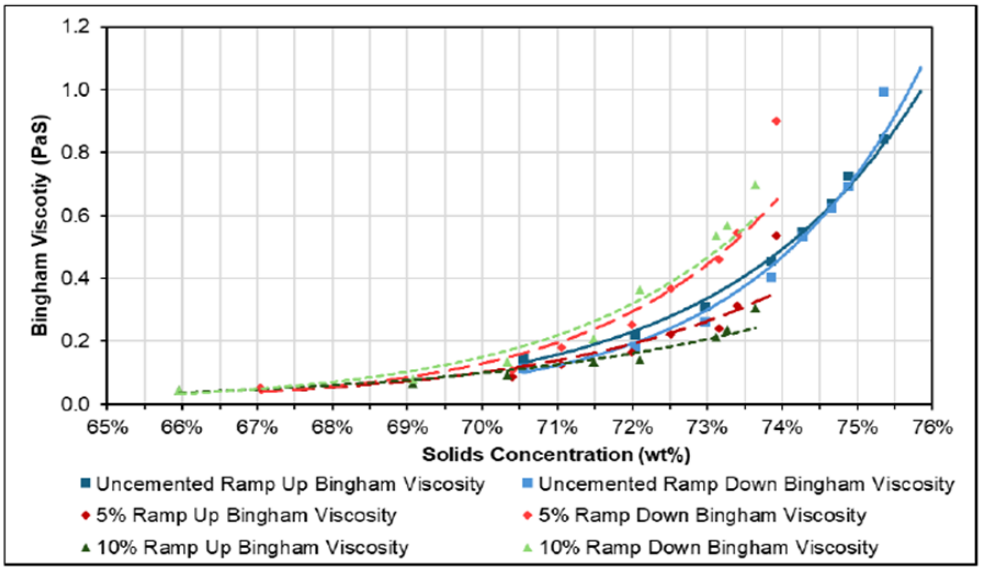


Source: RMS, 2024

13.6.3 Viscosity and Dynamic Yield Stress

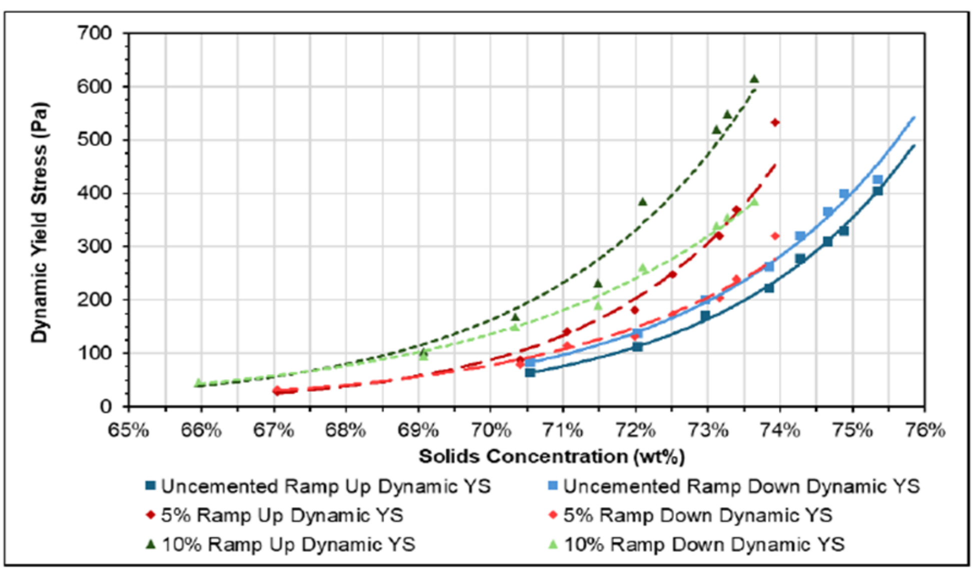
Viscosity measures the internal friction within a fluid, providing insight into its resistance to flow, and is useful for estimating friction losses and material transport requirements. Paste is generally considered a non-Newtonian fluid with plastic properties, so a Bingham regression is applied to the data to determine both the viscosity and dynamic yield stress. The results for viscosity and dynamic yield stress are shown graphically in Figure 13.27 and Figure 13.28, respectively.

Figure 13.27 – Bingham Viscosity vs. Solids Concentration



Source: RMS, 2024

Figure 13.28 – Dynamic Yield Stress vs. Solids Concentration



Source: RMS, 2024

13.6.4 THICKENING

Thickening tests were conducted to determine the sedimentation characteristics of the sample of FPT. AN 923 VHM was identified as the most effective flocculant during screening tests. Static thickening tests revealed that a feed solids concentration of 15% produced optimal results, with an underflow density of up to 65%w/w solids and clear overflow. The dynamic thickening tests, performed at various flux rates, confirmed that underflow densities of 65% could be achieved with a flux rate as low as 0.5 t/m²/h. This result aligns with the PEA test result where a density of 60 % w/w was achieved at the higher flux rate of 1.0 t/m²/h.

13.6.5 FILTRATION

Scoping filtration tests were conducted to evaluate the dewatering performance of FPT comparing vacuum disc and belt filtration methods. At higher feed densities (65 wt% solids or more), the FPT sample exhibited excellent filtration properties, producing filter cakes with an acceptably low moisture content of 17 to 19% and high filtration rates. The Sefar 05-678-122-NFG filter cloth was selected for its ability to produce thicker and drier cakes, making it ideal for use in industrial filtration equipment. Overall, the filtration tests demonstrated that FPT can be efficiently dewatered using vacuum filtration systems, reducing the water content to acceptable levels for backfill use. Table 13.18 shows a summary of this test.

Table 13.18 – Membrane Screening Results Summary

Cloth ID	Cloth Type	Feed Solids (wt%)	Vacuum (bar)	Cycle Time (s)	Cake Thickness (mm)	Cake Moisture (%)	Corrected Filtration Rate (kg/m²/h)	Filtrate Quality
A	05-405-245-NFG	65	0.7	90	9.1	19	503	Cloudy
B	05-610-15-NFG				8.3	18	460	Clear
C	05-610-460-NFG				9.5	19	533	Opaque
D	05-678-122-NFG				9.7	17	504	Clear
E	05-814-122-NFG				9	18	471	Slightly Cloudy

Source: RMS, 2024

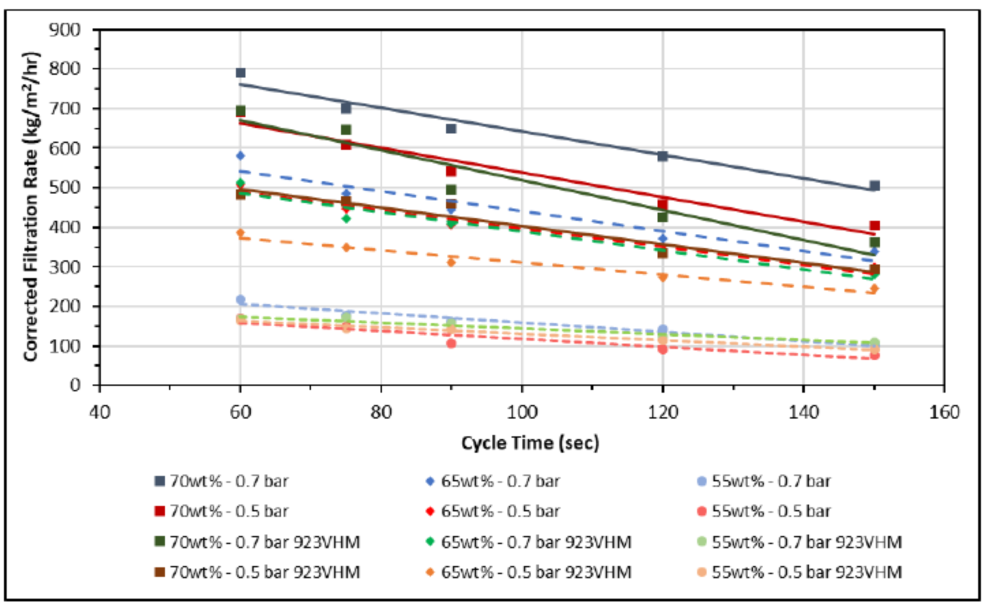
13.6.5.1 Vacuum Disc Filtration

Vacuum disc filtration testing was conducted at three different feed densities, within the range produced during the preceding dynamic thickening tests. The tests were carried out at two (2) vacuum pressures using a filter leaf dip test apparatus, both with and without the addition of flocculant. The procedure simulated a vacuum disc filtration process, where the disc sector is submerged in slurry for specific time intervals, followed by drying cycles. By testing at different filter feed densities, a range of operating parameters was established to predict throughput under various conditions—nominal, optimal, and less efficient. The tests also demonstrated how vacuum pressure influences the overall performance of the filtration process.

These results can be used for equipment sizing, including thickeners, feed tanks, filter sizes, and determining vacuum pump and auxiliary equipment needs. The data provided a range of values for efficiency, power consumption, and desired cake production rates. A vacuum pressure of 0.7 bar, corrected for site elevation with an 80% efficiency factor, was deemed optimal, while a lower pressure of 0.5 bar was tested for conditions where the vacuum pump might run at reduced efficiency. Key parameters included filter feed densities of 70 wt%, 65 wt%, and 55 wt% solids, a laboratory temperature of 20°C ± 3, and a pH of 7.5. Results are summarised in Figures 13.29 and 13.30, with a 5 mm cake thickness identified as ideal for disc filters.

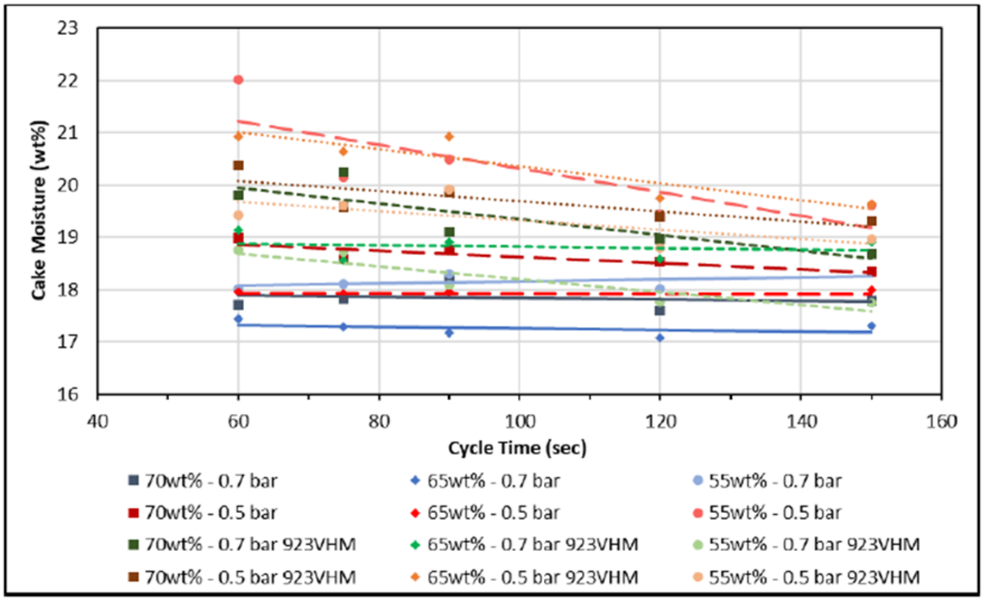
The vacuum disc filtration tests show that performance is adequate to good when the filter feed density is at least 65 wt%. However, at 55 wt%, the filtration results were poor, with the least dry filter cakes, suggesting challenges at lower densities. The use of a filtration aid (flocculant) did not enhance the filtration rate or reduce cake moisture. To optimise filtration performance, the filter feed density should be maintained at a minimum of 65 wt% solids. The Ada Tepe tailings thickener which will be re-used at Čoka Rakita has a relatively high depth to diameter ratio which implies that it will allow for a high underflow density due to compaction in the thickener.

Figure 13.29 – Corrected Filtration Rate vs. Cycle Time



Source: RMS, 2024

Figure 13.30 – Cake Moisture vs. Cycle Time



Source: RMS, 2024

13.6.5.2 Vacuum Horizontal Belt Filtration

Horizontal belt filtration testing was performed at three different densities. The testing aimed to determine whether flocculant addition would improve filter performance, with tests conducted both with and without flocculant. A Buchner-type filter setup was used to simulate full-scale belt filtration, and subsamples of 300 g, 200 g, and 100 g were selected to analyse varying cake thicknesses. Two drying times (90 s and 30 s) were tested, with vacuum pressures of 0.7 bar and 0.5 bar, adjusted for site elevation and an 80% efficiency factor.

The filter feed density increments were 70 wt%, 65 wt%, and 55 wt%. The filtration cloth used was a Sefar 05-678-122-NFG, and a flocculant (AN 923 VHM) was added at 40 g/t. Filtration capacities were derated by 70% to account for realistic continuous production conditions on a belt filter.

The belt filtration results demonstrate a clear relationship between sample size, cycle time, and filtration rate, as expected. Longer cycle times lead to lower filtration rates but result in drier filter cakes. The addition of a flocculant enhances the filtration rate with minimal effect on cake moisture. To optimise belt filtration performance, it is important to maintain a consistent filter feed density of at least 65 wt% solids. This strategy would help reduce the footprint of the belt filtration plant.

13.6.6 DESICCATION

The desiccation tests simulated how quickly the FPT would dry under various environmental conditions, representing different seasons and climates. The results showed that the FPT desiccates much faster in warm, moderate-humidity conditions (40°C and 65% humidity), which are typical of summer and fall. In these conditions, the tailings were ready for the next layer of deposition within two (2) weeks. In contrast, colder and more humid environments, such as those found in winter or spring, slowed the desiccation process, with tailings requiring up to three (3) weeks or more to dry sufficiently. The desiccation tests also revealed that there was no significant segregation of the material during drying, and compaction occurred naturally as the water evaporated. These findings suggest that FPT can be effectively managed in dry seasons, but may require longer drying times in wetter conditions, impacting the timing of tailings deposition. Table 13.19 summarises the desiccation environments and parameters.

Table 13.19 – Desiccation Environments and Parameters

Environment	Temperature (+/-5, °C)	Relative Humidity (+/-5, %)
Greenhouse 1 (GH1)	40	90
Greenhouse 2 (GH2)	20	90
Greenhouse 3 (GH3)	20	65
Greenhouse 4 (GH4)	40	65
Fridge (FR)	4	Variable
Freeze/Thaw (FT)	-20	Variable

Source: RMS, 2024

RMS calculated a bearing compaction of approximately 100 kPa (10 t/m²), assuming DPM is using a D9 dozer and applying a safety factor of 2. This value was used to assess whether the lift was compact enough for heavy equipment to operate on and if the sample was ready for the next lift. The FPT was prepared with a paste density of around 250 mm slump (~74 wt%) and subsampled into 200 x 200 mm round containers with 63 mm thick lifts. Each sample was placed in a controlled environment to settle and desiccate naturally, with temperature and humidity monitored. Periodically, the lifts were weighed, photographed, and tested for compaction using a pocket penetrometer. Once a lift reached the target bearing compaction of 100 kPa, it was considered complete, and the next 63 mm lift was added.

Cold jointing was observed between the layers when analysing the desiccated material after testing was completed. It is important to note that the layers were not altered or compressed after deposition. No material segregation was detected either during the deposition of the layers or upon visual inspection over time.

13.6.7 Unconfined Compressive Strength (UCS)

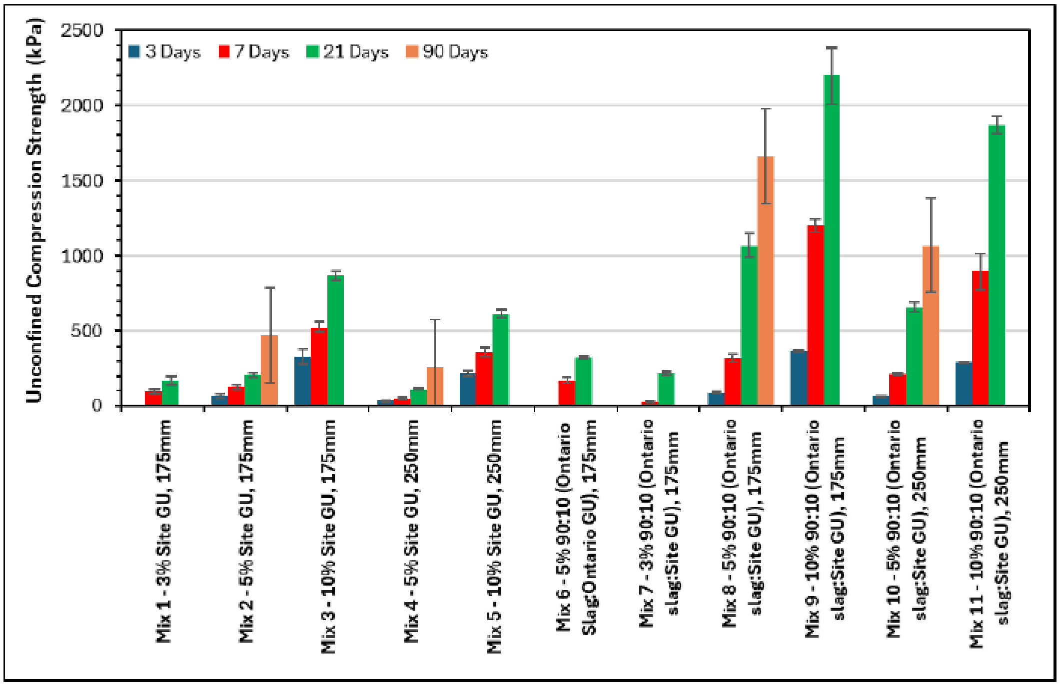
UCS testing measured the mechanical strength of the backfill material over time at various binder additions and curing periods. The results clearly demonstrated that the blend of 90% Ground Granulated Blast Furnace Slag (GGIBFS) and 10% CEM II cement provided superior strength compared to mixes with cement alone. For instance, after 21 days of curing, a 10% binder addition resulted in a strength of 2.197 MPa, while a 5% addition achieved 1.662 MPa. These strength levels ensures that the material will provide long-term stability. Additionally, no strength loss was observed after 90 days of curing, indicating that the backfill will maintain its integrity over time. These results which are shown in Table 13.20 and Figure 13.31 suggest that the GGIBFS II blend is an optimal solution for producing high-strength paste backfill for underground mine operations.

Table 13.20 – UCS Test Results Summary

Mix #	Binder	Slump (mm)	Binder Conc (wt%)	Curing Period (days)			
				3	7	21	90
1	CEM II	175	3	-	96	169	-
2	CEM II	175	5	68	123	206	468
3	CEM II	175	10	332	525	867	-
4	CEM II	250	5	34	53	113	259
5	CEM II	250	10	217	359	612	-
6	90:10 (Ont GGIBFS: Ont GU)	175	5	-	172	322	-
7	90:10 (Ont GGIBFS: CEM II)	175	3	-	27	219	-
8	90:10 (Ont GGIBFS: CEM II)	175	5	91	318	1,069	1,662
9	90:10 (Ont GGIBFS: CEM II)	175	10	368	1,199	2,197	-
10	90:10 (Ont GGIBFS: CEM II)	175	5	67	212	656	1,070
11	90:10 (Ont GGIBFS: CEM II)	250	10	290	894	1,871	-

Source: RMS, 2024

Figure 13.31 – UCS Results Summary



Source: RMS, 2024

13.7 Process Design Criteria Derived from Test Results

- Axb = 61.5 (average from all test programs) and 44.6 (design value of 15th percentile of variability tests).
- Ai = 0.114 (average from variability testing and PEA tests for Opex).
- Bond Ball Mill Work Index = 13.7 kWh/t (average) and 15.22 kWh/t (design).
- Optimum primary grind for gravity plus rougher flotation route is 53 µm as confirmed by the variability testing performed during phase 3 (2024).
- The following LCT (PEA) conditions were adopted as design criteria for the flotation circuits:
  - 5 min conditioning with 250 g/t $CuSO_4$.
  - 125 g/t PAX and A3477 collector addition to roughers.
  - Total rougher float time = 30 min.
  - Cleaner 1 duration = 15 min.
  - Cleaner scavenger = 6 min.
  - Cleaner 2 duration = 10 min.
  - 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/m² can be used with underflow densities of 60% w/w solids based on the Phase 2 test results. The testwork performed by RMS during 2024 showed that it would be beneficial for the downstream filtration process if the underflow density can be increased to 65% w/w solids, but the solids loading would need to be decreased to around 0.5 tph/m². It should be noted that the Ada Tepe tailings thickener is oversized at the design rate of 1 tph/m² so an elevated density should be achievable.
- Flocculant addition rates of 50 g/t for the tailings thickener and 20 g/t for the concentrate thickener.

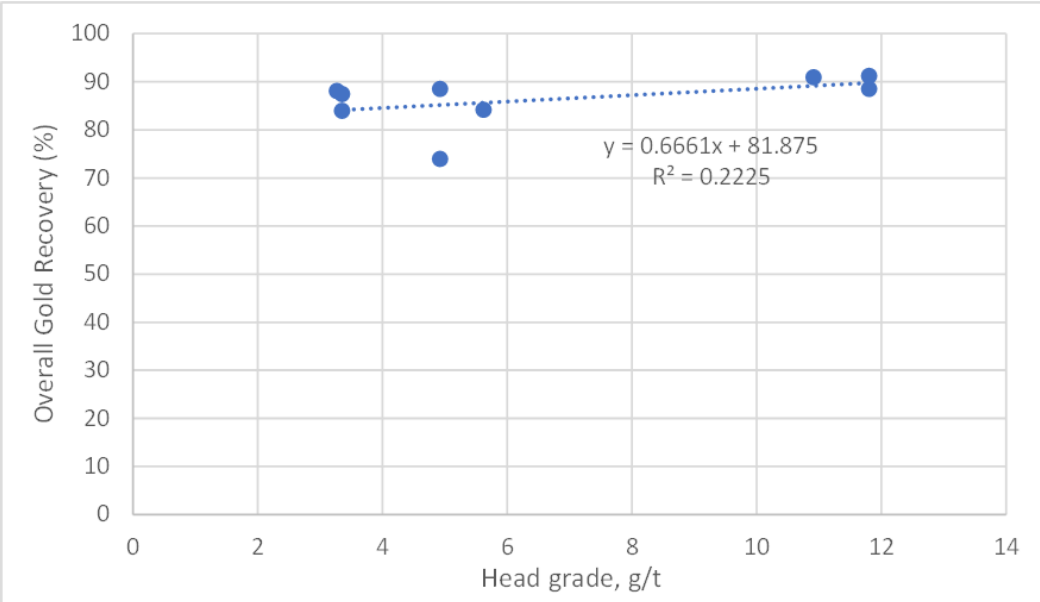
13.8 Recovery Model

With respect to gravity concentration, two (2) observations made from the PEA test results were 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 different sets of tests. For the process design criteria an upgrade ratio was selected that yields gravity concentrate grades that are more aligned with typical industry values. These grades are typically higher than what was achieved during the batch testing of the samples.
- During the PEA, it was observed that there was a good correlation between gravity recovery and head grade. The empirical correlation derived from the PEA results were: Gravity Recovery = 41 x [Head Au]^0.14 with [Head Au] in g/t and Recovery as %. This equation was retained for the PFS but the constant term of 41 was adjusted to 35 to re-align it with the lower gravity recoveries obtained during variability testing. The updated equation, as used for the PFS, is thus : Gravity Recovery = 35 x [Head Grade]^0.14.
- With both the upgrade ratio and gravity recovery defined the concentrate mass pull can be calculated at different head grades.

With respect to overall gold recovery, there is no clear correlation that could be established. Figure 13.32 shows the results for all tests performed by BaseMet Labs during the PEA 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%; this was again confirmed during the PFS phase of testing. Based on the variability results a constant overall recovery of 87% was adopted for PFS design purposes.

Figure 13.32 – Overall Gold Recovery versus Head Grade



Source: DRA, 2024

13.9 Deleterious Elements

Arsenic was present in the final product (concentrate) samples. For the three (3) samples tested by BaseMet Labs in 2023 during the PEA, 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. However, five (5) of the fifteen (15) variability samples tested by BaseMet Labs during 2024 for the PFS yielded arsenic grades in the final flotation concentrate that exceeded the threshold. All four (4) of the pyrrhotite-magnetite samples and one (1) of the high sulphide samples (SUL-2) produced high arsenic grades ranging from 2,204 g/t to 3,466 g/t. These represent less than 20% of the mineral resource volume so it should be possible to blend these with other concentrates to ensure shipped concentrate remains below the threshold. Provision has been made in the PFS design for concentrate storage to allow blending of the final product.

All other deleterious elements were insignificant.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The MRE update for Čoka Rakita was estimated in September 2024 using drilling data collected since 2009 but dominated by data collected in 2021, 2023, and 2024 when the mineralised skarn, that is the focus of this Report, was identified as a target with gold potential. It is an update to the Maiden MRE, reported in November 2023 (November 2023 Maiden MRE) and disclosed in a Technical Report prepared pursuant to the Canadian Securities Administrator's National Instrument 43-101 – Standards of Disclosure for Mineral Projects and filed on SEDAR+ on January 24, 2024.

The MRE update was estimated by QP (Maria O'Connor, MAIG) with support from other ERM 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 was 30 August 2024, which is also the effective date of the MRE. 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:

- CSA_Alteration.csv
- Assay_BM_30082024.csv – for gold, SFA used in preference to FA where available
- CSA_Assay_ME.csv – for gold, SFA used in preference to FA where available
- CSA_BulkDens.csv
- CSA_Collar.csv
- CSA_Geotech.csv
- CSA_Lithology.csv
- CSA_MagSusc.csv
- CSA_ScreenFireAssay.csv
- CSA_Structure.csv
- CSA_Sulphides.csv
- CSA_Survey.csv
- CSA_Vein.csv.

Files were loaded into Datamine and subjected to a series of validation checks.

Table 14.1 – Summary of Collar Data

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	64	40,390
	2024	72	34,073
Diamond – Subtotal		***177***	***96,082***
Diamond Tail	2023	30	16,520
	2024	16	5,934
Diamond Tail – Subtotal		***46***	***22,454***
RC	2008	4	474
	2023	44	7485
RC – Subtotal		***48***	***7,959***
Total		**271**	**126,495**

Notes:
Trenches and channels, depths = 0 m status=in progress drill holes excluded

Queries identified during the load-up and validation process for relevant data were discussed with DPM. Outcomes for the main data files – collar, survey, assay_bm, lithology, density - are summarised as follows:

- Five (5) drillhole had zero depth recorded in the database – RIDD007B, RIDT012, RIDT014, RIDT039, RADD043A. These were terminated due to technical drilling reasons and were excluded from the estimate.
- Four (4) drillholes were in progress – RADDHG008, RIDD089, RIDD090, RIDD091.
- For "missing" assays, there were three (3) categories:
  - Eleven (11) holes had no samples/assays recorded for any part of the drill hole– RADD043A, RIDD007B, RIDD021A, RIDD068A, RIDT012, RIDT014, RIDT030C, RIDT039, RIRC013, RIRC014, RIRC031
  - Samples in the assay file with no assay results – these were holes that had been sampled but were awaiting assay results for recently completed holes
  - or where there were voids (OID/OIS - 3,009 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.
- Four (4) diamond drillholes were missing geotechnical logging – one of which was recently drilled and the others had shallow depths < 60 m.
- All completed drillholes had logged geology and alteration except for 22. Of these, 2 are RC and 16 were drilled in 2024, meaning they are likely to be awaiting data.
- There are 125 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 density can be reliably estimated based on recorded measurements.

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.

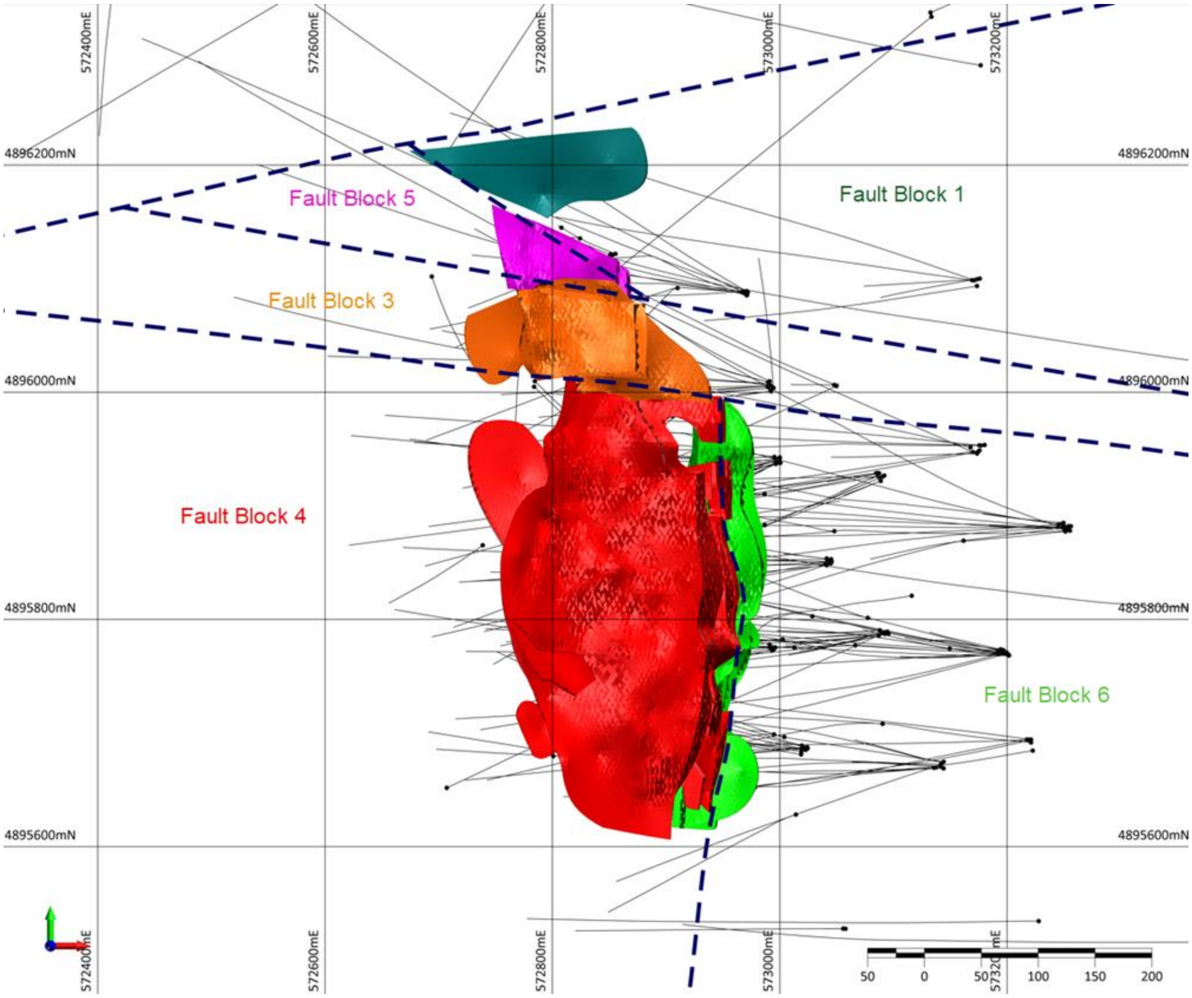
14.2.1 Data Excluded

Trenches and channels were excluded both because of limited quality control but more significantly because they are surface samples and are therefore not relevant to the deep skarn mineralisation modelled for this MRE. Aside from holes that had to be terminated, and holes that were in progress, 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: ERM, 2024

The Leapfrog Geo project containing the lithology, structure and oxidation model for the area was provided by DPM. The QP (Maria O'Connor, MAIG) reviewed and validated the models and made small adjustments to them to 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:

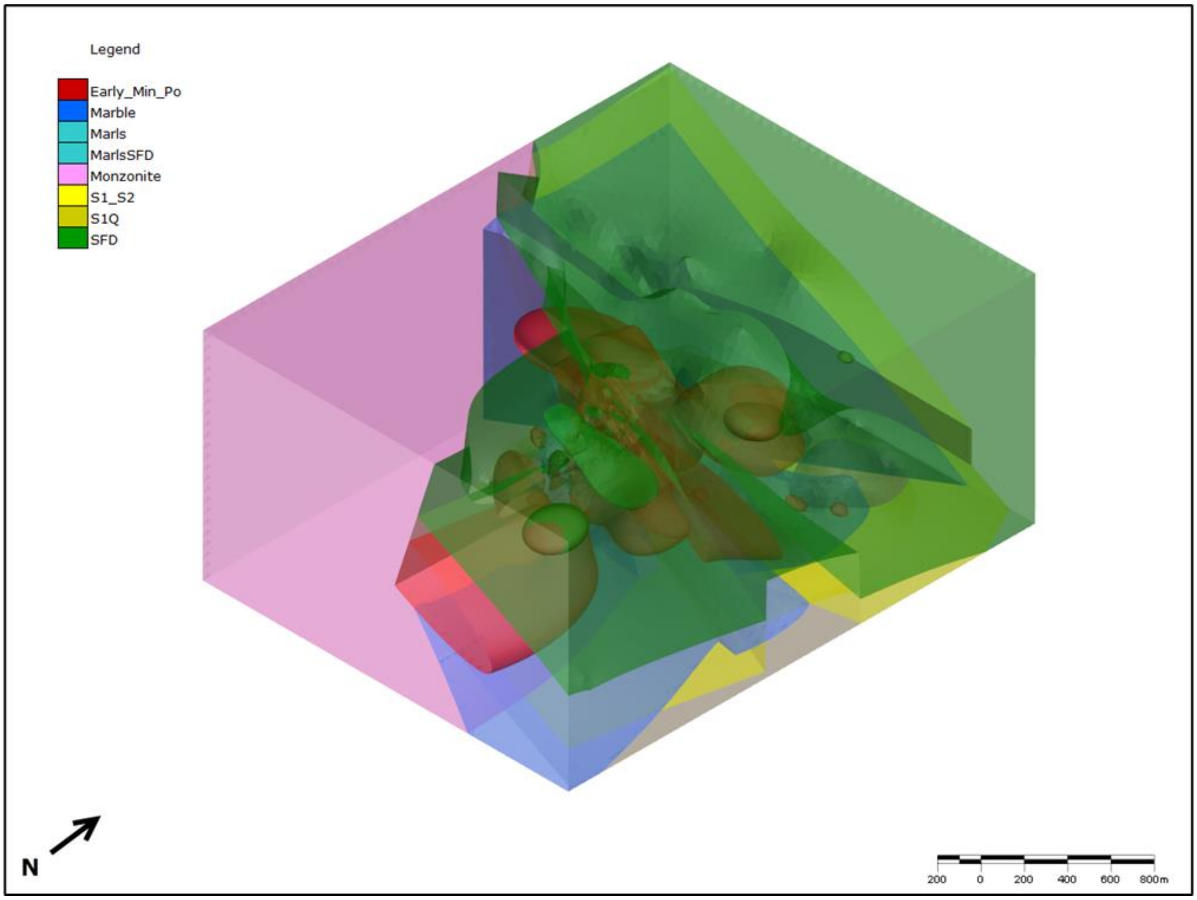
- The lithology model was exported individually so that the mineralisation wireframes could be reviewed by lithology, which was not possible to do when mineralisation was integrated into the lithology model.
- Removal of the topography used to constrain the upper limit of the model to remove any misalignment 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:

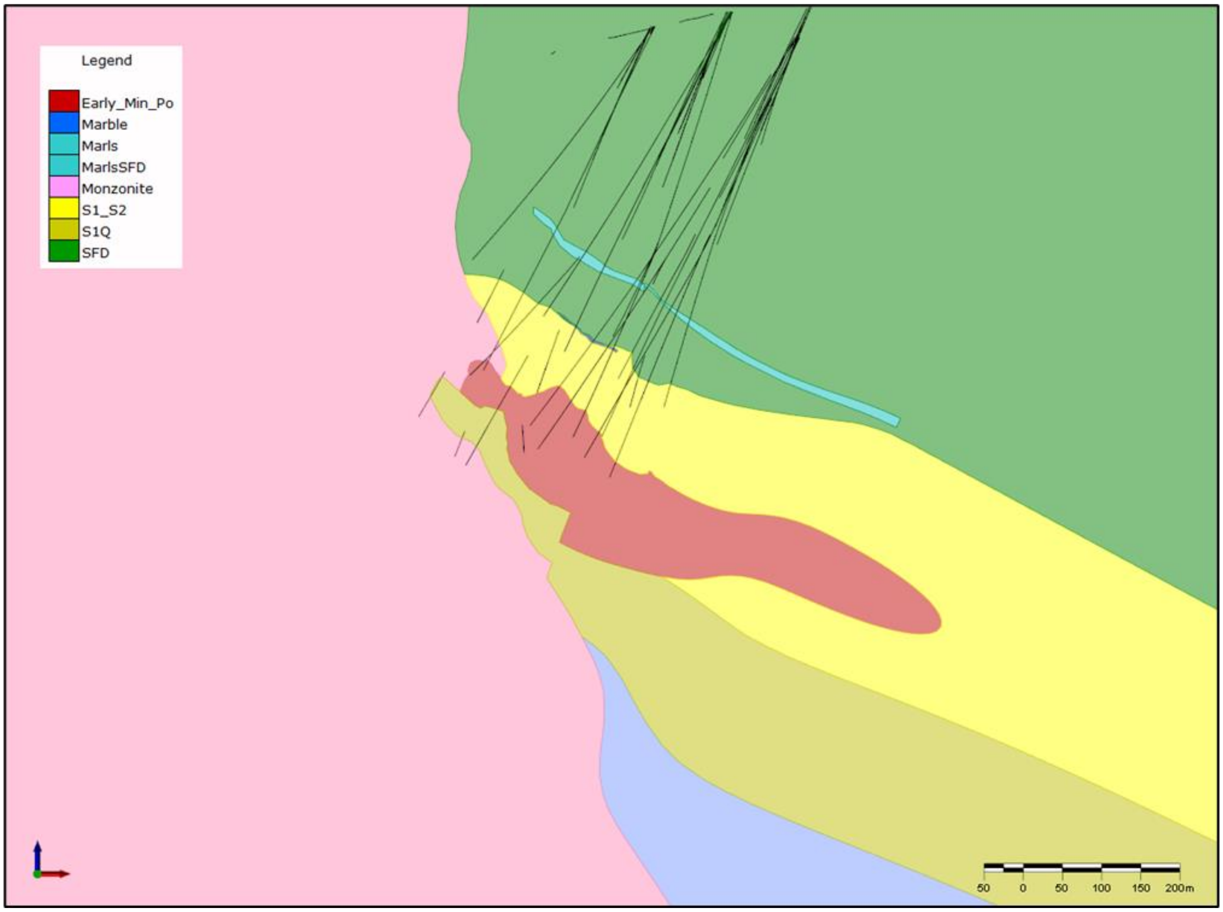
- Early mineralised porphyry (Early_Min_PO) – modelled as an intrusive body with a moderate dipping trend towards the east-northeast, trends varied slightly between faulted blocks, using an elongate spheroidal ellipse.
- Monzonite – modelled as an intrusive body using a spherical model.
- Marls – modelled 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: ERM, 2024

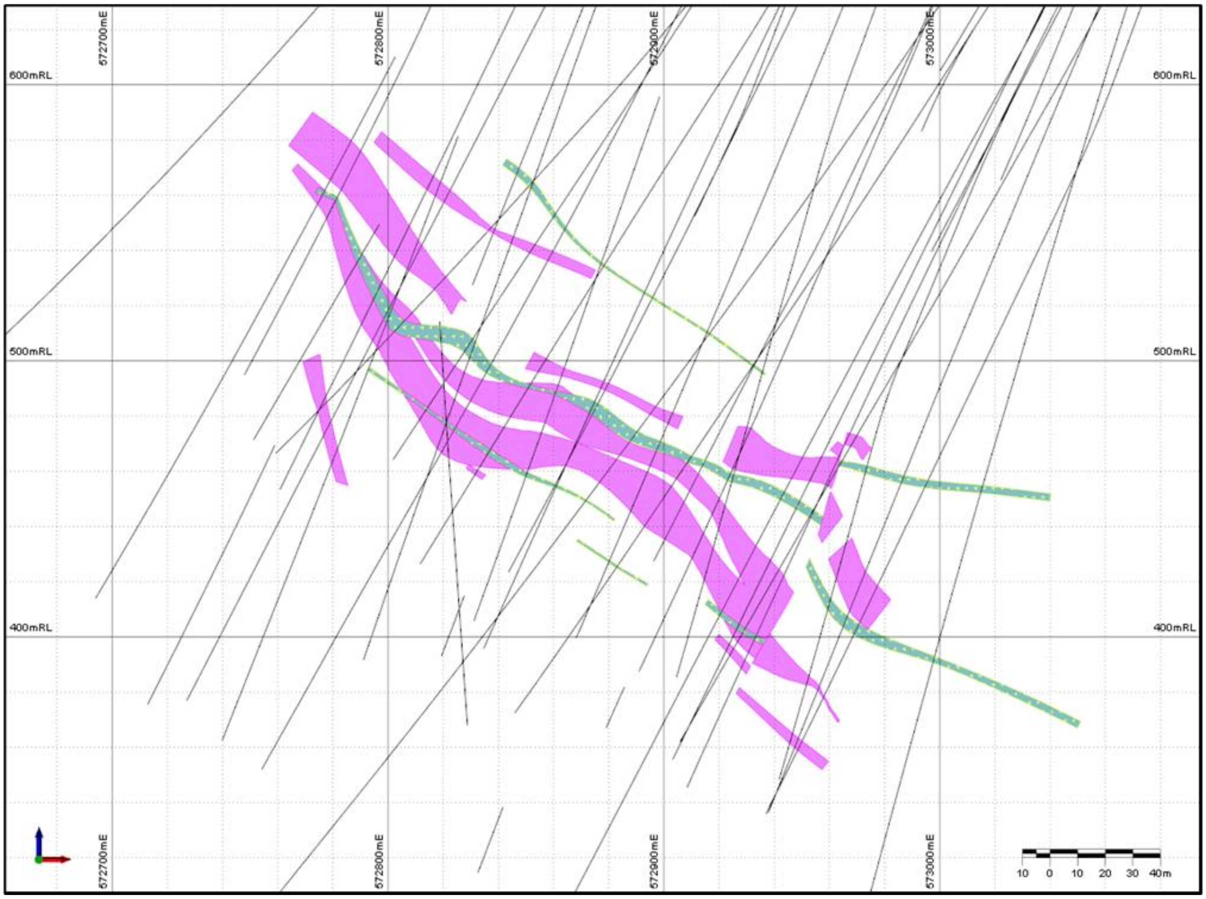
Figure 14.3 – Cross Section Showing Lithology Units and Drillholes, View Looking North, 4895780mN



Source: ERM, 2024

Late-stage crosscutting intrusive sills were modelled using the vein modelling tool. A total of six (6) 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: ERM, 2024

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, 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. Infill drilling undertaken after the PEA in 2024 supported the model, with zones of waste and mineralisation being encountered where the model had predicted.

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 the QP and in close collaboration with DPM geologists. Three broad zones of higher mineralised intensity were described – one at the basal skarn contact, one just below the main PXP layer and one near the S1_S2 contact with the overlying marl units.

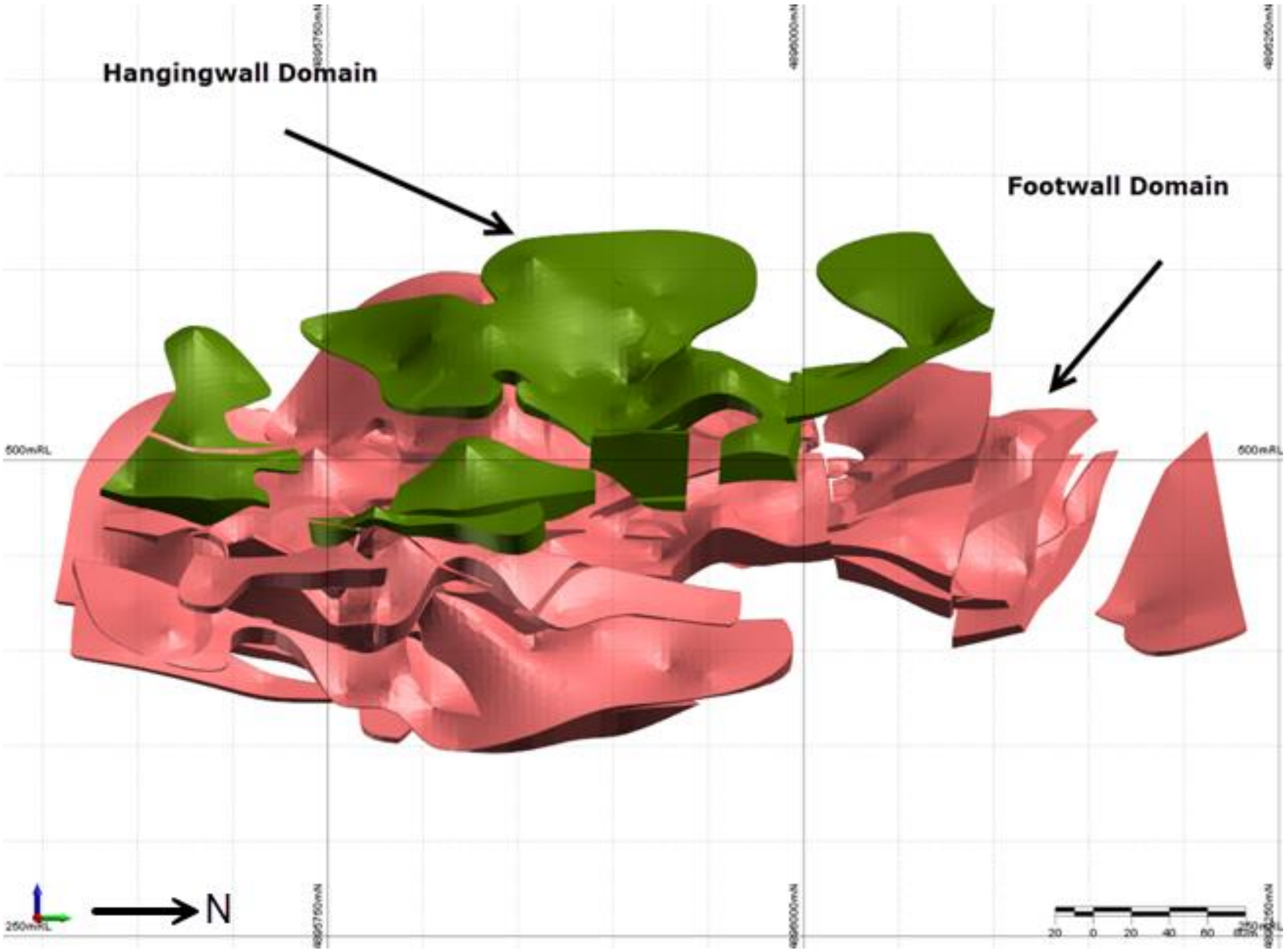
Three (3) large coherent zones were modelled using the vein modelling tool as follows:

- 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 just below the initial PXP later stage intrusive.
- Hangingwall 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.

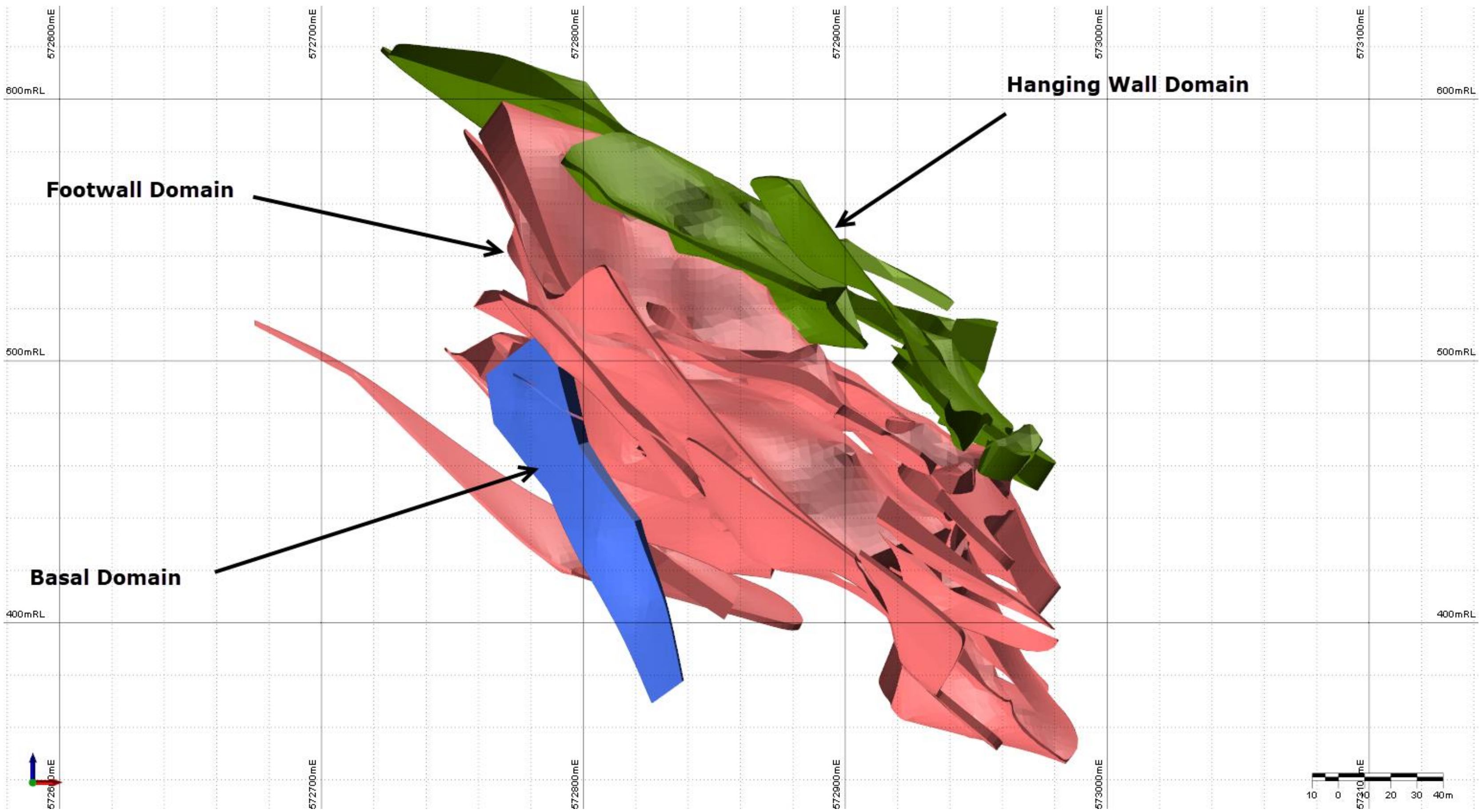
The mineralisation model is shown in Figure 14.5 to Figure 14.7.

Figure 14.5 – Mineralised Domains, View Looking West



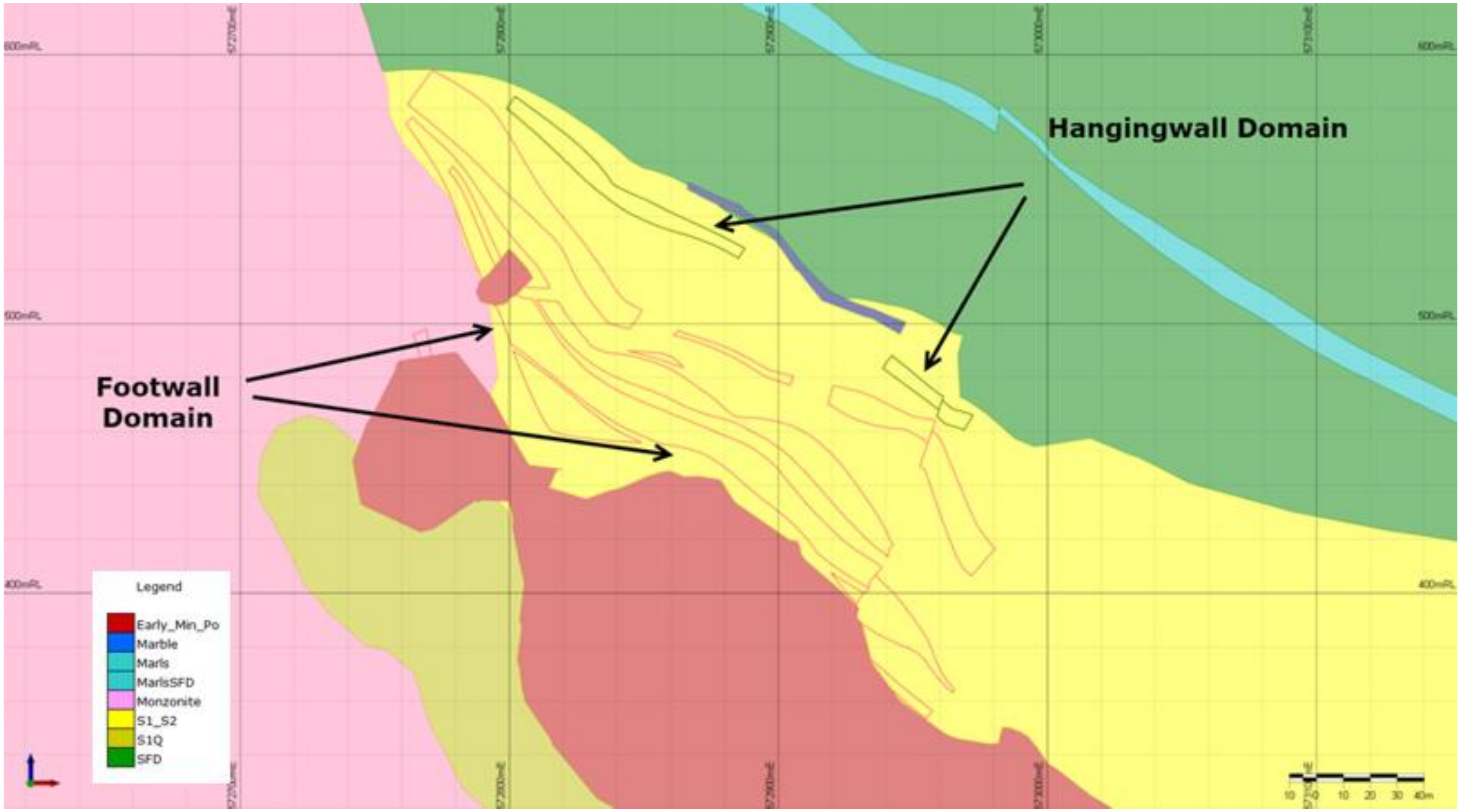
Source: ERM, 2024

Figure 14.6 – Mineralised Domains, View Looking North



Source: ERM, 2024

Figure 14.7 – Cross Section Looking North (4895790mN) Lithology Model; Mineralisation Overlain as 2D Lines



Source: ERM, 2024

14.3.3 LITHOGEOCHEMICAL DOMAINS

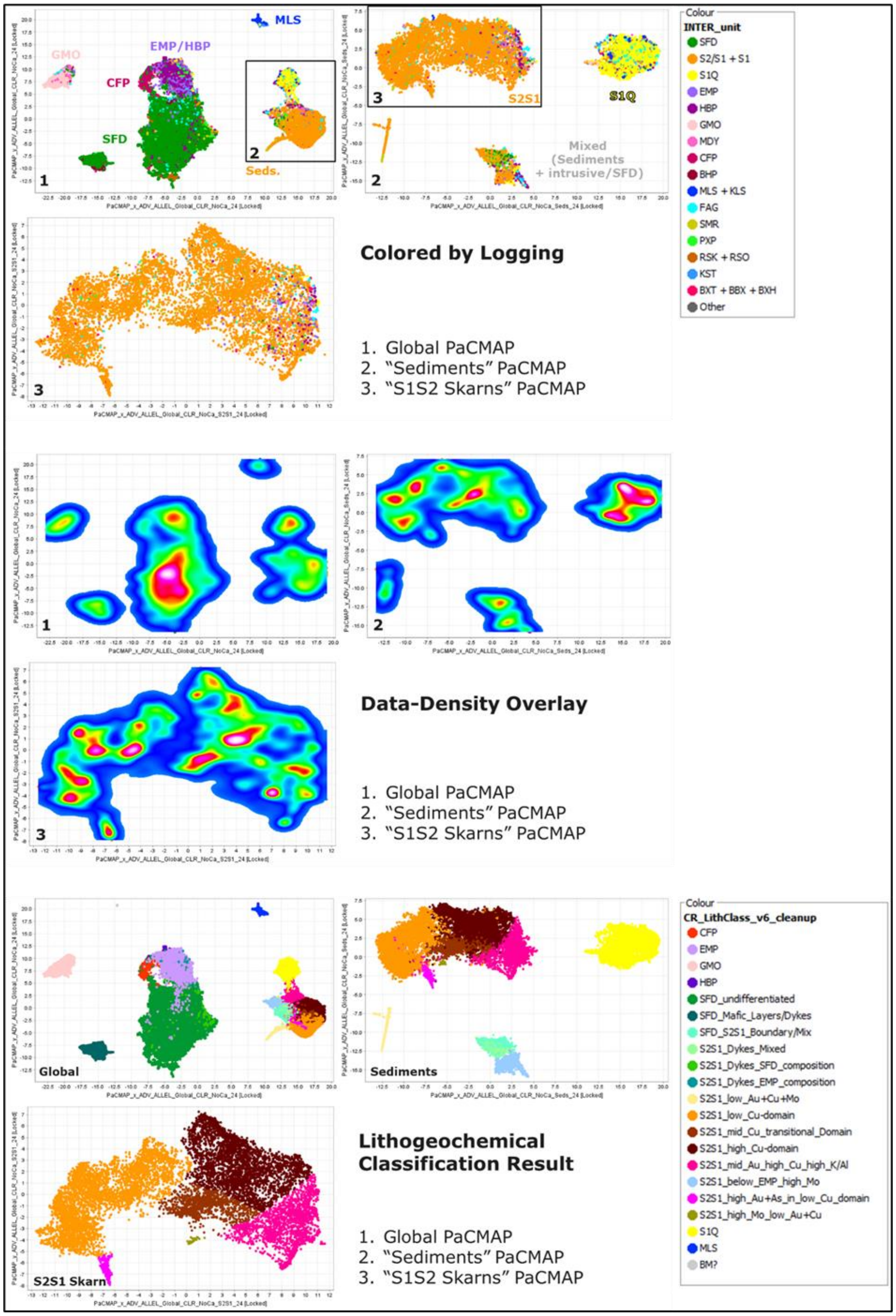
Multi-element assay data were reviewed for their suitability to generate lithogeochemical classification to inform geological modelling, resource domaining, and geometallurgical assessments. Data quality and coverage of the multi-element assay database was considered adequate for the purposes of mineral 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 14.8). 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. Furthermore, to generate the final lithogeochemical classification, the PaCMAP dimension reduction process was repeated for subsets of the data, first for the samples in the cluster representing siliclastic sediments, and then only for samples corresponding to S2S1 skarns. This iterative process was applied to better discriminate subgroupings within the skarns in order to assess if these have any relationship to the distribution of the mineralisation.

To generate the classification ("PaCMAP_LithClass"), the compositional clusters identified in the PaCMAP X-Y projection were reviewed against geological logging and their average compositions, 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. Labels for the S2S1 skarn subgroups are based on their composition.

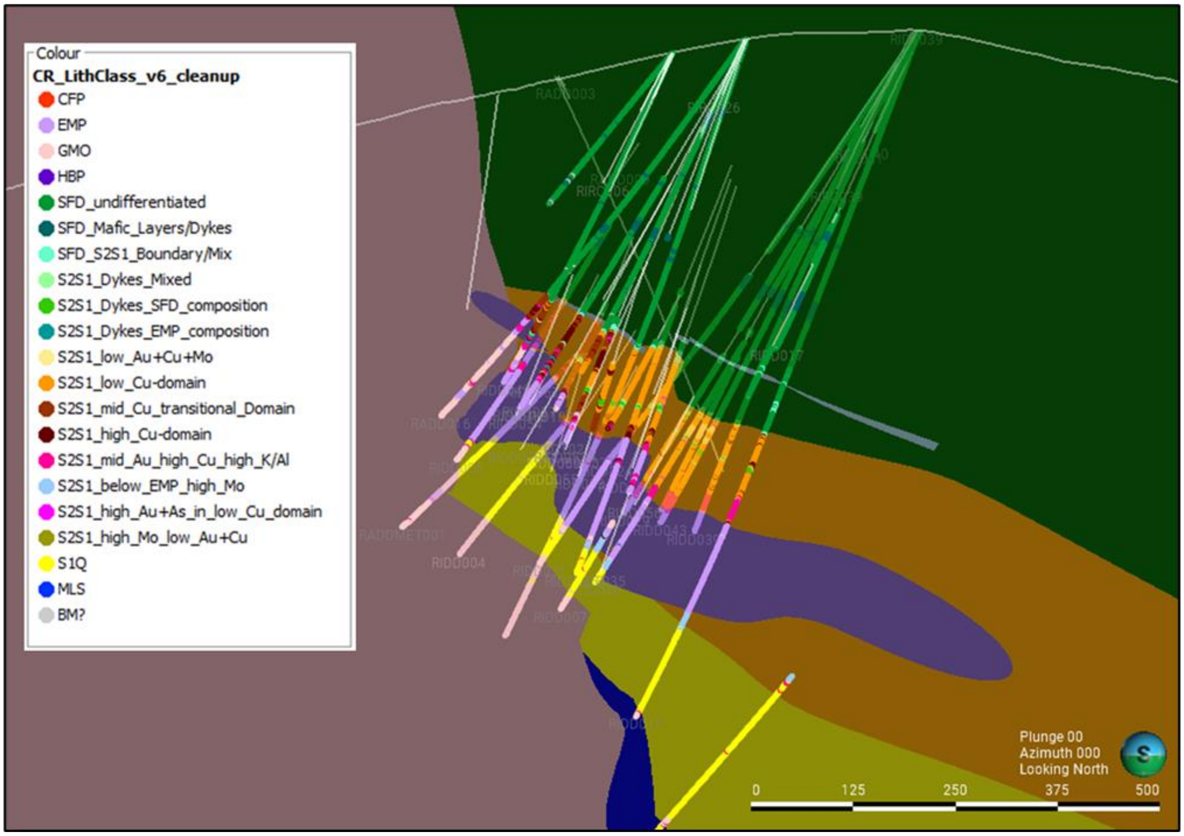
Figure 14.8 – PaCMAP (Wang et al., 2021) x-y Projection and Classification of CLR-Transformed Multi-Element Data



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.

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 is shown in Figure 14.9.

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: ERM, 2024

The S2S1 skarns can be split into several sub-domains, most notably a high-Cu domain and a low-Cu domain with a somewhat distinct transitional domain with intermediate Cu contents between the two. Volumetrically smaller domains include a zone characterised by elevated Au, As, and S that is located along the inferred N-S fault in the eastern part of the deposit. Most of the very-high-grade gold appears to sit in the high-Cu and transitional domains, near the contact to the low-Cu domain.

Gold contents within the S2S1 unit are not correlated to any of the chalcophile elements or any of the element ratios that may capture geochemical trends in response to skarn-alteration (e.g., Fe/Al, K/Al, Mn/Al). However, skarn-alteration of S2S1 unit is difficult to assess given the censored Ca data above 15 wt% and absence of carbon data. Similarly, no distinct compositional signature of low-grade material (aside from the PXP dykes) within high-grade domains could be identified. Contents of Au are not correlated to contents to Cu, which supports the assumption that the bulk of the Au mineralisation may not be linked to the event that generated the Cu enrichment in the S2S1 unit.

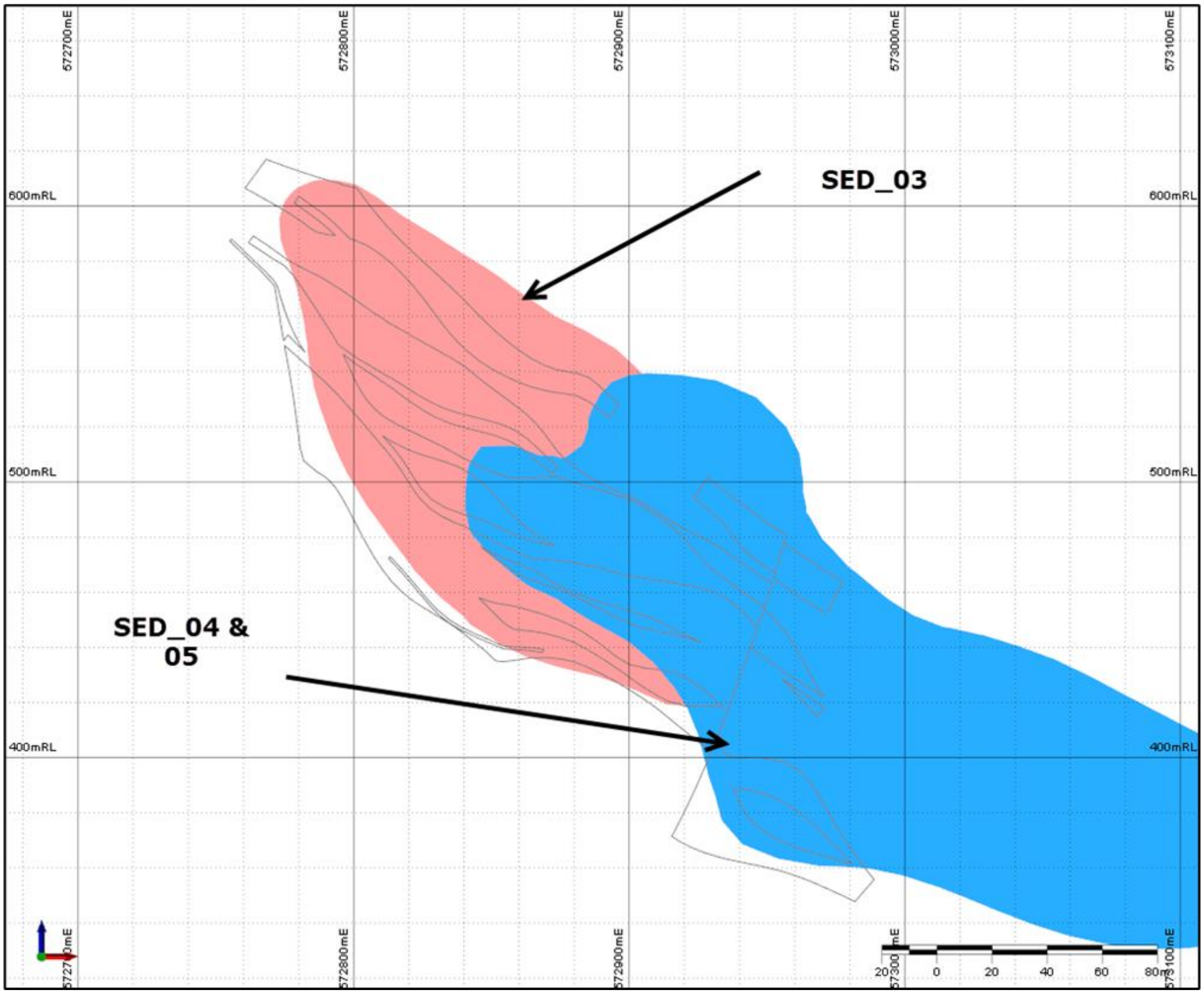
Very-high gold grades are generally associated with relatively low As, and zones with high values for the Au/As ratio may contain more free gold, whereas gold associated or hosted by sulphides are likely to contain higher As contents and thereby lower Au/As ratios.

Given the lack of clearly identifiable geochemical controls and correlations, very high-grade Au mineralisation may be largely structurally controlled, but this is still open to interpretation. The location and shape of the intrusions (which may be affected by pre-existing structures as well) and associated implications for temperature gradients and fluid migration as well as the PXP dykes, which may have acted as less permeable layers within the S2S1 unit, might have additionally contributed to focusing high-grade Au mineralisation.

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 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: ERM, 2024

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. This remains unchanged from that used in the November 2023 Maiden MRE.

14.5 Domaining

For mineralisation, there were three main domains overall – footwall mineralisation (several sub-parallel units interpreted to be part of the same mineralisation event); hangingwall mineralisation and a monzonite/early mineralised porphyry (EMP) hosted steep mineralisation interpreted beneath the footwall in Fault Block 4.

Footwall mineralisation is the largest zone and was coded ESTZON 101 while the hanging wall mineralisation was coded ESTZON 102. The monzonite/EMP hosted mineralisation was coded ESTZON 103.

Mafic intrusions, interpreted to be sills, were grouped into a domain (ESTZON 200) and internal waste units continuous over at least two drill holes in-section and across two cross-sections, were grouped into a single domain (ESTZON 201).

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 14.2 presents domain codes used for mineralisation and waste.

Table 14.2 – Domain Codes

Wireframe	Flagging Field	Flagging Code	ESTZON
min_fb1_fw_	MINZON	101	101[3]
min_fb3_fw1_	MINZON	301	101[3]
min_fb3_fw3_	MINZON	303	101[3]
min_fb3_hw_	MINZON	302	102
min_fb4_fw1_	MINZON	401	101[3]
min_fb4_fw3_	MINZON	403	101[3]
min_fb4_fw5_	MINZON	405	101[3]
min_fb4_hw_	MINZON	402	102
min_fb4_hw1_	MINZON	404	102
min_fb5_fw1_	MINZON	501	101[3]
min_fb5_fw3_	MINZON	503	101[3]
min_fb6_fw1_	MINZON	601	101[3]
min_fb6_fw3_	MINZON	603	101[3]
min_fb6_fw5_	MINZON	605	101[3]
min_fb6_hw_	MINZON	602	102
min_fb4_DPM_hw_	MINZON	701	103
iw_fb3_fw1_	MINZON	901	201
iw_fb3_fw2_	MINZON	902	201
iw_fb4_fw1_	MINZON	903	201
iw_fb4_fw2_	MINZON	904	201
iw_fb4_fw3_	MINZON	905	201
iw_fb4_fw6_	MINZON	906	201
iw_fb5_fw2_	MINZON	907	201

Wireframe	Flagging Field	Flagging Code	ESTZON
iw_fb6_fw2_	MINZON	908	201
geo_fb1_pxp2_	LATE_INT	12	200
geo_fb3_pxp1_	LATE_INT	31	200
geo_fb3_pxp2_	LATE_INT	32	200
geo_fb4_mdy1_	LATE_INT	45	200
geo_fb4_mdy2_	LATE_INT	46	200
geo_fb4_pxp1_	LATE_INT	41	200
geo_fb4_pxp2_	LATE_INT	42	200
geo_fb4_pxp3_	LATE_INT	43	200
geo_fb4_pxp4_	LATE_INT	44	200
geo_fb5_pxp1_	LATE_INT	51	200
geo_fb5_pxp2_	LATE_INT	52	200
geo_fb6_pxp1_	LATE_INT	61	200
geo_fb6_pxp2_	LATE_INT	62	200
geo_fb6_pxp3_	LATE_INT	63	200
geo_fb6_pxp4_	LATE_INT	64	200
cr_geo_sfd_	GEOL	1	3017
cr_geo_vhm_	GEOL	2	302
cr_geo_marls_sfd_	GEOL	3	303
cr_geo_marls_	GEOL	4	3045
cr_geo_s1_s2_	GEOL	5	3045
geo_fb1_pxp2_	GEOL	6	
geo_fb3_pxp1_	GEOL	6	
geo_fb3_pxp2_	GEOL	6	
geo_fb4_mdy1_	GEOL	6	
geo_fb4_mdy2_	GEOL	6	
geo_fb4_pxp1_	GEOL	6	
geo_fb4_pxp2_	GEOL	6	
geo_fb4_pxp3_	GEOL	6	
geo_fb4_pxp4_	GEOL	6	
geo_fb5_pxp1_	GEOL	6	
geo_fb5_pxp2_	GEOL	6	
geo_fb6_pxp1_	GEOL	6	
geo_fb6_pxp2_	GEOL	6	

Wireframe	Flagging Field	Flagging Code	ESTZON
geo_fb6_pxp3_	GEOL	6	
geo_fb6_pxp4_	GEOL	6	
cr_geo_early_min_po_	GEOL	7	3017
geo_s1q_	GEOL	8	308
geo_marble_	GEOL	9	309
geo_monzonite_	GEOL	10	310
geo_metased_ [2]	GEOL	11	311

Notes:

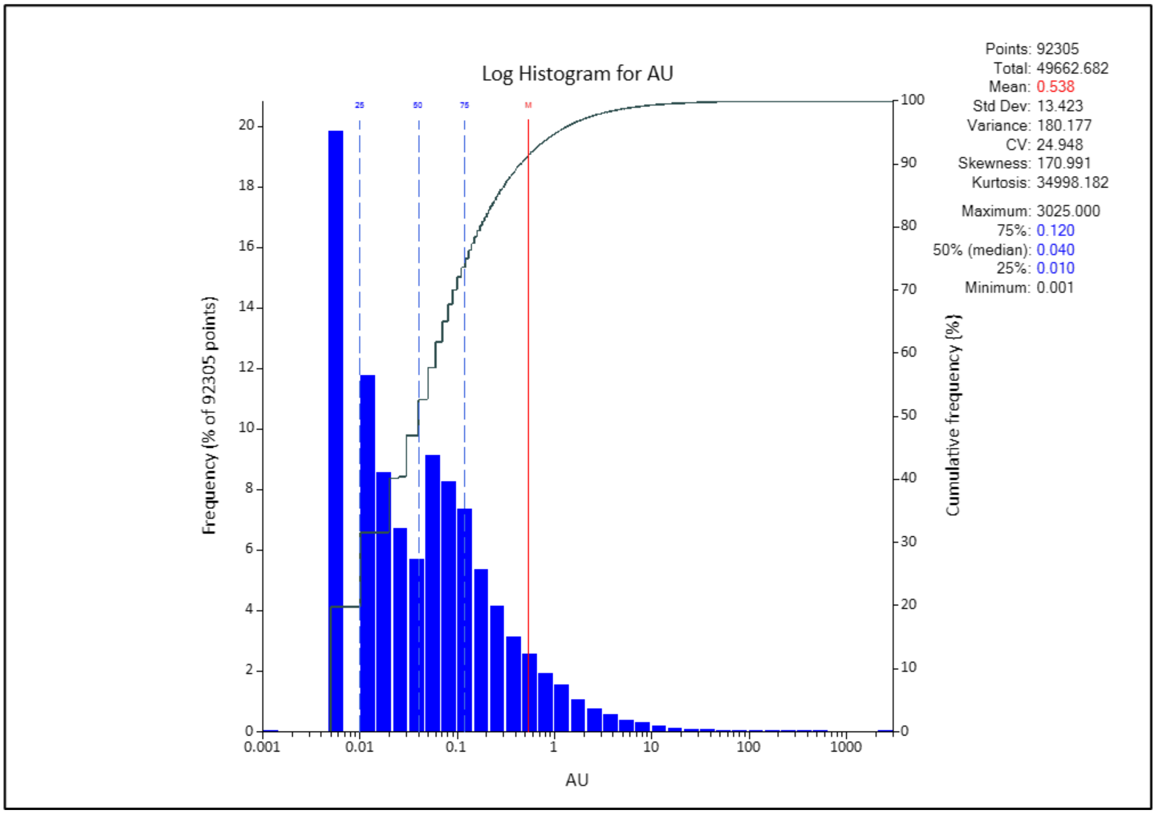
1) ESTZON 200 is a waste domain (late-stage intrusives) but deviates from the naming convention for other waste domains due to its location largely crosscutting mineralisation.
2) While there is data in the metasediments, the block model does not extend into them.
3) ESTZON 101 is subsequently split further by lithology into 101 (S1/S2), 104 (marls), 107 (EMP), 110 (monzonite).

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 to Figure 14.14. Histograms for silver (whole dataset) and mineralised domains are shown in Figure 14.15 to Figure 14.18,.

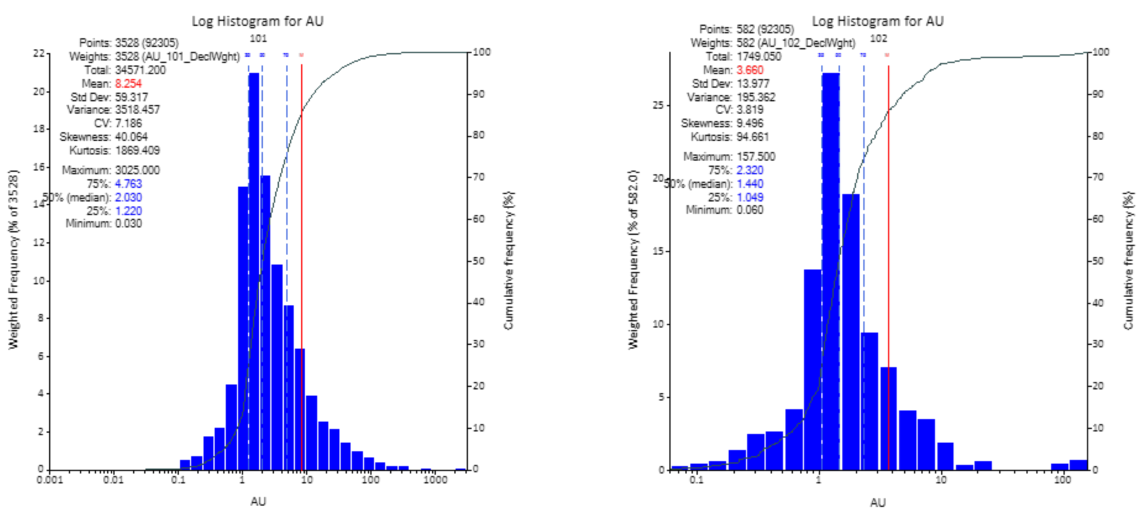
A cell declustering weight of 20 x 20 x 5 m was used to account for clustering in the dataset. Statistics of raw gold data are presented in Table 14.3.

Figure 14.11 – Log Normal Histogram of Gold – Whole Dataset



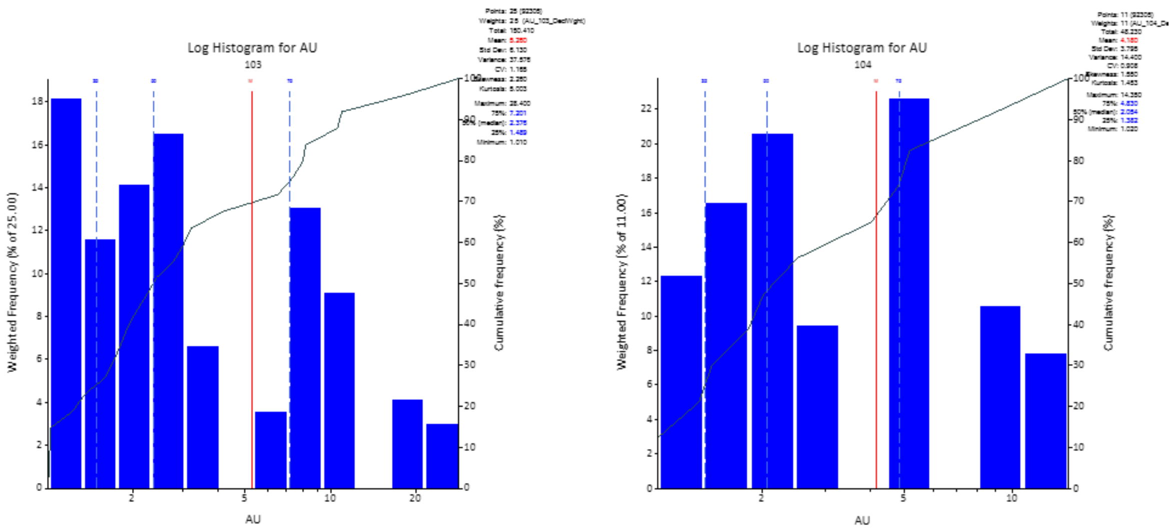
Source: ERM, 2024

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



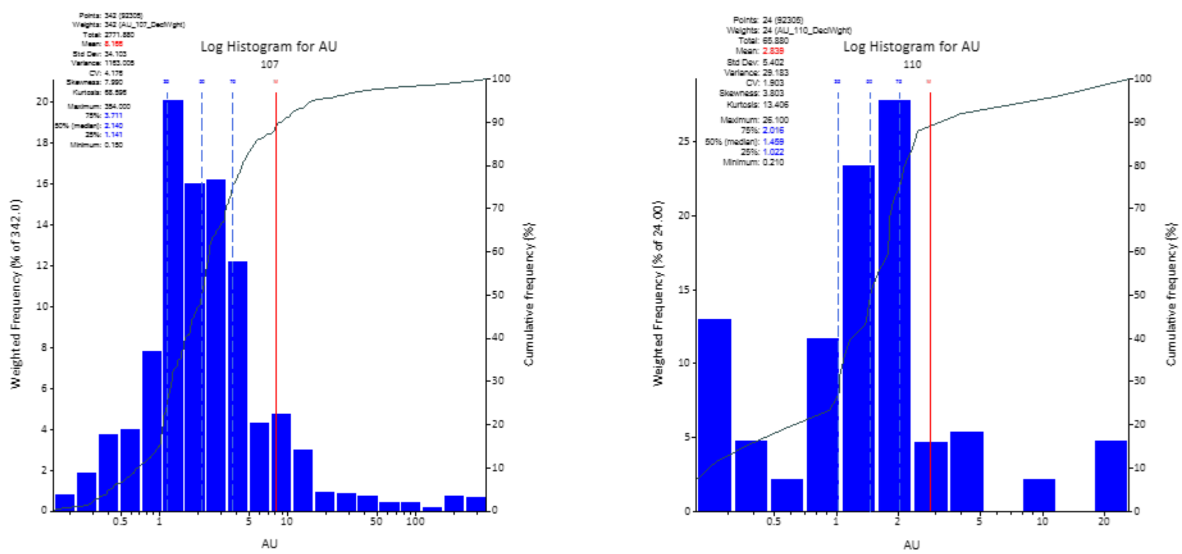
Source: ERM, 2024

Figure 14.13 – Log Normal Histogram for Gold for ESTZON 103 (Left) and ESTZON 104 (Right)



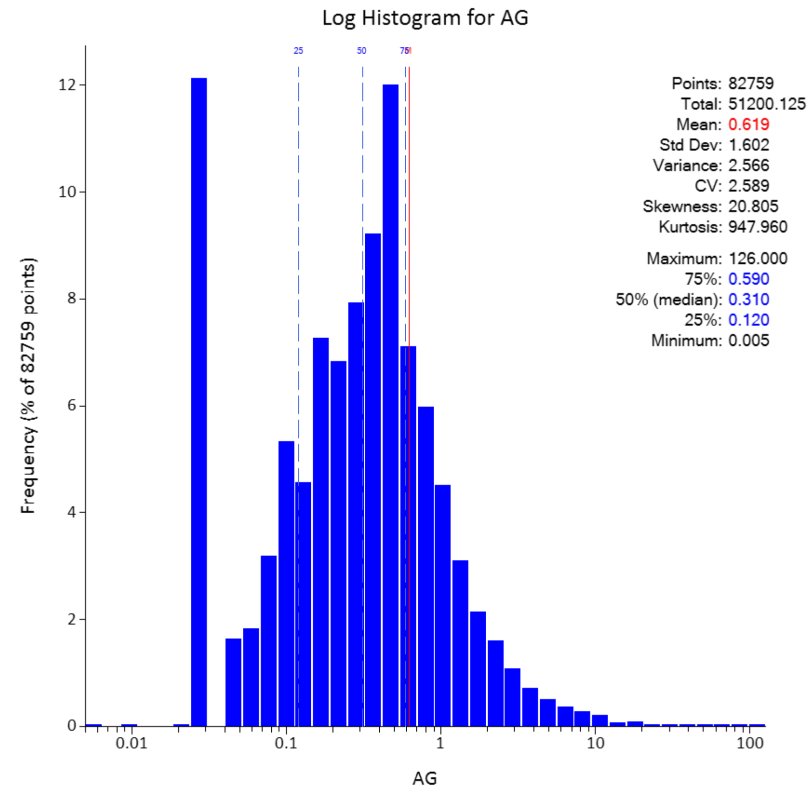
Source: ERM, 2024

Figure 14.14 – Log Normal Histogram for Gold for ESTZON 107 (Left) and ESTZON 110 (Right)



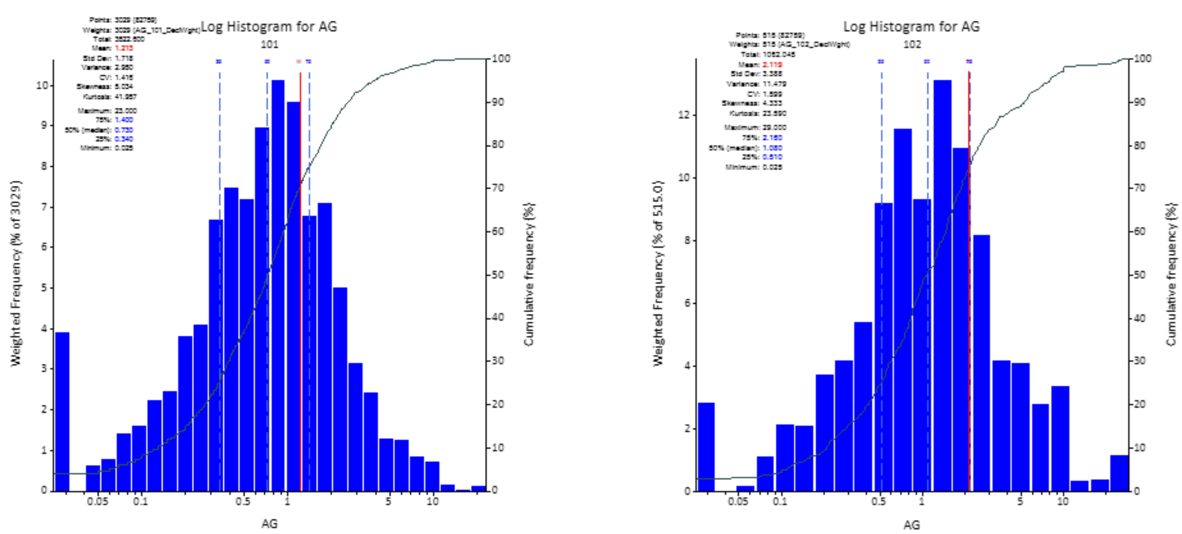
Source: ERM, 2024

Figure 14.15 – Log Normal Histogram of Silver – Whole Dataset



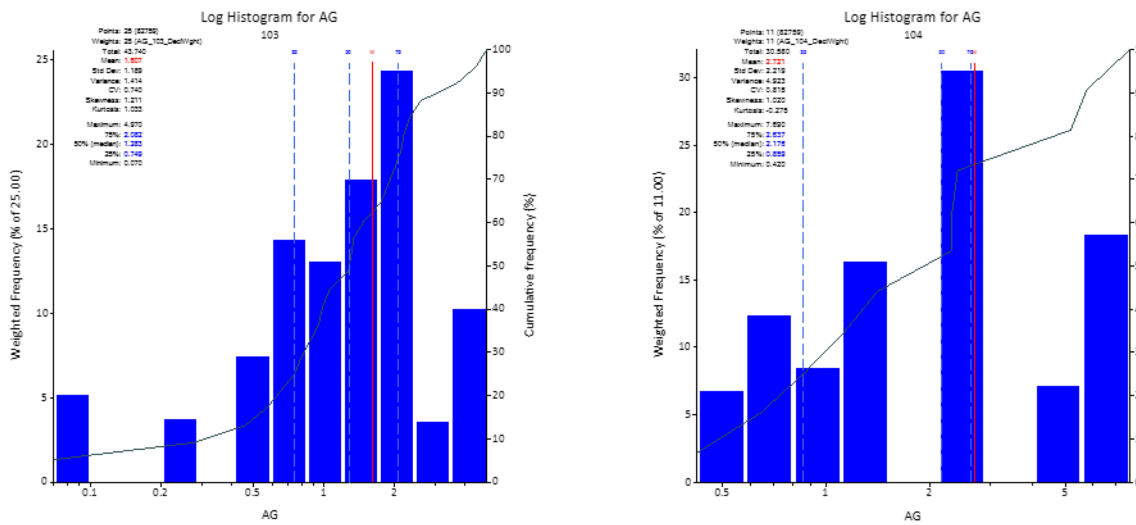
Source: ERM, 2024

Figure 14.16 – Log Normal Histograms for Silver for ESTZON 101 (Left) and ESTZON 102 (Right)



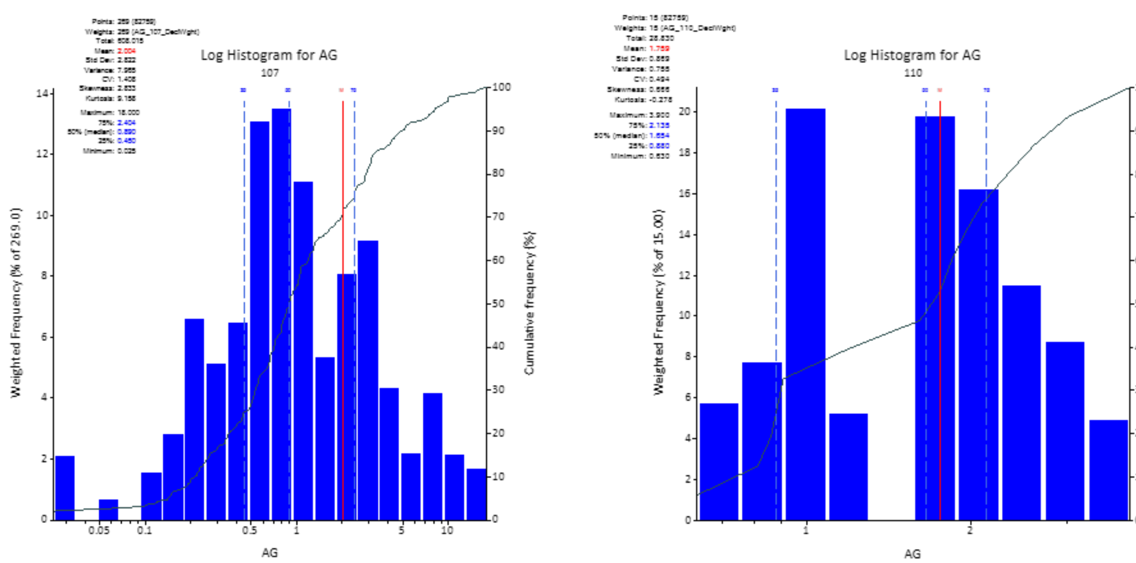
Source: ERM, 2024

Figure 14.17 – Log Normal Histograms for Silver for ESTZON 103 (Left) and ESTZON 104 (Right)



Source: ERM, 2024

Figure 14.18 – Log Normal Histograms for Silver for ESTZON 107 (Left) and ESTZON 110 (Right)



Source: ERM, 2024

Table 14.3 – Summary Raw Statistics for Gold – Clustered and Declustered Where Applicable

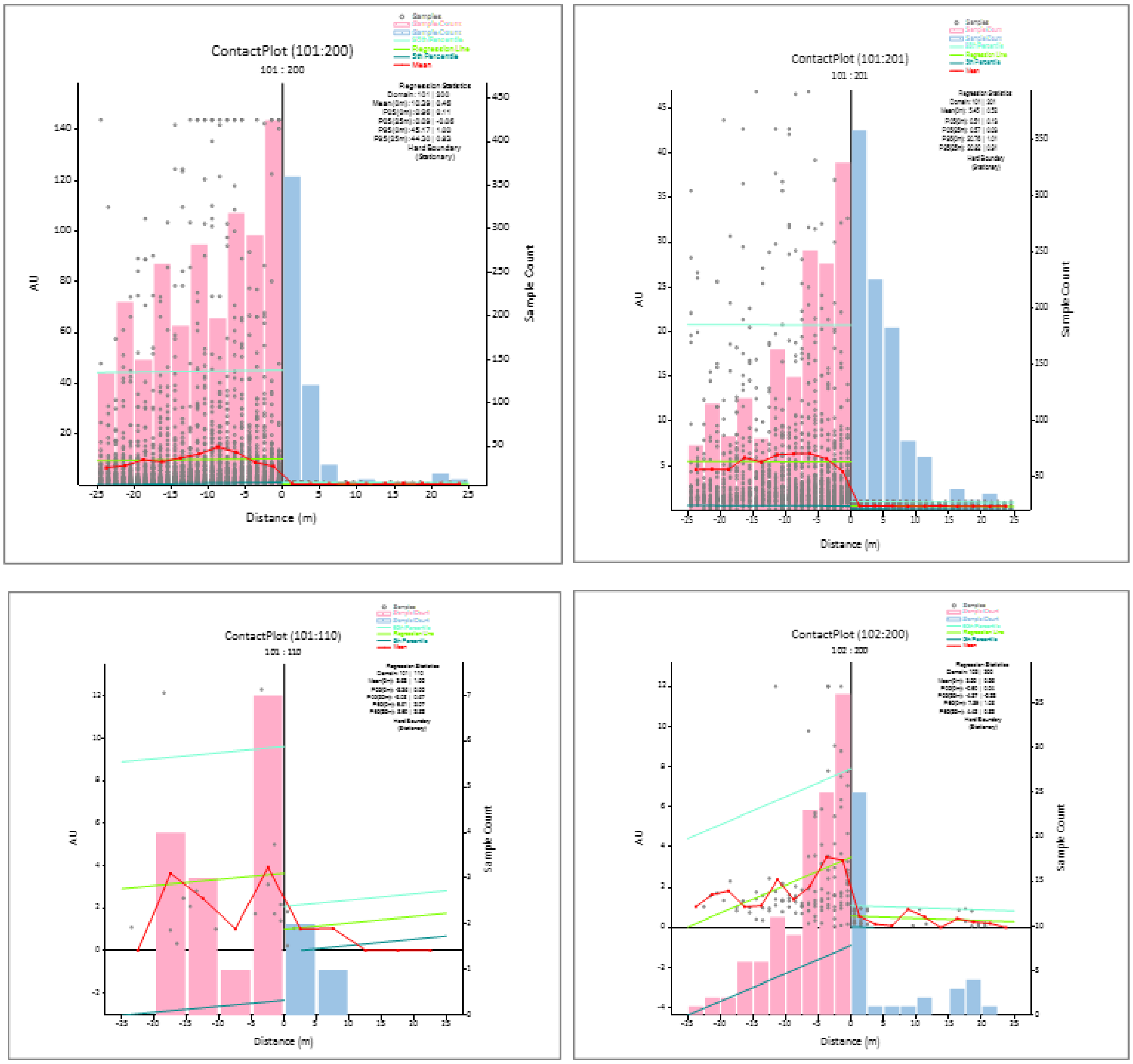
Statistic (Gold)	ESTZON							
	101	104	107	110	103	102	200	201
Declustering cell size	20 x 20 x 5 m							
Samples	3,545	11	345	24	25	582	850	643
Minimum	0.03	1.02	0.15	0.21	1.01	0.06	0.01	0.01
Maximum	3,025.00	14.35	354.00	26.10	28.40	157.50	4.89	4.98
Mean	8.24	4.18	8.13	2.84	5.26	3.66	0.33	0.50
Standard deviation	59.31	3.80	34.03	5.40	6.13	13.98	0.39	0.36
Coefficient of variation	7.20	0.91	4.19	1.90	1.17	3.82	1.20	0.73
Variance	3,518.10	14.40	1,158.25	29.18	37.58	195.36	0.16	0.13
Skewness	40.07	1.55	8.01	3.80	2.26	9.50	5.26	3.34
Log samples	3,545.00	11.00	345.00	24.00	25.00	582.00	850.00	643.00
Log mean	0.94	1.09	0.84	0.34	1.17	0.48	-1.72	-0.96
Log variance	1.45	0.65	1.35	1.13	0.88	0.91	1.62	0.61
Geometric mean	2.56	2.97	2.31	1.40	3.23	1.62	0.18	0.39
10%	0.79	1.02	0.68	0.23	1.02	0.63	0.03	0.13
20%	1.11	1.29	1.06	0.63	1.27	0.99	0.07	0.20
30%	1.33	1.45	1.24	1.04	1.69	1.10	0.11	0.27
40%	1.61	1.88	1.57	1.19	1.95	1.26	0.16	0.35
50%	2.03	2.05	2.14	1.46	2.38	1.44	0.22	0.44
60%	2.64	2.94	2.43	1.79	3.03	1.71	0.29	0.54
70%	3.78	4.68	3.28	1.85	5.85	2.04	0.39	0.63
80%	5.91	4.83	4.42	2.13	8.01	2.85	0.55	0.76
90%	12.28	8.21	8.92	3.27	10.91	4.84	0.75	0.88
95%	27.33	10.98	15.82	8.67	16.13	7.69	0.88	0.98
97.50%	49.76	12.67	45.68	17.22	22.18	10.93	0.97	1.08
99%	100.18	13.68	184.76	22.55	25.91	55.65	1.57	1.89

Table 14.4 – Summary Raw Statistics for Silver

Statistic (Silver)	ESTZON							
	101	104	107	110	103	102	200	201
Declustering cell size	20 x 20 x 5 m							
Samples	3,044	11	272	15	25	515	692	526
Minimum	0.03	0.42	0.03	0.63	0.07	0.03	0.03	0.03
Maximum	23.00	7.69	21.00	3.90	4.97	29.00	20.00	10.00
Mean	1.16	2.78	2.36	1.92	1.75	2.06	0.72	0.64
Standard deviation	1.51	2.29	3.40	0.87	1.36	3.31	1.32	0.90
Coefficient of variation	1.30	0.83	1.44	0.45	0.78	1.61	1.83	1.41
Variance	2.28	5.26	11.53	0.76	1.85	10.96	1.74	0.82
Skewness	4.15	0.96	2.64	0.56	1.05	4.42	8.15	5.04
Log samples	3,044.00	11.00	272.00	15.00	25.00	515.00	692.00	526.00
Log mean	-0.45	0.66	0.07	0.54	0.21	0.00	-1.08	-1.10
Log variance	1.44	0.77	1.67	0.24	0.90	1.59	1.70	1.54
Geometric mean	0.64	1.94	1.07	1.72	1.23	1.00	0.34	0.33
10%	0.13	0.45	0.22	0.74	0.36	0.21	0.05	0.05
20%	0.27	0.76	0.37	0.91	0.63	0.42	0.12	0.13
30%	0.40	1.01	0.54	1.37	0.80	0.61	0.21	0.21
40%	0.55	1.29	0.71	1.63	0.98	0.80	0.30	0.30
50%	0.73	1.86	0.91	1.81	1.28	1.02	0.41	0.39
60%	0.93	2.33	1.29	1.89	1.55	1.42	0.55	0.50
70%	1.18	2.42	2.20	2.05	2.17	1.79	0.73	0.64
80%	1.65	4.71	3.10	2.59	2.43	2.49	0.99	0.90
90%	2.50	5.78	7.17	2.99	4.05	5.01	1.49	1.35
95%	3.73	6.67	10.00	3.31	4.56	7.77	2.14	1.99
97.50%	5.55	7.18	12.20	3.60	4.75	10.00	2.99	2.68
99%	8.07	7.49	16.28	3.78	4.88	19.70	6.17	4.58

Contact analysis was completed to establish if soft or hard boundaries between domains should be used in the estimate. Hard boundaries were maintained between domains based on the results. Some examples are shown in the following image (Figure 14.19).

Figure 14.19 – Contact Analysis Results

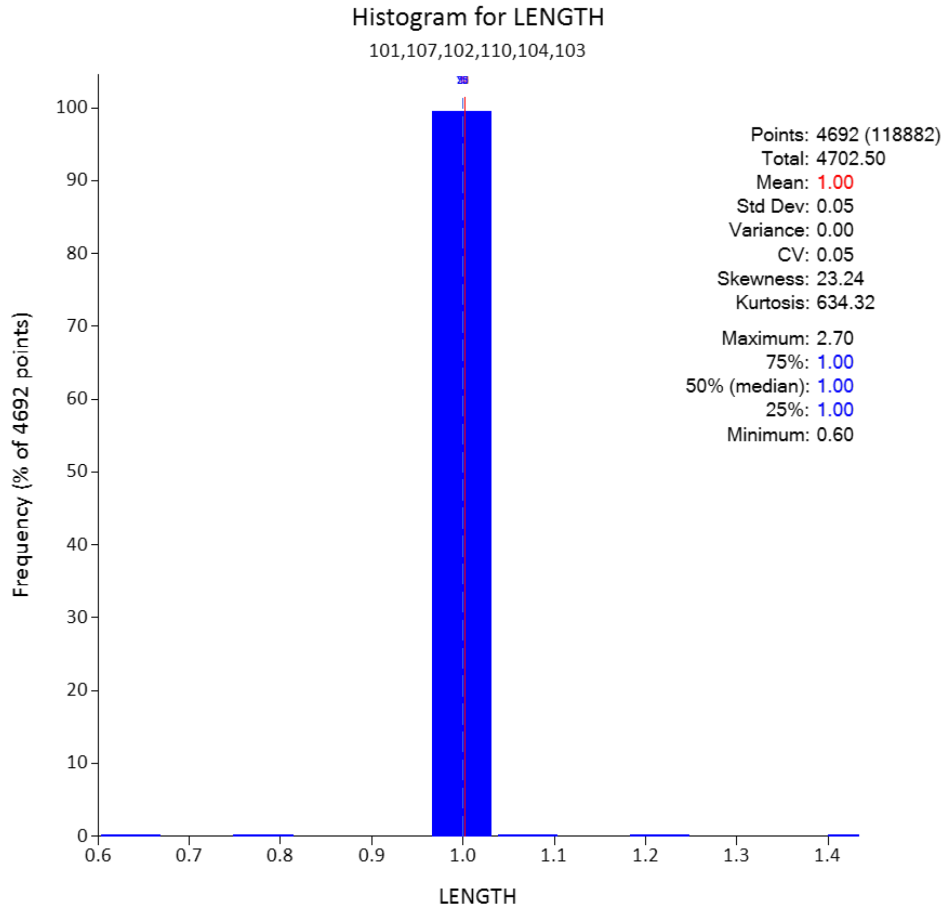


Source: ERM, 2024

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 14.20; therefore 1.0 m composite length was chosen with only two residuals arising in ESTZON 103 (both 0.75 m), which were excluded.

Figure 14.20 – Histogram Showing Length of Raw Data in Mineralised Domain



Source: ERM, 2024

14.8 Global and Domain Statistics

Domain statistics of the uncut 1 m composites are presented in Table 14.5 (Gold) and Table 14.6 (Silver). Uncut gold is characterised by a very high coefficient of variation.

Table 14.5 – Summary of 1 m Composite Statistics - Gold

Statistic (Gold)	ESTZON							
	101	104	107	110	103	102	200	201
Declustering cell size	20 x 20 x 5 m							
Samples	3,554	11	345	24	24	583	853	643
Minimum	0.03	1.02	0.15	0.21	1.01	0.06	0.01	0.01
Maximum	3025.00	14.35	354.00	26.10	28.40	157.50	4.89	4.98

Statistic (Gold)	ESTZON							
	101	104	107	110	103	102	200	201
Mean	8.21	4.18	8.13	2.84	5.93	3.67	0.33	0.50
Standard deviation	54.60	3.80	34.03	5.40	6.37	13.98	0.39	0.36
Coefficient of variation	6.65	0.91	4.19	1.90	1.07	3.81	1.20	0.73
Variance	2980.81	14.40	1158.25	29.18	40.52	195.42	0.15	0.13
Skewness	41.81	1.55	8.01	3.80	2.06	9.49	5.27	3.34
Log samples	3554.00	11.00	345.00	24.00	24.00	583.00	853.00	643.00
Log mean	0.95	1.09	0.84	0.34	1.35	0.48	-1.72	-0.96
Log variance	1.47	0.65	1.35	1.13	0.79	0.91	1.62	0.61
Geometric mean	2.59	2.97	2.31	1.40	3.85	1.62	0.18	0.39
10%	0.78	1.02	0.68	0.23	1.32	0.63	0.03	0.13
20%	1.12	1.29	1.06	0.63	1.70	0.99	0.07	0.20
30%	1.33	1.45	1.25	1.04	1.92	1.10	0.11	0.27
40%	1.62	1.88	1.57	1.19	2.30	1.26	0.16	0.35
50%	2.05	2.05	2.14	1.46	3.01	1.44	0.22	0.44
60%	2.68	2.94	2.44	1.79	3.52	1.71	0.29	0.54
70%	3.86	4.68	3.28	1.85	7.02	2.04	0.39	0.63
80%	6.08	4.83	4.42	2.13	8.16	2.86	0.54	0.76
90%	12.70	8.21	8.92	3.27	10.96	4.88	0.75	0.88
95%	28.88	10.98	15.82	8.67	17.75	7.72	0.88	0.98
97.50%	50.43	12.67	45.68	17.22	22.76	11.43	0.97	1.08
99%	102.35	13.68	184.76	22.55	26.15	55.65	1.57	1.89

Table 14.6 – Summary of 1 m Composite Statistics – Silver

Statistic (Silver)	ESTZON							
	101	104	107	110	103	102	200	201
Declustering cell size	20 x 20 x 5 m							
Samples	3,049	11	272	15	24	516	695	526
Minimum	0.03	0.42	0.03	0.63	0.07	0.03	0.03	0.03
Maximum	23.00	7.69	21.00	3.90	4.97	29.00	20.00	10.00
Mean	1.24	2.77	1.97	1.69	1.47	2.18	0.75	0.65
Standard deviation	1.76	2.30	2.84	0.87	1.24	3.51	1.35	0.94
Coefficient of variation	1.42	0.83	1.45	0.51	0.84	1.62	1.81	1.44
Variance	3.09	5.30	8.08	0.75	1.54	12.34	1.82	0.89

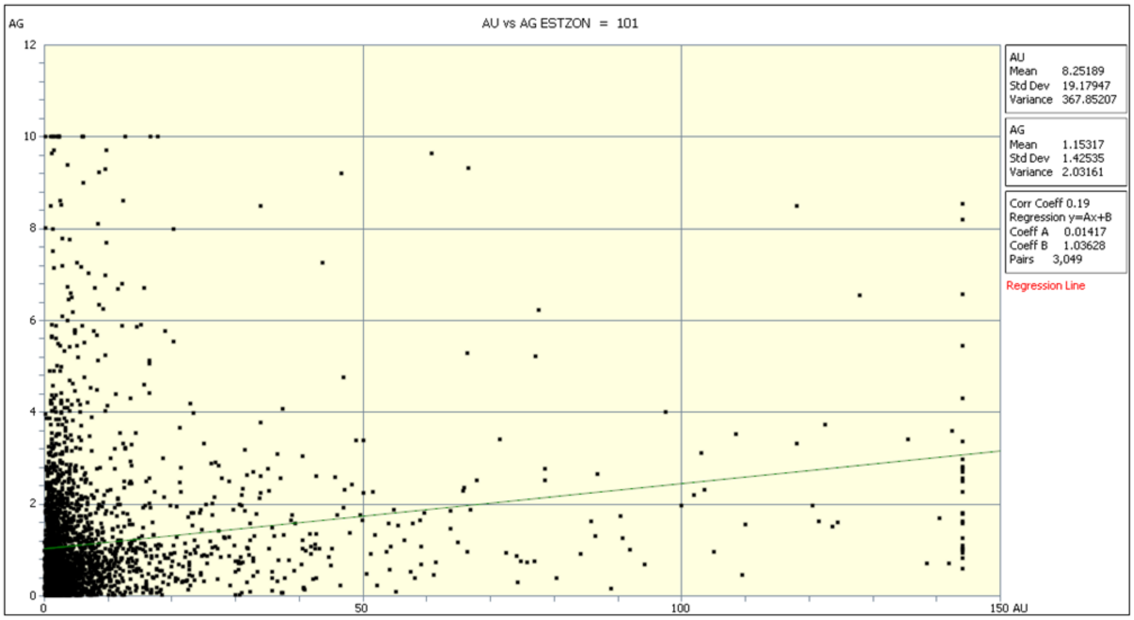
Statistic (Silver)	ESTZON							
	101	104	107	110	103	102	200	201
Skewness	4.85	0.96	3.14	0.84	1.52	4.24	8.47	4.96
Log samples	3049.00	11.00	272.00	15.00	24.00	516.00	695.00	526.00
Log mean	-0.43	0.66	-0.06	0.40	0.03	0.03	-1.03	-1.06
Log variance	1.52	0.76	1.65	0.26	0.87	1.65	1.71	1.47
Geometric mean	0.65	1.94	0.94	1.48	1.03	1.03	0.36	0.35
10%	0.13	0.48	0.20	0.74	0.28	0.21	0.05	0.06
20%	0.27	0.75	0.36	0.86	0.58	0.41	0.12	0.14
30%	0.40	0.98	0.54	0.89	0.75	0.61	0.23	0.24
40%	0.56	1.24	0.71	1.01	0.85	0.84	0.31	0.30
50%	0.74	1.83	0.89	1.60	0.99	1.08	0.43	0.39
60%	0.95	2.33	1.18	1.81	1.28	1.47	0.57	0.50
70%	1.22	2.41	1.93	1.94	1.42	1.90	0.76	0.64
80%	1.73	5.25	2.82	2.22	2.19	2.58	1.00	0.89
90%	2.66	5.78	4.70	2.79	3.35	5.46	1.51	1.46
95%	4.09	6.63	8.18	3.10	4.42	8.07	2.25	2.00
97.50%	6.17	7.16	10.00	3.50	4.68	10.00	3.02	2.70
99%	9.36	7.48	15.04	3.74	4.85	23.63	4.44	4.72

14.9 Variables and Correlations

The reported variables are gold and silver. Gold and silver are not well correlated within the skarn mineralisation (correlation coefficient of 0.19 in main footwall unit (Figure 14.21) 0.26 in hangingwall).

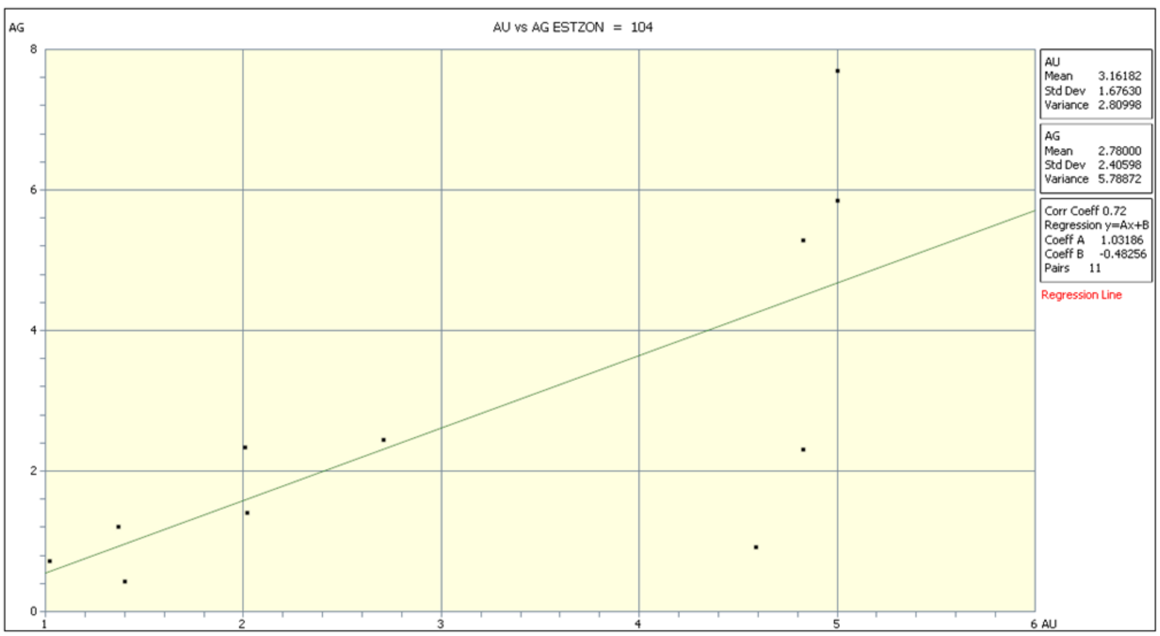
There is a moderate to strong correlation in the smaller footwall units - marls (ESTZON 104 (Figure 14.22), correlation coefficient of 0.72 and monzonite, correlation coefficient of 0.53).

Figure 14.21 – Scatterplot of Au vs Ag in ESTZON 101 Showing Weak Correlation



Source: ERM, 2024

Figure 14.22 – Scatterplot of Au vs Ag in ESTZON 104 Showing a Strong Correlation



Source: ERM, 2024

14.10 Treatment of Outliers (Top Cuts)

The treatment of outliers is one of the key factors that the MRE is most 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 (in which case it may be possible to domain them out). Equally, imposing a top cut that is too harsh leads to estimated metal that is too low.

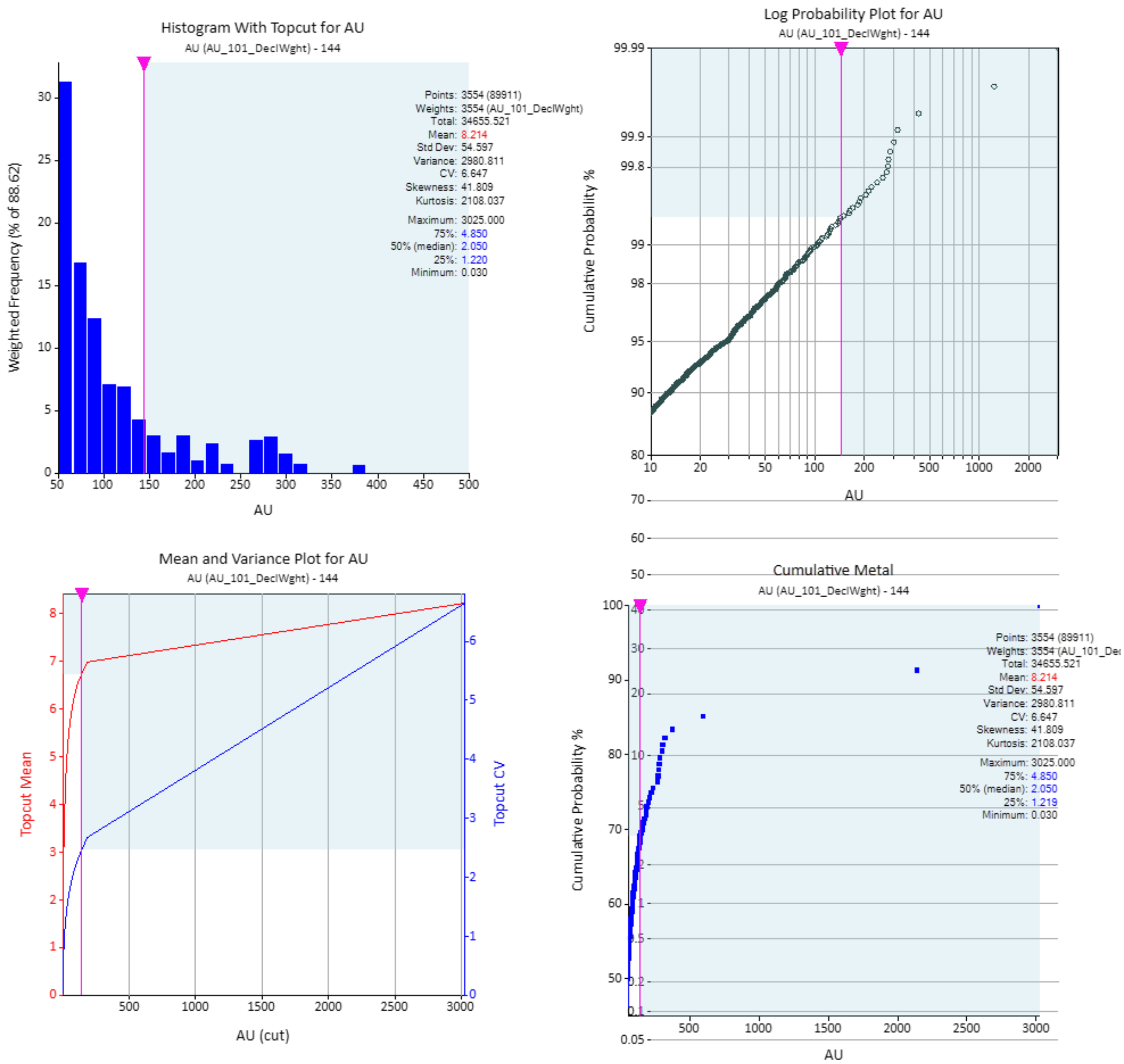
Global top cut analysis was completed in Supervisor software and outliers were reviewed spatially prior to a top cut (grade cap) being applied. Global top cut analysis comprises the simultaneous review of the grade histogram, log probability, mean-variance and cumulative metal plots, and having a balanced approach to choosing based on inflection points and spatial review.

Sensitivity reviews were completed on several top cuts for gold in the main footwall domain (ESTZON 101) ranging from 110 g/t to 175 g/t Au before a top cut of 144 g/t being chosen. Further drilling completed since the Maiden MRE (November 2023) has increased the top cut, with more higher grades being assayed. Test estimates run on the various top cuts shows the chosen top cut of 144 g/t Au results in a metal estimate approximately 3% higher than the lowest tested (110 g/t Au) and 2% lower metal than the highest tested (175 g/t Au).

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 14.23 to Figure 14.28, while Table 14.7 presents the top cuts applied to the mineralisation domains. Tables 14.8 and 14.9 show the summary statistics for the top cut estimation composites, showing a reduction in the coefficient of variation for the gold domains, though it remains high for major domains.

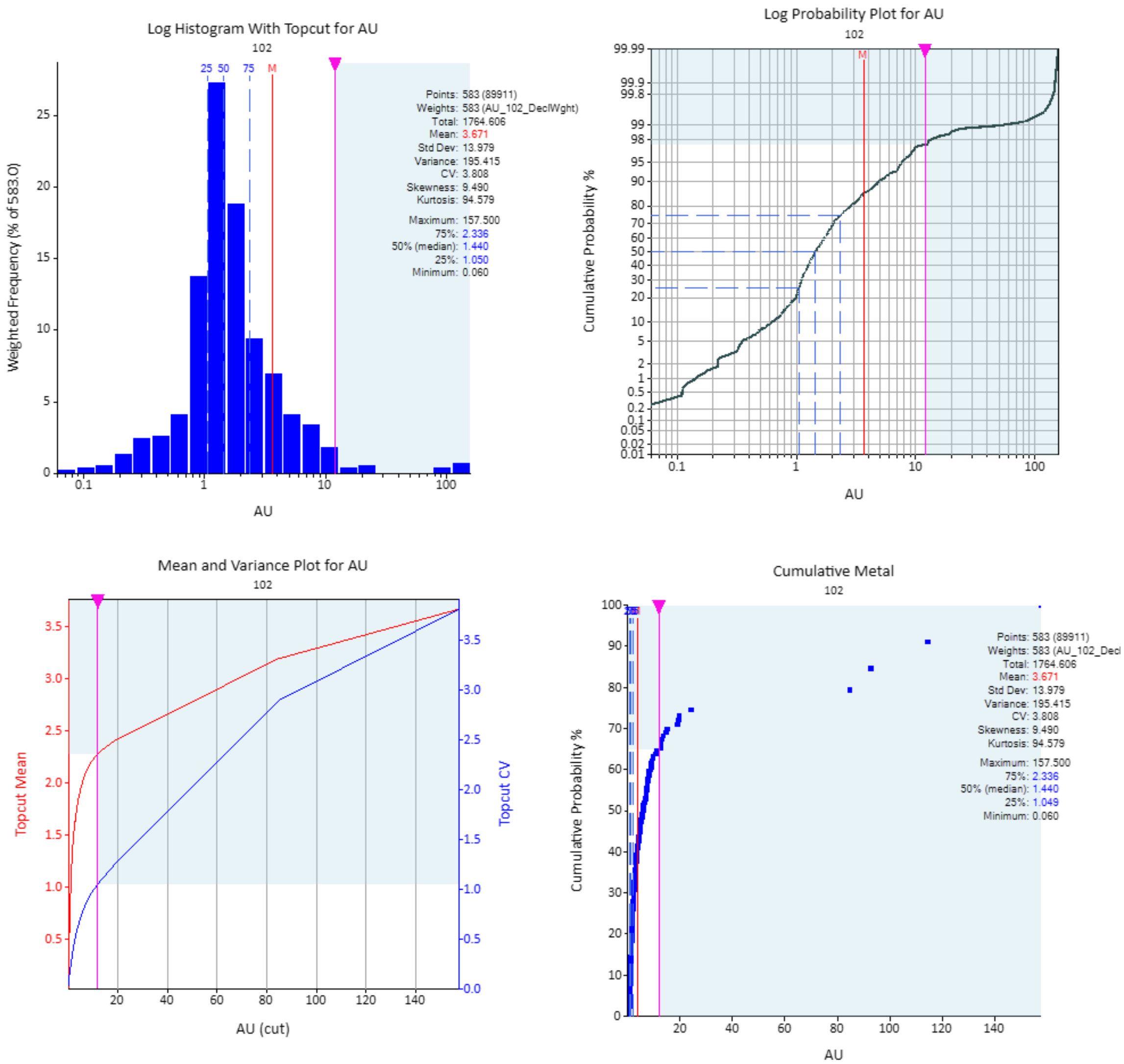
Additional capping within the estimate is also used. This is defined in the search parameters and is described further in Section 14.14.

Figure 14.23 – Global Top Cut Analysis for Gold for ESTZON 101



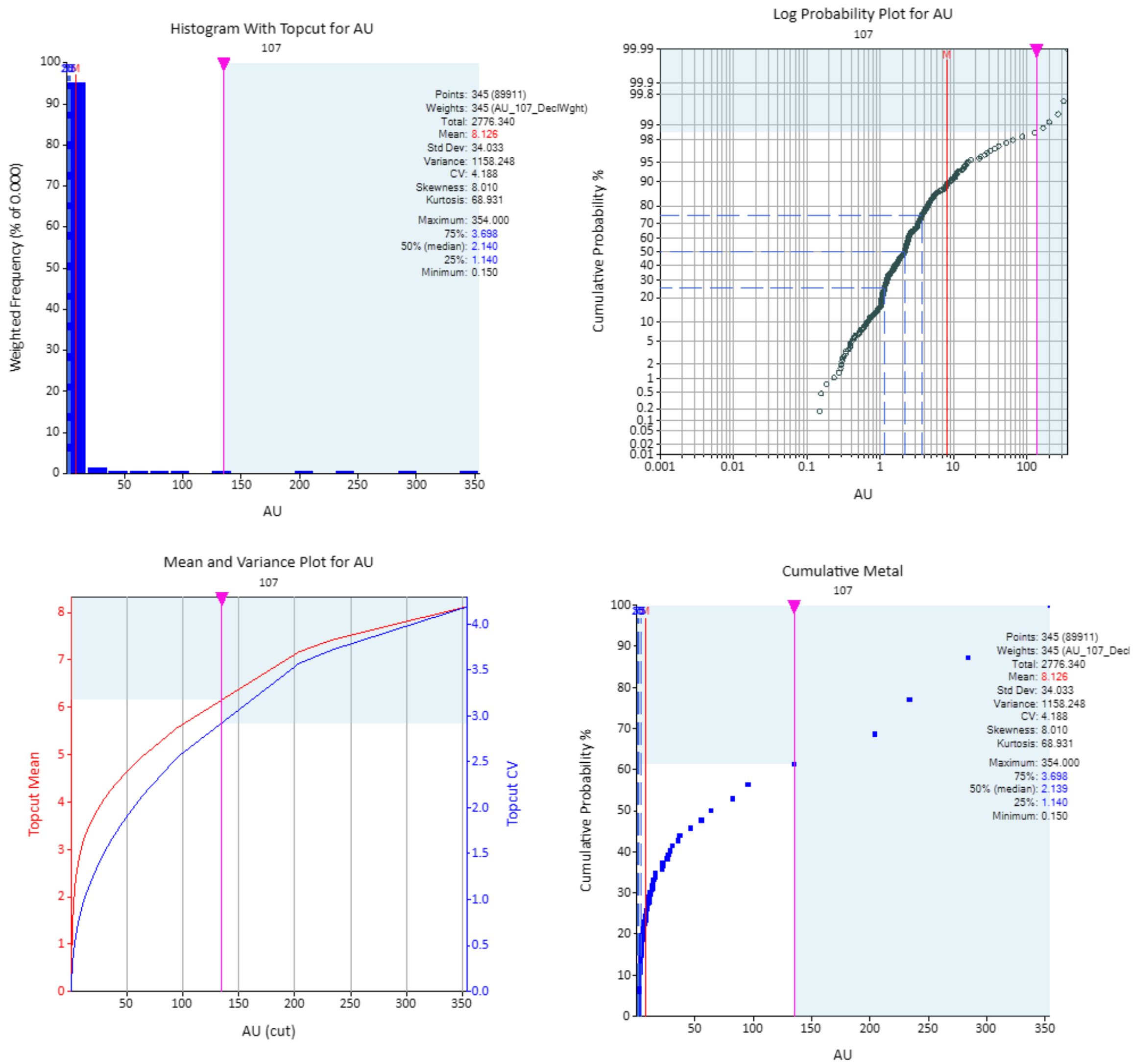
Source: ERM, 2024

Figure 14.24 – Global Top Cut Analysis for Gold for ESTZON 102



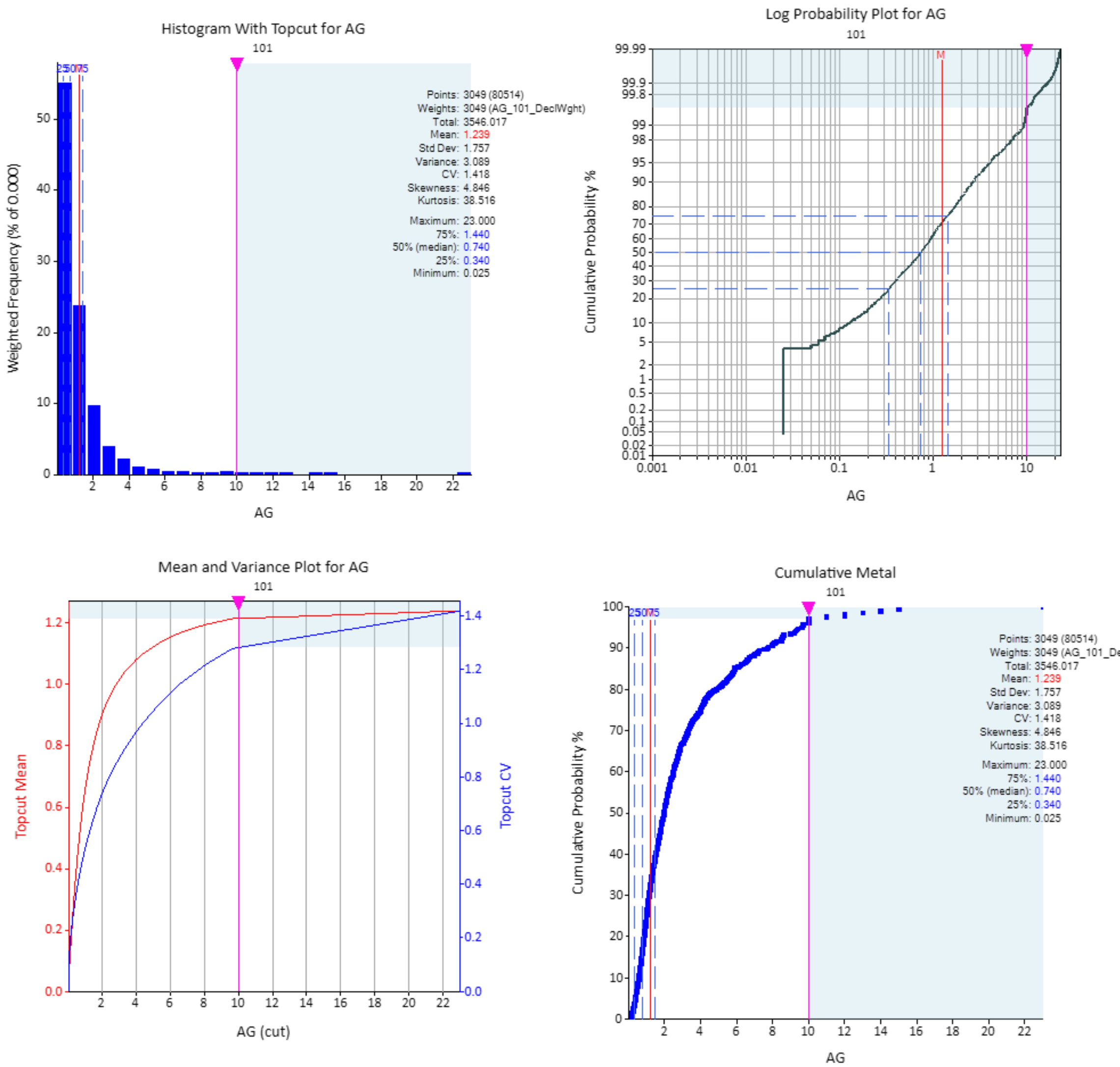
Source: ERM, 2024

Figure 14.25 – Global Top Cut Analysis for Gold for ESTZON 107



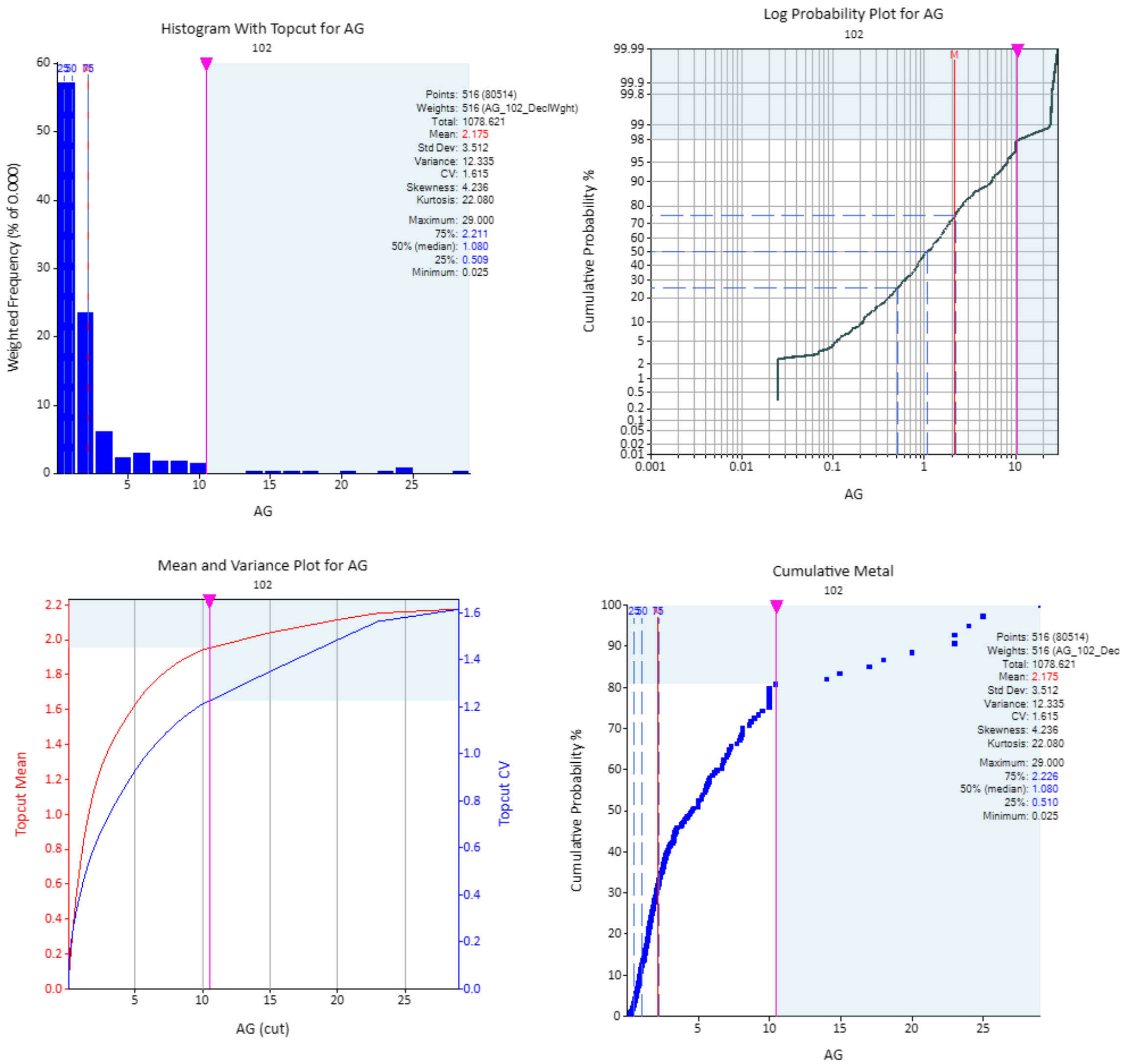
Source: ERM, 2024

Figure 14.26 – Global Top Cut Analysis for Silver for ESTZON 101



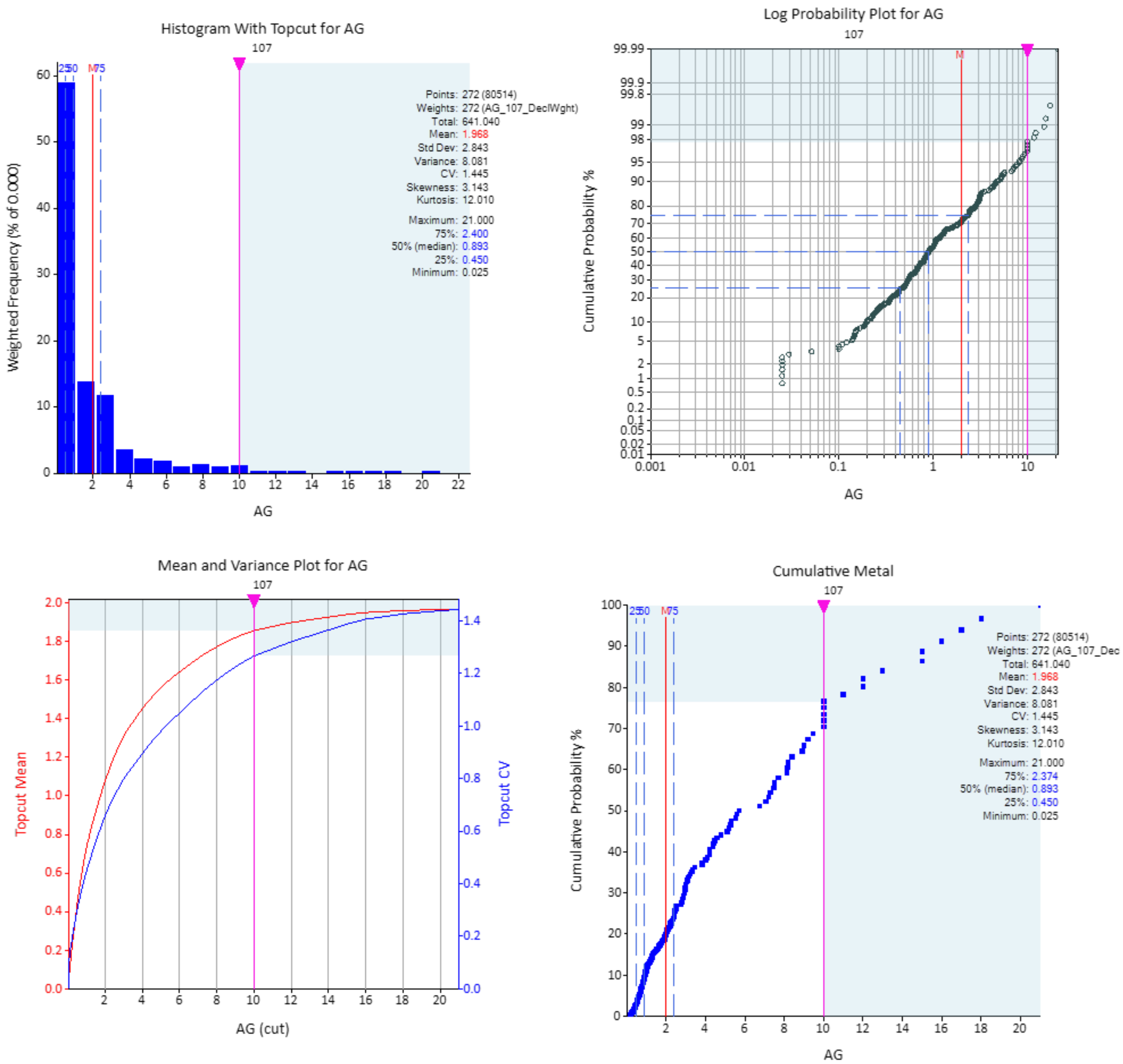
Source: ERM, 2024

Figure 14.27 – Global Top Cut Analysis for Silver for ESTZON 102



Source: ERM, 2024

Figure 14.28 – Global Top Cut Analysis for Silver for ESTZON 107



Source: ERM, 2024

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

Domain	Element	Number (Data)	Uncut Mean	Uncut CV	Top Cut	Cut Mean	Cut CV	Number (Data Cut)	% Data Cut	% Change Mean / Metal
101	Gold	3,554	8.24	7.19	144	6.64	2.50	24	0.70	19.50
	Silver	3,049	1.24	1.42	10	1.22	1.29	7	0.20	1.90
104	Gold	11	4.18	0.91	11	3.02	0.53	2	18.20	27.80
	Silver	11	2.77	0.83	7.69	2.77	0.83	1	9.10	0
107	Gold	345	8.13	4.19	135	6.20	2.97	4	1.20	23.70
	Silver	272	1.97	1.44	10	1.86	1.27	10	3.70	5.60
110	Gold	24	2.84	1.90	-	-	-	-	-	-
	Silver	15	1.69	0.51	3.9	1.69	0.51	1	6.70	0
102	Gold	583	3.67	3.81	12	2.28	1.05	15	2.60	38.00
	Silver	516	2.18	1.61	10.5	1.95	1.22	10	1.90	10.20
103	Gold	24	5.93	1.07	11	4.83	0.74	2	8.30	18.50
	Silver				-	-	-	-	-	-
200	Gold	853	0.33	1.20	1	0.31	0.88	13	1.50	6.70
	Silver	695	0.75	1.81	4	0.67	1.12	8	1.20	9.80
201	Gold	643	0.5	0.73	1	0.48	0.58	26	4.00	4.40
	Silver	526	0.65	1.44	3	0.60	1.06	10	1.90	8.40

Table 14.8 – Summary Statistics for Top Cut Composites – Gold

Statistic (Gold)	ESTZON							
	101	104	107	110	103	102	200	201
Declustering cell size	20 x 20 x 5 m							
Samples	3,554	11	345	24	24	583	853	643
Minimum	0.03	1.02	0.15	0.21	1.01	0.06	0.01	0.01
Maximum	144.00	5.00	135.00	26.10	11.00	12.00	1.00	1.00
Mean	6.64	3.02	6.20	2.75	4.83	2.28	0.31	0.48
Standard deviation	16.61	1.60	18.42	5.19	3.60	2.39	0.27	0.28
Coefficient of variation	2.50	0.53	2.97	1.89	0.75	1.05	0.88	0.58
Variance	275.88	2.55	339.44	26.97	12.93	5.70	0.07	0.08
Skewness	5.78	0.17	6.00	3.84	0.70	2.55	0.99	0.32
Log samples	3554.00	11.00	345.00	24.00	24.00	583.00	853.00	643.00
Log mean	0.94	0.94	0.83	0.28	1.28	0.46	-1.73	-0.97

Statistic (Gold)	ESTZON							
	101	104	107	110	103	102	200	201
Log variance	1.42	0.35	1.27	1.16	0.59	0.71	1.57	0.57
Geometric mean	2.55	2.57	2.29	1.33	3.61	1.58	0.18	0.38
10%	0.79	1.02	0.68	0.26	1.32	0.65	0.03	0.13
20%	1.11	1.29	1.06	0.44	1.70	1.00	0.07	0.20
30%	1.33	1.45	1.25	0.87	1.92	1.10	0.11	0.27
40%	1.61	1.88	1.57	1.06	2.30	1.26	0.16	0.35
50%	2.03	2.05	2.14	1.39	3.01	1.44	0.22	0.44
60%	2.64	2.94	2.44	1.64	3.52	1.71	0.29	0.54
70%	3.78	4.68	3.28	1.87	7.02	2.04	0.39	0.63
80%	5.91	4.83	4.42	2.18	8.16	2.87	0.54	0.76
90%	12.28	5.00	8.92	3.60	10.96	4.83	0.75	0.88
95%	27.41	5.00	15.82	8.24	11.00	7.73	0.88	0.98
97.50%	49.97	5.00	45.68	15.96	11.00	11.91	0.97	1.00
99%	100.13	5.00	135.00	22.04	11.00	12.00	1.00	1.00

Table 14.9 – Summary Statistics for Top Cut Composites – Silver

Statistic (Silver)	ESTZON							
	101	104	107	110	103	102	200	201
Declustering cell size	20 x 20 x 5 m							
Samples	3,049	11	272	15	24	516	695	526
Minimum	0.03	0.42	0.03	0.63	0.07	0.03	0.03	0.03
Maximum	10.00	7.69	10.00	3.90	4.97	10.50	4.00	3.00
Mean	1.19	2.72	1.94	1.76	1.53	1.92	0.67	0.60
Standard deviation	1.52	2.22	2.48	0.87	1.27	2.36	0.76	0.63
Coefficient of variation	1.28	0.82	1.28	0.49	0.83	1.23	1.13	1.06
Variance	2.31	4.92	6.14	0.76	1.61	5.55	0.57	0.40
Skewness	3.22	1.02	2.07	0.67	1.39	2.20	2.24	1.99
Log samples	3049.00	11.00	272.00	15.00	24.00	516.00	695.00	526.00
Log mean	-0.45	0.66	-0.05	0.44	0.05	0.00	-1.04	-1.09
Log variance	1.49	0.72	1.63	0.26	0.94	1.59	1.64	1.44
Geometric mean	0.64	1.94	0.95	1.55	1.05	1.00	0.35	0.34
10%	0.13	0.50	0.20	0.75	0.27	0.21	0.05	0.05
20%	0.27	0.74	0.35	0.86	0.59	0.41	0.12	0.14

Statistic (Silver)	ESTZON							
	101	104	107	110	103	102	200	201
30%	0.39	0.99	0.53	0.90	0.78	0.61	0.23	0.23
40%	0.55	1.29	0.71	1.24	0.92	0.84	0.31	0.30
50%	0.73	2.18	0.89	1.65	0.99	1.08	0.42	0.39
60%	0.93	2.33	1.18	1.84	1.30	1.46	0.57	0.50
70%	1.20	2.40	1.96	2.02	1.49	1.87	0.76	0.64
80%	1.69	4.63	2.90	2.33	2.22	2.55	1.00	0.89
90%	2.57	5.72	5.19	2.82	3.50	5.28	1.51	1.46
95%	3.98	6.50	8.67	3.10	4.44	8.03	2.21	1.99
97.50%	5.88	7.10	10.00	3.50	4.68	10.00	3.02	2.69
99%	9.18	7.45	10.00	3.74	4.85	10.50	4.00	3.00

14.11 Variography

Experimental semi-variograms (variograms) were generated for gold and silver in Supervisor software for use in grade estimation for all domains that had sufficient samples available using 1 m composites, with extreme grades excluded. Composites had extreme grades excluded rather than capped because extreme grades are not representative of the underlying grade continuity, and indeed generally represent pure nugget effect. Including these, or capping them, can unduly impact the nugget and sills once variograms are back-transformed for use in the estimate. Variography was completed on ESTZON 101, 102, 107, 200, 201 for gold and 101, 102, 200 and 201 for silver (presented in Table 14.10 to Table 14.14). For the smaller footwall domains that did not have sufficient samples to complete variography, the variogram from 101 (the largest footwall domain) was applied during estimation.

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 spherical structures.

The modelled orientations were consistent with the geological understanding of the mineralisation (as depicted in Figure 14.29 to Figure 14.32). Variograms for gold in the mineralised skarns were characterised by a moderate to high nugget and a dominant first structure, isotropic in the major and semi-major directions and averaging 40 m. Approximately 90% of the spatial continuity is within

50 m. This has remained constant since the last MRE, albeit with higher nugget due to more higher grades being intersected in recent drilling. Silver is characterised by longer ranges and lower nugget.

Table 14.10 – 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	0.44	0.42	80	60	180	41	30	10	0.14	148	109	45
Silver	0.35	0.43	80	50	180	41	46	41	0.22	53	47	0.22

Table 14.11 – 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	0.30	0.39	90	40	180	16	57	4	0.31	52	100	19
Silver	0.31	0.32	90	50	180	54	41	2	0.37	116	90	25

Table 14.12 – Variogram Parameters in Datamine ZYZ Rotation – ESTZON 107

Element	C0	C1	Rotation			Range			C2	Range		
			Z	X	Z	Major	Semi	Minor		Major	Semi	Minor
Gold	0.45	0.42	90	40	180	68	108	15	0.13	148	109	45
Silver	-	-	-	-	-	-	-	-	-	-	-	-

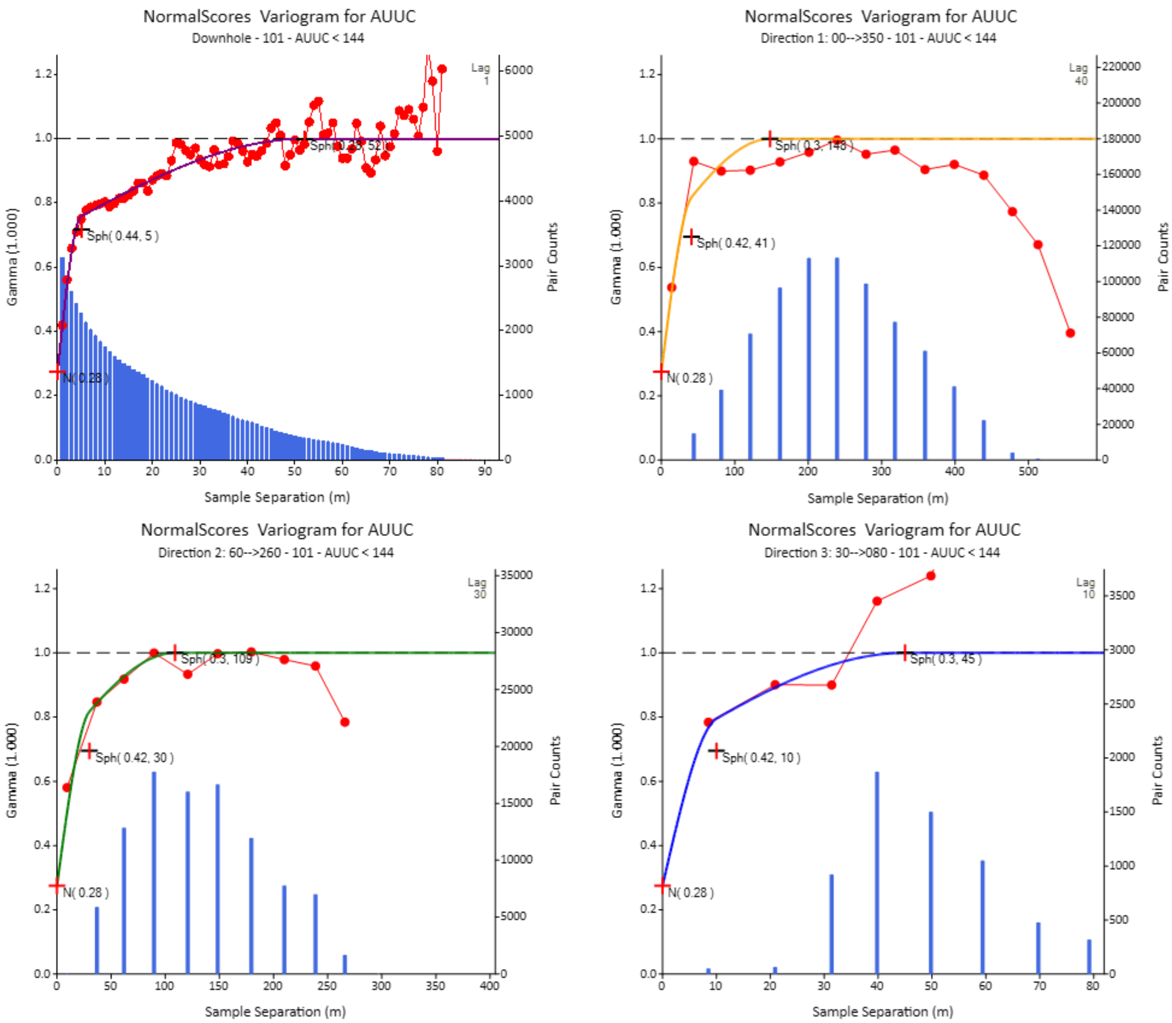
Table 14.13 – Variogram Parameters in Datamine ZYZ Rotation – ESTZON 200

Element	C0	C1	Rotation			Range			C2	Range		
			Z	X	Z	Major	Semi	Minor		Major	Semi	Minor
Gold	0.17	0.28	110	30	130	46	35	4	0.55	252	188	39
Silver	0.39	0.18	120	30	130	46	90	4	0.43	179	105	39

Table 14.14 – Variogram Parameters in Datamine ZXZ Rotation – ESTZON 201

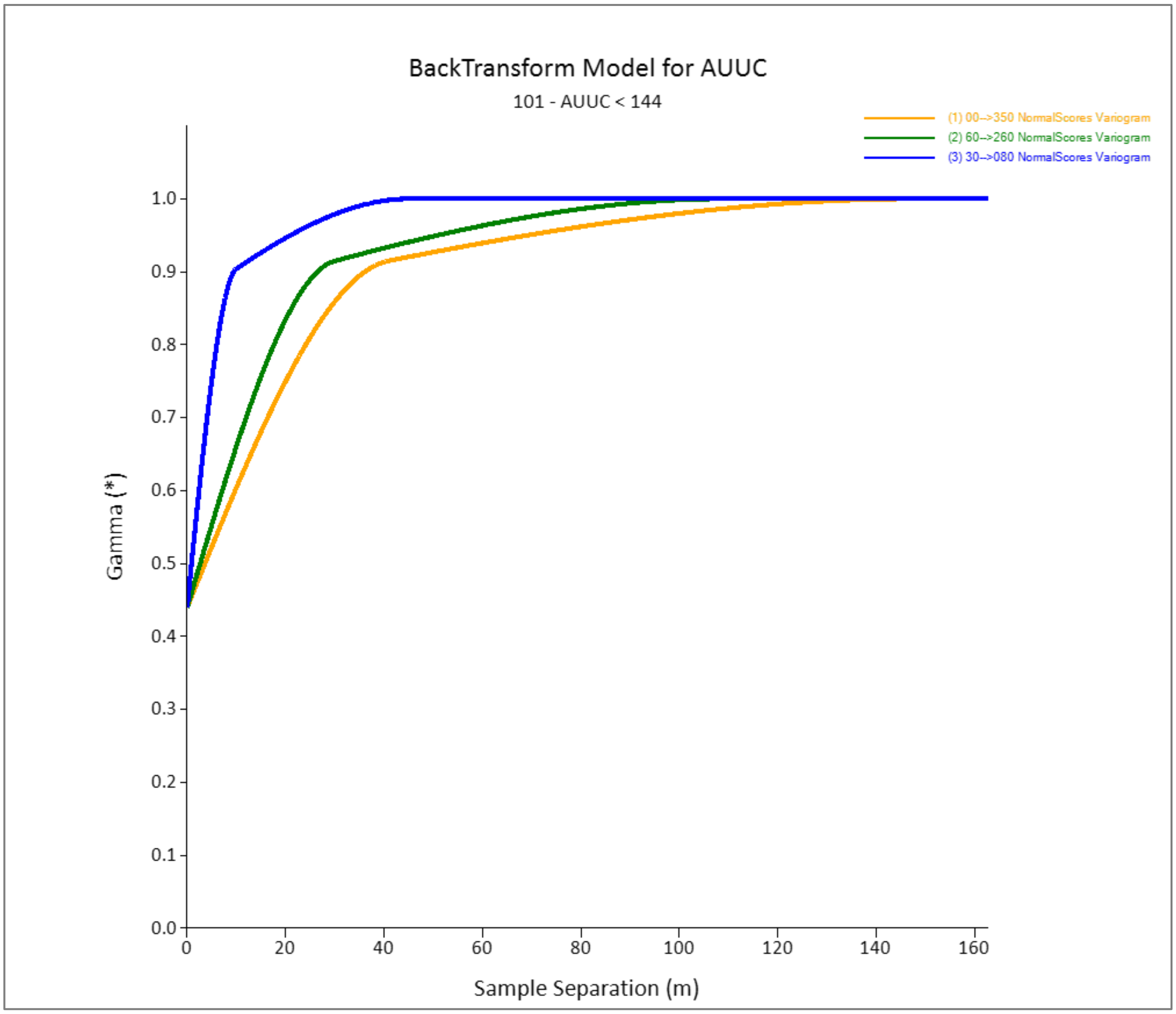
Element	C0	C1	Rotation			Range			C2	Range		
			Z	X	Z	Major	Semi	Minor		Major	Semi	Minor
Gold	0.28	0.32	100	30	180	34	48	2	0.40	82	57	11
Silver	0.30	0.40	80	40	180	87	55	10	0.30	119	89	15

Figure 14.29 – Experimental Semi-Variograms for Gold in ESTZON 101



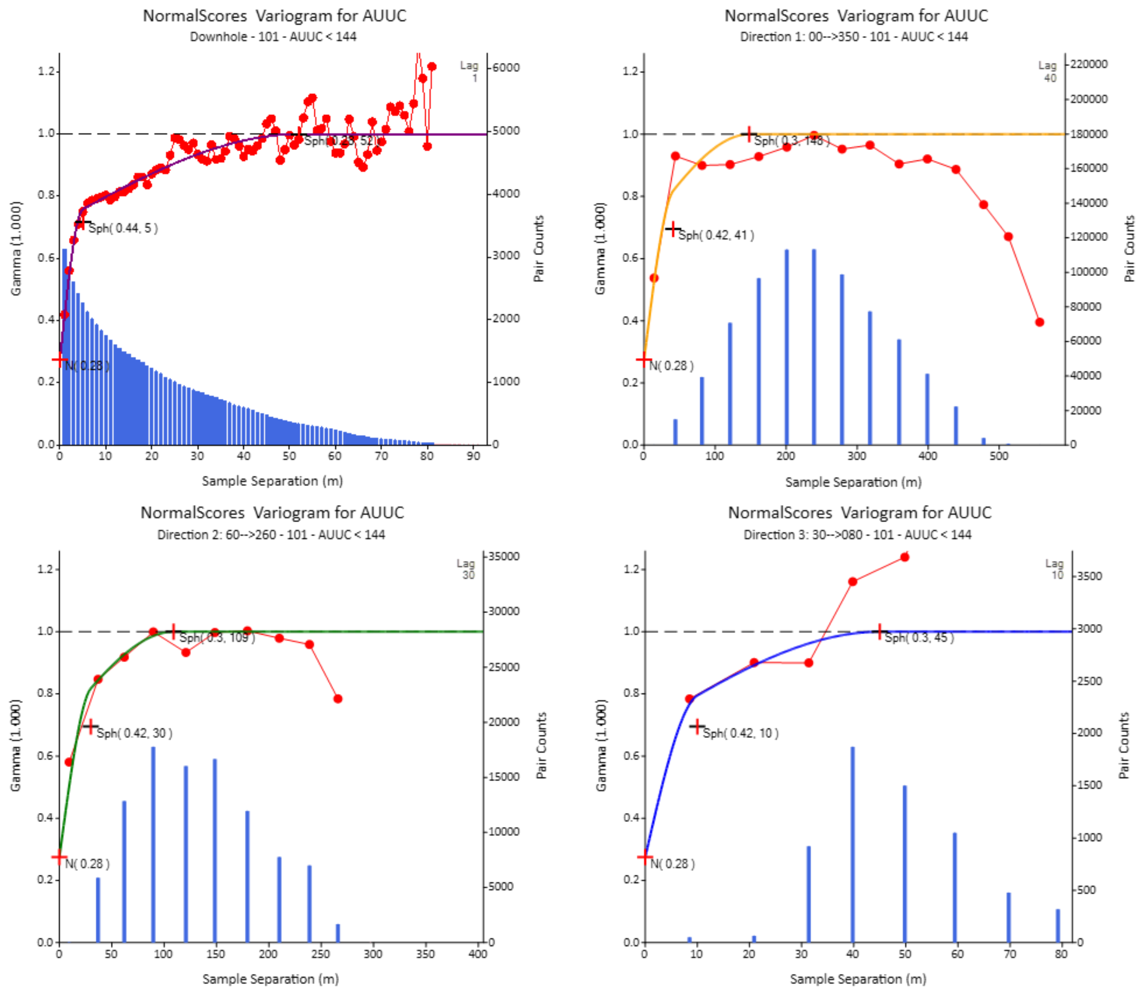
Source: ERM, 2024

Figure 14.30 – Variogram Model for Gold in ESTZON 101



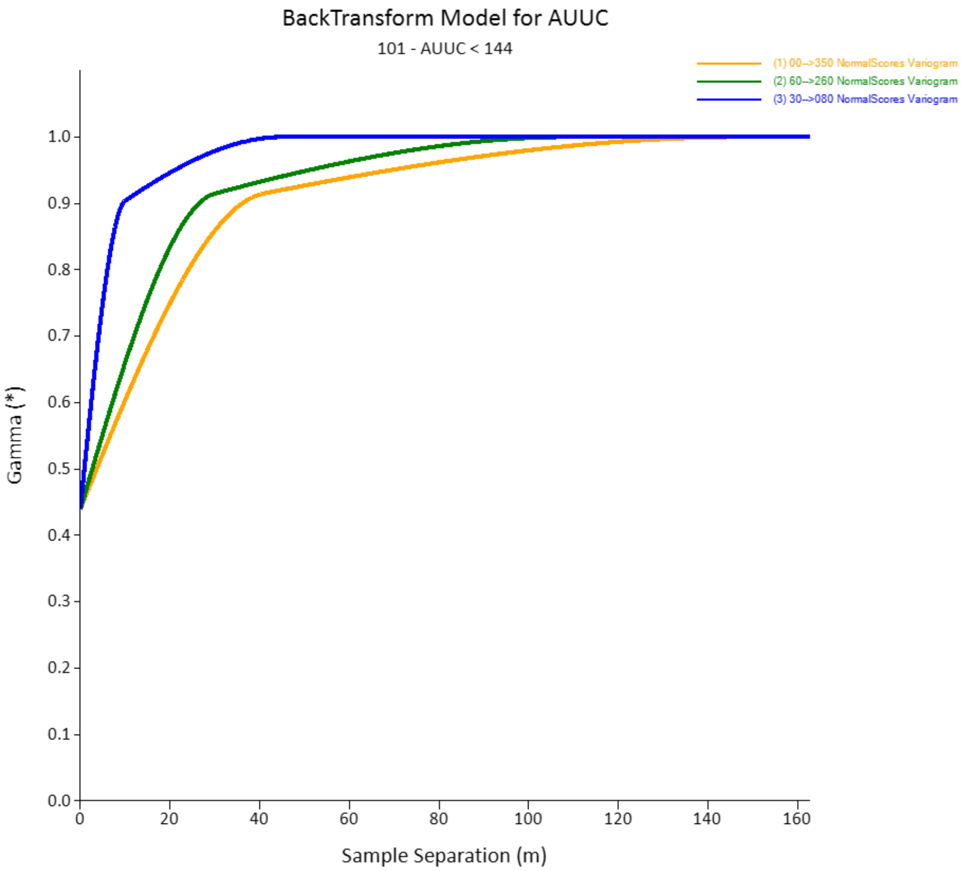
Source: ERM, 2024

Figure 14.31 – Experimental Semi-Variograms for Silver in ESTZON 101



Source: ERM, 2024

Figure 14.32 – Variogram Model for Silver in ESTZON 101



Source: ERM, 2024

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.

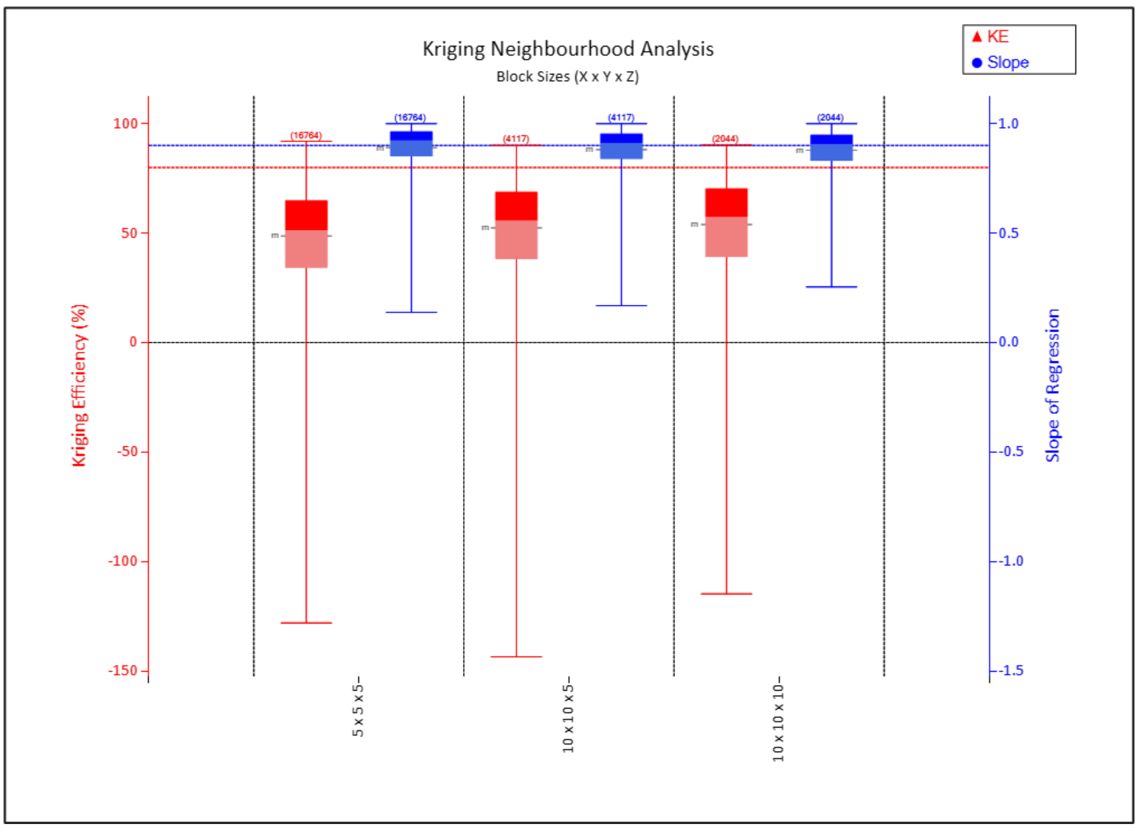
2. choosing a single block location representing a well-informed block, a moderately informed block and a poorly informed block and reviewing the kriging quality statistics.

A block size of 10 m x 10 m x 5 m was chosen which is compatible with the proposed mining method and also results in good slopes of regression and kriging efficiencies when compared to other blocks reviewed (Figure 14.33).

Minimum and maximum composite selection of 12 and 32 were chosen (Figure 14.34). High numbers were used in the previous MRE (20/40) to intentionally smooth the estimate, given the presence of coarse gold and associated risks in being able to precisely locate blocks above cut-off. Further work was completed by DPM in 2024 to more fully understand the presence of coarse gold, and while it certainly is present, the Čoka Rakita deposit is considered a 'Medium' type coarse gold deposit which will have 'Medium' sampling problems (Dominy, 2024). While this needs to be considered when judging the level of selectivity possible in the deposit, it was determined that a certain measure of selectivity in the search would be beneficial in more accurately estimating the tonnes and grade above cutoff.

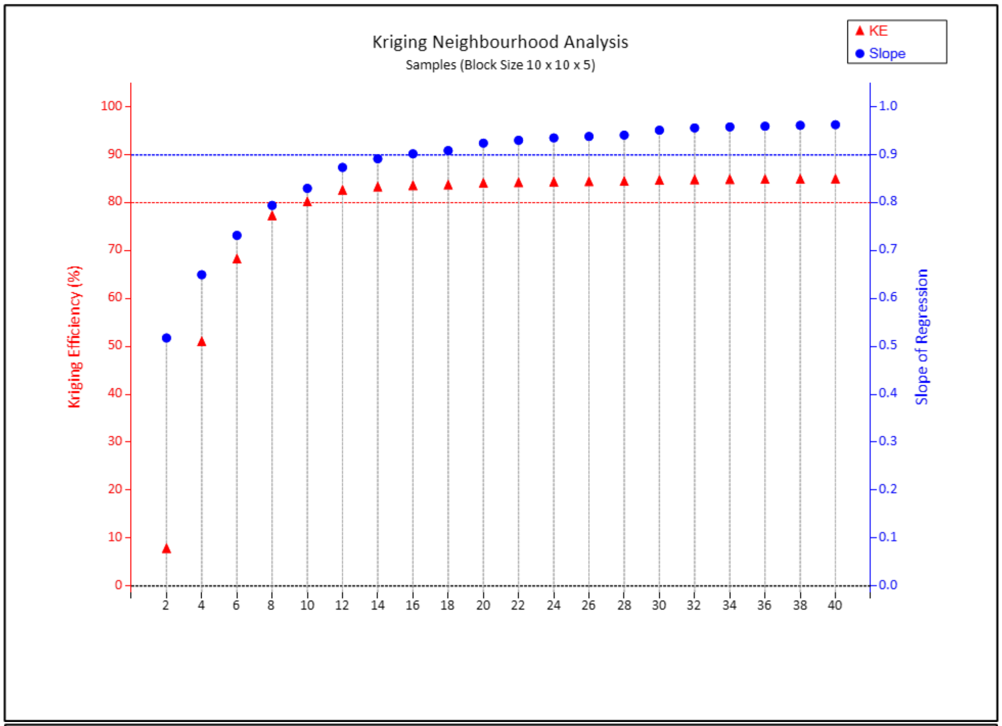
The extent of the second search pass was chosen to align broadly with the variogram ranges. The first was chosen to reflect the closer drill spacing (20 – 30 m).

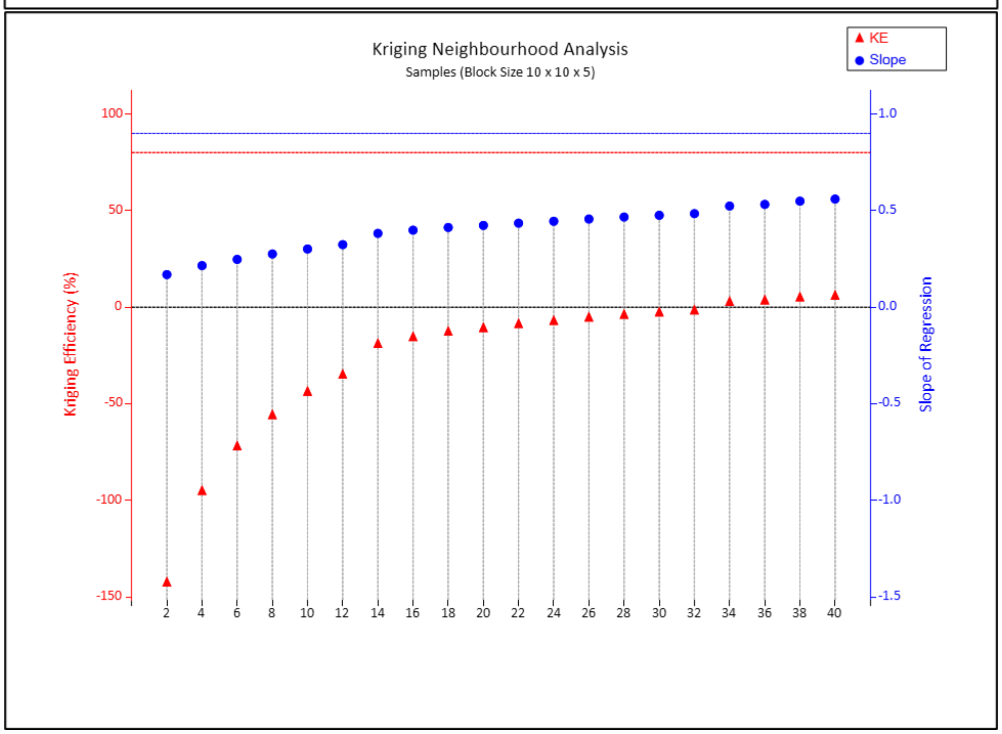
Figure 14.33 – KNA Block Size Review



Source: ERM, 2024

**Figure 14.34 – KNA Sample Search Strategy Review
('Good' Block – Above, 'Bad' Block – Below)**





Source: ERM, 2024

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 compared well. The block model prototype is presented in Table 14.15.

Table 14.15 – Block Model Prototype

Dimension	**Minimum** (m)	**Maximum** (m)	**Extent** (m)	**Block Size**	
				Parent Cell	Sub-Cell
Easting	572,600	573,200	600	10	0.5
Northing	4,895,500	4,896,400	900	10	0.5
Elevation	250	1,000	750	5	1

Table 14.16 – Block Model Attributes – Final Model (cr_md241001.mre.dm)

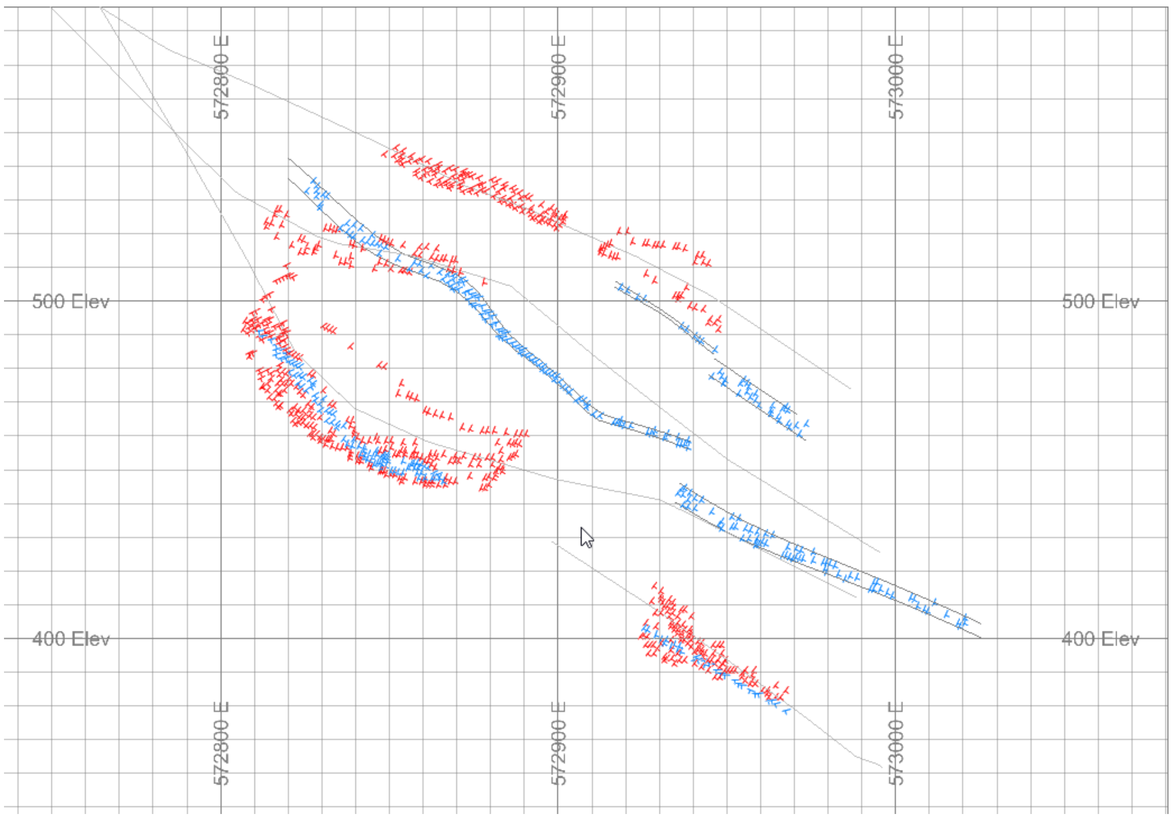
Field Description	**Field Name**	**Type/Unit**	**Values/Meaning**	
Gold estimate	AU	Numeric: g/t	Variable	
Silver estimate	AG	Numeric: g/t	Variable	
Arsenic estimate	AS	Numeric: ppm	Variable	
Sulphur estimate	S	Numeric: %	Variable	
Copper estimate	CU	Numeric: %	Variable	
Mineralisation domains	MINZON	Numeric (Integer)	101-301-303-401-403-405-406-501-503-601-603-605	Hanging wall mineralisation (including sills and internal waste)
			102-302-402-404-502-602	Footwall mineralisation (including sills and internal waste)
			701	Monzonite/EMP hosted mineralisation
			901-910	Internal waste units in footwall
			999	Waste
Oxidation	OXIDE	Numeric (Integer)	3	Strongly oxidised
			2	Partially oxidised
			1	Fresh
Mineral Resource classification	CLASS	Numeric (Integer)	2	Indicated
			3	Inferred
			9	Unclassified

Field Description	Field Name	Type/Unit	Values/Meaning	
Density	DENSITY	Numeric (t/m^3)	2.71	Sequence felsic debris flow deposit unit
			2.65	VHM
			2.78	Marls – SFD
			Estimated Default 2.99	Marls
			Estimated Default 2.97	S1/S2
			Estimated Default 2.69	Late Sills
			2.67	EMP
			2.64	Quartzite
			2.73	Marble
			2.68	Monzonite
			Estimated Default 3.00	FW mineralisation
			Estimated Default 3.10	HW mineralisation
			Estimated Default 2.69	Monzonite/EMP hosted mineralisation
Geology	GEOL	Numeric (Integer)	1	Sequence felsic debris flow deposit unit
			2	VHM
			3	Marls – SFD
			4	Marls
			5	S1/S2
			6	Sills
			7	Early mineralised porphyry
			8	Quartzite
			9	Marble
			10	Monzonite
Pyrrhotite-Magnetite geometallurgical domain	PYR_MT	Numeric (Integer)	0	Outside the domain
			1	Inside the domain
Mineral Resource	MRE	Numeric (Integer)	1	In MRE (supported by RPEEE)
			0	Outside of MRE (not classified)

14.14 Dynamic Anisotropy

Dynamic anisotropy (DA) was implemented in Studio RM to allow the search ellipsoid to be locally oriented according to variable dips and dip directions of the estimation domains. Dips and dip directions were estimated into the model using wireframes and used in the estimate to avoid a single orientation being applied to a domain that undulates.

Figure 14.35 – Cross Section showing DA Surfaces and Resultant Model Dip and Dip Direction Displayed as Ticks



Source: ERM, 2024

14.15 Sensitivity Analysis – 2024

Sensitivity analysis on assumptions and parameters used in the estimate was completed by ERM in 2024 prior to the MRE update. Sensitivities tested included:

- Top cuts applied to composites and grade capping using distance thresholds within the estimate.
- Composite length.
- Block size.
- Search parameters.

- Domaining strategy, including the interpretation of a 'very high grade' (VHG) domain in the core of the deposit, characterised by grades exceeding approximately 10 g/t Au.

The conclusions of the sensitivity analysis were as follows:

1. The tonnage and grade estimate is highly sensitive to:
   - Use or non-use of VHG domain. While there is some conceptual evidence of the domain having geological support (e.g. fault-bounded, broad continuity of high grades, capped by mafic sills which may act as potential fluid traps), there remains insufficient empirical evidence from the drill core that this zone hangs together.
   - When using VHG domain, use of hard vs semi-hard boundaries (to varying degrees).
   - Top cuts imposed in individual domains.
   - Use of 1m (lower grades) vs 2 m (higher grade) composites.
2. The tonnage and grade estimate is moderately sensitive to:
   - Use of a capping distance.
   - Use of more selective searches.
   - Block size of 10 x 10 x 5 vs 10 x 10 x 10 m (metal moderately sensitive, tonnes and grade are more impacted).
3. At this point, the scenarios that are currently more supported by geological evidence and logging is considered to be the skarn domain estimate (used in the PFS), rather than the VHG domain. 1 m composites were preferred to 2 m because they more closely align with length weighted grades of raw assays, due to the predominant sampling length of 1 m.

The recommendations from this analysis were implemented for this MRE and are described in relevant sections. There were as follows:

- The undifferentiated skarn domain, and not the VHG domain, should be used as the basis for the PFS MRE due to current level of geological logging and interpretation, and understanding of continuity. This continues the methodology used in the PEA.
- Block size to be changed from 10 x 10 x 10 m (X x Y x Z) to 10 x 10 x 5 m, informed by favourable KNA and the benefits of modelling additional selectivity in the Z dimension.
- Composite length to remain 1 m, aligning with the dominant sampling length.
- Grade capping (top cuts) to be imposed for Au as in the PEA, but an additional within-estimate capping strategy based on distance is recommended to be implemented, to further constrain the influence of very high grades.
- A more selective search to be implemented to align with infill drilling (smaller ranges) and lower minimum and maximum samples in a balanced approach to somewhat reduce smoothing, while cognisant of the presence of coarse gold.

14.16 Grade Estimation

Grades were estimated into mineralisation and waste blocks using Ordinary Kriging in Datamine Studio RM. A three-pass search strategy was used. Table 14.17 shows the percentage of blocks estimated in each search pass and Figure 14.36 shows the model coloured by search pass. Searches and minimum/maximum sample numbers were supported by KNA (Section 14.12), and were reduced compared to the November 2023 Maiden MRE to account for tighter infill drilling and increased selectivity to avoid smoothing of high grade areas into lower grade areas. 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 14.17 – Percentage of Blocks Estimated in Each Search Pass

Variable	Search Pass	% Volume
Gold	1	62
	2	33
	3	5
Silver	1	54
	2	38
	3	8

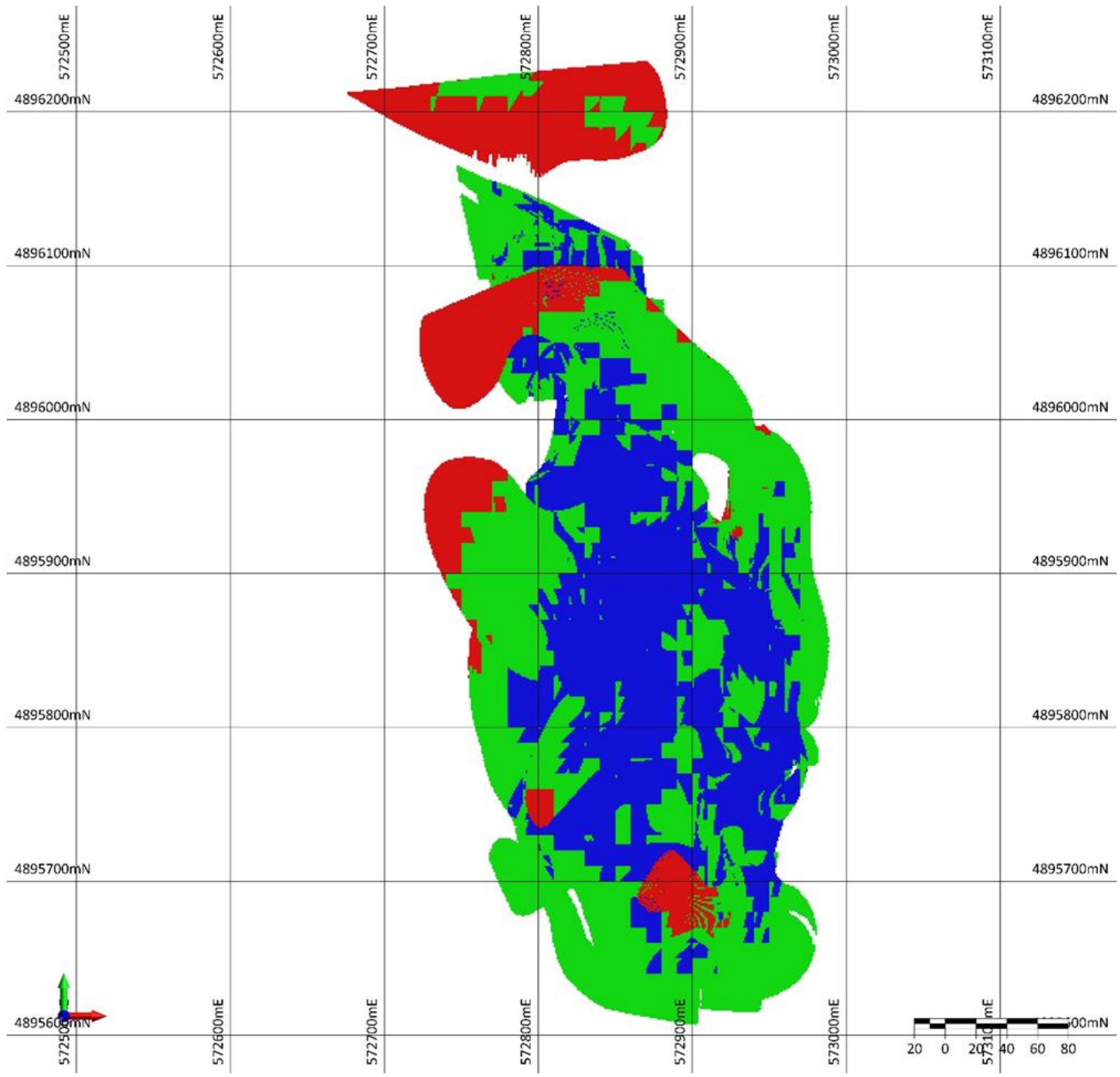
Discretisation of 4 x 4 x 5 (X x Y x Z) was used to stabilise the block variance in the estimate – 5 in the Z direction chosen because 1 m composites were used, and best practice is to align composites with the height of the block in the Z direction.

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 evaluated during validation and classification but are removed from the final model issued to engineers for downstream use.

The sample search neighbourhood is presented in Table 14.18. 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 (IDW2) to assess sensitivities.

Figure 14.36 – Plan View of the Model, Looking Northeast, Coloured by Search Pass for Gold



Source: ERM, 2024

Table 14.18 – Sample Search Neighbourhood for Grade Estimates

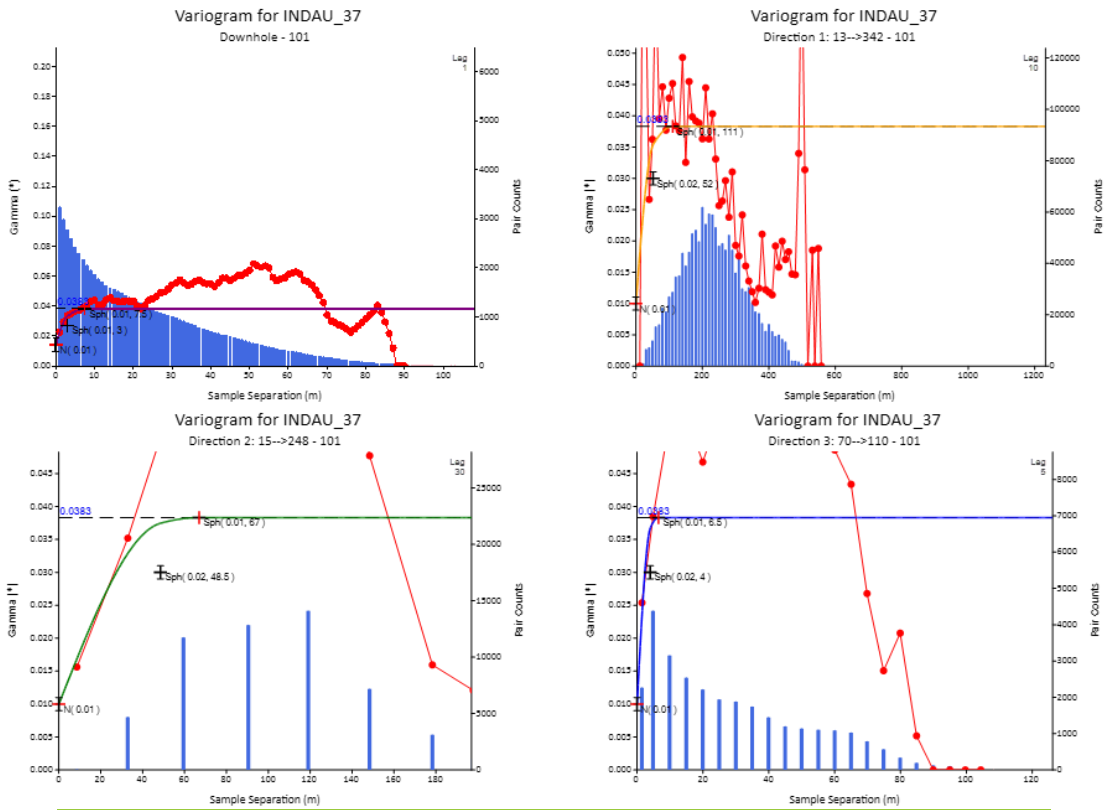
Domains	Search Pass	Search 1	Search 2	Search 3	Minimum Composites	Maximum Composites	Maximum per DH	In-Estimate Capping
Gold*, silver estimate (and arsenic, sulphur)								
101, 110	1	45	30	10	12	30	5	37 g/t Au, >30 m
	2	135	90	30	12	30		
	3	315	210	70	5	30		
102	1	45	30	10	12	30	5	None
	2	90	60	20	12	30		
	3	180	120	40	5	30		
103	1	45	30	10	12	30	5	4 g/t Au, >30 m
	2	90	60	20	12	30		
	3	180	120	40	5	30		
104	1	30	30	10	6	30		None
	2	60	60	20	6	30		
	3	120	120	40	6	30		
107	1	75	30	10	10	30	5	37 g/t Au, >15 m
	2	150	60	20	10	30		
	3	225	90	30	5	30		
200, 201	1	75	50	10	16	30	4	None
	2	150	100	20	16	30		
	3	375	250	50	8	30		
Waste domains	1	30	30	10	12	30	5	None
	2	60	60	20	12	30		
	3	300	300	100	8	30		
Copper Estimate								
All domains	1	45	30	10	12	30	5	None
	2	135	90	30	12	30		
	3	315	210	70	5	30		

*in-estimate capping thresholds imposed only on gold estimates in selected domains

Within the estimate, a further distance restriction of 30 m was imposed on top cut composites that exceeded 37 g/t Au. The grade was chosen by reviewing a set of simple over cross variograms at the following percentiles – 70th, 75th, 80th, 85th, 90th, 95th, 97.5th, and 99th. The structures of these variograms were reviewed to assess at what percentile the structure deteriorates, which helps identify the grade above which there is a very limited or no spatial relationship. This grade was identified as 37 g/t Au, which equates to the 95th percentile (Figure 14.37).

An indicator was then generated at this grade to model the variogram to identify a suitable distance beyond which the grade should be capped within the estimate (Figure 14 36). 30 m was chosen to be well within the ranges of the 1st and 2nd directions modelled.

Figure 14.37 – Modelled Variogram Ranges for the 37 g/t Au Indicator

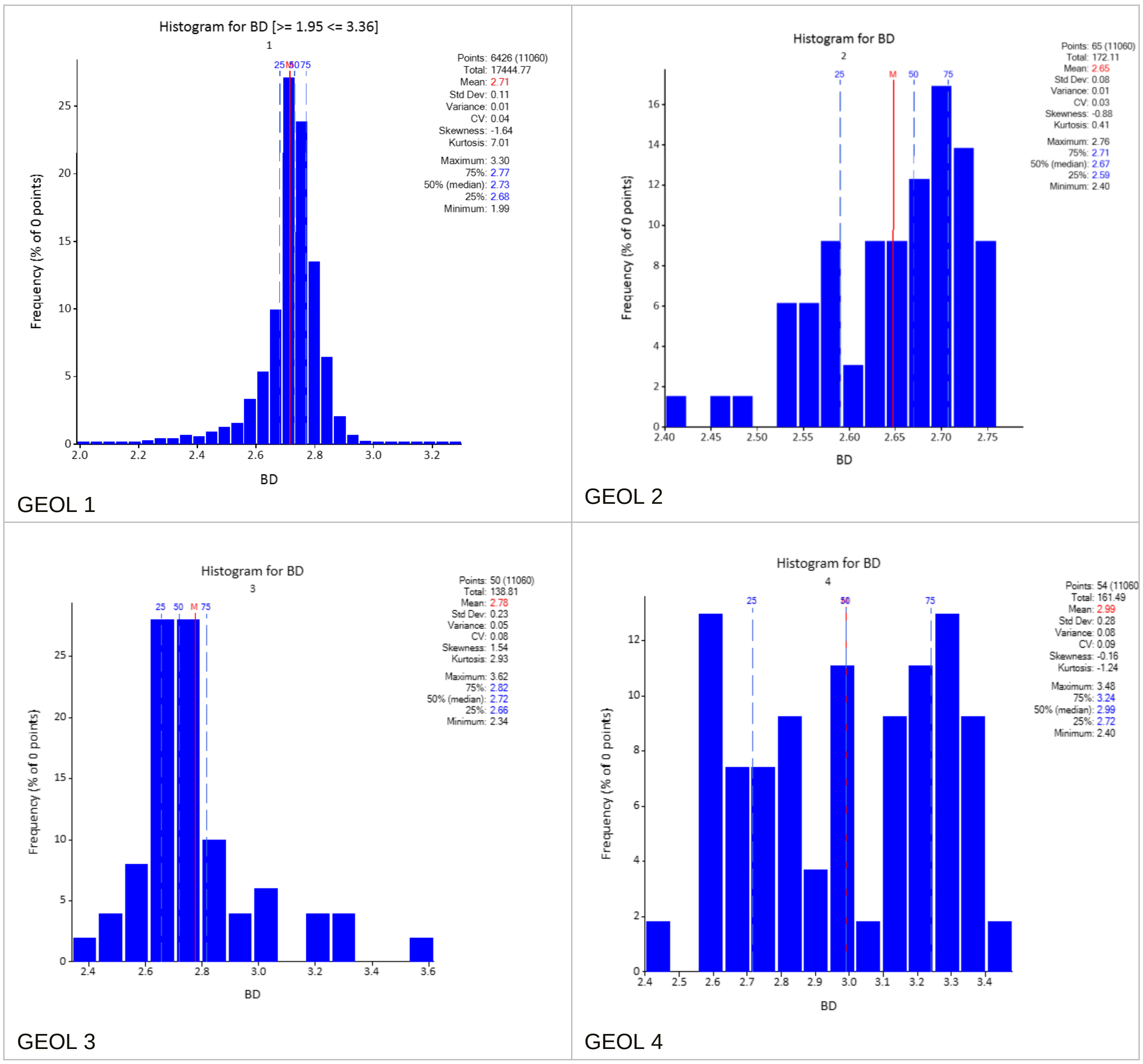


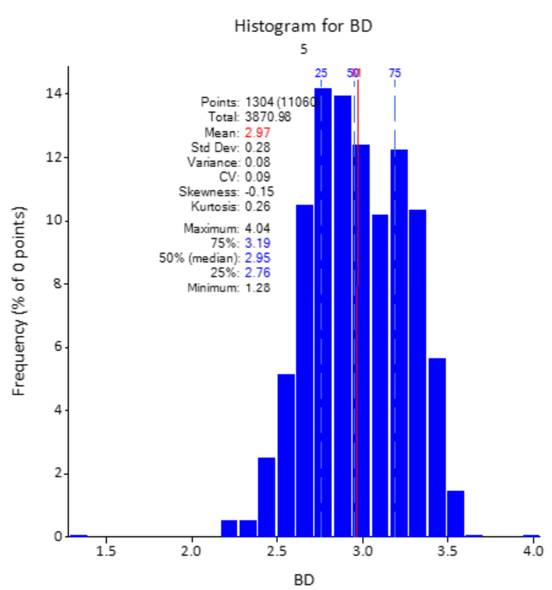
Variable	C0	C1	Range			C2	Range		
			Major	Semi	Minor		Major	Semi	Minor
IND_37	0.01	0.02	52	48.5	4	0.0008	111	67	6.5

14.17 Bulk Density Estimate/Assignment

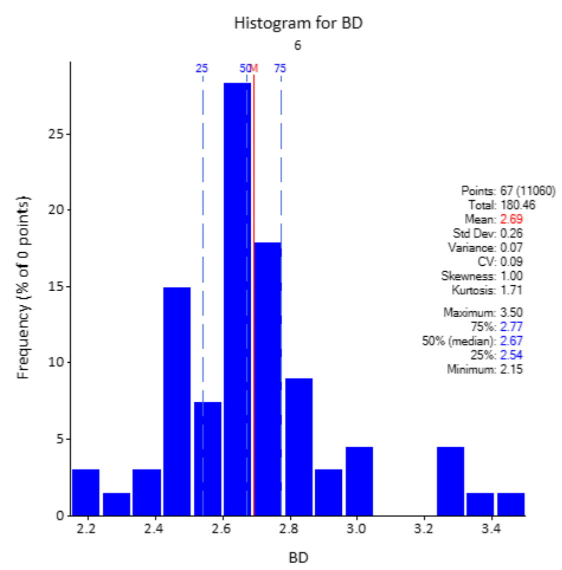
In-situ dry bulk density (BD) measurements were analysed by reviewing histograms by modelled lithology (Figure 14.38). Some zones had a narrow range of measured densities, in which case the mean BD was assigned to blocks within that lithology. Certain lithologies had a wide range of BD values, often reflective of the degree of alteration and in particular skarnification that the rock had been subject to. In those cases, BD was estimated using Ordinary Kriging to reflect the internal variability observed in the given unit.

Figure 14.38 – Histograms Showing Measured BD

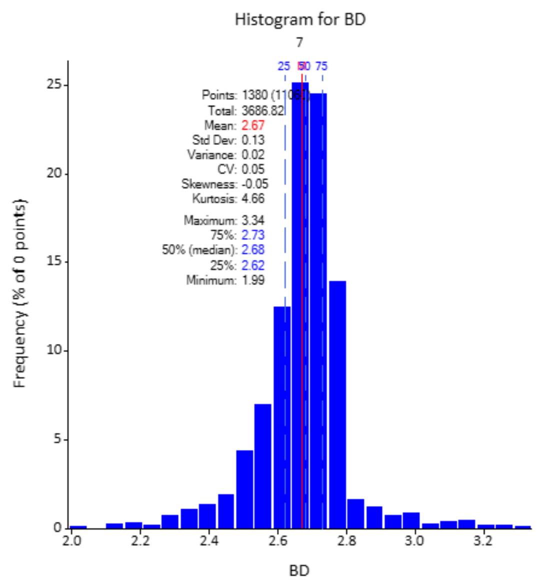




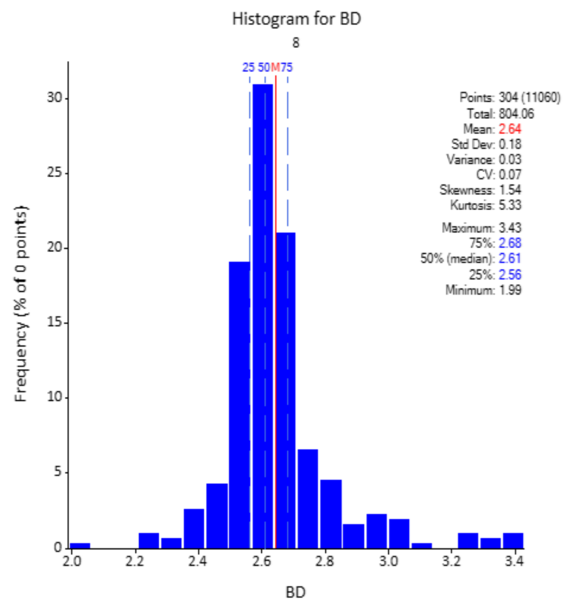
GEOL 5



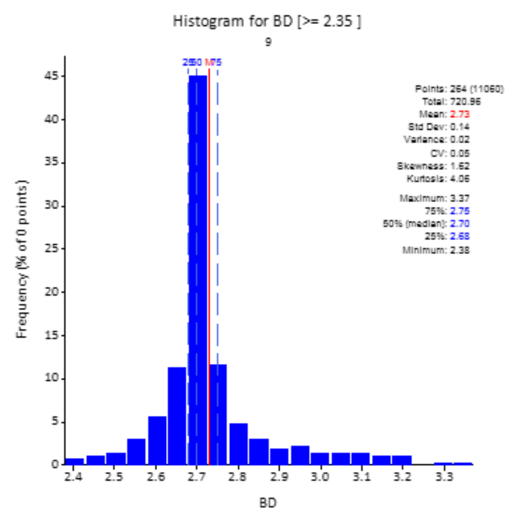
GEOL 6



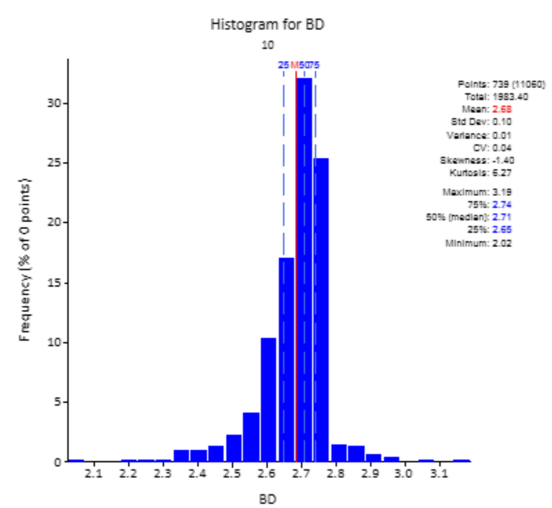
GEOL 7



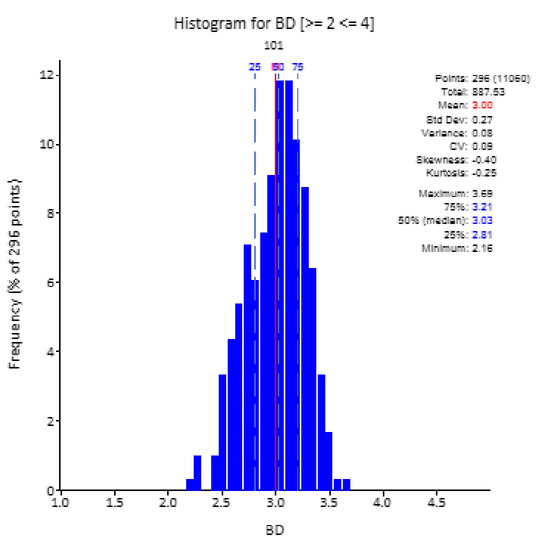
GEOL 8



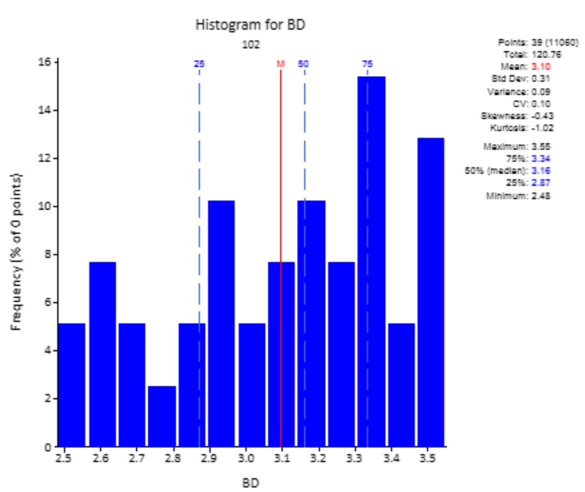
GEOL 9



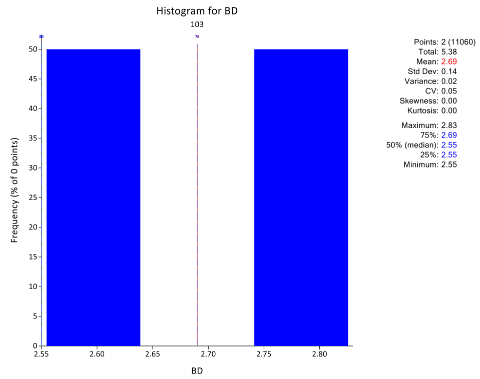
GEOL 10



ESTZON 101



ESTZON 102



ESTZON 103

Source: ERM, 2024

Table 14.19 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 (rather than capped) when reviewing histograms and preparing estimation composites, since in most cases they are likely to be measurement errors instead of true outliers.

Table 14.19 – Methods to Assign BD by Lithology

Domain	Description	Method	Mean assigned if applicable (t/m³)
GEOL=1	SFD	Mean BD assigned	2.71
GEOL=2	VHM	Mean BD assigned	2.65
GEOL=3	Marls – SFD	Mean BD assigned	2.78
GEOL=4	Marls	Estimated using OK	2.99 (only assigned to small number of un-estimated blocks)
GEOL=5	S1/S2	Estimated using OK	2.97 (only assigned to small number of un-estimated blocks)
GEOL=6	Mafic sills	Estimated using OK	2.69 (only assigned to small number of un-estimated blocks)
GEOL=7	Early mineralised porphyry	Mean BD assigned	2.67
GEOL=8	Quartzite	Mean BD assigned	2.64
GEOL=9	Marble	Mean BD assigned	2.73
GEOL=10	Monzonite	Mean BD assigned	2.68
ESTZON=101	Mineralised skarn footwall	Estimated using OK	-
ESTZON=102	Mineralised skarn hanging-wall	Estimated using OK	-
ESTZON=103	Mineralisation in monzonite/emp	Estimated using OK	

For those domains where BD was estimated, the search engaged is presented in Table 14.20. 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 14.23), visual inspection and swath plots (Figure 14.46 to Figure 14.48).

Table 14.20 – Sample Search Neighbourhood for BD Estimate

Search Pass	Search 1	Search 2	Search 3	Minimum Composites	Maximum Composites	Maximum per DH
1	75	60	10	12	28	5
2	150	120	20	12	28	
3	525	420	70	6	30	

14.18 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 14.21).
- Visual inspection of estimated grades in plan and in cross sections and comparison with the input composites (example cross sections presented in Figure 14.39 to Figure 14.42).
- Check for global bias by estimation pass and by domain – comparison of estimated and declustered composite statistics (Table 14.22).
- Check for local bias considering the supporting information – analysis of local trends in estimates using swath plots (Figure 14.43 to Figure 14.48).
- Checks to ensure the boundary conditions between estimation domains are honoured.

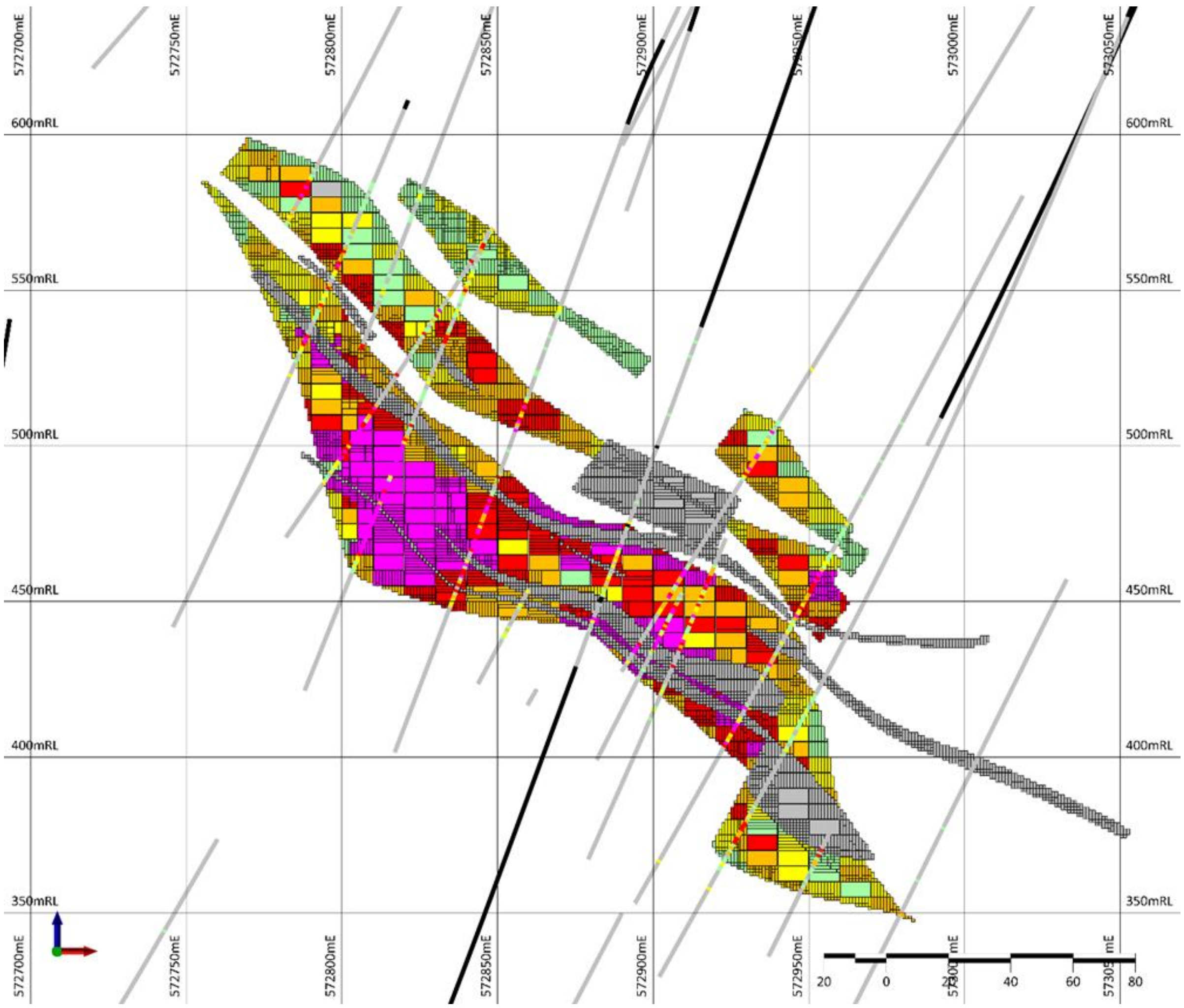
Volume checks show that the model has been built correctly based on the wireframes used. Grades are comparable on a global and domain-by-domain basis between estimation composites and blocks and are well within the 10% threshold that is considered reasonable.

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 14.21 – Block Model vs. Wireframe Volumes

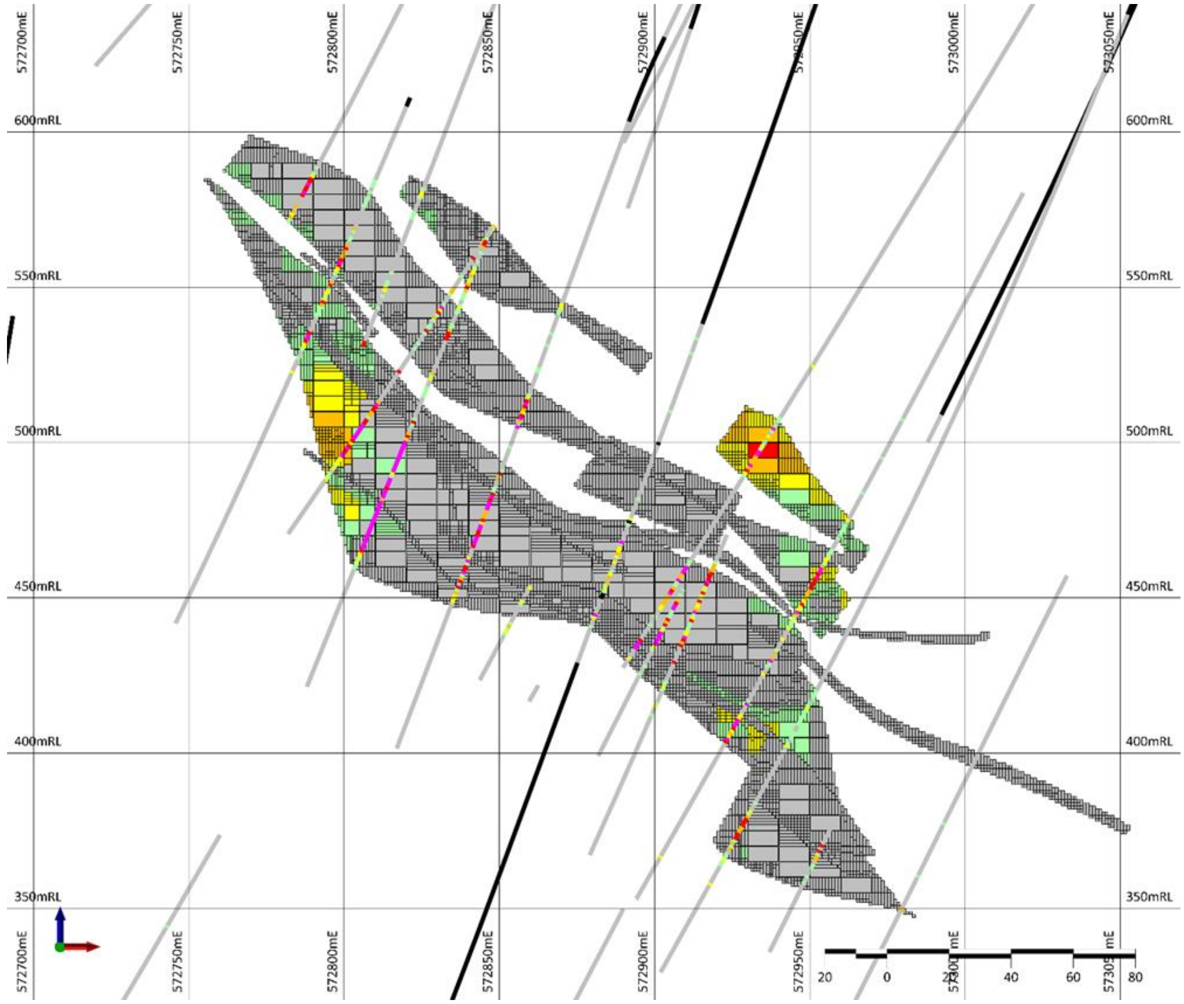
Domain	Wireframe volume	Model volume	% Difference
101+104+107+110	2,628,562	2,626,357	0%
103	20,557	20,568	0%
102	378,122	378,178	0%
200	744,940	749,147	+1%
201	336,607	333,104	-1%

Figure 14.39 – Cross Section at 4895820 m (±15 m) Looking North Showing Estimated Gold Grade and Input Composites



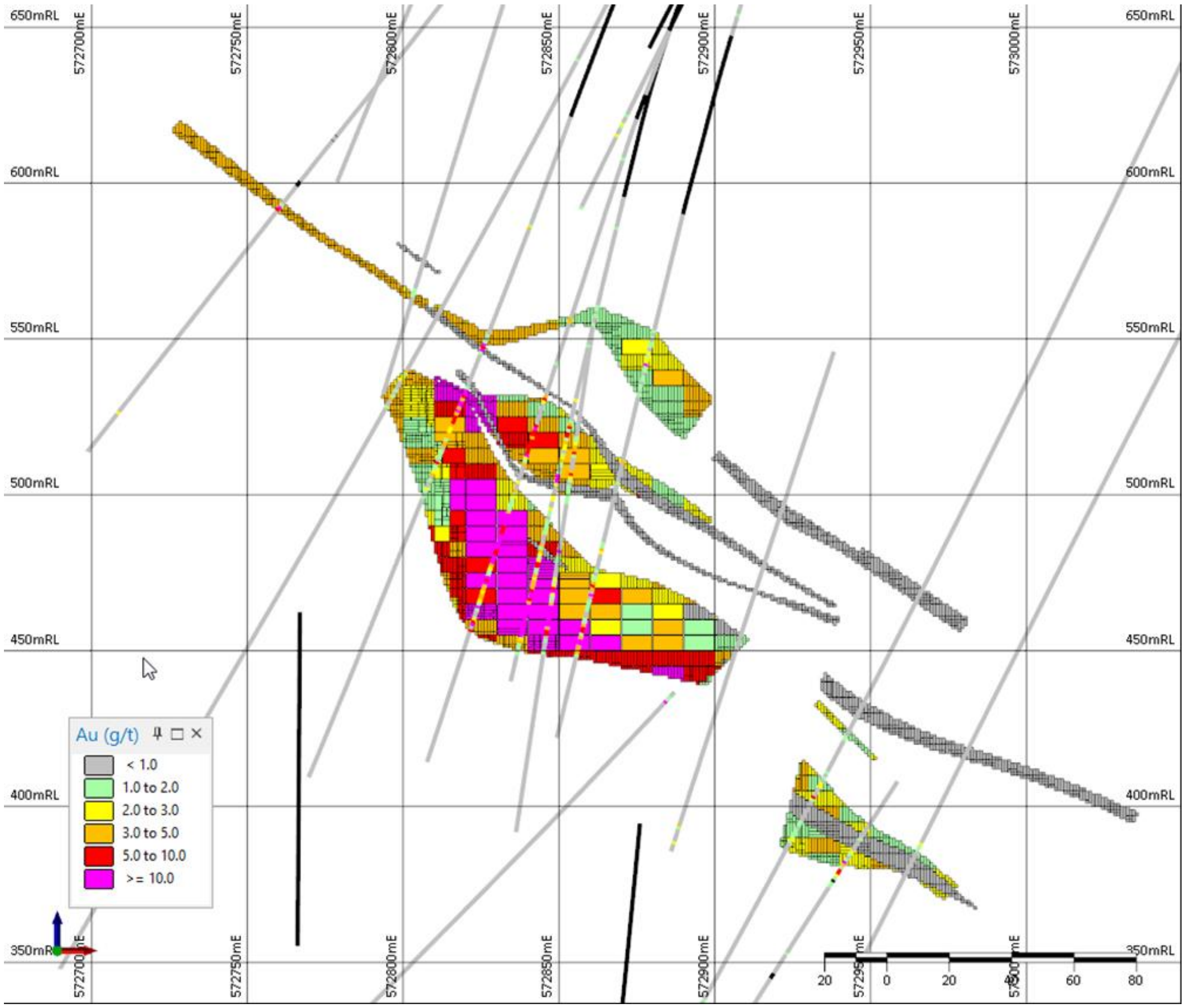
Source: CSA Global, 2023

Figure 14.40 – Cross Section at 4895820 m (±15 m) looking North Showing Estimated Silver Grade and Input Composites



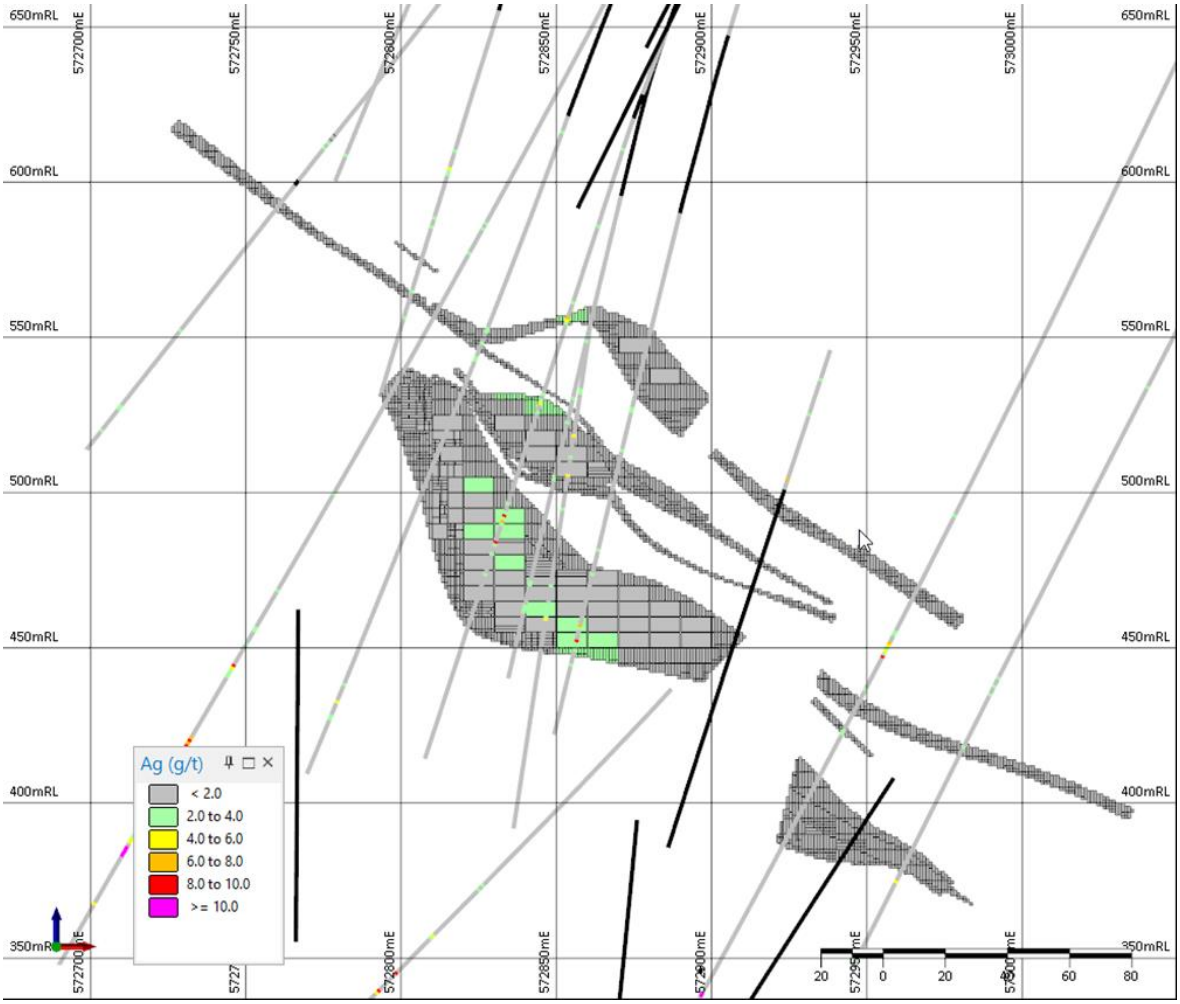
Source: CSA Global, 2023

Figure 14.41 – Cross Section at 4895940 m (±15 m) Looking North Showing Estimated Gold Grade and Input Composites



Source: CSA Global, 2023

Figure 14.42 – Cross Section at 4895940 m (±15 m) Looking North Showing Estimated Silver Grade and Input Composites



Source: CSA Global, 2023

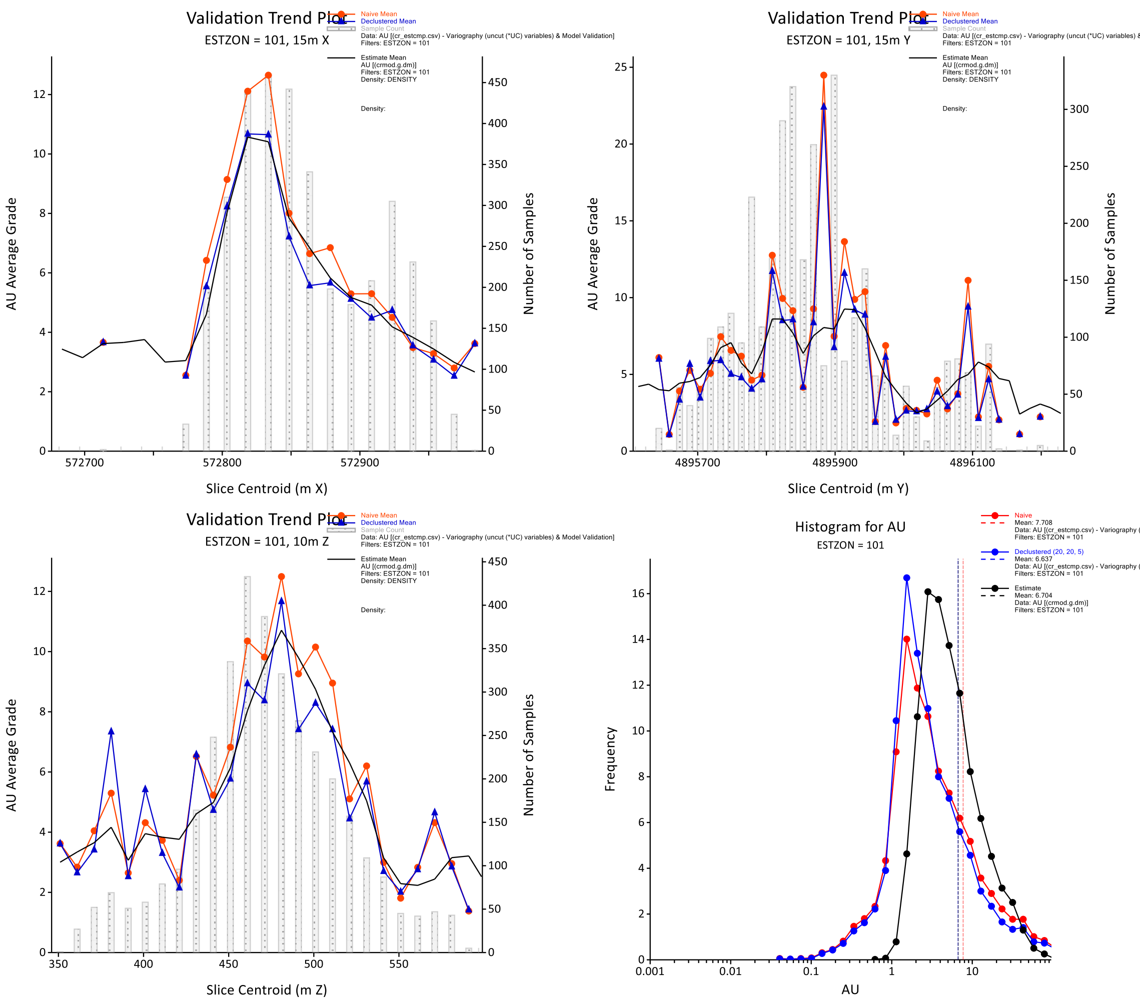
Table 14.22 – Global Statistics – Comparison of Block and Composite Grades

Variable	Domain name	Domain	Declustered composite grade	Block grade	% Difference	% Metal[1]
Au	ESTZON	101	6.64	6.70	1%	88%
	ESTZON	107	6.20	6.43	4%	7%
	ESTZON	102	2.28	2.34	3%	4%
Ag	ESTZON	101	1.19	1.15	-4%	73%
	ESTZON	107	1.94	1.78	-8%	10%
	ESTZON	102	1.92	1.60	0%	15%
Density	EZON_DEN	4	2.99	2.93	-2%	

Variable	Domain name	Domain	Declustered composite grade	Block grade	% Difference	% Metal[1]
	EZON_DEN	5	2.97	2.96	0%	
	EZON_DEN	6	2.69	2.71	1%	
	EZON_DEN	101	3.00	3.00	0%	
	EZON_DEN	102	3.10	3.13	1%	

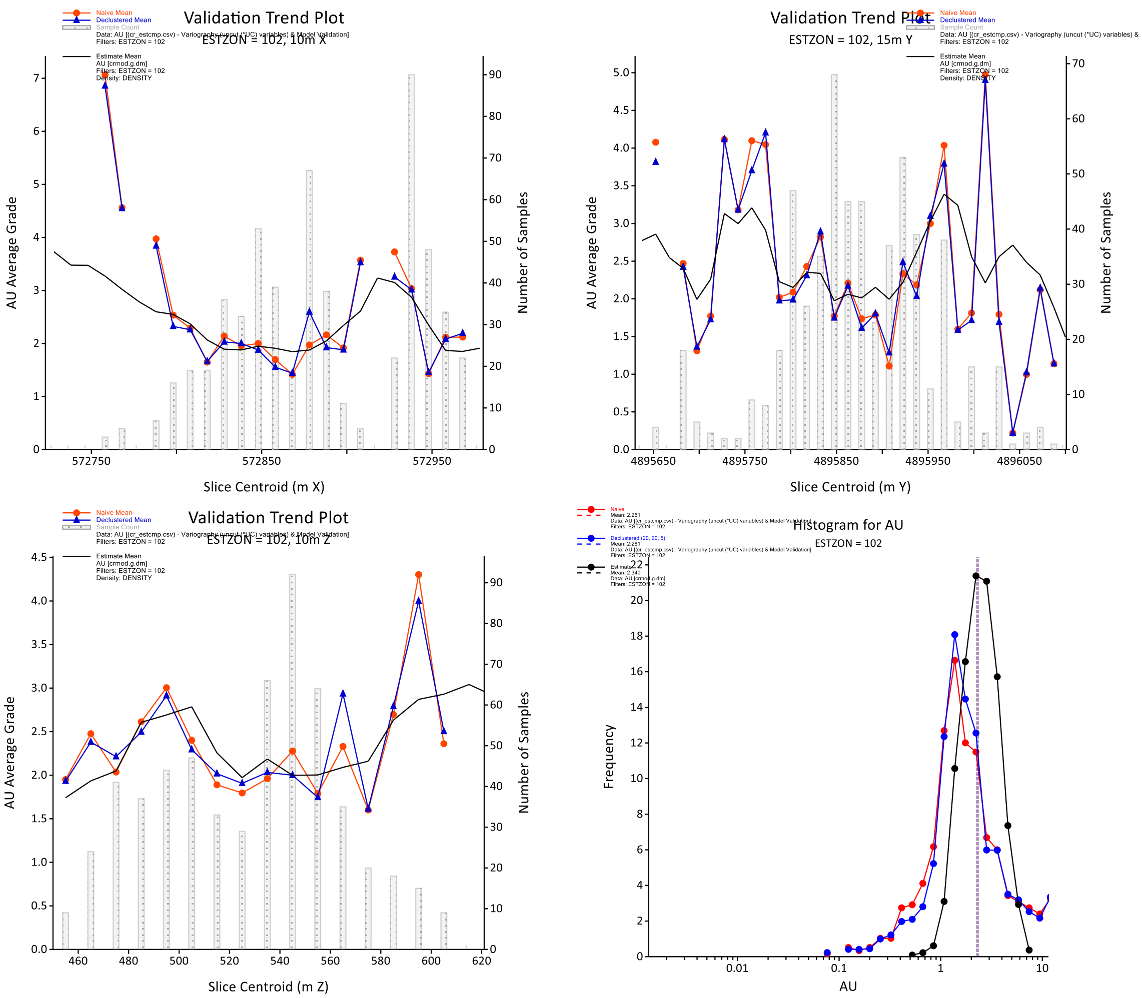
[1] % metal represents how significant a particular domain is in terms of metal content.

Figure 14.43 – Swath Plots and Log Histogram for Au ESTZON 101



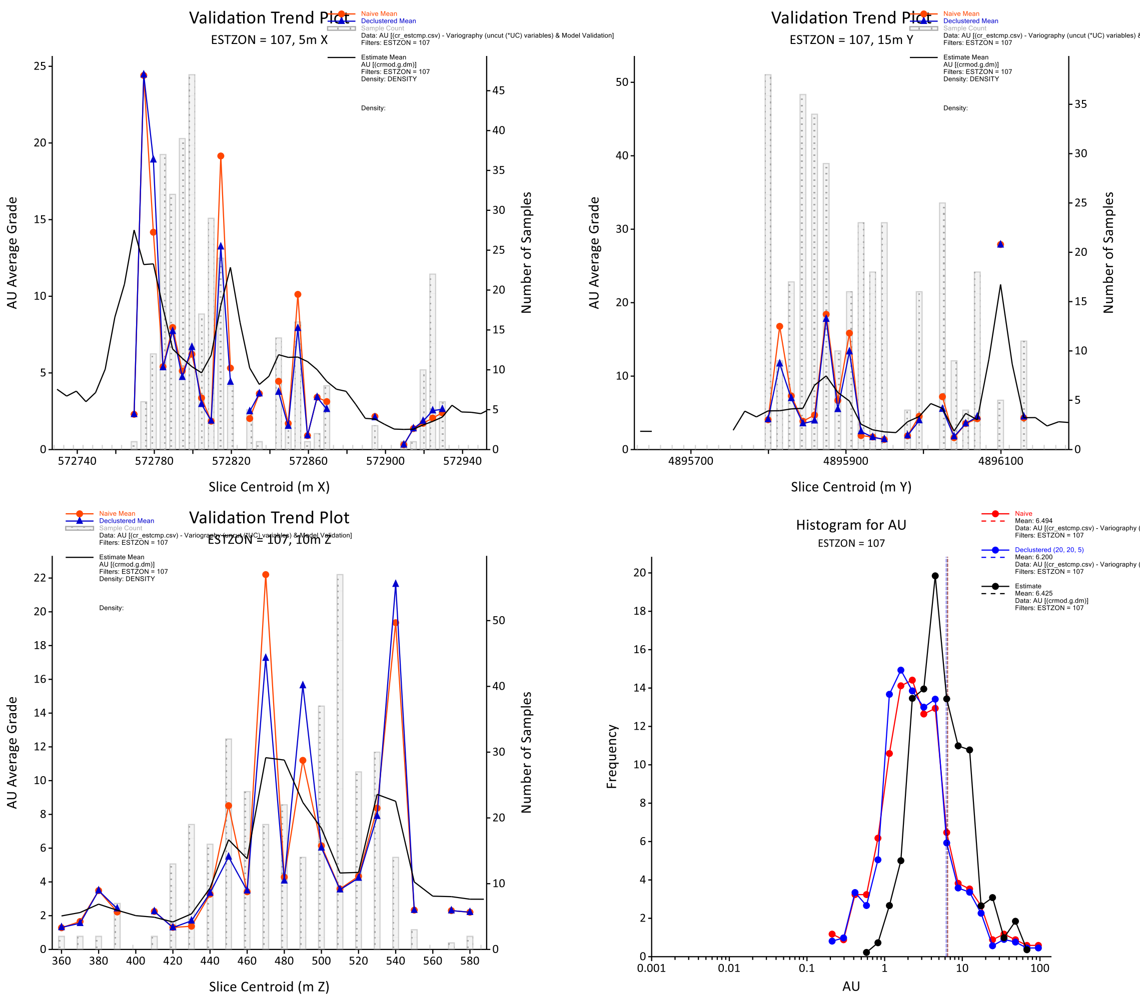
Source: CSA Global, 2024

Figure 14.44 – Swath Plots and Log Histogram for Au ESTZON 102



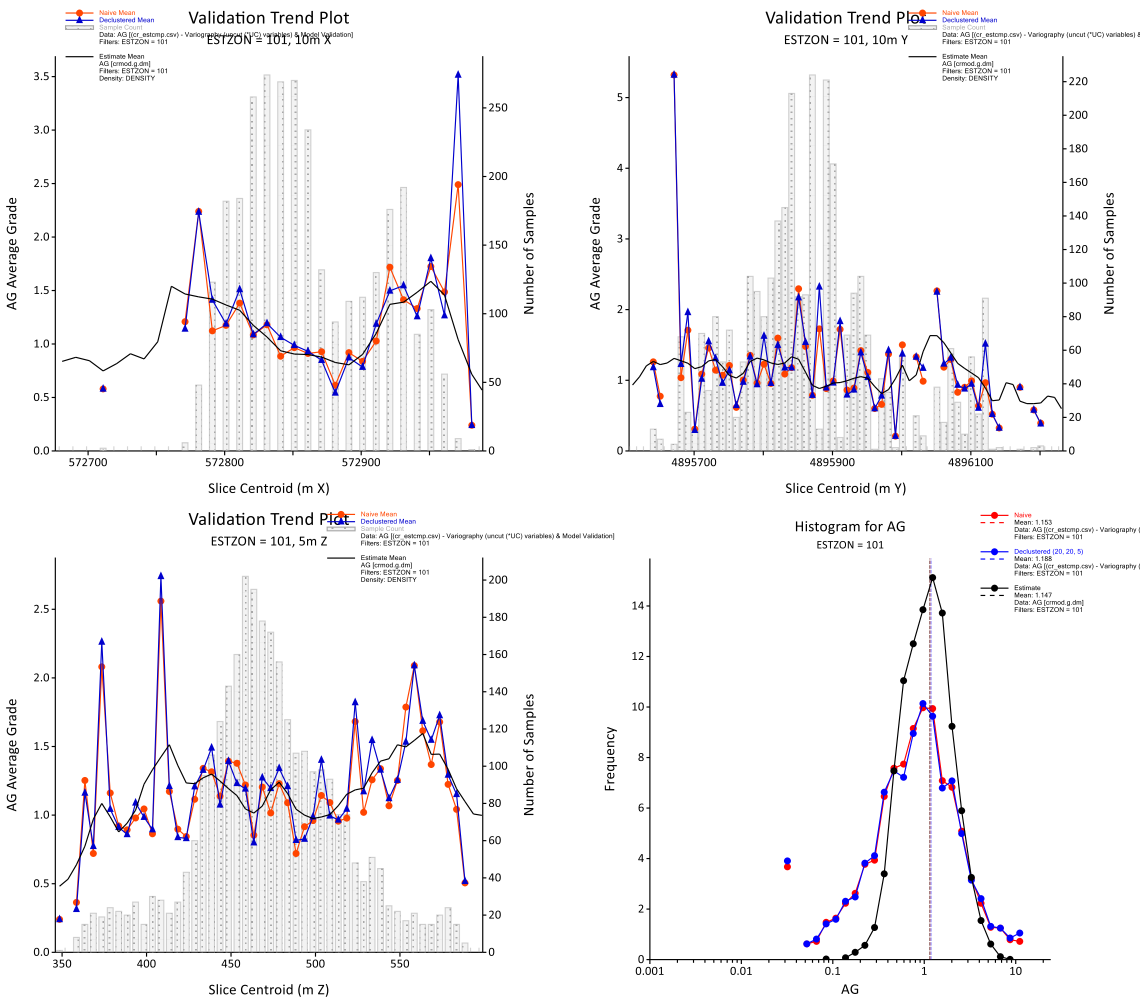
Source: CSA Global, 2024

Figure 14.45 – Swath Plots and Log Histogram for Au ESTZON 107



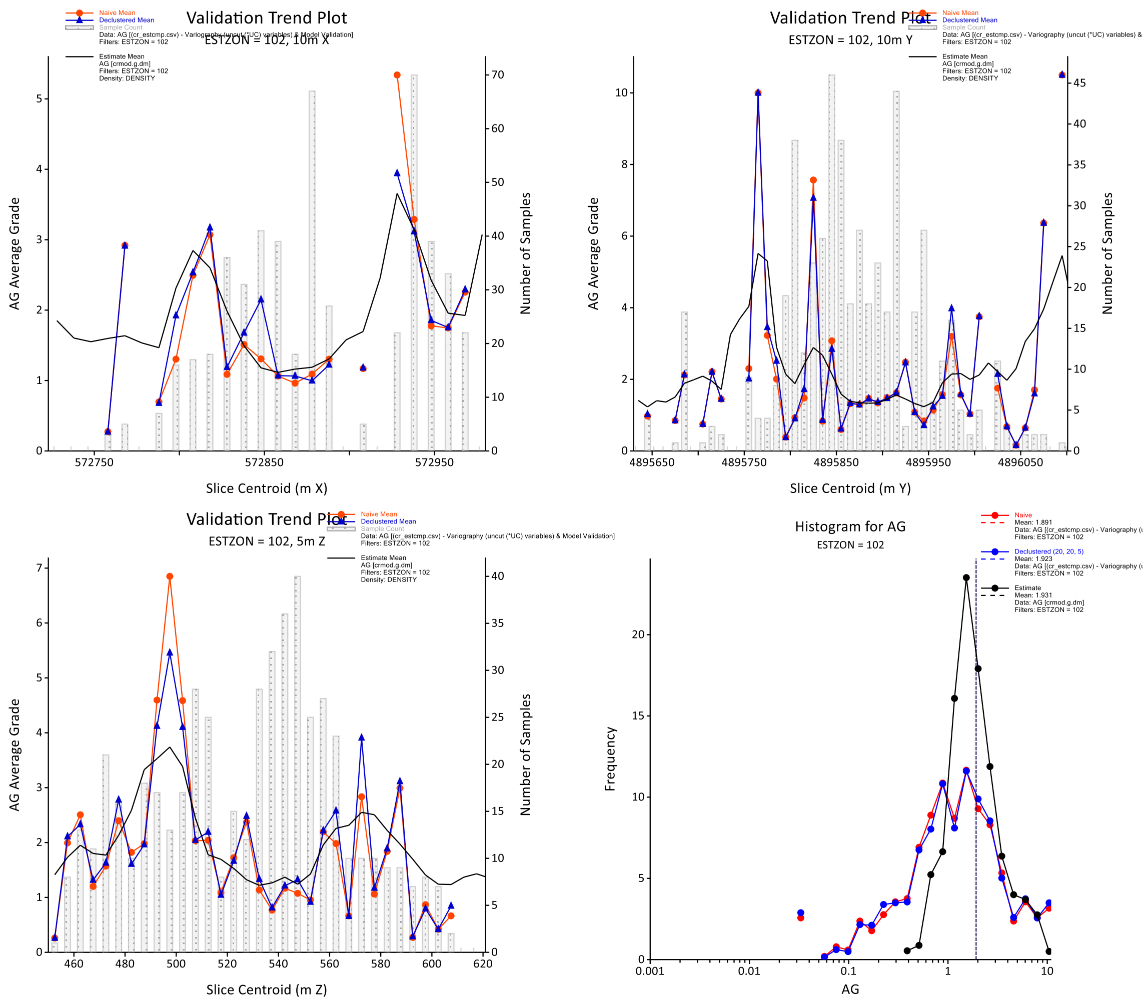
Source: CSA Global, 2024

Figure 14.46 – Swath Plots and Log Histogram for Ag ESTZON 101



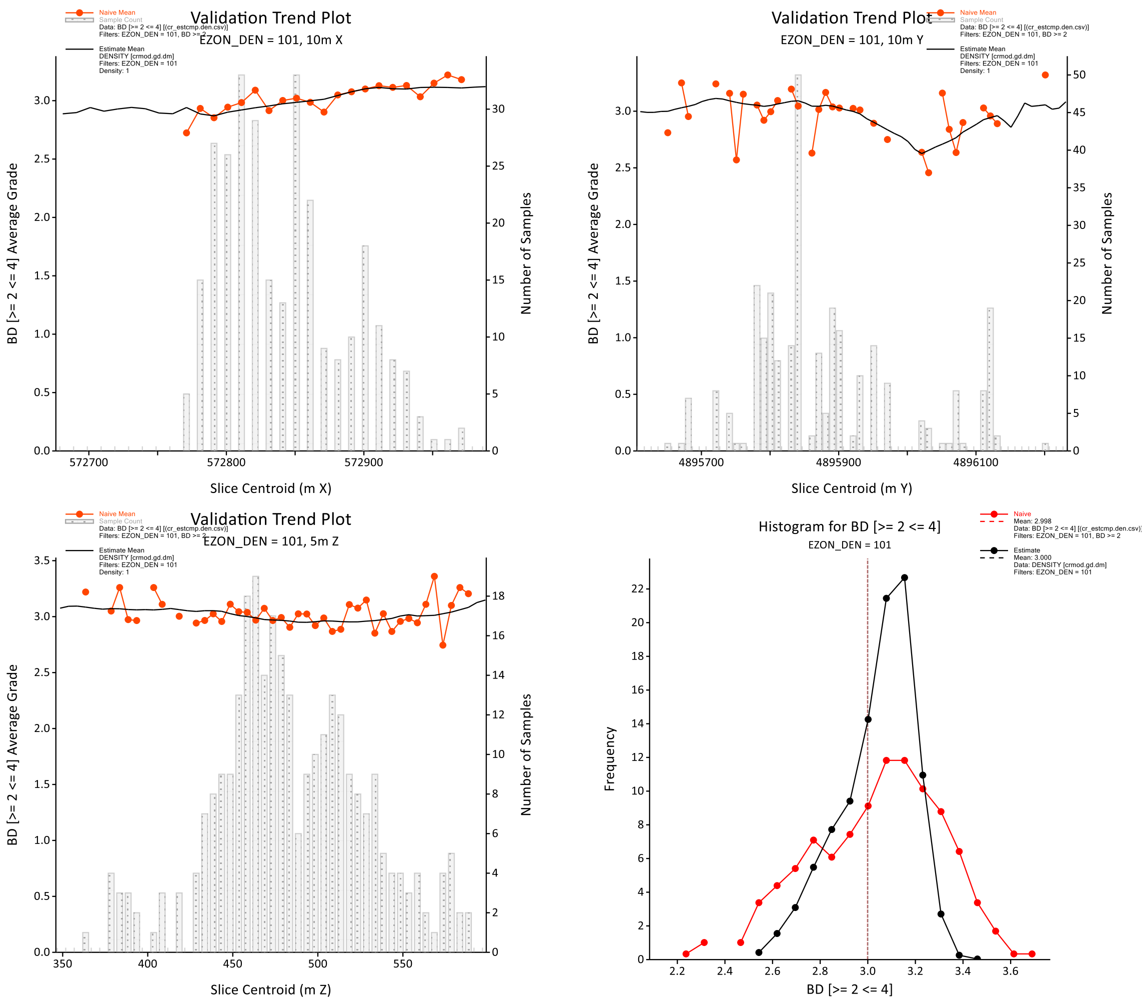
Source: CSA Global, 2024

Figure 14.47 – Swath Plots and Log Histogram for Ag ESTZON 102



Source: CSA Global, 2024

Figure 14.48 – Swath Plots and Log Histogram for Density ESTZON 101



Source: CSA Global, 2024

14.19 Mineral Resource Classification

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

A breakeven cut-off grade ("COG") of 2 g/t Au (rounded from 1.93 g/t Au) at US$1,700/oz of gold and a minimum width constraint of 3 m was used to define optimised underground potentially mineable shapes using Deswik's Stope Optimizer (DSO) to determine RPEEE of the block model and classify and report Mineral Resources for the Project. The cut-off grade breakdown and cost assumptions are shown in Table 14.23 to Table 14.25. In collaboration with the QP (Maria O'Connor, MAIG) and DPM's Mining Engineers, reasonable parameters were chosen for the MSO process and these are presented in Table 14.26.

Table 14.23 – COG Calculation and Cost Assumptions for Čoka Rakita MRE

Item	Cost per tonne (US$/t)
Underground mining costs	37.13
Process costs	22.11
G&A costs	12.46
Sustaining capital	11.19
Total	**82.89**

Table 14.24 – Commercial Terms Used Within the COG Calculation

Item	Unit	Cost per gold ounce
Concentrate transportation	US$/dmt	60.31
Concentrate treatment	US$/dmt	151.16
Concentrate refining	US$/oz	6.76
Concentrate penalty	US$/dmt	0
Concentrate grade – gold	g/t	101
Concentrate gold payable	%	98.4
Royalty	%	5
	Total US$ per gold ounce	**71.88**
	Total US$ per gold gram	**2.31**

Table 14.25 – COG Calculation for MRE

Item	Unit	Value
Gold price (ounce)	US$/oz	$1,700
Gold price (gram)	US$/g	$54.66
Revenue	US$/g	$47.76
Less royalty	5% of Sales	($2.39)
Less per gold gram costs	US$/g	($2.31)
Realised revenue	US$/g	$43.06
Cost per tonne to produce	US$/t	$82.89
MRE cut-off grade	**g/t**	**1.93 (rounded to 2 g/t)**

Notes:

- Processing costs: US$22.11/t Process (including Dry Tailings + Paste Fill costs).
- Transportation 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 88.8% and gold payability of 98.4%.
- Calculated cut-off grade for mineral resource reporting assumes mining dilution of 0%.

Table 14.26 – MSO Parameters

Parameter Name	Parameter Setting	Value	Units
Block Model Settings			
Input block model			
Optimisation field/default value	AU	0	g/t Au
Density field/default value	DENSITY	2.7	t/m³
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)	20	m
	V (Z axis/level height)	10	m
Stope depth (Z axis/stope width in MSO)	Minimum	3	m
	Maximum	20	m
Dilution	ELOS dilution	0	m
Stope dip angles	Minimum	0	°

Parameter Name	Parameter Setting	Value	Units
	Maximum	65	°
Stope strike angles	Minimum	-15	°
	Maximum	15	°
Maximum Stope Thickness Ratio	Top to Bottom	1.9	m
	Left to Right	1.4	m
Create sub shapes (Variable Shape Fraction)	U Min1	0	m
	U Max1	0.5	m
	V Min1	0.0	m
	V Max1	1.0	m
	U Min2	0.0	m
	U Max2	1.0	m
	V Min2	0.0	m
	V Max2	0.5	m
Material configuration	Include CLASS 9 material		
Advanced slice options	Ignore pillar requirements between full shapes		
	Ignore pillar requirements in split stope shapes		

14.19.2 Mineral Resource Classification

The MRE update for the Project has been classified as Indicated and 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 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

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 *An 'Indicated Mineral Resource' is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics are estimated with sufficient confidence to allow the application of Modifying Factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation.*

An Indicated Mineral Resource has a lower level of confidence than that applying to a Measured Mineral Resource and may only be converted to a Probable Mineral Reserve.

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. The following was taken into consideration by the QP (Maria O'Connor, MAIG):

- 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 (Kriging Efficiency, slope of regression, average number of samples etc.).
- Validation of the estimation of in-situ grades for gold.
- Validation of the tonnage factors derived from estimation of the in-situ dry bulk density.
- Drill Hole Spacing Study using conditional simulation to inform what drill hole spacing is required for an annual production volume variance of less than 15% for Indicated Mineral Resources.

- All Mineral Resources have Reasonable Prospects for Eventual Economic Extraction as demonstrated through stope optimisation, described in Section 14.19.1.

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 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 generally at 20 m to 30 m centres, which aligns with the outcomes of a Drill Hole Spacing Study (DHSS) using conditional simulation on the PEA dataset completed earlier in 2024 which determined a drill spacing of 30 m to be suitable for Indicated Mineral Resources on the basis of achieving a production deviation of less than 15% annually at a 90% confidence (ERM, 2024). Certain parts of the model in the hangingwall and at the margins do not achieve the required drill spacing and has been classified as Inferred Mineral Resources.

14.20 Mineral Resource Reporting

The MRE for the Čoka Rakita Gold Project is presented in Table 14.27. The Mineral Resource has been reported exclusive of Mineral Reserves.

Table 14.27 – Čoka Rakita MRE Using Underground Mining Scenario

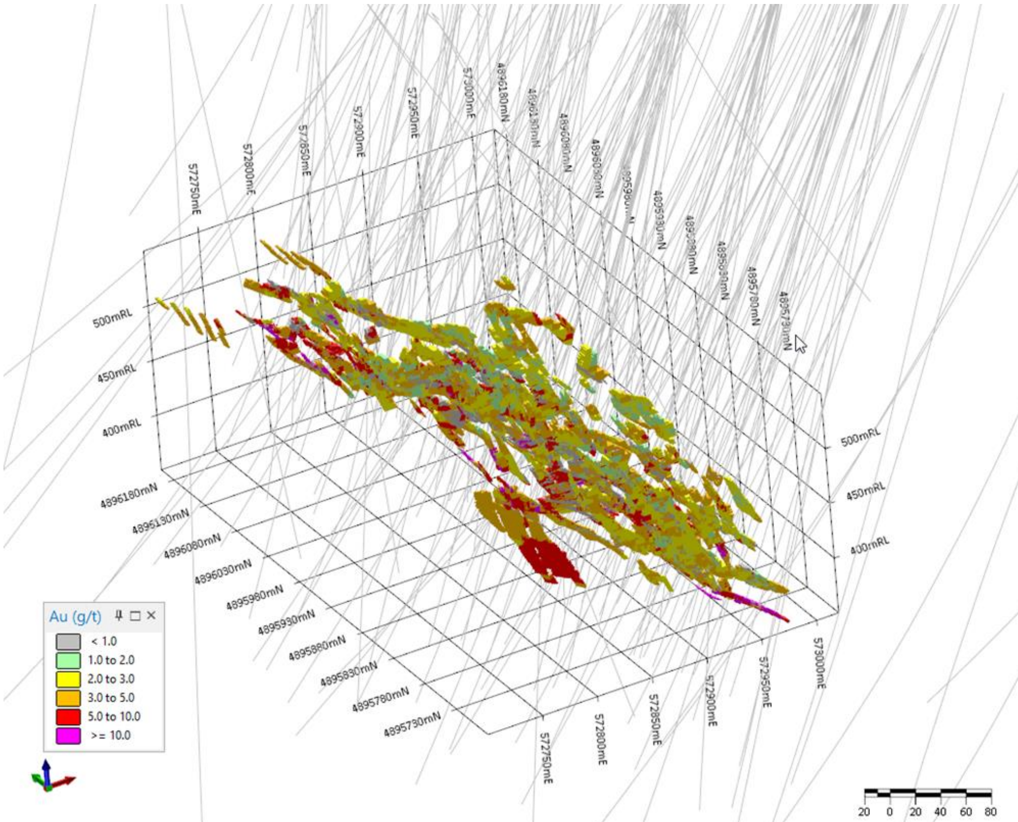
Čoka Rakita Mineral Resource Estimate Effective Date of 30 August 2024					
Mineral Resource category	Tonnes (Mt)	Gold Grade (g/t)	Contained Gold (koz)	Silver Grade (g/t)	Contained Silver (koz)
Indicated	1.45	3.30	154	1.30	61
Inferred	0.11	3.11	11	1.24	5

Notes:

- The cut-off grade value of 2 g/t assumes US$1,700/oz gold price, 88.8% gold recovery, 0% dilution, US$71.7/t operating cost (mining, process and G&A), US$11.19/t sustaining capital cost, as well as offsite and royalty costs.
- Mineral Resources are reported within DSO underground mining shapes generated at a 2 g/t Au cut-off grade, to ensure Mineral Resources meet RPEEE. The stope optimisation 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 (Maria O'Connor, MAIG) is not aware of any legal, permitting, title, taxation, socio-economic, marketing, political, environmental, or other risk factors that might materially affect the estimate of Mineral Resources, other than those specified in Section 14.21.
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
- Mineral Resources are reported exclusive of Mineral Reserves.
- Figures have been rounded to reflect that this is an estimate and totals may not match the sum of all components.

A 3D view of the Indicated and Inferred Mineral Resource model is presented in Figure 14.49.

Figure 14.49 – 3D view of the Classified Model, supported by RPEEE, Coloured by Gold (Looking Northeast with Supporting Drillholes)



Source: ERM, 2024

14.21 Risk Factors that May Affect the Mineral Resource

The QP (Maria O'Connor, MAIG) 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 14.28:

- 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 sensitive to the choice of top cut grades; therefore, changes to those values and how they are treated within the estimate could impact the grade and tonnage above the cut-off grade of the MRE. This means that if the top cut chosen is too high, metal content could be overestimated; if top cut chosen is too low, metal content would be underestimated. Sensitivity analysis on choice of top cuts (110, 125, 144, 175 g/t Au) shows variance in metal estimated to be in the order of 5% between lowest and highest top cut tested.

- 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 14.28. The overall risk to the Čoka Rakita MRE is reflected in the current resource classification as Indicated and Inferred Mineral Resources and is considered moderate, which is consistent with the high-grade nature of the Project.

Table 14.28 – Qualitative Risk Assessment

Factor	Risk	Comment
Sample collection, preparation and assaying	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. The level of coarse gold has been characterised as being a 'Medium' type coarse gold mineralisation, which is likely to have 'Medium' sampling problems. This generally indicates that a specialised protocol will be required to achieve quality assay data (e.g. SFA) (Dominy, 2024). Analysis for the vast majority of samples has been completed using screen fire assay which requires larger volumes which is considered good practice for this kind of deposit. The majority of the gold is associated with finer fractions, but coarse gold is associated with higher grades. The mineralised systems that have problematic levels of coarse gold generally lie within the "High-Med" to "Extreme" types. These generally require specialised sampling systems/protocols which may include whole core or large RC lot sampling followed by large assays such as SFA, LeachWELL or PhotonAssay. For High to Extreme types, tonnage-scale bulk sampling is likely required to validate resource/reserve estimates (Dominy, 2024).
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. A focussed duplicates program should be undertaken to monitor the transition from core shed to assay. This should include half diamond core duplicates, RC rig duplicates (where applicable), laboratory coarse split duplicates and assay/pulp duplicates (Dominy, 2024).
Geological model	Low	Uncertainty in accuracy of location of intrusives modelled is moderate based on core. However this is mitigated by the use of lithogeochemical analysis and modelling of multi-element geochemistry which allows for differentiation of these intrusives from S2/S1 unit, which is difficult by viewing core alone. The fact that core can look very similar in terms of skarnification and intensity of alteration but have different grade character across short distances.

Factor	Risk	Comment
Mineralisation model	Moderate	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 is mitigated by the low cutoff grade of the Mineral Resource and Mineral Reserve.
Treatment of outliers (grade caps)	Moderate	The MRE is very sensitive to the choice of grade cap. A relatively conservative grade cap was applied, which cuts 0.7% of the data and c. 20% of the metal. When data is top cut (at 144 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. An additional cap and distance threshold has been imposed within the estimate to reflect the fact that higher grades are unlikely to have the same continuity as lower grades. Sensitivity work on various top cuts conducted during this MRE (110, 125, 144 and 175 g/t Au) has indicated that the chosen top cut (144 g/t Au) results in a metal estimate approximately 3% higher than the lowest tested (110 g/t Au) and 2% lower metal than the highest tested (175 g/t Au).
Grade estimate	Moderate	The coefficient of variation (CV) is on the upper end of what is considered tolerable for Ordinary Kriging, largely due to the presence of ahigh grades tail. 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. The classification as Indicated and Inferred Mineral Resources reflects this risk.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

This section presents the Mineral Reserve Estimate for the Čoka Rakita Project with an effective date of August 30, 2024, and details the key assumptions, parameters, and methods used for converting Mineral Resources to Mineral Reserves.

A Mineral Reserve is an estimate of tonnage and grade of measured and indicated mineral resources that, in the opinion of the QP (Mr. Khalid Mounhir, P.Eng.), can be economically mineable including dilution and allowances for losses that may occur when the material is mined or extracted.

The Čoka Rakita Deposit Mineral Reserve Estimate was prepared by WSP and is based only on Indicated Mineral Resources identified in the block model provided by ERM. All mineral resource material in the block model that was classified as Inferred was assigned a zero grade.

The mining method of longitudinal long-hole stoping was chosen and the Geotechnical analysis using empirical methods (Mawdesley et. al (2001) and (Hadjigeorgiou (1995)) has recommended a 20 m level interval, a maximum strike length of 30 m, and a maximum width of 20 m.

15.2 Estimation Procedure

The process consists of converting Measured and Indicated Mineral Resources to Proven and Probable Reserves by identifying material that exceeds the COG value while conforming to the geometrical constraints determined by the mining method and applying modifying factors such as dilution and mining recovery. The conversion of Measured and Indicated Mineral Resources to Proven and Probable Mineral Reserves involves the following procedures:

- Review the geological block model of the resource received from ERM;
- Review the gold selling price assumption to ensure it is reasonable;
- Estimate the on-site production costs for the mining method and mining situation;
- Estimate the economic modifying factors;
- Exclude Inferred Mineral Resources;
- Estimate mining modifying factors: dilution and mining recovery;
- Determine mine design parameters, such as stope dimensions, minimum mining width, and minimum footwall angle for long hole stopes;
- Outline potentially mineable shapes in the block model based on the resource value exceeding the cut-off grade;
- Screen potentially mineable shapes with the Mineable Shape Optimiser application in Deswik software;

- Refine potentially mineable shapes by removing un-mineable resource material;
- Design mine development and mine infrastructure in Deswik. CAD software;
- Determine production sequencing with Deswik Scheduler software;
- Prepare a life-of-mine plan for development and production;
- Estimate capital, operating, and sustaining capital costs associated with the life-of-mine plan;
- Prepare the Mineral Reserve statement.

15.3 Modifying Factors

15.3.1 Cut-Off Grade Estimation

Gold is the only payable commodity recovered, and the COG was calculated using the breakeven method.

Gold commodity price of US$1,500 /oz proposed by DPM is in line with the 10-year average gold price calculated based using World Gold Council data and presented in Table 15.1.

Table 15.1 – Average Gold Price

Year	Gold Price (US$/oz)
2014	1,266.4
2015	1,160.1
2016	1,250.8
2017	1,257.2
2018	1,268.5
2019	1,392.6
2020	1,769.6
2021	1,798.6
2022	1,800.1
2023	1,940.5
Average	**1,490.4**

The gold price that has been used for the design was reviewed by the QP from WSP and is seen as conservative. All costs used for the design are in US dollars.

The costs per ore tonne, the marketing costs and the mill recovery used in the COG estimation were compiled from the PEA economic model. These inputs were reviewed and confirmed as being appropriate for use to inform the PFS COG estimate.

The transportation, treatment and refining cost used are calculated as a weighted average cost of gravity gold concentrate and flotation gold concentrate.

The QP (Mr. Khalid Mounhir, P.Eng.) judged it necessary to apply a 5% contingency factor to the PEA costs. With the adjusted costs the calculated in-situ stope COG is 2.5 g/t and the development COG is 1 g/t.

The COG estimation parameters are shown in Table 15.2.

Table 15.2 – COG Estimation Parameters

Costs per Ore Tonne (US$)		
Category	**Units**	**Value**
Mining	US$/ t milled	37.13
Processing	US$/t milled	22.11
G&A	US$/t milled	12.46
Sustaining capital	US$/t milled	11.19
Total Cost	**US$/t milled**	**82.89**
Total Cost with 5% Contingency	**US$/t milled**	**87.03**
Marketing Costs (US$)		
Category	**Units**	**Value**
Concentrate transportation	US$/dmt	60.31
Concentrate treatment	US$/dmt	151.16
Concentrate refining	US$/oz	6.76
Concentrate penalty	US$/dmt	0.00
Concentrate grade Au	g/t	101
Concentrate Au payable	%	98.4
Mill recovery	%	88.8
Royalty	%	5.0
Total Cost	**US$/oz**	**71.88**

COG Calculation			
Category	Units	Value (Stope)	Value (Development)
Gold price (ounce)	US$/oz	1,500	1,500
Gold price (gram)	US$/g	48.23	48.23
Revenue	US$/g	42.14	42.14
Less royalty 5% of sales	US$	(2.11)	(2.11)
Less per gold gram costs	US$	(2.31)	(2.31)

COG Calculation			
Realised revenue	US$/g	37.72	37.72
Cost per tonne to produce	US$/t	87.03	36.3
Mill feed COG	g/t	2.31	0.96
Mining dilution	%	10	-
Dilution grade	g/t	0.50	-
Reserve (In-situ) COG	**g/t**	**2.5**	**1.00**

15.3.2 Mining Dilution

Mining dilution comes from three (3) principal sources: planned (or internal) dilution; unplanned (or external) dilution; and backfill dilution.

The rock dilution was based on geotechnical recommendations of HW failure of 1.0 m and FW failure less than 0.5 m for a stope of 20 m high, 20 m span and 30 m strike at 65° dipping angle.

The backfill dilution comes from adjacent and below mined out stopes and was estimated based on benchmarking.

The overall dilution used is 10%, with a 6% estimated rock dilution plus 4% backfill dilution. Grades for internal dilution were estimated from the Mineral Resource block model. The external dilution was estimated to have an average grade of 0.5 g/t, which was the weighted average of backfill and wall rock dilution proportions.

15.3.3 Mining Recovery

Mining recovery represents material loss in the mining process and is defined as the percentage of actual mineable material extracted from the planned mining shape.

Mining losses in the excavation of the ore development drifts are assumed to be zero. For the long-hole stopes, the mining recovery is estimated to be 95% which is in line with industry best practices for the Čoka Rakita Deposit type and mining method.

15.4 Stope Optimisation

Mineral Reserves were estimated by generating a preliminary stope shape using Deswik Stope Optimizer (DSO) Version 4 software with respect to the design and economic criteria established such as COG, deposit geometry criteria and stope shape parameters. Multiple optimisation scenarios were evaluated to obtain the optimal results in terms of tonnage and grade. The DSO parameters used to prepare the stope are provided in Table 15.3.

The stopes produced are visually checked against the geological block model for consistency, general capital development and stope access are subsequently designed and optimised.

The stopes are then sequenced to suit the mining method (long-hole longitudinal retreat) and scheduled in Deswik scheduler to produce the production profile and life-of-mine (LOM).

Table 15.3 – DSO Parameters

Parameter	Units	Value
Optimisation field (in BM)	g/t	AU
Default density	t/m³	2.93
Stope orientation plane	NA	YZ
Level spacing	m	20
Section spacing	m	30
Stope minimum width	m	3
Stope maximum width	m	20
Stope pillar width	m	0.001
Near wall dilution	m	0
Far wall dilution	m	0
Default dip angle	deg	65
Minimum dip angle	deg	65
Maximum dip angle	deg	115
Maximum strike angle	deg	15
Maximum strike angle change	deg	15
Maximum side length ratio	NA	1.9
Cut-off grade	g/t	2.5

15.5 Mineral Reserve Statement

The mine design and Mineral Reserve Estimate were completed to a level appropriate to support a PFS. The Mineral Reserve Estimate stated herein has been prepared in accordance with the 2014 CIM Definition Standards and 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines.

The Mineral Reserve for the Čoka Rakita Deposit is estimated at **6.633 Mt** and diluted ore grading **6.38 g/t Au**, This Mineral Reserve estimate includes dilution, mining recovery factors, and incremental development tonnes and is summarised in Table 15.4.

Table 15.4 – Mineral Reserve Estimate Effective as of August 30, 2024

Classification	Tonnes (Mt)	Au (g/t)	Contained Gold (Moz)
Proven	-	-	-
Probable	6.633	6.38	1.359
Proven + Probable	**6.633**	**6.38**	**1.359**

1. At the time of this Report, there are no Proven Mineral Reserves for the Čoka Rakita Project.
2. The Mineral Reserves disclosed are classified as Probable and are based on the 2014 CIM Definition Standards and 2019 CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines.
3. The Inferred Mineral Resources are treated as waste and do not contribute to reserves estimation.
4. Mineral Reserves has an effective date of August 30, 2024.
5. The reference point at which the Mineral Reserves are defined is where the ore is delivered to the process plant and therefore not inclusive of milling recoveries or payable metal deductions.
6. Long-term metal price assumed for the evaluation of the Mineral Reserves is US$1,500/oz for gold.
7. Mineral Reserves are based on a global rounded stoping cut-off grade of 2.5 g/t and development COG of 1g/t (in-situ). Refer to Subsection 15.3.1 for COG estimation details.
8. Mineral Reserves account for 10% external mining dilution and 95% mining recovery applied to the stopes
9. Contained Metal is calculated as follows: Au Contained Metal, (oz) = Tonnage (Mt) * Grade (g/t) / 31.1035.
10. The Mineral Reserve Estimation was completed under the supervision of Mr. Khalid Mounhir, P.Eng., Lead Mining Engineer at WSP Canada Inc., who is a QP as defined under NI 43-101.
11. The QP is not aware of any mining, metallurgical, infrastructure, environmental, permitting, legal, title, taxation, socio-economic, marketing or political factors that might affect the estimate of Mineral Reserves, other than those specified in Section 15.6.
12. Sum of individual table values may not equal due to rounding.

15.6 Factors That May Affect the Mineral Reserve Estimate

The Mineral Reserves are subject to the types of risks common to underground gold mining operations and include:

- Geological complexity, geological interpretation, and Mineral Resource block modelling.
- Changes in Market conditions, commodity prices and COG estimation assumptions.
- Changes in rock quality and geotechnical assumptions.
- Changes in hydrogeological conditions and water inflow assumptions
- Input factors used to assess dilution and mining recovery factors.
- Assumptions regarding social, permitting, and environmental conditions.

To the extent known to the responsible QP (Mr. Khalid Mounhir, P.Eng.), there are no known mining, metallurgical, infrastructure, environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors that could affect the Mineral Reserve estimate that are not documented in this section.

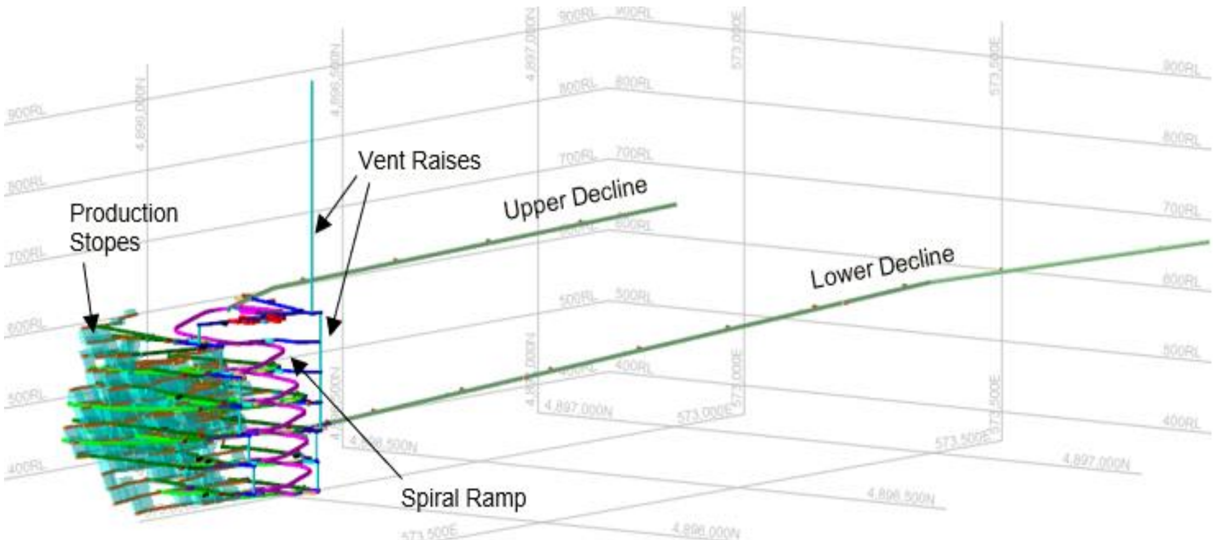
16 MINING METHODS

16.1 General Description of the Deposit and Mining Project

Čoka Rakita is an underground mining project, The gold ore occurs in a skarn-type deposit approximately 250 m to 600 m below the surface. The ore 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.

Figure 16.1 provides a 3D illustration of the underground mine planned for the Čoka Rakita Project.

Figure 16.1 – 3D Representation of the Čoka Rakita Mine



Source: WSP, 2024

The host rock of the deposit is a calcareous clastic sedimentary rock, with ore occurring in skarn-altered calcareous sandstone. The ore is primarily controlled by stratigraphy and, to a lesser degree, by structural factors. The footwall boundary of the ore closely parallels a sill-like diorite intrusion.

A geotechnical trade-off study carried out by WSP on positioning the main underground infrastructure at either the HW or FW showed that the quality of the sedimentary host rock in the hanging wall generally ranges from fair to good, while the footwall of the deposit exhibits poor ground conditions. Consequently, the decision was made to position the mine development and the underground infrastructure on the hanging wall side of the deposit.

DPM plans to mine the deposit using 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 9.79 million tonnes from the PEA, the planned production rate is reasonably conservative compared to Taylor's Rule and Long's Relationship outputs.

Taylor's Rule

- Production Rate = 0.0143 * Tst 0.75
- Referential Tonnage = 9,790,000 t
- Short Tons (st) = 10,791,000 st
- Production Rate = 2,692 st/day
- = 2,442 t/day

Long's Relationship for Underground Mines

- Production Rate = 0.297 * $T^{0.563}$
- Referential Tonnage = 9,790,000 t
- Production Rate = 2,562 t/day

An updated empirical estimation for the optimal production rate should be done with the PFS resource numbers once available.

The deposit will be accessed by driving upper and lower declines from the east side of the ore in the HW. The upper decline will give access to the 620 sublevel, and the lower decline to the 440 sublevel. A spiral ramp will be developed to connect both declines and give 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 high intervals.

16.2 Geotechnical

WSP completed a Geotechnical PFS 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 within the proposed mine workings.
- **Stope Stability and Dilution Review** – Review mining methods based on the EGM. Stope stability sizes were evaluated using empirical methods Mawdesley et al, (2001) and dilution using Clark and Pakalnis (1997).
- **Stand-Off Distance, Stope Sequence, and Infrastructure Stability Review** – Completed 3D numerical elastic induced stress modelling and completed a comparison of infrastructure locations against areas of poor ground.
- **Crown Pillar and Temporary Sill Pillar Review** – Scaled Span empirical assessment of crown pillar and 3D elastic numerical model assessment of the temporary sill pillar.
- **Underground Infrastructure Ground Support and Review of Vertical Stability** – Using empirical, analytical, and numerical methods.
- **Portal Ground Support and Development Guidance** – Using empirical and analytical methods.
- **Backfill Strength Estimate** – Using empirical methods (Mitchell, 1982).

The data available for the PFS included geotechnical core logging data, downhole televiewer surveys, laboratory testing, and supplementary data sets from infill drilling for the Project area and information on in-situ stress testing from the Chelopech mine site. The geotechnical investigation data was provided to WSP as it was collected by DPM and their local consultants. Methods of geotechnical data collection were provided by WSP and regular check-ins and reviews of data collected and sent for laboratory analysis were completed by WSP. The QP completed a site visit to the project site and observed data collection, laboratory testing procedures and visit to the underground Chelopech mine were completed on June 17 to 20, 2024.

The results of the investigation were not completely available for inclusion into the PFS rock mechanics assessments when the assessments were completed; as such, the EGM data presented in this report (which used data available as of October 2024) do not necessarily align with input parameters used for each rock mechanics design assessment (which were completed from July through to October 2024). More data was collected after completion of the EGM and design assessments.

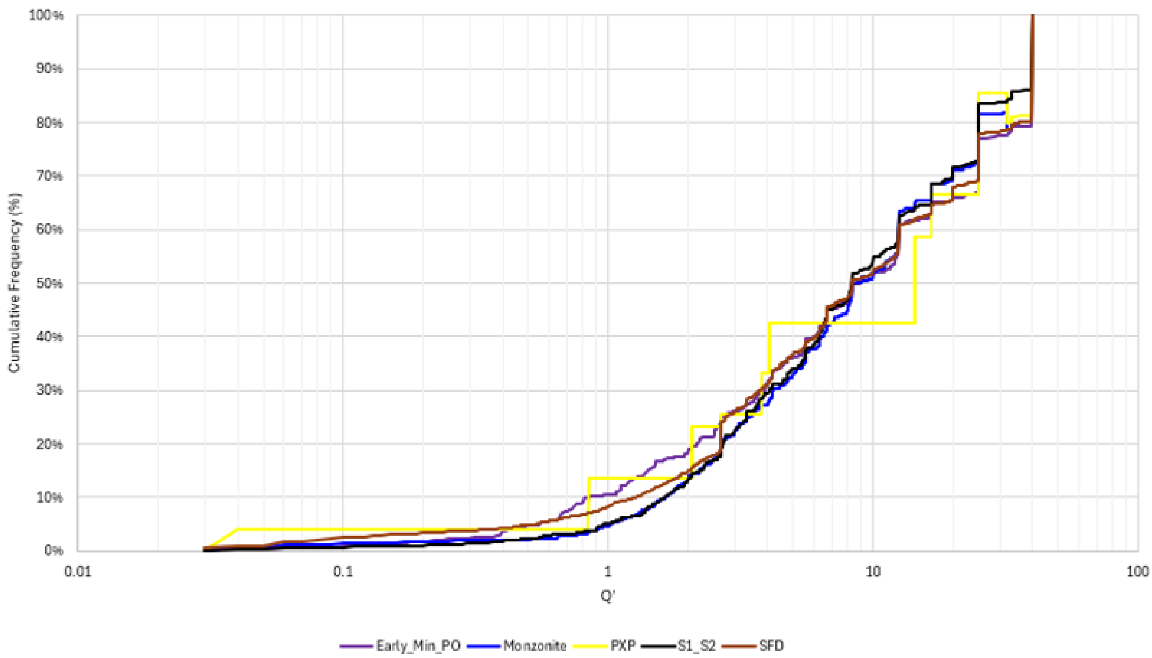
16.2.1 Engineering Geology Model – Rock Mass Quality

The EGM was examined based on two (2) separate divisions. The first division was based on the PFS geotechnical core logged lithology and second was based on the geologic formation wireframes (created by DPM). The results for each division were compared and the values developed for the geologic formation wireframes were ultimately used for the design assessments. The geologic formation wireframes enabled a better estimate of spatial distribution of the overall geotechnical character because the logged lithologies were spatially defined in the geologic model by DPM. 98%

of the proposed underground workings are within the SFD, S1_S2, and Early_Min_PO geologic formation wireframes while approximately 97% of the geotechnical drill core was collected in those same formations.

In general, the geotechnical domains had similar classification values and ranged from Good (Barton Q'>10) quality for 50% of the logged core, to Fair to Poor (Barton 10>Q'>1) for 40% of the logged core, and Very Poor (Barton Q'<1) for 10% of the logged core as illustrated on Figure 16.2.

Figure 16.2 – Barton Q' Distribution by Geologic Formation



Source: WSP, 2024

The intact strength characteristics were also similar for the geologic formations except for the S1_S2 unit (Table 16.1). The S1_S2 (which comprises sandstone and other sedimentary structures) had about half the strength of the other formations. This is unusual since the sandstone lithology (when examined independently) also had a much higher σ_{ci} (but lower mi) than estimated for S1_S2.

Table 16.1 – Intact Strength Parameters by Geologic Formation

Geologic Formation	Mean σ_{ci} (MPa)	Mean m_i	Tension Cut-Off (MPa)
SFD	154.2	26.6	5.4 (1)
S1-S2	75.6	17.8	3.5 (1)
Early Min	145.6	11.6	11.8

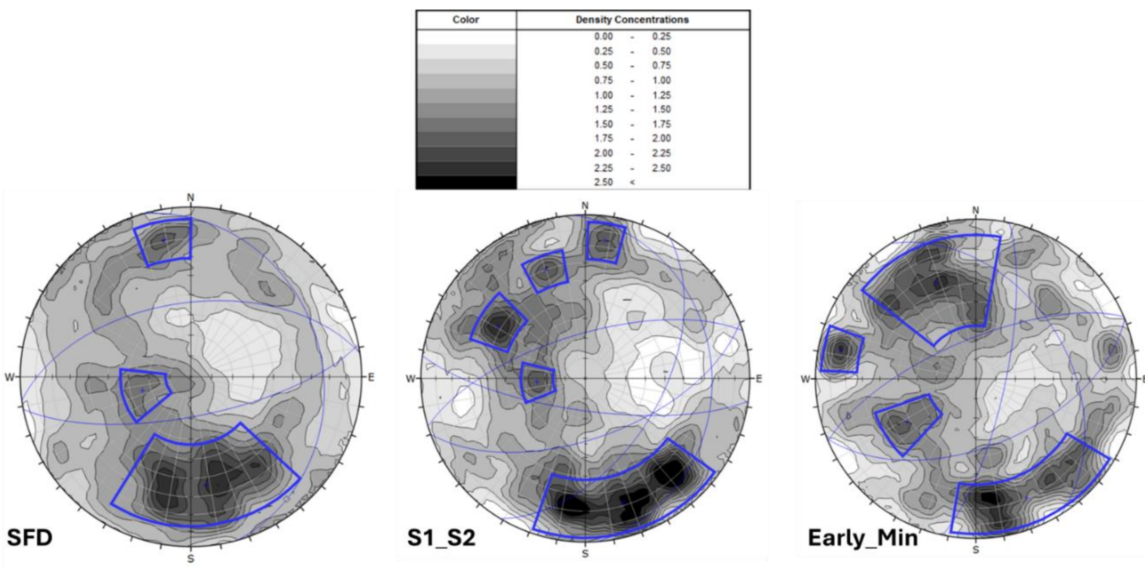
Note:

(1) The intersection of this fit with a confining stress of 0 is below the cut-off estimated from tensile tests.

Assessment of major structures suggest that there are a substantial number of faults with relatively poor ground character intersecting the proposed mine workings. The fault model and interpretation of fault character and size were developed by DPM and were used in the rock mechanics study assessments. It is assumed that much of the Very Poor (Barton Q) geotechnical core can be attributed to fault activity. It is also assumed that it is unreasonable to move all the mine infrastructure to avoid faulting. As such, fault intersections were assumed to be Very Poor (Barton Q) ground with conservative intact strengths for the purposes of the rock mechanics stability assessments.

Rock mass fabric (joints and veins) was also examined and it was found that the major open joint sets are aligned North and Southeast and are fairly consistent across the geologic formations (Figure 16.3). Closed structure was generally aligned with the open structure pole concentrations (with the exception of Monzonite, which only intersects < 0.5% of the mine workings). In addition to orientation, it was noted that in some of the field strength tests, point load tests, and laboratory tests that closed defects (typically veins) can influence the rock mass strength. The majority of the samples from the high quality dataset (laboratory testing) did not report influence from defects; therefore, the strength reduction caused by veining was assumed to have a isolated impacts on the overall mine stability and will require more strength data (from laboratory testing and field strength estimates) to improve confidence on the influence of veining.

Figure 16.3 – Stereonet Pole Concentrations by Geologic Formation



Source: WSP, 2024

In situ stress data was not available at Čoka Rakita, available public datasets and information from the nearby, DPM owned, Chelopech mine were reviewed. In general, the Chelopech data does not have consistent results and could not be confidently applied to Čoka Rakita additionally, the world stress map has limited data near the project site but does suggest the general tectonic character of the area is either a strike-slip or normal faulting regime. The following was assumed for the purpose of the PFS rock mechanics assessments:

- Sigma 3 is vertical and lithostatic;
- Sigma 1 is horizontal (East – West trending) and has a K ratio of 1.5; and
- Sigma 2 is horizontal (North – South trending) and has a K ratio of 1.5.

16.2.2 Stope Stability Review and Dilution

A stope stability graph method and dilution assessment were completed using the PFS mine design using empirical methods Mawdesley et. al (2001) and Hadjigeorgiou (1995). For the PFS design, the mining method considered is longitudinal open stoping with backfill (cemented rock, rock fill and paste backfill).

The stability graph plots the stope Hydraulic Radius (HR) versus Modified Stability Number (N'). The key input data are geometry of stope, induced stress, rock strength, rock mass classification and joint orientation. Stopes located in the "stable" zone are expected to exhibit little or no wall deterioration during the mining cycle, while stopes plotting to the right, in the "caving zone" may be unstable even with support. Some dilution may nonetheless be encountered even for stopes that plot within the stable zone on the stability graph. Dilution may originate from the formation of unstable blocks or wedges that will fall into the stope with the blast, or later, depending on the length of time the stope remains open, or from rock mass damage originating from poor drilling and blasting practices.

Plots were created for varying hanging wall/footwall angles, stope size, rock mass rating (Barton Q') and with or without ground support in the back and/or hanging wall. WSP considers a stope geometry stable if the stope plots on and above the 70% Probability of Failure (PoF) curve for the Mawdesley stability plot.

The range in rock mass properties reflect two (2) general types of ground conditions in the mineralised zone. The typical ground condition in the mineralised zone area representing Barton Q' = 4 to 12.5 (Fair to Good) and a second zone represented by Poor to Very Poor quality rock where Barton Q'=1 (or less) likely due to fault zones or areas of low RQD not related to known faults. The hanging wall and footwall were assessed at face inclination angles of 45°, 65° and vertical. Based on the assessment the hanging wall will need to be designed at 65° or steeper in order to achieve reasonable strike lengths unless hanging wall ground support is included. Hanging walls from 45° to 65° require hanging wall support (cable bolts). Based on a standard stope span of 15 m the back will require ground support which will be based on the standard support for development drifts in ore (no cable bolts). A selection of stope configurations assessed are provide in the following Table 16.2.

Table 16.2 – Stable Stope Configurations

Scenario	Face Inclination	Stope Height (m)	Max Span (m)	Max Strike (m)
Unsupported HW	65°-90°	20	30	20
			20	30
			15	50
Supported HW (cables bolts)	45°-65°	20	30	20
			20	25

The final stope configuration selected to be used in the PFS mine design is 20 m Span x 20 m Height x 30 m Length. The hanging wall (in rock) of the final stope in each block will need to be designed at 65° or steeper unless hanging wall support is planned. All interior stope walls are designed vertical and will be backfilled with cemented paste and will not require wall support. The back of the drill drifts will need to be supported with standard ground support.

An estimate of stope dilution was completed using the PFS mine design stope size (20 m Span x 20 m Height x 30 m Length) on the Clark and Pakalnis stability charts (ELOS) and the following is the estimated dilution:

- Rock mass quality Barton Q'=4 (Fair) results in < 1.0 m of dilution in the hanging wall and 2.0 m of dilution in the back without support and reduced to 0.5 m in the back with support for stope dips of 65° and 90°.
- Rock mass quality Barton Q'>4 (Fair to Good) results in 0.5 m of dilution in the stope hanging wall and 1 m of dilution in the back without support and reduced to 0.5 m in the back for stope dips of 65° and 90°.
- Rock mass quality Barton Q'< 4 (Q' = 1 - Poor to Very Poor) has > 2 m of dilution in the stope back and hanging wall for stope dips of 65° and 90°.

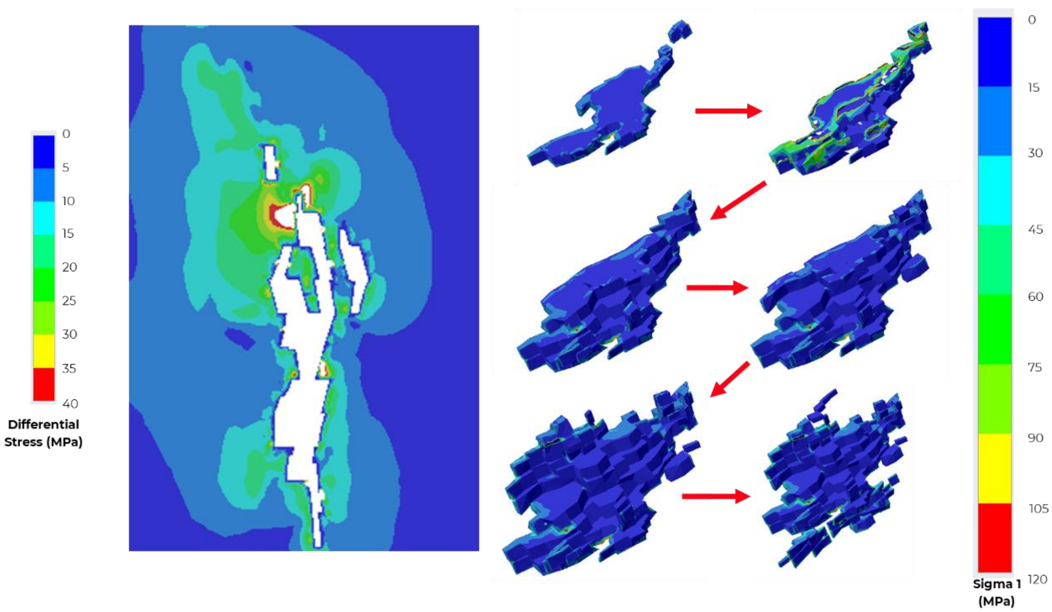
Based on the geotechnical information reviewed an estimate of up 10% of the stopes could be in Poor to Very Poor (Barton Q'<4) quality rock mass zones which could result in dilution up to 20% if stope strike length is not reduced and/or additional ground support not applied to the hanging wall. Stopes with rock mass quality of Barton Q'≥4 could result in dilution on the order of 5-10%. Internal stopes with vertical walls and backfilled with cemented paste fill are planned with 0% dilution.

16.2.3 Stand-Off Distance, Stope Sequence, and Infrastructure Stability Review

The PFS 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 3D elastic numerical (RocScience EX3) induced stress model was developed to assess stand off distances for long-term critical infrastructure to be placed outside the influence of induced stresses from stope extraction. The extraction sequence for the mine follows a longitudinal open stoping method, meaning no other infrastructure (aside from development drift accesses) will be located in the hanging wall. The spiral ramp and other critical infrastructure are required to be a minimum of 50 m away (laterally) from the stoping extraction area.

The stope sequence was also reviewed for areas in which induced stress may become problematic due to the chosen stope sequencing. In general, the induced stress hazard is low for Čoka Rakita (considering the assumed in-situ stress parameters) during early stages of mining. Near the end of mine life, when mining advances through the temporary sill pillar, the numerical model suggests (Figure 16.4) that the mining sequence should follow a North to South extraction process and avoid diminishing pillars on the level; although, not following these recommendations does not represent a fatal flaw at this stage. The proposed sequence has not been optimised to reduce induced stress build up (e.g., there are some diminishing pillars) but instead to prioritise operational considerations. Future study stages will examine the sequence in more detail with plastic behaviour and a better understanding of in situ stress; after which the sequence near the end of mine life can be optimised (to the extent possible) to balance induced stress and operational considerations.

Figure 16.4 – Stand-off Distance Assessment. Left: Plan View of the Temporary Sill Pillar after Extraction. Right: Example Model Steps of the Modelled Extraction Sequence

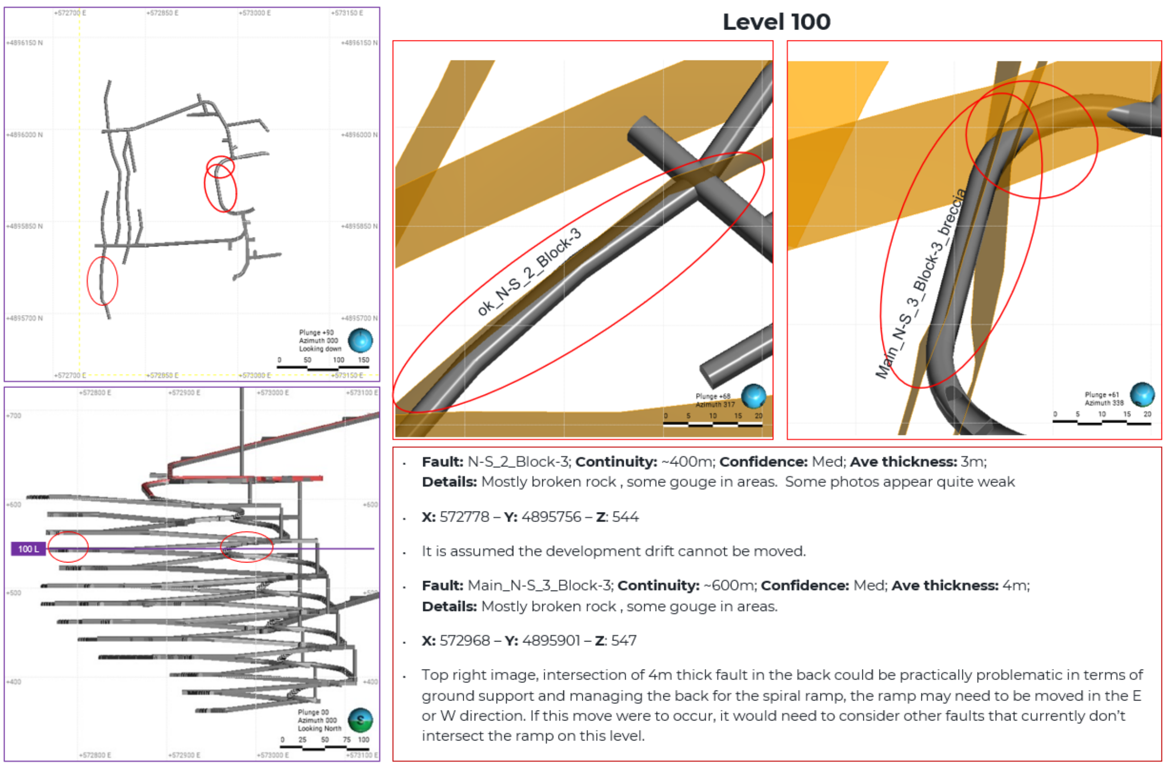


Source: WSP, 2024

For infrastructure stability, the intersection of faults with the proposed PFS mine workings was completed. An initial check with the fault model (by DPM) available the summer of 2024 was completed and resulted in some recommendations for moving critical infrastructure (e.g., the maintenance shop and the surface ventilation raise); however, a review was then completed in October 2024 after the latest fault model was provided by DPM (also in October 2024). The latest review found new intersections of faults with underground workings (example section on

Figure 16.5) which are potentially problematic to drift stability and may require adjustments to underground mine workings. These adjustments were not completed for the PFS mine workings as the geotechnical data was made available after the mine design was completed; however, the review lays out the groundwork for the next stage of study to focus on the confidence and character of the fault shapes at each point of intersection with the mine workings identified as being problematic for stability.

Figure 16.5 – Example Fault Intersection Review of Level 100 of the Proposed PFS Mine Workings Using the Most Recent Fault Model (October 2024)



Source: WSP, 2024

16.2.4 Crown and Temporary Sill Pillar Review

A simple empirical crown pillar stability assessment (Scale Span) and temporary sill pillar excavation assessment using 3D numerical stress model (RocScience EX3) was completed to identify potential instability issues with the PFS 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, Barton 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. The EX3 model also did not indicate any stability related issues in the crown pillar.

Based on the PFS 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 at the end of the mine sequence. Sill pillar stability was reviewed using the 3D elastic numerical induced stress model (RocScience EX3) also used to assess infrastructure offset and mine sequencing in the previous section. The stability assessment does not focus on how to sequence mining through the sill pillar (that is discussed in the previous section), but rather the overall stability of the pillar as mining approaches it from below. The pillar stability was reviewed by comparing Sigma 1 and Sigma 3 against the rock mass Hoek-Brown failure curve (considering mean GSI). The pillar core is estimated to remain stable as mining approaches from the below. Small areas on the north edge of the pillar may experience rock mass failure for up to half the depth of the pillar, but this should not influence stability or ground support requirements on the levels beneath the sill pillar.

As mining progresses through the pillar, higher induced stress is possible and may result in further rock mass failure. Based on the EGM and numerical modelling it is unlikely the rock mass will build up significant stress to fail violently or with high energy (Differential stress is typically < 30 MPa, intact strength is between 80 MPa to 100 MPa, and mean GSI is approximately 60); rather, the rock mass is more likely to fail through block rotation and loosening which could be supported with standard ground support elements (albeit at heavier densities / spacing).

16.2.5 UNDERGROUND INFRASTRUCTURE GROUND SUPPORT

A series of ground support assessments were completed based on a range of underground excavation types and anticipated ground conditions. Assessment methods include empirical charts (Barton and Grimstad), RocScience Unwedge assessment, and RocScience RS2 assessments based on induced stress inputs from the EX3 model. The ground support recommendations are based on two geologic formations (SFD and S1_S2) which house the majority of the infrastructure (>95%), three ground character categories (Q'< 1, 1 <Q'< 10, and 10 <Q'), and infrastructure type (long-term, temporary, unique, intersections). Key considerations are that temporary headings prioritise installation of split-sets and long-term and unique infrastructure use fully grouted resin rebar (or equivalent). Fibre reinforced shotcrete (FRS) is only considered for the worst ground support category (Q' < 1) or for specific critical chambers, while welded wire mesh (WWM) is used as surface support for the other categories. The recommended ground support follows Table 16.3 to Table 16.6.

Table 16.3 – Recommended Ground Support for Temporary Headings

Development	Formation	Rock Mass Quality (Q')	Minimum Bolt Pattern Density	Surface Support
Sub-Level Ore Drift	S1_S2	> 10	Back/Shoulders: SS-33 2.7 m long at 1.0 m x 1.0 m Ribs: SS-33 2.7 m long at 1.0 m x 1.0 m	6 Gauge WWM
		1 < Q' < 10	Back/Shoulders: SS-33 3.0 m long at 1.0 m x 1.0 m Ribs: SS-33 2.7 m long at 1.0 m x 1.0 m	6 Gauge WWM
		< 1	Back/Shoulders: Resin Rebar 2.4 m long at 1.2 m x 1.2 m Ribs: Resin Rebar 2.4 m long at 1.2 m x 1.2 m	75 mm FRS
Crosscut Incline and Decline	SFD	> 10	Back/Shoulders: SS-33 2.4 m long at 1.2 m x 1.2 m Ribs: SS-33 3.0 m long at 1.0 m x 1.0 m	6 Gauge WWM
		1 < Q' < 10	Back/Shoulders: SS-33 3.3 m long at 1.0 m x 1.0 m Ribs: Resin Rebar 3.0 m long at 1.4 m x 1.4 m	6 Gauge WWM
		< 1	Back/Shoulders: Resin Rebar 3.0 m long at 1.2 m x 1.2 m Ribs: Resin Rebar 3.0 m long at 1.2 m x 1.2 m	75 mm FRS
	S1_S2	> 10	Back/Shoulders: SS-33 2.7 m long at 1.0 m x 1.0 m Ribs: SS-33 2.7 m long at 1.0 m x 1.0 m	6 Gauge WWM
		1 < Q' < 10	Back/Shoulders: SS-33 3.3 m long at 1.0 m x 1.0 m Ribs: Resin Rebar 3.0 m long at 1.4 m x 1.4 m	6 Gauge WWM
		< 1	Back/Shoulders: Resin Rebar 3.0 m long at 1.2 m x 1.2 m Ribs: Resin Rebar 3.0 m long at 1.2 m x 1.2 m	75 mm FRS
First 6 months of Upper and Lower Decline	SFD	> 10	Back/Shoulders: SS-33 2.4 m long at 1.4 m x 1.4 m Ribs: SS-33 2.4 m long at 1.4 m x 1.4 m	6 Gauge WWM
		1 < Q' < 10	Back/Shoulders: SS-33 2.4 m long at 1.4 m x 1.4 m Ribs: SS-33 2.4 m long at 1.4 m x 1.4 m	6 Gauge WWM
		< 1	N/A	N/A
	S1_S2	> 10	Back/Shoulders: SS-33 2.7 m long at 1.0 m x 1.0 m Ribs: SS-33 2.7 m long at 1.0 m x 1.0 m	6 Gauge WWM
		1 < Q' < 10	Back/Shoulders: SS-33 2.7 m long at 1.0 m x 1.0 m Ribs: SS-33 2.7 m long at 1.0 m x 1.0 m	6 Gauge WWM
		< 1	N/A	N/A

Notes:

- Temporary drifting support classes may be applied for up to 6 months in long-term infrastructure. Long-term ground support requirements (see next table) must be installed prior to exceeding 6 months. Intersections are excluded from this rule; long-term intersections must follow the long-term ground support requirements prior to building the intersection.
- A Factor of Safety FoS of 1.1 was used for design acceptance criteria of temporary drifting.
- Crosscut incline and declines should follow < 1 Rock Mass Quality ground support if within 30 m of the stopes (regardless of mapped rock mass quality), prior to beginning stope extraction due to anticipated loss of confinement during extraction.
- Of note, temporary headings are mostly within the induced stress field generated by extraction resulting in less favourable stress conditions.
- SS-33 bond strength is assumed at 35 tonnes / meter assuming excellent installation procedures.

Table 16.4 – Recommended Ground Support for Long-Term Headings

Development	Formation	Rock Mass Quality (Q')	Minimum Bolt Pattern Density	Surface Support
Long-term headings (e.g., upper and lower decline, spiral ramp, ramp level accesses, stockpile/remuck bay, cuddies, etc.	SFD	> 10	Back/Shoulders: Resin Rebar 2.4 m long at 1.4 m x 1.4 m Ribs: Resin Rebar 2.1 m long at 1.4 m x 1.4 m	6 Gauge WWM
		1 < Q' < 10	Back/Shoulders: Resin Rebar 2.4 m long at 1.4 m x 1.4 m Ribs: Resin Rebar 2.1 m long at 1.4 m x 1.4 m	6 Gauge WWM
		< 1	Back/Shoulders: Resin Rebar 3.0 m long at 1.4 m x 1.4 m Ribs: Resin Rebar 2.4 m long at 1.4 m x 1.4 m	100 mm FRS
	S1_S2	> 10	Back/Shoulders: Resin Rebar 2.4 m long at 1.4 m x 1.4 m Ribs: Resin Rebar 2.4 m long at 1.4 m x 1.4 m	6 Gauge WWM
		1 < Q' < 10	Back/Shoulders: Resin Rebar 2.4 m long at 1.4 m x 1.4 m Ribs: Resin Rebar 2.4 m long at 1.4 m x 1.4 m	6 Gauge WWM
		< 1	Back/Shoulders: Resin Rebar 3.0 m long at 1.4 m x 1.4 m Ribs: Resin Rebar 2.7 m long at 1.4 m x 1.4 m	100 mm FRS

Notes:
- A FoS of 1.5 was used for design acceptance criteria of long-term drifting.
- Of note, long-term headings are outside of the induced stress field generated by extraction resulting in more favourable stress conditions.

Table 16.5 – Recommended Ground Support for Unique Infrastructure

Development	Minimum Bolt Pattern Density	Surface Support
Fuel and Lube Bay (6 mW x 5 mH)	Back/Shoulders: Resin Rebar 3.0 m long at 1.2 m x 1.2 m Ribs: Resin Rebar 2.4 m long at 1.4 m x 1.4 m	50 mm FRS
Explosive Storage (7 mW x 6 mH)	Back/Shoulders: Resin Rebar 2.4 m long at 1.2 m x 1.2 m Back/Shoulders: Cable Bolts (15.6 mm dia.) 5.0 m long at 2.0 m x 2.0 m Ribs: Resin Rebar 3.0 m long at 1.2 m x 1.2 m	50 mm FRS
Maintenance Bay (9 mW x 7.5m H)	Back/Shoulders: Resin Rebar 2.4 m long at 1.2 m x 1.2 m Back/Shoulders: Cable Bolts (15.6 mm dia.) 6.0 m long at 2.0 m x 2.0 m Ribs: Resin Rebar 2.4 m long at 1.2 m x 1.2 m Rib: Cable Bolts (15.6 mm dia.) 5.0 m long at 2.0 m x 2.0 m	50 mm FRS

Notes:
- A FoS of 1.5 was used for design acceptance criteria of long-term drifting.
- Construction of these facilities assume they will only be placed in good ground (Q' > 10).

Table 16.6 – Recommended Ground Support for Intersections

Intersection Inscribed Circle Span	Minimum Bolt Pattern Density – Secondary Support
> 6.0 m and <= 7.5 m	Back/Shoulders: Cable Bolts (15.6 mm dia.) 5.0 m long at 2.0 m x 2.0 m
> 7.5 m and <= 12.5 m	Back/Shoulders: Cable Bolts (15.6 mm dia.) 6.0 m long at 2.0 m x 2.0 m

Notes:

- A FoS of 1.5 was used for design acceptance criteria of infrastructure.
- Primary ground support for intersections will follow primary ground support requirements laid out for temporary / long-term drifting in the above tables.
- There is no difference between temporary or long-term drifting in regard to secondary support requirements.
- The PFS mine design does not include intersections with inscribed circle spans > 12.5 m
- Construction of intersections assume they will only be placed in good ground (Q' > 10)

16.2.5.1 Vertical Stability and Ground Support

Based on the PFS mine design long term vertical developments were classified as either a surface vent raise and underground vent raises. There is only one surface vent raise and it has a planned diameter of 4 m, depth of 300 m and the base case location is approximately north of the spiral ramp. The underground vent raises have a planned diameter of 3 m, (assumed depths of 35-40 m) and connect the ventilation from level to level within the footprint of the spiral ramp.

An empirical vertical stability assessment was completed using McCracken and Stacey (1989). This assessment method is based on a modified Barton Q value defined as Q_R. Q_R is plotted against raise diameter and a ventilation raise was considered stable (10 years) if the PoF is 5% or less. During the PFS there was limited information on the rock mass quality of the raise locations and data was based on the general rock mass conditions from the PFS underground geotechnical program and the nearest exploration drillholes.

The following is concluded and recommended from the surface vent raise stability assessment:

- A 4 m diameter unsupported opening is unlikely to be stable for the surface vent raise specifically from surface to 150 m vertical depth.
- Based on the estimated rock mass quality from surface to 150 m vertically ground support in the raise walls will be required (e.g. 50mm thick shotcrete and resin grouted 1.8 m long bolts on 1.5 m spacing) to maintain stability for 10 years.
- Based on the estimated rock mass a top-down excavation method (e.g. Alimak) should be planned from surface to 150 m vertically. Vertical raise boring excavation methods can be completed from 150 m to 300 m vertically (or sooner if good rock mass is encountered earlier).
- Drill a 300 m long pilot drill hole to collect local geotechnical data and update the vertical stability assessments. The pilot hole is planned to be drilling Q1 2025 by DPM.

The following is concluded and recommended from the underground vent raise stability assessment:

- Drill pilot holes and collect geotechnical data prior to excavation of each 3 m underground vent raise. This drilling can occur during mine operations.

- If poor rock mass quality is encountered in the pilot hole relocate the underground vent raise or install wall support (e.g., 500 mm thick shotcrete).

16.2.6 Portal Design Guidance

An Upper and a Lower Portal are required for the project to facilitate access to the underground. The declines dimensions for the PFS at the portals are the same dimension at 5.5 m Wide x 6 m High. During the data review no PFS underground geotechnical drillholes were near the portal locations. As such, all available PFS underground geotechnical drill holes and nearby exploration drill holes (drill hole photographs) were used to evaluate the near surface conditions for the portal designs.

The expected lithological unit for the upper and lower declines is the SFD (Monomict Breccia unit). For the selected locations, both portals are designed to be installed within an upwards sloping terrain. The portal designs are the same for both portals and will be required to be updated considering site specific data upon completion of geotechnical data acquisition. Based on the rock mass qualities (Barton Q of Extremely Poor to Poor from 2.5 m to 30 m vertically from surface) and exploration drill hole photographs the following Table 16.7 outlines the estimated percentage of recommended ground support type for the initial development of the tunnel. The ground support will be updated based on the depth of terrace design to the portal areas and actual conditions encountered.

The boxcut design for each portal area is conceptualised to be benched, with an initial 8.5 m high bench, followed by a 10 m high bench with an additional 6.3 m to allow for the decline excavation. The two benches are separated by a horizontal 4 m wide bench. The overburden at surface (if present), will be sloped at 25° and removed from the crest for a distance of 4 m. Bench faces are planned at 70°. The upper bench height is lower than the lower bench height due to expected lower quality rock. The benches are conceptualised to be excavated in 2.5 m lifts, with the requirement of support to be installed prior to proceeding to the next lift. A bolt spacing of 1.4 m and a bolt length of 5.4 m has been assumed for the portal design. Fibre reinforced shotcrete will be required on all surfaces to protect against rockfalls, with an average thickness of 15 cm.

The following is recommended for the next level of study:

- Site investigation program of the portal entrances and along the tunnel alignments. A program has been developed with DPM and will be completed during the next stage of study.
- An assessment for optimisation of the boxcuts to minimise material extraction volumes, while ensuring that conditions at the portal design depth are suitable for establishing a stable boxcut and portal/decline.
- A comprehensive assessment including consideration for kinematics and other types of analyses (i.e., plug failure, dead weight loading, numerical studies) after site specific data is available.

Table 16.7 – Support Recommendations for the Initial Declines (after NGI, 2015)

Q_{portal} Minimum	Q_{portal} Maximum	Support Category (After NGI, 2015)	Back		Walls		% of Tunnel	Other Support
Barton Q Descriptor			Bolt Spacing (m)	Proposed Bolting Length (m)	Bolt Spacing (m)	Proposed Bolting Length (m)		
0.01	0.04	7	1.1	3.0	1.2	2.1	1	Fibre reinforced shotcrete >15 cm and systematic bolting plus shotcrete pillars (double layer of rebar, with 45 cm of sprayed concrete) or lattice girders with an average centre to centre spacing of 2.3 m
Extremely Poor								
0.04	0.15	6	1.2	3.0	1.4	2.1	19	Fibre reinforced shotcrete 12 to 15 cm and systematic bolting plus shotcrete pillars (single layer of rebar with 30 cm of sprayed concrete) or lattice girders with an average centre to centre spacing of 2.6 m
Extremely Poor to Very Poor								
0.15	0.5	5	1.3	3.0	1.4	2.1	40	Fibre reinforced shotcrete 9 to 12 cm and systematic bolting plus shotcrete pillars or lattice girders with an average centre to centre spacing of 4.0 m
Very Poor								
0.5	1.5	4	1.4	3.0	1.4	2.1	40	Fibre reinforced shotcrete 6 to 9 cm and systematic bolting
Very Poor to Poor								

16.2.7 Backfill Strength Estimate

A backfill strength assessment was completed using the Mitchell et. al,1982 method using the following PFS mine design parameters:

- Overhand mining method;
- Backfill density = 2000 kg/m³;
- UCS strength = 0.35 and 0.5 MPa;
- Backfill cohesion = 30°;
- Stope plug thickness (stope height) = 20 m;
- Maximum stope span = 15 m;
- Maximum stope strike = 30 m; and
- Stope dip = 65° and 90°.

The Factor of Safety (FoS) associated with the following backfill strengths and stope dips are as follows:

- 0.5 MPa for 65° and 90° stope dip = FoS of 2.0; and
- 0.35 MPa for 65° and 90° stope dip = FoS of 1.5.

A backfill strength of 0.35 or 0.5 MPa is acceptable for the overhand mining method and the selected strengths can be based on other mine design factors (e.g., mine production schedule, backfill laboratory strengths achieved). The minimum backfill strength should not be below 0.35 MPa for all planned stope configurations for the overhand mining method.

16.3 Hydrogeology

The objectives of the hydrogeological investigations carried out in support of the PFS for the Čoka Rakita Project were to assess the groundwater conditions within the area of the proposed surface and underground mine infrastructure development.

Hydrogeological data collection including hydraulic testing and borehole instrumentation for groundwater monitoring was completed in select drillholes throughout the 2024 field investigation campaign by the University of Belgrade field staff and reviewed by WSP. WSP's role included QA/QC of the field data, and review of field documentation, data analyses, and factual reporting completed by University of Belgrade.

Following completion of the PFS field investigation program, WSP compiled and reviewed the data and reports provided by the University of Belgrade as part of a conceptual hydrogeological assessment of the project area. The conceptual hydrogeological model formed the bases for development of a 3D numerical groundwater flow model that was used to predict groundwater inflows into the underground development during the LOM in support of the underground water balance and design of the dewatering infrastructure.

The data used for the development of the conceptual and numerical hydrogeological models are summarised in the following subsections.

16.3.1 GROUNDWATER LEVELS AND FLOW DIRECTIONS

Vibrating wire piezometers (VWP) and standpipe piezometers were installed at the project site as part of the 2024 PFS field investigations to obtain information on groundwater levels, gradients, and flow directions. The collected data indicate the groundwater flow direction 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. 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. No information is currently available on water levels or hydraulic gradients associated with the limestone unit below the planned mine development.

The water levels collected from standpipe monitoring wells installed in the project area vary between 8 m below ground surface in the valleys up to 98 m below ground surface at higher elevations. Based on data obtained from the nearby Timok project, 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 groundwater levels was recorded in the area of the Bigar Hill deposit, approximately 3 km NW from the Čoka Rakita Project.

The preliminary VWP data indicate a strong upward hydraulic gradient in the area of boreholes HG004, HG005, HG006 and HG007, and a downward hydraulic gradient in the area of borehole HG002. Natural groundwater gradients reflect recharge over the mountain tops and groundwater discharge along the valley floor. Continued drilling within the Project area may be impacting local groundwater conditions, and the limited datasets obtained from the recently installed VWPs may not represent stabilised conditions. Further monitoring is required to confirm the stabilisation of the local groundwater conditions.

16.3.2 HYDROSTRATIGRAPHY AND HYDRAULIC PARAMETERS

Based on the data obtained from the field investigations, a total of six (6) stratigraphic units with similar hydraulic conductivities have been identified within the project area:

- Volcanic Breccia (XMO);
- Sandstone (SSS);

- Sandstone and Diorite (SSS/GDI);
- Diorite (GDI);
- Quartzite (MQZ);
- Monzonite (GMO); and
- Faults and fractures (FAG).

The hydraulic conductivities of the individual hydrostratigraphic units were derived from in situ single-well response testing in open boreholes using packers. The test intervals extended from near ground surface up to 714.7 m depth, and the majority of testing was performed within the volcanic breccia (46%) and monzonite units (19%). A summary of the hydraulic conductivity values assigned to the individual hydrostratigraphic units including number of tests conducted in each unit is presented in Table 16.8.

Table 16.8 – Summary of Hydraulic Conductivity Results

Basic Statistics Kf (m/s)	XMO	SSS	SSS, GDI	GDI	MQZ	GMO	ALL	FAG
	Volcanic breccia	Sandstone	Sandstone, Diorite	Diorite	Quartzite	Monzonite		Structures
Average	1.7E-07	2.6E-07	8.5E-08	3.5E-08	2.7E-07	2.0E-07	1.7E-07	2.4E-07
Geo Mean	4.5E-08	1.2E-07	2.1E-08	1.8E-08	3.3E-08	2.6E-08	4.0E-08	4.7E-08
Median	4.9E-08	2.IE-07	9.4E-09	2.0E-08	1.1E-07	1.7E-08	4.6E-08	5.4E-08
max	1.5E-06	6.7E-07	3.2E-07	1.7E-07	8.8E-07	1.3E-06	1.5E-06	1.3E-06
min	5.0E-10	1.4E-09	1.4E-09	1.9E-09	2.1E-10	4.2E-10	2.1E-10	2.1E-10
# of Samples	77	22	9	17	9	32	166	64

The highest hydraulic conductivity value of 2E-06 m/s is associated with the volcanic breccia, and the lowest hydraulic conductivity of 4E-10 m/s was calculated for the monzonite unit. The geometric mean of hydraulic conductivities calculated for the individual hydrostratigraphic units varies between 6E-09 m/s and 1E-07m/s. The geometric mean of hydraulic conductivity results for all in situ permeability tests completed as part of the Čoka Rakita PFS is approximately 3E-08 m/s. A slight decrease in hydraulic conductivity with depth was observed in most of the hydrostratigraphic units.

Four (4) major structural trends (E-W, ENE-WNW, N-S and NW-SE) were identified by the DPM in the Čoka Rakita project area. A total of 64 test intervals included brittle zones such as faults, joints, or fractures. The geometric mean of hydraulic conductivity values calculated for all the tested intervals with structures is similar to the geometric mean value of the hydraulic conductivities calculated for the individual hydrostratigraphic units. Zones of increased hydraulic conductivity appear to be localised and cannot be attributed to any major regional structures.

Karstic limestone was identified in the area of the Timok project located to the north from the Čoka Rakita study area. Based on geotechnical data obtained during the field investigation at Čoka Rakita, limestone is present at depth below the planned mine development. As part of the hydrogeological data gap analyses conducted in June 2024 it was recommended to extend one geotechnical borehole (RADDHG008) by approximately 350 m to reach the underlying limestone unit with an objective to identify potential connection of the ore body zone with the limestone. Hydrogeological testing, geophysical logging, and installation of two VWP piezometers was proposed in this borehole.

16.3.3 Numerical Groundwater Flow Model and Inflow Predictions

The objectives of the numerical groundwater flow modelling were to develop a model to reflect the current understanding of site hydrogeology and use the model to estimate the quantity of potential groundwater inflows to the planned underground mine development. To achieve this objective, a deterministic approach was used where a 3D numerical groundwater model was constructed and calibrated to represent the "best estimate" of groundwater flow conditions based on the data collected during the 2024 PFS field investigations. The calibrated model was subsequently adapted to include the mine components so that it could be used for the predictive simulations.

The FEFLOW modelling package was selected for the numerical groundwater flow modelling at the project site. FEFLOW (Version 8.1) is a commercially available 3D finite element groundwater flow and solute transport modelling package developed by DHI-WASY. FEFLOW is capable of simulating saturated and unsaturated groundwater flow, solute and heat transport in three dimensions. It also has capability to simulate discrete fracture flow (in 1D linear and/or 2D planar fractures). FEFLOW was selected given its capabilities to efficiently discretise the local features around the mine in parallel with the regional extent of the catchments that contribute to the overall groundwater balance.

The current-conditions calibrated model was used as a starting point for the development of forecast simulations. Seepage boundaries were added to represent the underground workings, and these were activated progressively with time and depth on an annual basis according to the preliminary Phase 2 mine plan and schedule (available at the time of the model development). The forecast simulation was run transiently over a period of 8 years (-2 to +6) with groundwater flow rates to the seepage boundaries tracked based on mine level and section of the mine development. Simulated groundwater drawdown was calculated relative to the predevelopment conditions simulation. Due to the schedule difference between Phase 2 (preliminary) and Phase 3 (final) PFS mine plan, Year 7 inflows were added based on an average of Year 5 and Year 6 inflows.

Volumes of the simulated groundwater inflows to the mine workings are provided in Table 16.9. The values shown on the table are approximating an average flow rate over an annual time period. Total predicted annual groundwater inflows to the mine reached a peak of approximately 20 L/s during the Mine Year 1, then remained relatively steady until the end of the simulation with inflows varying from about 17.5 L/s to 19.0 L/s.

Table 16.9 – Inflow Predictions over Life-of-Mine

Year	Inflow Rate (m^3/d)	Inflow Rate (L/s)
-2	1,187	13.7
-1	1,721	19.9
1	1,654	19.1
2	1,589	18.4
3	1,548	17.9
4	1,521	17.6
5	1,594	18.5
6	1,527	17.7
7*	1,555	18.1

*Average of Year 5 and Year 6 inflow rate

16.4 Mining Method

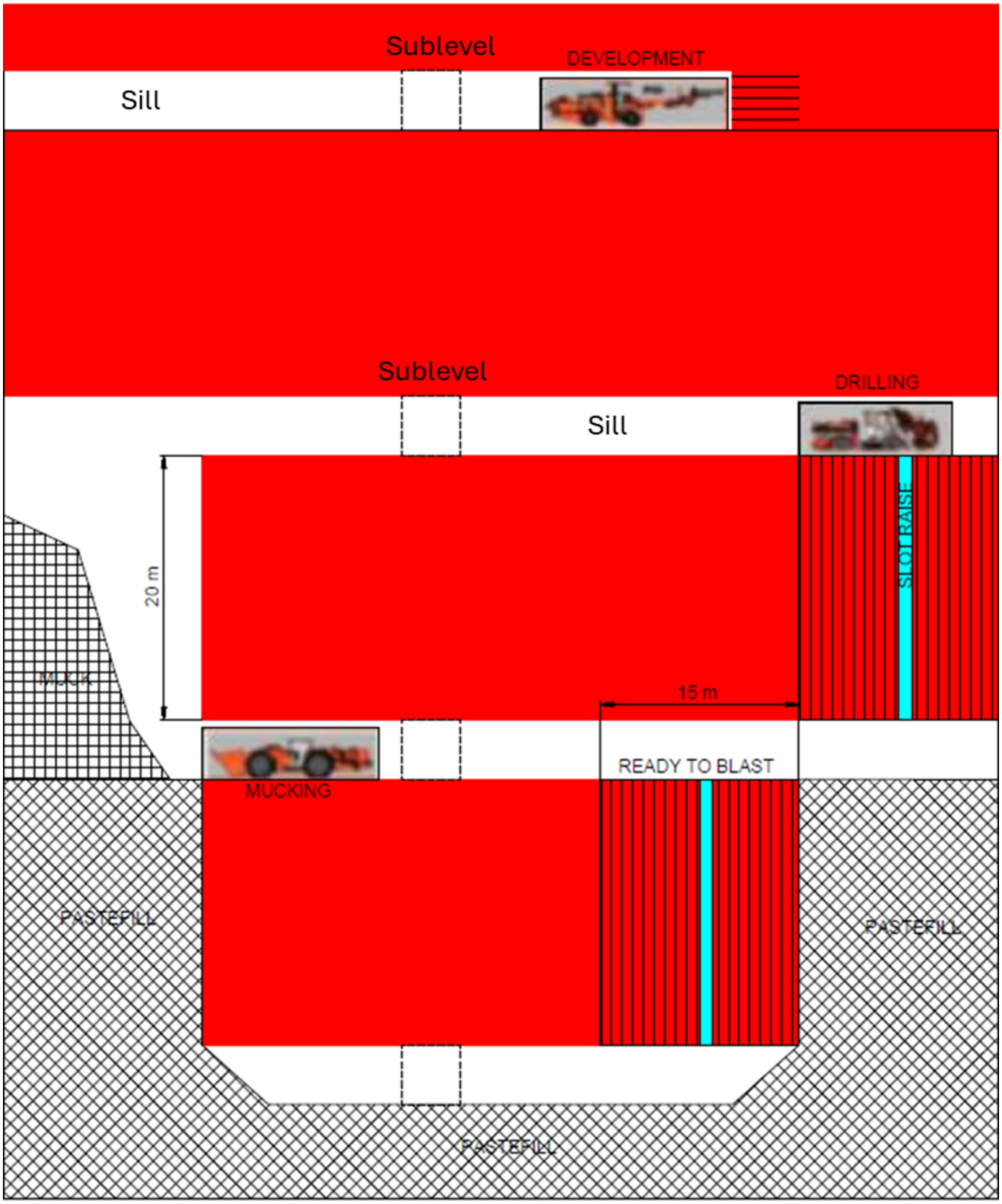
16.4.1 Sublevel Stoping

DPM plans to mine the deposit with sublevel long-hole open stoping. This method divides the deposit into sublevels, and the ore 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. A trade-off analysis comparing the longitudinal and transverse mining methods revealed no advantages in terms of schedule or overall development length for the transverse method. Consequently, the decision was made to proceed with the long-hole longitudinal mining method as illustrated in Figure 16.6.

Figure 16.7 illustrates the strategy for developing the sublevels, using a plan view of the 440 and 460 sublevels as an example. Most of the ore 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 ore. These production drives will be driven from crosscuts extending across the deposit from the hanging wall to the footwall. The mining sequence will be from Deposit's FW to HW and bottom up, the sill pillar is located in 420L.

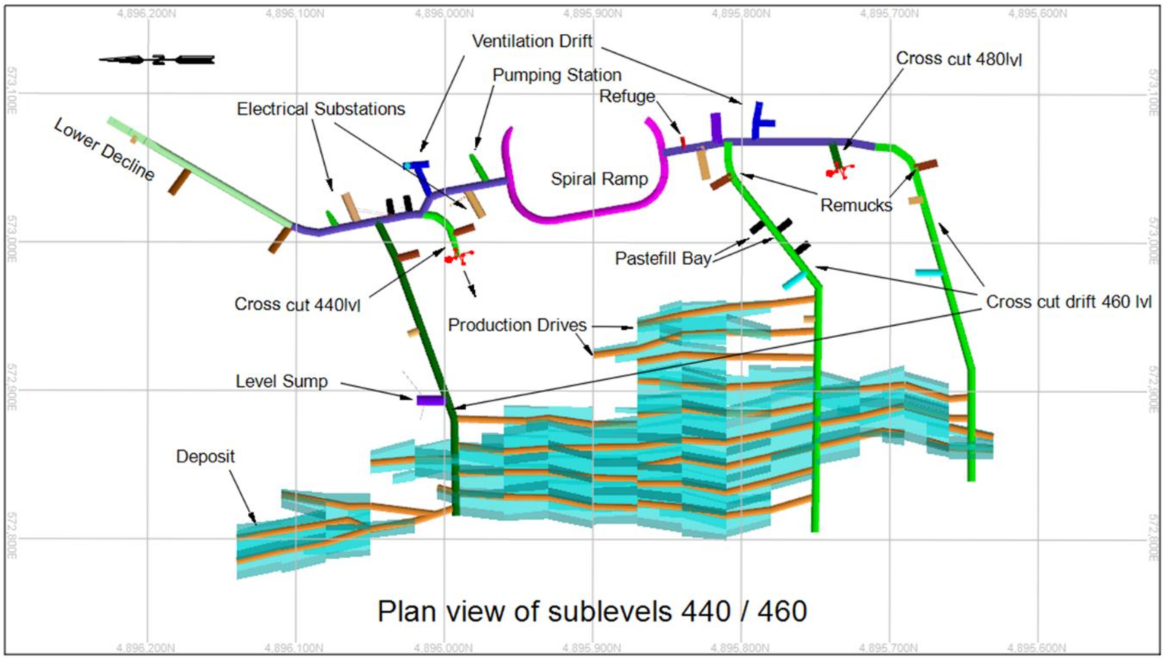
The longitudinal stopes will be mined in a retreating fashion, advancing in series, one after the other, towards the crosscuts. In compliance with geotechnical guidelines, the dimensions of the stopes will be 30 m long, 20 m wide (from Footwall to Hanging wall) and 20 m high, corresponding to the sublevel interval.

Figure 16.6 – Longitudinal Long-Hole Mining Method – Long Section



Source: WSP, 2024

Figure 16.7 – Example of the Development Required for Mining a Sublevel



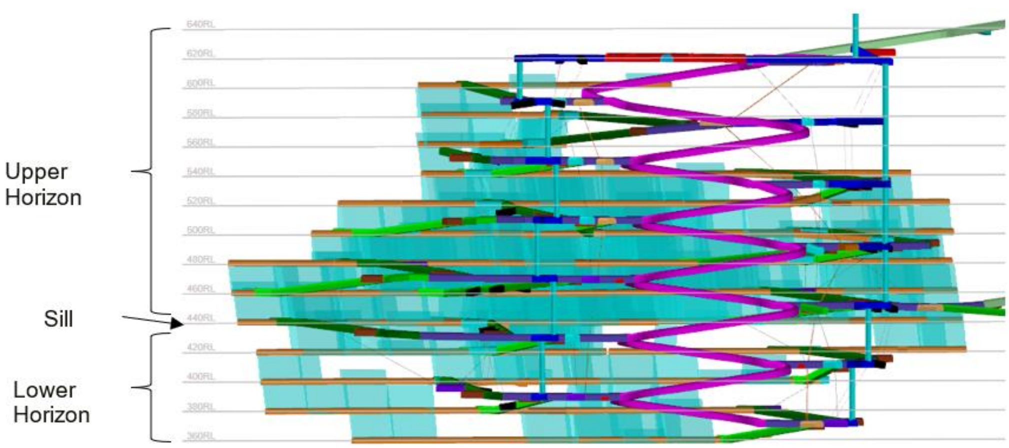
Source: WSP, 2024

16.4.2 SEQUENCING

The mine is divided into two horizons: the upper horizon (440L to 620L) and the lower horizon (360L to 440L), with a sill pillar at 420L as presented in Figure 16.8.

Mining will commence at the bottom of the upper horizon on 440L and progress upward in an ascending sequence. Once the upper horizon reaches 560L, mining of the lower horizon will begin at 360L, also progressing upward in a bottom-up sequence.

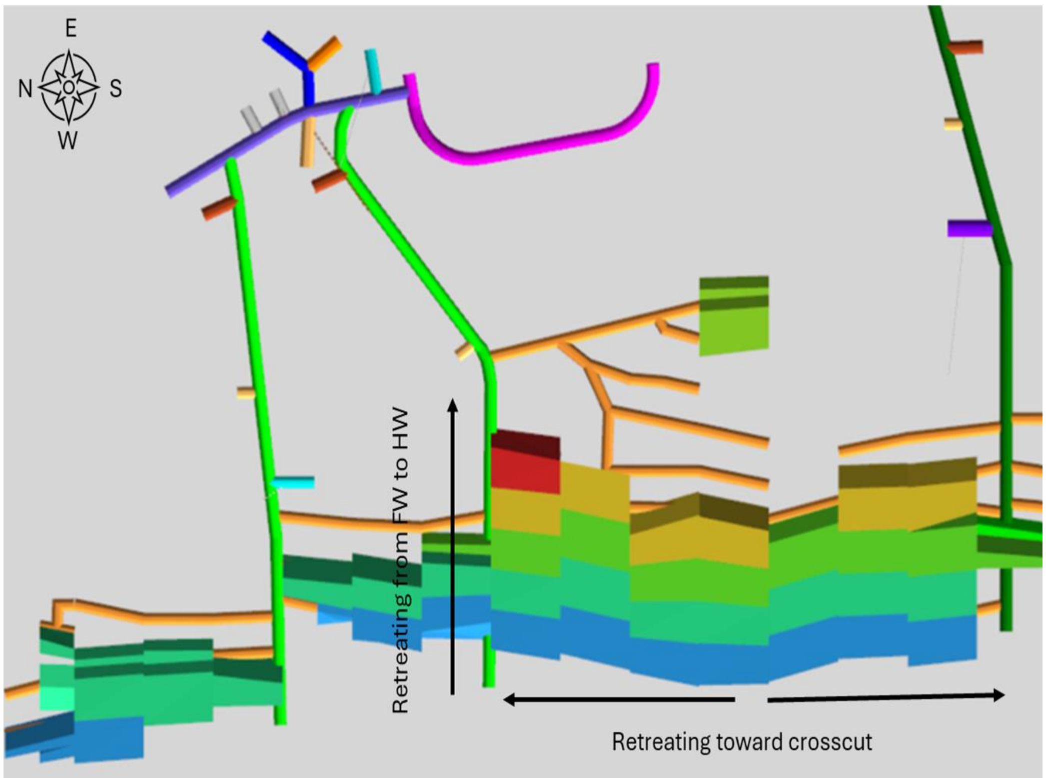
Figure 16.8 – East to West View of the Mining Horizons



Source: WSP, 2024

In the East-West direction, the mining sequence will retreat from FW to HW towards the crosscut drifts as represented on a typical level in Figure 16.9. Stoping operations will generally occur on more than one sublevel at a time to ensure enough areas are active to meet production targets.

Figure 16.9 – Sequence Advancement on Typical Level



Source: WSP, 2024

16.4.3 DRILLING AND BLASTING

All decline and lateral excavations will be developed using drill and blast methods using three (3) two-boom electric-hydraulic jumbos and two (2) mobile explosives loaders.

The mine will have three (3) production drill rigs, two (2) versatile top hammer rigs will be used for drilling downholes as well as uppers and one support top hammer rig will be used as a spare for service holes and longholes when needed.

At first a 2.1m x 2.1 m conventional drop raise is drilled and then the 89 mm diameter production fanned holes will be drilled in rings with a 2.1 m burden and 2.1 m spacing.

Production ring blast holes will be loaded with bulk emulsion explosives and fired from the slot raise, retreating towards the mucking access.

16.4.4 Mucking and Hauling

WSP has completed material handling analysis to optimise the mine efficiency and profitability. The study has examined different underground ore transport systems, including diesel, electric, and articulated truck.

A selection matrix based on decision key factors, showed that the best solution (with the assumptions of the study) is the combination of the TH545i Sandvik 45t truck and the LH515i Sandvik 15 t LHD.

Ore will be mucked using remote-controlled LHDs, transported, and dumped into the nearest remuck bay. The ore will be loaded onto muck trucks in the level access and hauled to the surface via the spiral ramp and the declines.

Production tonnes from 480 to 600 level will be trucked via the upper decline while those from 460 to 360 level will be trucked via the lower decline, this division is based on the shortest distance to the surface stockpile.

16.4.5 Backfill

Backfill is required to maximise mineral extraction and maintain stope stability. Three (3) types of backfill are used at Čoka Rakita: unconsolidated waste (rock fill); cemented rock fill (CRF); and cemented paste fill (CPF). Rock fill and CRF are used where the mining sequence allows it (no adjacent or beyond stope).

Unconsolidated rock fill (URF) is used to fill empty stopes where possible, minimising costs associated with transporting waste rock to the surface. Development waste material is utilised for both URF and CRF. Rock fill is transported to stopes via LHDs. Cement slurry for CRF is produced in batches at a surface plant and transferred underground using a transmixer. CRF is mixed by LHDs in an excavated development bay and placed with mobile equipment.

A surface paste backfill plant will process available tailings mixed with a cement binder to produce CPF. The paste fill will be delivered underground through a pipeline in the upper decline and distributed throughout the mine via pipelines and a network of boreholes.

16.5 Mine Design

16.5.1 Stope Design

The stope dimensions design parameters were established through consultation with the WSP geotechnical specialist and are summarised in Table 16.10.

The average stope size is 17,500 t, and the annual production is to be 855,000 t, as such many stopes must be mined concurrently. Stopes design and scheduling were performed using Deswik software.

Table 16.10 – Mine Design Parameters – Stopes

Stope Design	Parameter
Mining Method	Sublevel Stoping
External Dilution	10%
Mining Recovery	95%
Stope Dimensions	
Width	20 m
Height	20 m
Length	30 m
Longhole Drilling	
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)	6.40 t/m-drilled
Blasting	
Explosive	Bulk Emulsion
Powder Factor	1.97 kg/m³

Source: DPM, 2024

16.5.2 Development Design

The main infrastructure is placed in the orebody hanging wall with two (2) access points, consisting of declines totaling 2,192 m in length connected to an internal ramp. Both the declines and the ramp are driven at a maximum gradient of -15%.

Lateral development will consist of all level access, crosscuts drives, ore sills, and excavations for infrastructure. The main development heading profiles for the underground workings are presented in Table 16.11.

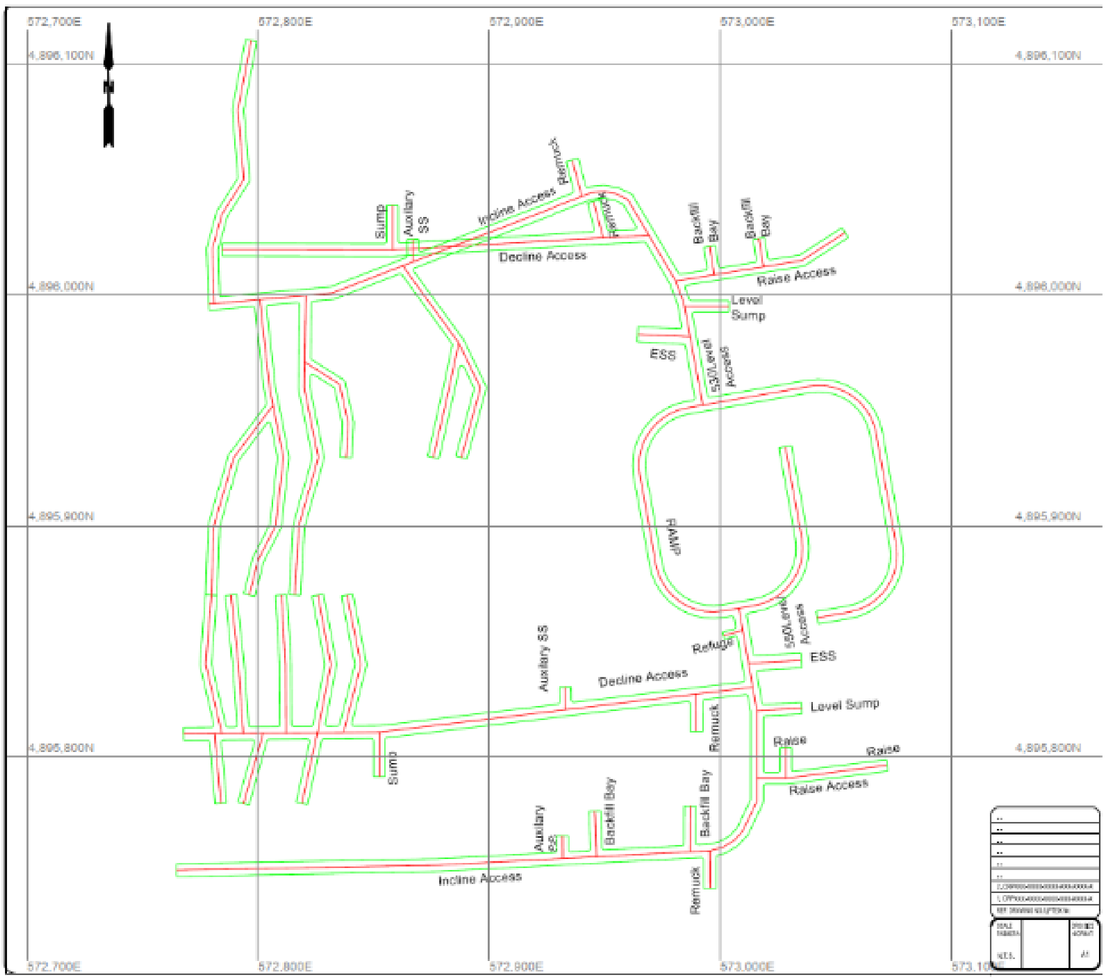
Table 16.11 – Main Development Heading Profiles

Item	Width (m)	Height (m)
Surface Decline	5.5	6.0
Spiral Ramp	5.5	5.5
Ramp Level Access	5.5	5.5
Xcut Decline	5.5	5.5

Item	Width (m)	Height (m)
Sublevel (Ore) Drift	5.0	4.5
Xcut Decline	5.5	5.5
Pumping Station	5.0	5.0
Maintenance Bay	9.0	7.5
Ventilation Raise	4 m diameter	

Figure 16.10 is a plan schematic showing a typical level. Each level consists of electrical stations and backfill bays, sumps, ventilation accesses and remuck bays.

Figure 16.10 – Typical Level Layout



Source: WSP, 2024

16.6 Mine Access and Underground Facilities

The deposit will be accessed by driving two (2) independent declines (upper and lower). Both declines will be utilised for material haulage and sized appropriately for all larger mobile equipment. Both upper and lower declines are to be developed in parallel which will provide underground access for personnel and equipment. Both declines will function as air intakes for the mine ventilation system at its end state.

The development of a spiral ramp will commence from both faces of the upper and lower declines. The development of the spiral ramp will continue to be developed until they converge and breakthrough near 500L. It is not expected that the lower ramp to the final levels will be developed until Year 3 of the mine life. The spiral ramp will provide access to all levels of the mine and enable the transportation of ore to surface throughout the life of the mine, and waste during the years prior to the Mill operation. Additionally, level access headings will be developed off the spiral ramp to access the sublevels for mining the deposit.

The mine also has numerous underground facilities and services to support the Čoka Rakita operation. The following items are a summary of the various infrastructure and services required underground.

16.6.1 Mine Dewatering System

The Mine dewatering system is made up of a hierarchy of sumps which collect all water generated from various sources and activities in the mine into intermediate sumps consisting of level sumps and secondary sumps, which are then pumped to the central Main Pumping Station and deliver it to the Mill via the lower decline, for treatment, reuse, or disposal.

The dewatering system is designed to utilise gravity flow between sumps as much as possible to minimise the number of pumps and pipelines required.

16.6.2 Electrical Requirements

Various electrical hardware is required throughout the underground mine workings. Routing of the main cables from surface are extended to main electrical substations at each level and branch out to other electrical equipment to form a redundant loop underground and provide power to operations. The electrical excavations consist of the main electrical substation, auxiliary electrical substation and Distribution Equipment.

16.6.3 Service Support Boreholes

Boreholes are required throughout the mine to direct services such as dewatering, electrical, communications and paste backfill. The development of services boreholes assist by lessening the distance from hole to hole which increases functionality and reduces the cost on hard piping other having to route cables at much greater lengths. Boreholes are preferable to be drilled at steeper dipping angle of ±70° which also helps with functionality and installation of services inside the holes

16.6.4 Service Support Infrastructure

There is to be one (1) fuel bay located on 480L. Most of the underground mobile fleet are to be fuelled on surface, however if required, equipment can access this area to fuel up. It is expected that slower moving equipment and the Toyota pickup trucks will utilise the fuel bay or be fuelled by the mine utility vehicle with the fuel cassette.

16.6.5 Maintenance Shop and Auxiliary Facilities

There is a maintenance shop on 620L at the bottom of the upper decline. The maintenance shop and its auxiliary facilities are listed below.

- Maintenance Bay.
- Wash Bay.
- Emergency Preventative Maintenance Bay.
- Warehouse.
- Office/Lunchroom.
- Lube Bay.
- Waste Oil Bay.
- Tire Changing Bay.
- Latrine.

The entire area of the maintenance shop and its auxiliary facilities will have concrete floor, 150 mm thick. The entrance is oversized to accommodate parking of various mine fleet units in the area to report to or utilise the office/lunchroom and or latrine. The flow of ventilation is from the entrance to the rear of the shop with the air existing the shop through a regulator just beyond the latrine. Typical routine maintenance on the underground mining fleet is to be done on surface while emergency preventative maintenance and occasional maintenance will be done on slower moving underground equipment in the underground shop with the assistance of overhead cranes, tools found in the warehouse and a dedicated tire changing bay. Small amounts of mobile fluids will be stored underground such as hydraulic oil and grease which also consists of a distribution system. Lastly, prior to entering the underground maintenance shop, the mobile fleet will be cleaned in the nearby wash bay.

16.6.6 Explosive Magazines

The mine explosive magazines consist of three (3) differently identified areas which are the emulsion storage bay, explosive storage bay and detonator storage bay situated on 620L near the return air access drift. The explosive and detonator bays must be in an area such that they respect Serbian proximity regulations. The access corridors to these areas must have at least two (2) 90° corners and the chambers must have deflection chambers of at least 3 m depth.

Explosive materials are to be transported to site via the preferred vendor of DPM. There is to be a small emulsion storage transfer facility on surface for the vendor to transfer the emulsion from transportation vehicle into dedicated vessels which will be transported underground by the mine utility vehicle.

16.6.7 Paste Fill Underground Distribution System Infrastructure

The underground backfill system (UDS) infrastructure consists of two separate areas. The CPF distribution bay which is utilised to collar the CPF services boreholes and deliver paste backfill to a dedicated paste backfill receiving bay on the level below it, where the boreholes terminate.

It should be noted that there are two (2) CPF delivery bays of the same design as a typical CPF delivery bay, that serve to deliver paste to lower levels while bypassing intermediate levels which connect to two (2) CPF receiving bays of the same typical design. The bypass stations are from 620L to 470L and from 470L to 390L.

16.6.8 Miscellaneous Services and Infrastructure

The other various services and miscellaneous underground infrastructure required to assist the mine in its daily operations are:

- Compressed Air - Portable air compressors for underground mining equipment that require air and additional smaller compressors for tools and dispensing systems.
- Clarified Water (Mine Water) - Clarified water is to be used for production operations, washing the mobile fleet and dust suppression among other miscellaneous uses.
- LTE Network, VoiP Phone, Micro-seismic - Communications services are to be routed underground via messenger cable and boreholes as required to provide connectivity to micro-seismic instrumentation, leaky feeder, LTE, VoiP phone
- CRF Mixing Bay - The mine is to utilise dedicated repurposed drifts such that rock and cement slurry will be placed into the drift and mixed prior to dumping the mixture into an open stope as backfill.
- Underground Mine Roadbed - Aggregates from the nearby zagubica quarry will be used to construct and maintain the underground mine roads.
- Alarm System (Stench Gas) - An alarm system is to be situated on surface near the portal intake fans for direct injection at both the upper and lower declines in case of an emergency.

16.7 Mine Safety Infrastructure

The Čoka Rakita mine will have the following infrastructure related to mine safety:

- The ventilation return air raises connecting the sublevels will be equipped with ladderways and lighting, providing an emergency escape alternative to the spiral ramp.
- Both declines will provide a second means of egress from the mine if travel via the one of the two other declines is not feasible.
- Mine-rescue equipment will be provided for use by the mine-rescue team.
- The mine will have a dedicated mine rescue vehicle (ambulance) for emergency medical response.
- Three (3) portable mine rescue chambers will be strategically positioned throughout the mine, each designed and equipped to accommodate up to eight (8) people for 36 hours.
- The mine will have a stench gas system to provide warning in an emergency installed on surface connected directly to the air intakes for quick injection.

16.8 Mine Equipment

The Čoka Rakita Mine will be a mechanised operation utilising rubber-tired diesel equipment for all mining activities. Table 16.12 lists the mobile equipment planned for the mine, indicating the maximum number of units of each type required.

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 three (3) longhole drill rigs.

The 15-t 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; and two (2) utility vehicles with modular cassettes to act as a boom truck, fuel truck and lube truck. For tasks requiring work at height (installation of piping, ducting, hanging cable, etc.), personnel will operate from a platform elevated by a wheel loader and utilising a personnel basket in lieu of the loader bucks or forks.

Table 16.12 – Underground 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	Electro-hydraulic top hammer; capable for vertical and inclined fans; 115 mm longholes up to 54m in depth	2
Production Drill, Top Hammer	Electro-hydraulic top hammer; capable for vertical and inclined fans; 102 m longholes up to 38m in depth	1
Portable Air Compressor	Required for production drills as required	3
Emulsion Charger	Tunnels up to 65 m² cross section	2
Mucking and Haulage		
Mine Truck 45 t Payload	45 t payload	6
LHD 15 t Tramming Capacity	15 t tramming capacity	5
Ground Support		
Cable Bolter	Installs cable bolts up to 25 m long	1
Shotcrete Sprayer	20 m³/hour capacity	1
Transmixer	5.6 m³ capacity	2
Service		
Telehandler	4,000 kg capacity	3
Grader	Moldboard, 3658 mm Basic	1
Service Machine - Wheel Loader	Equipped with quick coupler for forks and bucket	2
Utility vehicle	Modular vehicle with different cassettes (boom, fuel tank and water tank options)	2
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	6-person capacity, two (2) wheels stretchers for injured persons	1

Source: DPM, 2024

16.9 Mine Personnel

Table 16.13 presents the annual staffing requirements for the underground mine for a full year production. This roster includes provisions for absences, vacations, and temporary vacancies. The underground mine will reach its peak workforce of 328 employees in Year 2 of the LOM plan.

The mine will operate on three (3) 8-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 its Chelopech mine in Bulgaria to provide training and ensure safe, and efficient operations. All development and production activities will be carried out by DPM employees, except raise boring.

Table 16.13 – Personnel Requirements at Mine Full Year Production

Department / Job Position	Total Number
Mining Production and Development	131
UG Site Services	66
Mechanical Maintenance	57
Electrical and Automation Maintenance	16
Technical Services	42
Total Personnel	**328**
Note: [1]HSE personnel included in admin cost in Section 21	

16.10 Pre-Production Schedule

The pre-production phase extends until the middle of Year 1, culminating with the mine attaining full production. The Project's pre-production phase will commence in the year prior to initiating the development of the declines. The principal activities during the year will include:

- Excavation of terrasses and portals for the upper and lower declines.
- Procurement of essential mining equipment, considering the lead time required for delivery from manufacturers.
- Recruitment and training of personnel for mine development.

During the first two years (Year -2 and Year -1), the primary focus will be on advancing the declines and the spiral ramp at a rate of 5 m/day. The spiral ramp will be driven simultaneously from the top and bottom, achieving a breakthrough at the 500 level by the end of Year -1. At this point, ramp development will pause and resume in Year 3 to continue accessing the lower horizon. Following the breakthrough, priority will shift to developing the infrastructure required to access the ore zones, enabling production to commence. During Year 1, development in both waste and ore will continue a rate of 3 m/d, with stope production ramping up throughout the year to contribute 84% of the total ore mined. The pre-production phase will conclude in Q2 when the mine achieves commercial production, defined as 75% of the planned capacity.

16.11 Life-of-Mine Plan

16.11.1 Production Plan

The mine plan is based on probable reserves and incorporate a dilution factor of 10% and a mining recovery of 95%. the LOM production plan extends over eight (8) years, during which the mine will produce ore at full capacity of 855 kt for six (6) years as indicated in Table 16.14.

The production plan will focus on mining high grade by starting production on 440 level and stoping operations will progressively advance to higher levels in an ascending sequence. The mine will operate at full production starting of Year 2. The spiral ramp will reactivate in Year 3, allowing development to proceed in the lower sublevels of the mine.

Stope production will commence on the 360 level in Year 4. Production in the lower horizon will progressively replace that of the upper horizon. In Year 7, the mine will maintain full output with production coming solely from stoping operations with the development contributing only about 4% of mineralised production. In Year 8, the final year of the LOM plan, production will be maintained to about 97 % of full capacity. The last sublevel to be mined will be the 420 level the sill pillar.

Table 16.15 provides a breakdown of the LOM production by level. Figure 16.11 illustrates the locations of the levels.

Table 16.14 – Life-of-Mine Production Plan

Description	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Total / Average
Development Ore	t	**17,317**	**92,735**	**96,803**	**87,070**	**81,406**	**101,044**	**80,762**	**35,672**	**-**	**592,809**
Grade	g/t	7.33	9.71	5.65	6.66	5.83	4.94	4.37	3.20	-	6.06
Contained Gold	oz	4,079	28,936	17,575	18,641	15,259	16,058	11,347	3,667	-	115,562
Stope Ore	t	**58,442**	**503,303**	**758,119**	**767,833**	**773,560**	**753,939**	**774,110**	**817,278**	**833,491**	**6,040,074**
Grade	g/t	10.97	10.87	7.35	8.54	6.43	5.28	5.73	4.82	3.73	6.41
Contained Gold	oz	20,606	175,898	179,266	210,754	159,933	128,099	142,686	126,782	99,903	1,243,926
Total Ore Mined	**t**	**75,759**	**596,038**	**854,922**	**854,902**	**854,966**	**854,983**	**854,872**	**852,950**	**833,491**	**6,632,883**
Grade	g/t	10.13	10.70	7.17	8.35	6.38	5.25	5.61	4.76	3.73	6.38
Contained Gold	oz	24,685	204,834	196,840	229,395	175,192	144,156	154,033	130,449	99,903	1,359,487

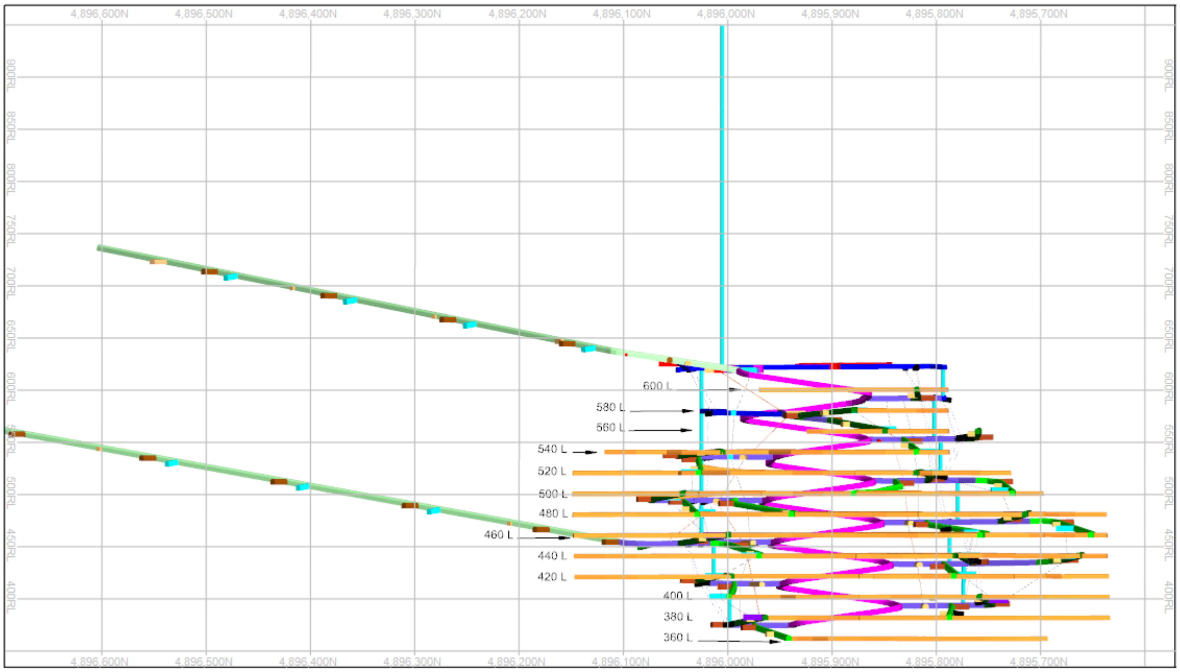
Source: WSP, 2024

Table 16.15 – Life-of-Mine Mined Ore Tonnes by Level

Description	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Totals
Level 360	-	-	-	-	11,367	74,375	56,372	35,069	-	177,183
Level 380	-	-	-	-	5,243	54,507	29,383	59,183	-	148,317
Level 400	-	-	-	-	348	15,093	56,361	102,881	28,156	202,839
Level 420	-	-	-	-	-	7,569	68,265	164,122	195,941	435,897
Level 440	60,853	298,196	225,836	185,423	106,960	231,656	76,031	59,215	89,096	1,333,264
Level 460	14,906	189,015	208,499	224,584	393,139	166,526	64,593	103,296	139,909	1,504,467
Level 480	-	94,868	180,453	189,540	138,400	240,077	203,464	54,055	132,108	1,232,965
Level 500	-	13,959	190,977	76,322	118,010	54,207	184,003	127,026	80,174	844,678
Level 520	-	-	34,133	147,203	51,028	10,972	53,704	85,344	130,104	512,488
Level 540	-	-	15,025	10,314	-	-	1,894	62,759	4,304	94,296
Level 560	-	-	-	21,515	19,061	-	-	-	31,985	72,561
Level 580	-	-	-	-	11,411	-	43,518	-	1,715	56,644
Level 600	-	-	-	-	-	-	17,284	-	-	17,284
Total Mined Mineralised Material Tonnes	**75,759**	**596,038**	**854,922**	**854,902**	**854,966**	**854,983**	**854,872**	**852,950**	**833,491**	**6,632,883**

Source: WSP, 2024

Figure 16.11 – East Longitudinal View of the Mine Levels



Source: WSP, 2024

16.11.2 Development Plan

Mine development will be intensive in the initial years of the mine life but diminishes in later years. The horizontal advance will average approximately 4,300 m/y during the initial two (2) years of pre-production development and two (2) years of operations, decreasing to 3,270 m/y from Years 1 to 5, it will average around 1,920 m annually for Years 6 and 7 and cease entirely during the final year of the mine life.

The primary development focus will be to develop the declines, the ramp, the vent system and establish the initial sills for each mining block. On each level, priority will be to establish the ventilation system as well as initial infrastructure, and ore development.

Table 16.16 presents the LOM development plan separated into Capital and Operating and summarised by development type.

Table 16.16 – Life-of-Mine Development Plan

Development Metres	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Total
Horizontal Capex Development:										
Surface Upper Decline	752									752
Surface Lower Decline	1,440									1,440
Spiral Ramp	253	1,188			413	38				1,893
Ramp Level Access	137	704	79	84	196	72	22			1,294
Horizontal Ventilation Development	396	326	13	71	199	15				1,019
Infrastructure	632	1,492	199	304	258	198	270	51	0	3,403
Total Horizontal Capex Metres	**3,610**	**3,710**	**291**	**459**	**1,066**	**323**	**292**	**51**	**0**	**9,801**
Horizontal Operating Development:										
Ore Operating Metres		254	1,452	1,524	1,374	1,273	1,604	1,312	580	9,373
Waste Operating Metres		1,082	1,535	1,287	837	1,669	1,398	1,606	264	9,678
Total Horizontal Operating Metres		**1,336**	**2,987**	**2,811**	**2,211**	**2,942**	**30,02**	**2,918**	**844**	**19,051**
Total Horizontal Development Metres	**3,610**	**5,046**	**3,278**	**3,270**	**3,277**	**3,265**	**3,294**	**2,969**	**844**	**28,852**
Vertical Development Metres	**323**	**293**	**25**		**71**	**35**				**748**

16.12 Electrical

Power for the underground mine loads will be supplied from the surface substation main 10 kV switchgear 6720-SWG-002. 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. One feeder cable will be routed down from the switchgear breaker on busbar A to the upper decline and the second feeder cable will be routed down from the switchgear second breaker on busbar B to the lower decline.

These two (2) mine power feeders will provide a redundant connection to improve reliability. UG main electrical substations are connected through the ring (radial) system. This redundancy on feeder cables and UG ring power distribution design is especially important for critical loads and mine operation as well as ventilation, dewatering and mobile equipment.

Each main electrical substation will be established with a concrete floor, fire rated wall, equipment and man doors, raised pads, lighting, ventilation, cable trays, and ground bar.

Each main UG electrical substation that supplies power to the loads will house a 10 kV Ring Main Unit switchgear that provides the ring design; a 10/1 or 0.69 kV transformer; a main distribution power centre; a 230 transformer and distribution board; and required DOL or VSD starters.

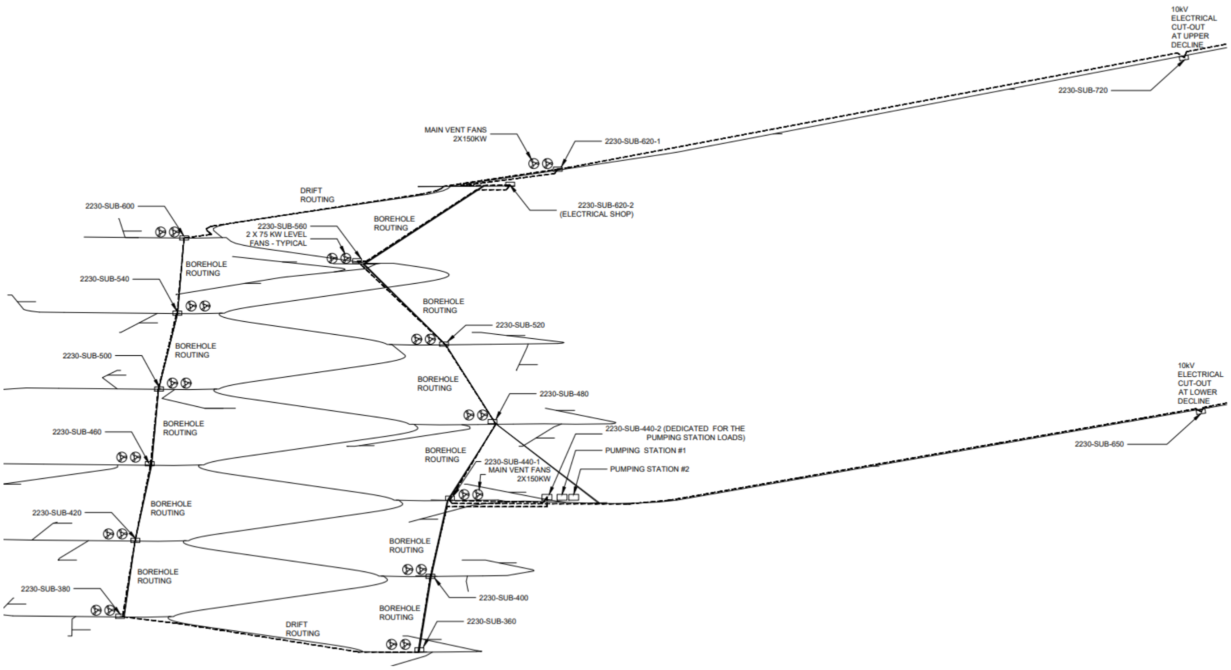
The 10 kV ring cable from the main electrical substation will terminate at the incoming feed-through terminals and feed through a borehole to the lower main electrical substation. Two (2) medium voltage feeders will be routed to the bottom of the mine through boreholes between main electrical substations 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 or 690 V output of the transformer will supply a distribution board that supports the 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 of the second main electrical substation in the production area that steps 10 kV down to 1000 V or 690 V.

Each production level is equipped with two (2) auxiliary electrical substations with minimum 500 m cable distance from the level’s main electrical substation. This design is allowed local 1000 V or 690 V power distribution to the infrastructure and mining equipment located in the upper and lower production drifts and faces.

Figure 16.12 provides the underground electrical reticulation schematic.

Figure 16.12 – Underground Electrical Reticulation Schematic



Source: WSP, 2024

16.13 Underground Communication

Underground communications solution will be designed to provide voice, video, and process connectivity across the mine. It would provide the operations with an overview and central control on mining process operations, real time tracking of the personnel and assets. The solution provides an underground communication network, which significantly increases operational efficiency, productivity, seamless data transfer while enhancing health, safety, and security of the personnel. As the mine is developed over the years, the proposed communications network and systems can be easily adapted to the changes.

The proposed communication system design would support for the following, but not limited to, applications and technologies:

- Realtime location of the mine personnel – For personnel safety and wellbeing.
- Phone/ Voice communications – For operations and mine safety.
- Process control and data acquisition of fixed and mobile infrastructure.
- Video footage at various vantage points – For real time information of the processes.
- Health, status, and location of the various mobile equipment.
- Visualisation, control and data acquisition of various storage and service facilities.
- Wireless access – To spread the communication across drifts and undercuts.

- Technologies to interconnect the process zones to local sitewide operational services and remote operations centres.
- Resilient, reliable wireless networks to connect sensing and condition monitoring applications (e.g., slope monitoring) and support operational personnel that may be working in these areas.
- A host system to assimilate all the data collected for analysis, predictive maintenance, and mine operations management.
- Data communicated among the peripherals will be using multiple standard protocols as required.

16.13.1 Fibre Optic Back Bone

- The underground fibre optic back bone will be a 48 / 72 strand single mode armoured cable. Further study on the data traffic, latency and bandwidth will determine the number of strands with sufficient spare capacity for future mine expansion. Depending on the criticality some of the services will have dedicated strands.
- The fibre optic back bone dedicated for underground communication will be isolated from the business network via a fire wall.
- The fibre optic back bone will be designed in a ring topology with 100% redundancy.
- Each fibre optic ring will originate from one of the redundant core layer switches and terminate on to the second core layer switch at the surface level. To achieve 100% redundancy each core layer switch will comprise of physical chassis and will be housed in two different buildings on the surface.
- To mitigate the possibility of cable severance and minimise down time, each fibre optic ring will originate from the surface core layer switch via upper decline and reach the second core switch via the lower decline (different physical fibre optic route). This approach may require routing the fibre optic cables via boreholes.
- The fibre optic cable will be installed along the electric power cables with necessary installation hardware (messenger wire, cable hangers etc.).
- For enhanced availability, the two (2) physical buildings will have diesel generator as a backup power source.
- The fibre optic back bone will daisy chain through a series of switches housed in a water and dust proof enclosure.
- The enclosure will also house 110VDC power supply and DC to DC convertor (110VDC to 48VDC) backed by a UPS power source for 24 hours.
- Each underground switch will be able to support IEEE 802.1, 802.3, 802.3at and PoE (for CCTV, WAPs etc.) on all ethernet ports. The switch will be able to support all Ethernet based industrial protocols like Ethernet/IP, PROFINET to name a few. Each underground switch will be capable of handling harsh environment like, dust, moisture and temperature up to 60ºC.

- For seamless integration and commissioning all the hardware, software and associated paraphernalia will be sourced from a single source vendor (Cisco).

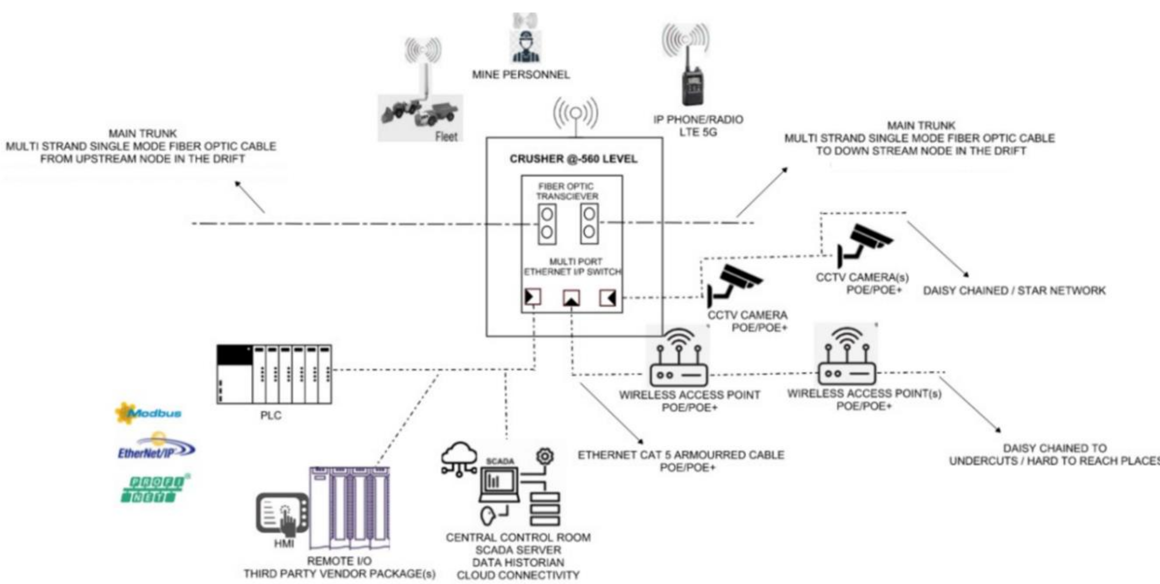
Fiber optic backbone will traverse through the switch cabinet installed at various critical infrastructure points and production areas throughout the mine to integrate the following services Table 16.17.

Table 16.17 – Integration of Services

Service Type	Integration
Audio telephony	Private / Public 4G LTE/ 5G cell phone network on the surface
Enterprise services	Public internet services on the surface
Video	CCTVs
Wi-Fi	Wireless Access Points connected to the switch cabinet
SCADA	PLCs, RIOs network
Asset & Personnel tracking system (Optional)	RFID tags

Figure 16.13 illustrates a typical switch cabinet integrating all the services listed above.

Figure 16.13 – Typical Switch Cabinet Integration



Source: WSP, 2024

16.13.2 LEAKY FEEDER

Traditional audio communication with a leaky feeder cable will be used as an alternate to cell phone network:

- The leaky feeder cable will be routed along the fibre optic cable path.
- To take advantage of the diesel generator back up power source, the line amplifier will be installed at the surface in the same building as the core switches.
- The repeaters along the path will be powered by the Ethernet switch cabinets which are backed up 24-hour UPS.

16.14 Ventilation

This section of the report outlines the ventilation design demand and strategy based on the proposed mine layout, peak diesel equipment fleet, and corresponding airflow demands for the Čoka Rakita project. Consideration was given to the peak primary airflow requirements of the project, and the subsequent fan duties needed to meet this. All applicable legislation, standards and guidelines were applied and referenced against the capacity of the circuit and the types of equipment to be used.

16.14.1 Design Criteria – Underground Mine Ventilation

Airflow requirements are regulated under the jurisdiction defined by the Serbian Government in a document entitled *PTN-Podzemnu-eksploataciju-metalicnih-i-nemetalicnih-lezista-MS* detailing a required rate of 0.067 m³/s/kW for diesel engines. WSP progressed the equipment fleet and applied a utilisation factor to capture the worst-case scenario with the resulting airflow demand for the project as shown in Table 16.18. A total of approximately 280 m³/s is required to be supplied to the project to meet the life of mine air supply demands.

Table 16.18 – Total Airflow Requirements

Mobile Equipment Type	Unit Power	Unit Air Flow	Qty	Utilisation	Total Air Flow	Standard Used
	kW	m³/s		%	m³/s	
Development Jumbo, 2 Boom	124	8.3	3	10	3.3	0.067 m³/s/kW
Long Hole Production Drill, DL421	110	7.4	1	10	0.7	0.067 m³/s/kW
Long Holle Production Drill, DL432i	110	7.4	1	10	0.7	0.067 m³/s/kW
Emulsion Charger	90	6.0	2	25	3.0	0.067 m³/s/kW
Mine Truck	450	30.2	6	65	117.6	0.067 m³/s/kW
LHD	256	17.2	5	65	55.7	0.067 m³/s/kW
Cable Bolter	110	7.4	1	20	1.5	0.067 m³/s/kW
Shotcrete Sprayer	155	10.4	1	20	2.1	0.067 m³/s/kW
Transmixer	185	12.4	2	25	6.2	0.067 m³/s/kW
Portable Screw Compressor	160	10.7	3	10	3.2	0.067 m³/s/kW

Mobile Equipment Type	Unit Power	Unit Air Flow	Qty	Utilisation	Total Air Flow	Standard Used
	kW	m³/s		%	m³/s	
Telehandler	75	5.0	1	25	1.3	0.067 m³/s/kW
Dimond Drill Rig	0	0	1	20	0	0.067 m³/s/kW
Grader	93	6.2	1	80	5.0	0.067 m³/s/kW
Service Machine/ Wheel Loader	111	7.4	2	50	7.4	0.067 m³/s/kW
Mobile Breaker	147	9.8	1	25	2.5	0.067 m³/s/kW
Pickup Truck	126	8.4	15	33	41.8	0.067 m³/s/kW
Mine Rescue Vehicle (Ambulance)	93	6.2	1	0	0.0	0.067 m³/s/kW
Utility Vehicle	126	8.4	2	70	11.8	0.067 m³/s/kW
Subtotal			*49*		*264*	
Leakage 5% (remaining leakage will serve as useful ventilation for the ramp)					13	
Total Ventilation Requirement					**277**	

Source: WSP 2024

Other design criteria are defined as follows:

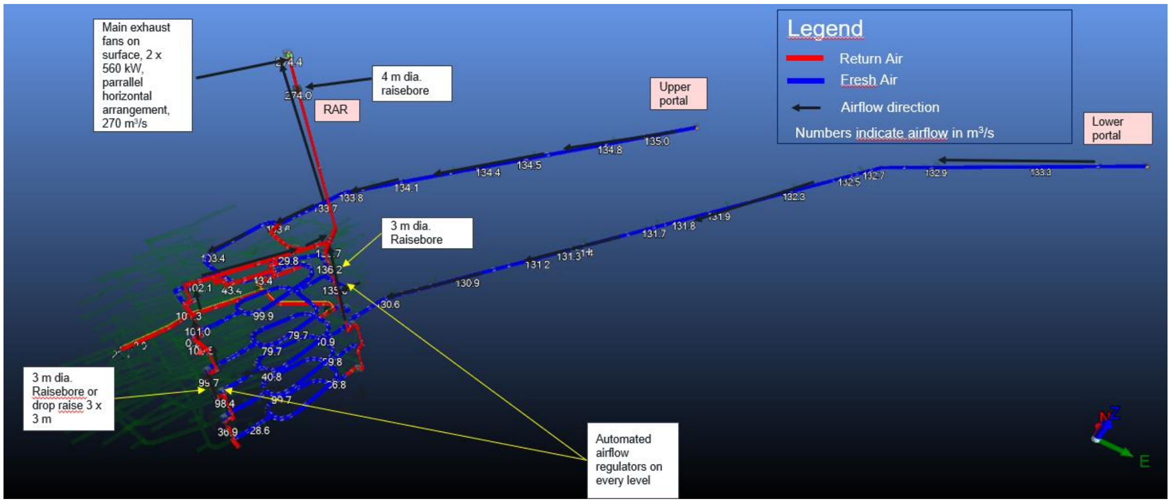
- Heating or cooling of mine air is not required;
- Any open stopes are sealed with a ventilation curtain to prevent short-circuiting;
- Maximum air velocity in ramps is 6 m/s;
- Foul air from production levels shall not mix with air from the ramp;
- Optimise blast clearing times; and
- Minimisation of the number of raises connected to surface.

16.14.2 Main Ventilation System Design

To comply with the maximum airspeed of 6 m/s and minimise the number of raises to surface, a design to have one dedicated Return Air Raise (RAR) and have the two portals as intake was selected as shown in Figure 16.14.

To ensure that the design does not mix the foul air with fresh air from the ramp, two sets of internal raises were required.

Figure 16.14 – Isometric View of the Selected Design, Single Exhaust Raise to Surface and Two Internal Raises



Source: WSP 2024

Raise sizing was assessed based on the typical sizes and to minimise excessive fan power. At the next stage, a raise sizing exercise should be performed to determine the most optimal size for the main RAR (currently 4 m diameter design) and the internal raises (currently 4 m diameter design). The escape raise is planned to be installed in the internal ventilation raises, and for the next level of study the sizing of the raises will have to be revised accordingly to consider the escape system infrastructure. The air velocity in the internal raises should consider a maximum velocity of 6 m/s when workers are present.

There are two (2) main fans each 560 kW installed on surface at the main exhaust raise in a parallel horizontal arrangement. The operating pressure of the main fans is 2,925 Pa and a total airflow of 277 m³/s. To size the main fan, the regulators were set for a worst-case scenario to send most of the air to the lower levels. Some leakage was considered at every regulator with 60% of the airflow to one set of internal raises and the remaining 40% to the other set of internal raises.

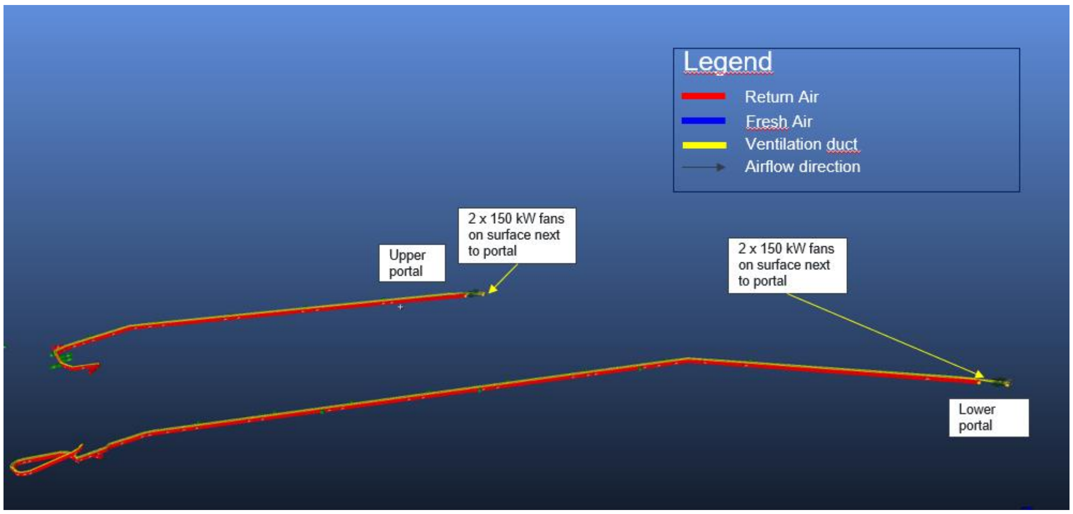
The system would be equipped with a state-of-the-art Ventilation-On-Demand system to modulate the speed of the main fans, auxiliary fans and with automated regulators based on airflow requirements determined considering the planned production, equipment fleet and utilisation factor. Each automated regulator is planned to be equipped with a gas and airflow sensor. An airflow sensor would also be installed in between levels in the ramps to detect stope leakage as well as to control the automated regulator to ensure that there is not any ‘dead spots’. As the two intakes from the portals will converge to a level, it could occur that one of the ramps has a low airflow which would provide sufficient airflow for a truck or other vehicles to operate if not controlled adequately.

This design would enable the possibility to access the ramps and unaffected production levels immediately following blasts.

16.14.3 AUXILIARY VENTILATION DESIGN

For the ramp development, the upper and lower decline would each have two lines of 1.4 m diameter flexible duct each supplied by 150 kW auxiliary fans as shown in Figure 16.15.

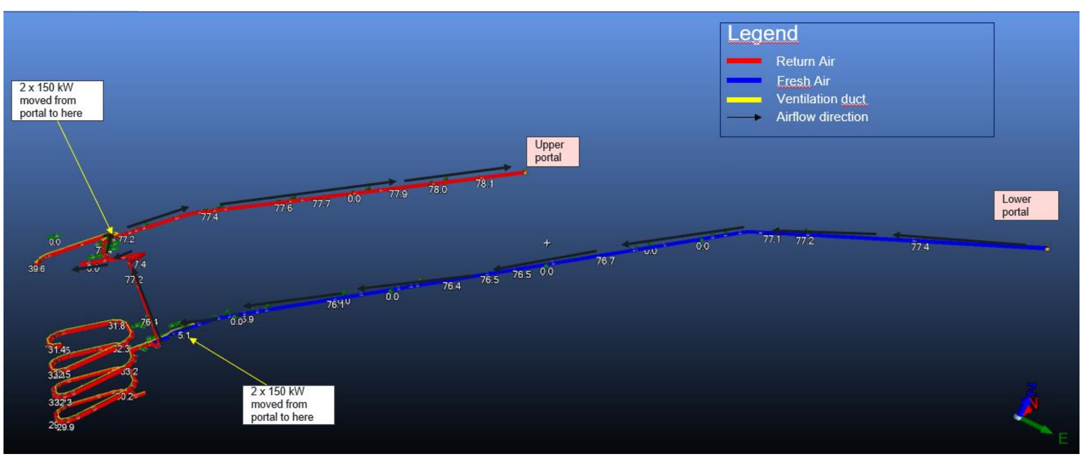
Figure 16.15 – Ramp Development Ventilation System



Source: WSP 2024

Once the main surface raise and internal ventilation raise is developed, a ventilation circuit will be created by installing the fans for the upper decline in a bulkhead as shown in Figure 16.16. The fans from both portals will be moved underground to develop the spiral ramp. To develop the production levels, 1.2 m diameter flexible duct will be used with 75 kW fans to supply enough air for an LHD.

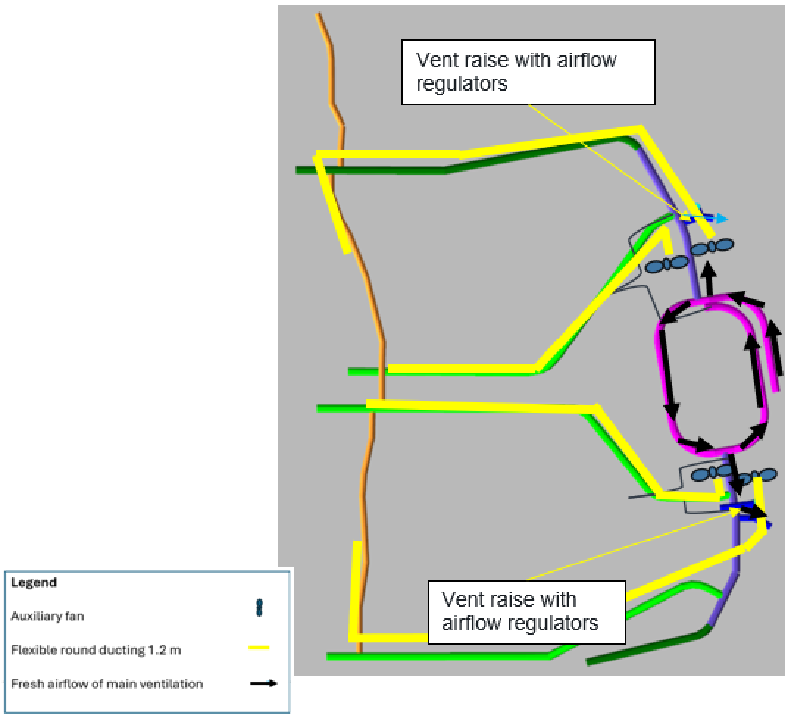
Figure 16.16 – Ramp Development Following Internal Ventilation Raise Connection



Source: WSP 2024

For production levels, the same 75 kW auxiliary fans to ventilate production levels would be installed upstream of the regulators as shown in Figure 16.17. There would be two fans installed on each level to develop the two faces.

Figure 16.17 – Auxiliary Ventilation on Production Levels



Source: WSP 2024

16.15 Mine Backfill

The primary mine backfill will be CPF formulated from mill tailings in the surface paste plant. The CPF will be pumped from the paste plant via a single section of surface piping (380 m long) to the upper decline portal. The CPF piping continues in the upper decline to its intersection with the orebody. The CPF pipeline in the decline will have an operational and a standby line. All remaining pipelines in the orebody will be single operational lines. Boreholes within the orebody will be cased and twinned with operational and standby boreholes.

The UDS comprises underground boreholes, mainline piping, level piping, and stope access piping. The design begins at the discharge of a positive displacement pump located at the paste plant, through a surface pipeline and down the upper decline, and stretches throughout the mine, allowing paste to be delivered into the mine's primary and secondary stopes.

Boreholes cascade downward from level to level between paste backfill bays.

The paste backfill delivery and receiving bays are designed to be 10 m long to allow sufficient room for two (2) borehole collars and breakthroughs (5 m apart), pipe fittings and instrumentation. The bay (5 m width and 5.5 m height) is designed to be aligned with the size of level drift. For further details on paste backfill bays (typical paste backfill receiving and delivery station).

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 and the hydraulic flow modelling of the paste backfill UDS for such things as pipeline diameter, pipeline materials of construction, and paste flow velocity.

The secondary mine backfill would be rock fill from the development waste rock generated from Year 1 onwards as either CRF or uncemented rock fill (RF) since starting in Year 1, none of the development waste rock will report to surface.

CRF will be placed into stopes prior to addition of CPF, and into stopes exposed by adjacent mining. The maximum proportion of CRF in a stope will be approximately 55% of the overall stope volume.

RF will be placed into stopes on an "opportunity" basis, which is to say that it will be placed into stopes where there will be no exposed faces to adjacent mining.

CRF slurry will be mixed in a surface batch plant that will also mix shotcrete for underground use. The batch plant consists of a high shear colloidal mixer that creates a homogeneous binder slurry at a 1:1 water to cement ratio along with a hydration retardant admixture. The cement slurry is then stored in an agitated tank for delivery into a transmixer.

The transmixer will be transported the CRF slurry underground and dump it onto a measured amount of waste rock stored in an available remuck bay in close proximity to the stope to be backfilled. It is important that the correct proportion of CRF slurry be added to the correct amount of waste rock to ensure the cured target strength of the CRF is achieved.

An LHD will thoroughly mix the CRF slurry into the waste rock, ensuring a homogeneous mixture is delivered to the stope.

Following the placement of the CRF in the stope, the remaining void will be filled with CPF and both will be allowed to cure to the target backfill strength required prior to adjacent mining activities.

16.15.1 KEY DESIGN COMPONENTS OF THE UNDERGROUND PASTE BACKFILL DISTRIBUTION SYSTEM

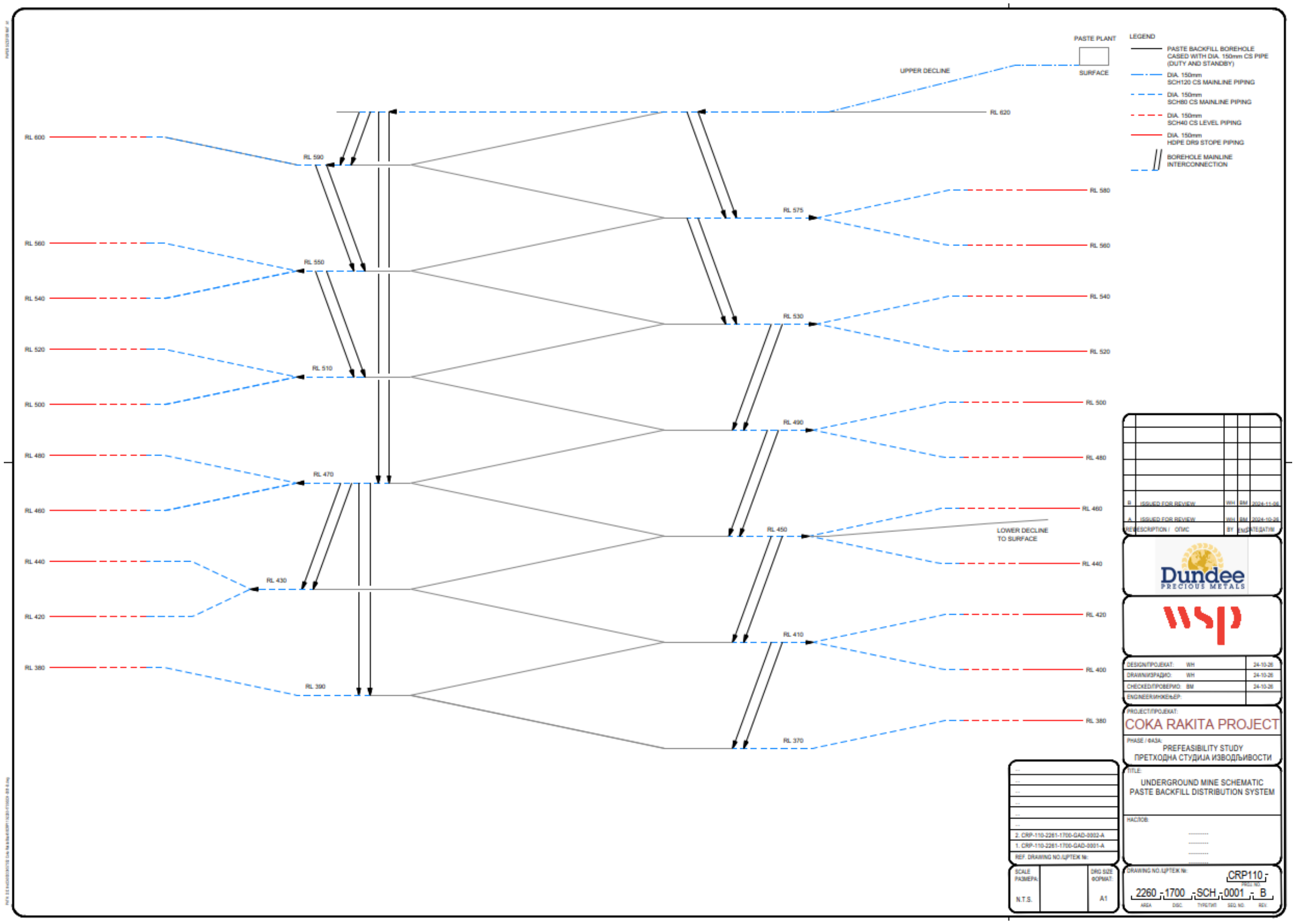
Common design elements within the entire design of the UDS are:

- All boreholes are “twinned” at 5 m spacing, 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, and a carbon steel (CS) pipe installed inside to mitigate the predicted questionable rock characteristics.
- The boreholes are drilled 230 mm in diameter and cased with a 150 mm CS Sch80 pipe.
- All boreholes originate or terminate in paste backfill bays to allow drilling equipment operation without interfering with mine vehicle traffic.
- The backfill delivery and receiving bays are in separate locations in a drift to allow level interconnection holes to be drilled with an acceptable dip
- CS pipe sections of 4-m lengths are used and connected using high-pressure Victaulic style couplings (HP70ES).
- Mainline piping is connected to the boreholes and is constructed of 150 mm CS Schedule 80 pipe to increase its longevity and withstand the potentially higher internal pressures.
- Level piping connected to the mainline piping is constructed of 150 m CS Schedule 40 pipe due to decreased longevity requirements and the lower internal pressures.
- Stope access piping connected to the level piping is constructed of 150 mm DR9 HDPE pipe due to low internal pressures.

The Mine Paste Fill UDS Schematic is presented Figures 16.18, and 16.19 illustrates the relative position of the various backfill boreholes and level piping with respect to the mine workings.

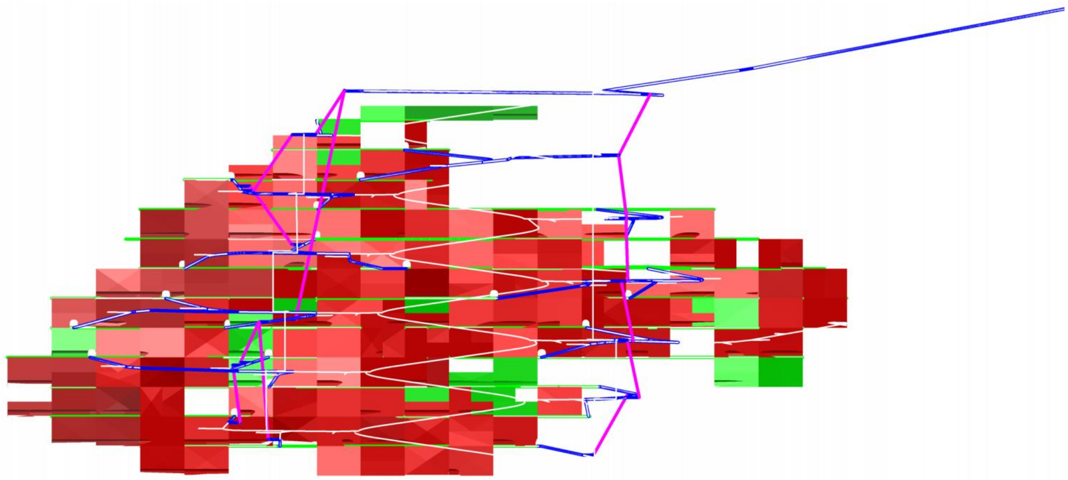
Generally, the interlevel boreholes dip 65° to 70° from the horizontal. The level piping connects to each operating borehole, as illustrated in Figure 16.20.

Figure 16.18 – Mine Paste Fill Underground Distribution System Schematic



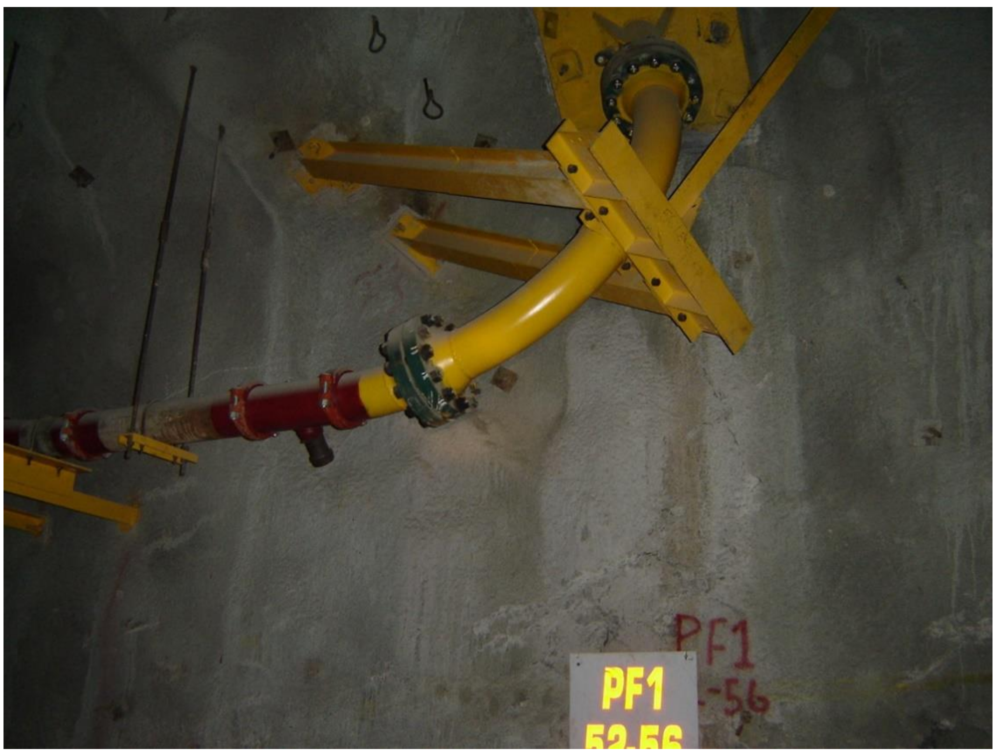
Source: WSP 2024

Figure 16.19 – Backfill Underground Distribution System on a Longitudinal Section



Legend: Blue and Purple – UDS system; Red and Green – Stope
Source: WSP 2024

Figure 16.20 – Typical Borehole to Level Piping Connection Configuration



Source: WSP 2024

16.15.2 KEY DESIGN COMPONENTS OF THE SHOTCRETE/CRF SLURRY BATCH PLANT

The batch plant proposed for the preparation of the Shotcrete/CRF Slurry is of modular construction that can be shipped and erected quickly on a relatively small footprint.

The components in the modules consist of:

Modules 1 and 2

- Colloidal Mixer
- Agitated CRF Slurry Tank
- CRF Slurry Pump
- Agitated Shotcrete Tank
- Shotcrete Pump
- Plasticizer Reagent Tote
- Plasticizer Metering Pump
- Retardant Reagent Tote
- Retardant Metering Pump
- Process Water Tank
- Process Water Pump
- Agitated Wastewater Tank
- Air Compressor
- Air Dryer
- Air Receiver
- Electrical and Instrumentation Room

Module 3

- Two (2) Cement Silos (with associated silo equipment and a Passive Dust Collection System)
- Two (2) Cement Screw Conveyors
- Binder Weigh Hopper

Module 4

- Shotcrete Aggregate Feed Bin
- Aggregate Conveyor with Weigh Element

Typically, Modules 1 and 2 are insulated “Sea Containers” with removable access panels and access doors.

16.16 Mine Dewatering

WSP refined the dewatering concept presented in previous PEA-level design to incorporate the Lower Decline by modifying the locations of Level Sumps, Secondary Sumps, and Main Dewatering Station, and determining the pump and pipeline sizing appropriate for the revised quantity of water to be discharged from the Mine to water treatment facility on surface.

In the PEA, the Upper Decline was the route for water leaving the Mine and it reported to the Mine dewatering pond near the processing facility. The dewatering route has now changed to the Lower Decline with water reporting to the Mill water treatment facility.

The mine dewatering design criteria were based on the revised water balance, which included:

- Groundwater seepage;
- Water required for mining operations, including drilling, dust suppression when mucking, and other ancillary water consumption;
- Paste backfill contained water release at placement and system flushing;
- Water generated from diesel fuel combustion emissions;
- Water in ore and waste leaving the Mine; and
- Water in the ventilation system leaving the Mine.

16.16.1 System Design Inputs

The following sections describe the inputs used to develop the dewatering system design.

16.16.1.1 Ground Water Seepage

Some geotechnical drilling at the mine site has taken place during this study, and the results were used to estimate of the quantity of groundwater inflow.

The hydrogeological group provided Table 16.19 while completing their scope of work under Section 16.3 of this Report.

Table 16.19 – Ground Water Seepage Estimate per Year (Base Case)

Year	Inflow Rate (m^3/d)	Inflow Rate (L/s)
-2	1,187	13.7
-1	1,721	19.9
1	1,654	19.1
2	1,589	18.4
3	1,548	17.9

Year	Inflow Rate (m^3/d)	Inflow Rate (L/s)
4	1,521	17.6
5	1,594	18.5
6	1,527	17.7
7*	1,555	18.1

*Value estimated from average between years 5 and 6

A base case average ground water inflow rate of 20 L/s, or 1,728 m^3/d was used in the dewatering system design.

16.16.1.2 *Water Required to Conduct Mining Operations*

The water reporting to the mine dewatering system from mining operations including drilling, dust suppression, and ancillary consumption was estimated to be 18.1 L/s or 1,564 m^3/d.

16.16.1.3 *Paste Backfill Contained Water Release and Distribution System Flushing*

The amount of water originating from paste backfill was estimated for paste having a 178 mm (7") slump and an average distribution system flushing frequency was estimated at once per day while backfilling. Most of the water contained in the paste backfill is consumed in the cement hydration process. The paste backfill and flushing water released into the mine dewatering system is estimated to be 1.7 L/s or 147 m^3/d.

The following additional factors were also included in mine dewatering calculation:

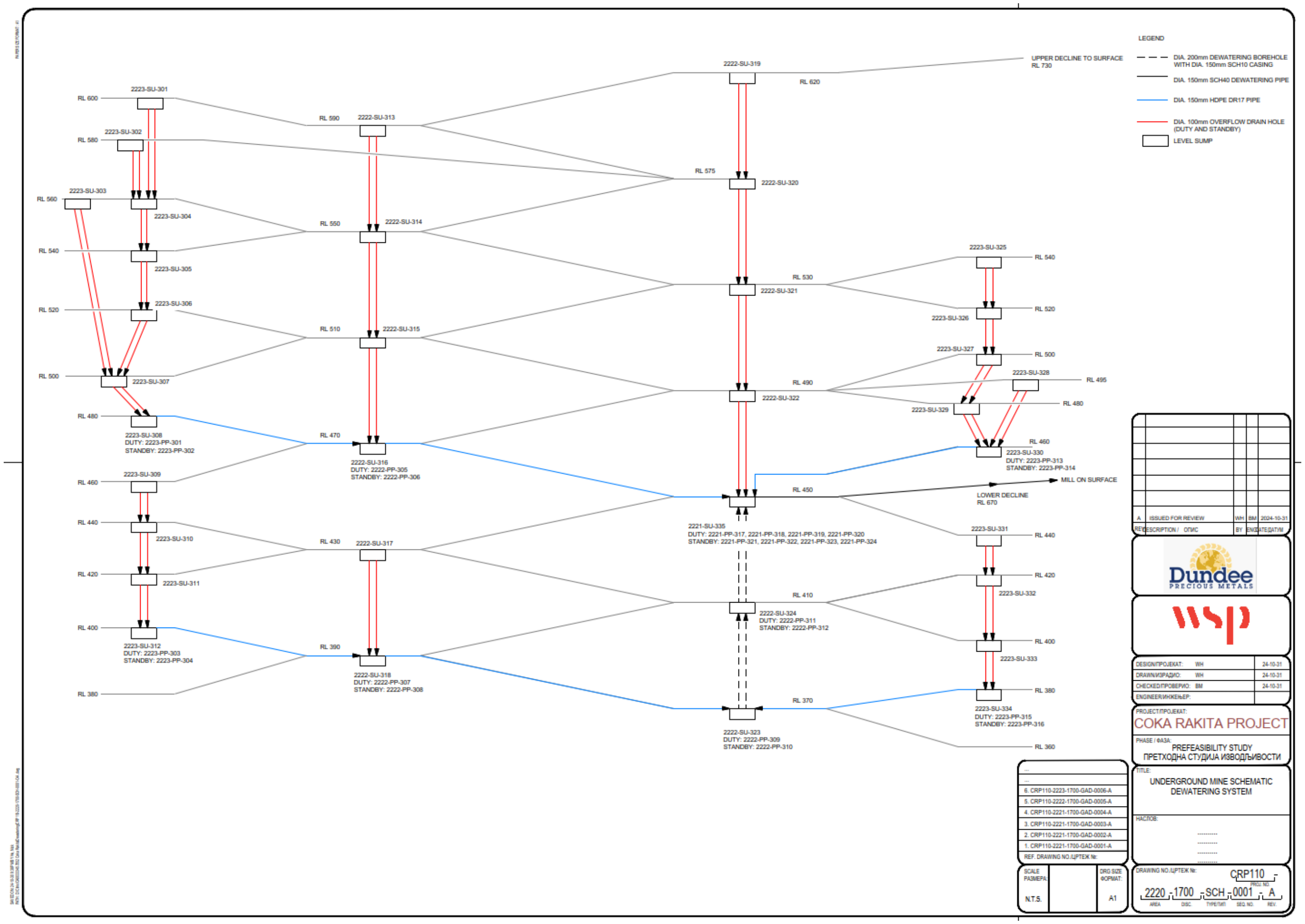
- Water generated from diesel fuel combustion is estimated at 0.02 L/s or 2 m^3/d;
- Water in ore and waste leaving the mine is estimated at 1.42 L/s or 123 m^3/d; and
- Water in the ventilation system leaving the mine is estimated at 0.33 L/s or 29 m^3/d.

Based on the total of the above water sources, the dewatering system capacity was designed for 38.1 L/s or 3,290 m^3/d of water pumped out of the mine.

16.16.2 Key Design Components of the Mine Dewatering System

Figure 16.21 illustrates the dewatering system design, which includes the level sumps, secondary sumps, main pumping station, drain holes, pipelines, and pump locations on the various levels of the mine.

Figure 16.21 – Mine Dewatering Schematic



Source: WSP 2024

The predominant design of level sumps used at some other mining operations incorporates a dirty/clean water sump design, as opposed to a single dirty water settling sump in this application. Therefore, the system can be considered a "dirty water" pumping system whose pump designs will be further refined in subsequent phases of the Project.

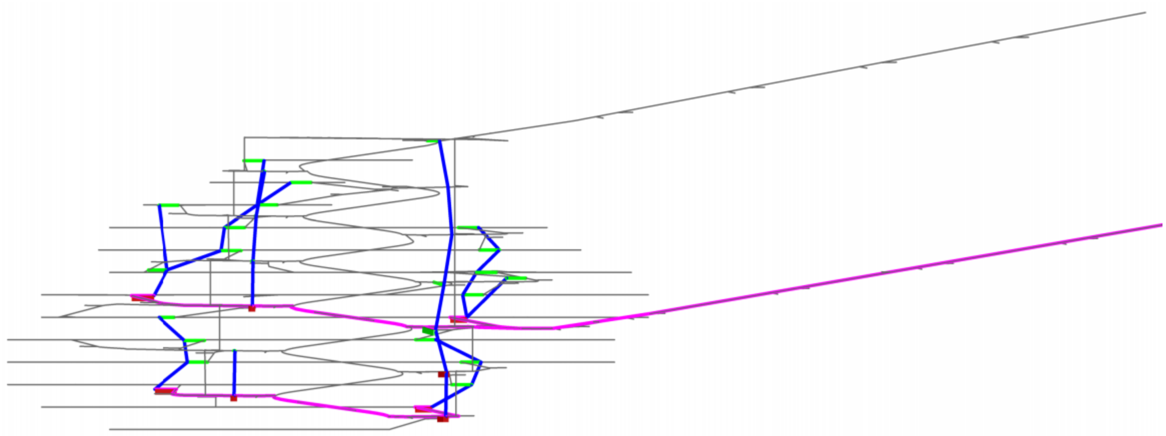
The intent of the current PFS design is to locate the sumps and associated lengths of pipelines and boreholes and to establish the type, size, and power requirements for the pumps within the system.

The design incorporates the following:

- The main dewatering station located on 440L receives water flowing by gravity or pumped from level sumps located at the intersections of stope drifts and access drifts, and water flowing by gravity or pumped from secondary sumps. The design incorporates:
    - Twined drain holes at each level sump (sump code 2223).
    - An agitated collection tank and two centrifugal pumps (one duty, one standby, sump code 2222) at each secondary sump.
    - Two (2) sets (trains) of main dewatering pumps (one train duty, one train standby, sump code 2221) at the main dewatering station (2221-SU-335). Each train consisted of four (4) centrifugal pumps in series. All the pumps operate at the same flow rate and dynamic head; and
    - Two dewatering pipelines (one duty, one standby) of CS, 150 mm, Sch40 pipe fitted with Victaulic couplings run up the lower decline to surface.
- All gravity drain boreholes from sumps are approximately 100 mm in diameter (DDH HQ size).
- Boreholes connected to pumps in sumps below 440L are 200 mm in diameter and contain a 150 mm CS casing to mitigate possible rock quality interactions in a pressurised pumping situation.
- Level pipelines are HDPE DR17 150 mm.

Figure 16.22 presents an overview of entire mine dewatering system.

Figure 16.22 – Mine Dewatering System Illustrated on a Longitudinal Section



Legend: Purple – Dewatering pipeline; Blue – Dewatering drain hole; Green – Level sump; Red: main sump
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 the process design as defined in the process design criteria, process mass balance and process area flowsheet.

The comminution circuit design is based upon the average throughput requirements and measured design (peak) material competency and hardness. The 85th percentile of hardness from variability samples testing was used for equipment sizing.

The design of the flotation circuit is based upon the testwork results, supplier recommendations, 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 at the start of the PFS to evaluate flowsheet options for gravity concentrate treatment, comminution and flotation.

These studies identified options for further on-site beneficiation of the primary gravity concentrate, focusing on gravity concentrate treatment, comminution, and flotation. Details of each ToS analysis are summarised below.

17.1.1 Gravity Concentrate Treatment

The PEA flowsheet has a portion of the grinding circuit cyclone underflow diverted to two parallel gravity concentration circuits. Each gravity circuit consists of a scalping screen with the screen undersize reporting to a centrifugal concentrator. Both scalping screen oversize and the centrifugal concentrator tailings return to the grinding circuit via the SAG mill discharge hopper. Gravity concentrate is periodically flushed from the bowls and discharged into a dedicated pumpbox. This concentrate is then pumped to a dewatering screen and filter before it is bagged. The problem with this arrangement is that the gravity concentrate would be difficult to sample and analyse accurately and this could lead to a loss in revenue. To mitigate this risk additional beneficiation of the primary concentrate was explored.

The following options have been identified for additional on-site beneficiation of the primary gravity concentrate:

- **Option 1**: Intensive cyanidation followed by electrowinning (EW) and smelting of the gold sludge to produce doré bars. Thiosulphate was also explored as an alternative lixiviant.

- **Option 2**: Secondary gravity concentration followed by direct smelting of the upgraded concentrate to produce doré bars.
- **Option 3**: Secondary gravity concentration followed by filtration and bagging for direct sale of the concentrate.
- **Base Case**: No beneficiation of primary concentrate.

Intensive leaching testwork was performed using both cyanide and thiosulphate to support the trade-off study. The results showed nearly complete leaching (>99.7%) using cyanide but less than 7% dissolution when using thiosulphate. The testwork results were used to estimate the capital and operating expenses of each option.

FLSmidth was provided with the Extended Gravity Recoverable Gold (EGRG) test results from both the PEA and PFS testwork programs and used this, augmented by their in-house benchmarking, to model various gravity configurations based on the size-by-size gold data contained in the EGRG results. Model projections show that a single-stage gravity concentrator yields about mid-30% gravity recovery, slightly lower than the 43% used in the PEA and trade-off study.

The primary gravity concentration stage for all three (3) options were identical with two parallel centrifugal concentrators producing a primary concentrate with a grade of 3.4 and 6.0 kg/t Au for the nominal (5.7 g/t) and design (10.0 g/t) feed gold grades, respectively.

DRA performed equipment sizing calculations based on industry best practice and the available testwork results.

Based on the financial results, only the base case or Option 3 is considered as the preferred option. However, both options face significant commercial sampling risks stemming from challenges in accurately analysing concentrates containing coarse gold. Option 1 is deemed undesirable due to permitting constraints associated with cyanide and the suboptimal recovery recorded when using thiosulphate. Option 2 emerges as the technical feasible option that is not hampered by either permitting or sampling risks.

17.1.2 COMMINUTION

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 ore is of average competency. The nameplate capacity of the treatment plant is 850,000 tpa. The following flowsheet options have been evaluated:

Option 1: New SAG and Ball milling (SAB) – No pebble crush

This option features two (2) mills with a combined motor power of 2,500 kW, evenly distributed between them. A SAG mill with a diameter of 5.94 m and an equivalent grinding length (EGL) of 2.97 m is selected as the primary mill. The ball mill has a diameter of 3.9 m and an EGL of 5.85 m.

Option 2: New SAG and Ball milling with pebble crushing (SABC)

Option 2 includes a HP100 (or equivalent) pebble crusher with a 90 kW motor drawing approximately 25 kW, integrated into the primary grinding circuit. The mills remain identical to those of Option 1, with the total power also matching that of Option 1.

Option 3: Refurbished SAG mill and one refurbished Vertimill

This option utilises two (2) refurbished mills from the Ada Tepe mine. The SAG mill, with a diameter of 5.94 m and an EGL of 2.97 m, is driven by a 3 MW motor. Two (2) VTM1250-WB Vertimills and a Sandvik CH420 pebble crusher will be sourced from Ada Tepe. The SAG mill and one (1) Vertimill will be installed at Čoka Rakita, along with a pebble recycle conveyor and associated chutes. While the pebble crusher is omitted, space is allocated for potential future installation.

Option 4: AG mill with pebble crusher and one Vertimill (all refurbished)

This option involves operating the existing Ada Tepe primary mill in autogenous grinding (AG) mode. A Sandvik CH420 pebble crusher will be installed to manage the accumulation of critical-size pebbles in the primary circuit. Both the primary and secondary grinding circuits will be completed with dedicated cyclone clusters.

Option 5: Refurbished single stage SAG (SSAG) mill only (with a coarser product)

This option represents the lowest capital expenditure (Capex) configuration, featuring a SSAG mill without a Vertimill. The grind size for flotation will be coarser, with a P_{80} of 72 µm for design hardness and approximately 69 µm for average hardness. This configuration will not achieve the targeted grind of 53 µm, leading to an estimated recovery decrease from 53 to 69 µm (approximately 1.5%) based on grind series testwork conducted during the PFS testwork program.

In all options, pebbles are recycled to the primary mill feed chute. Options 1 and 2 consider using new, purpose-built mills as alternatives to potentially oversized equipment from Ada Tepe, providing a benchmark for savings from reusing refurbished equipment. Option 3, the most capital-efficient for refurbished equipment, but excludes a pebble crusher. Option 4 minimises operating costs for new equipment by running the primary mill without a ball charge, reducing liner wear and grinding media costs, though it requires a pebble crusher for critical-size handling, raising capital costs compared to Option 3. Option 5 was included to assess whether the targeted grind could be achieved with this configuration.

The 85th percentile of hardness as determined from the variability testing conducted in 2024 was used for sizing all equipment.

Option 4, the refurbished AG, is identified as the most economical choice, providing significant expansion capacity. The primary mill operates well below its rated motor capacity, allowing for increased throughput by adding grinding media to the mill, requiring only minor modifications to the liners and pebble circuit. Consequently, Option 4 was selected as the basis moving forward.

17.1.3 FLOTATION OPTIONS

The Čoka Rakita PEA flowsheet proposed using Jameson cell flotation technology for rougher, scavenger, and cleaner stages. This circuit includes a scalping bulk sulphide rougher flotation cell followed by a similarly sized scavenger cell. The scavenger concentrate is directed to a smaller, dedicated cleaner cell, while its tailings go to a tailings thickener. Cleaner tailings are recycled back to the rougher scalper cell, and both the scalper and cleaner concentrates flow to the final concentrate thickener. The design does not require regrinding any streams within the flotation circuit.

DRA performed a high-level flotation technology ToS considering three (3) options:

- **Option 1**: New Jameson cells.
- **Option 2**: Refurbished the Staged Flotation Reactors (SFRs) from Ada Tepe, with additional new SFR units as recommended by Woodgrove Technology.
- **Option 3**: A hybrid approach combining refurbished SFRs for rougher and rougher scavenger duties, with a single Jameson cell replacing the entire two-stage cleaning circuit from Ada Tepe, alongside an additional SFR cell for both the rougher and rougher scavenger banks.

The number and size of flotation cells required for each option were determined provided by their suppliers based on testwork results from the PEA phase. Budgetary quotations were also obtained from suppliers. DRA performed preliminary layout analyses to estimate footprint variations, which were then used to adjust the estimates for construction indirect costs. DRA developed the Opex estimate for all three (3) options based on the mass balance information provided by the suppliers.

Following the financial comparison, Option 1, which features new Jameson cells, is identified as the preferred choice due to its operational advantages. In terms of risks and environmental impacts the technologies are comparable. The limited number of Jameson cells makes this circuit significantly easier to operate, construct and commission. Given these advantages and the associated Net Present Cost (NPC) benefits, the recommendation is to proceed with Jameson cells.

17.2 Production Profile

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 applies:

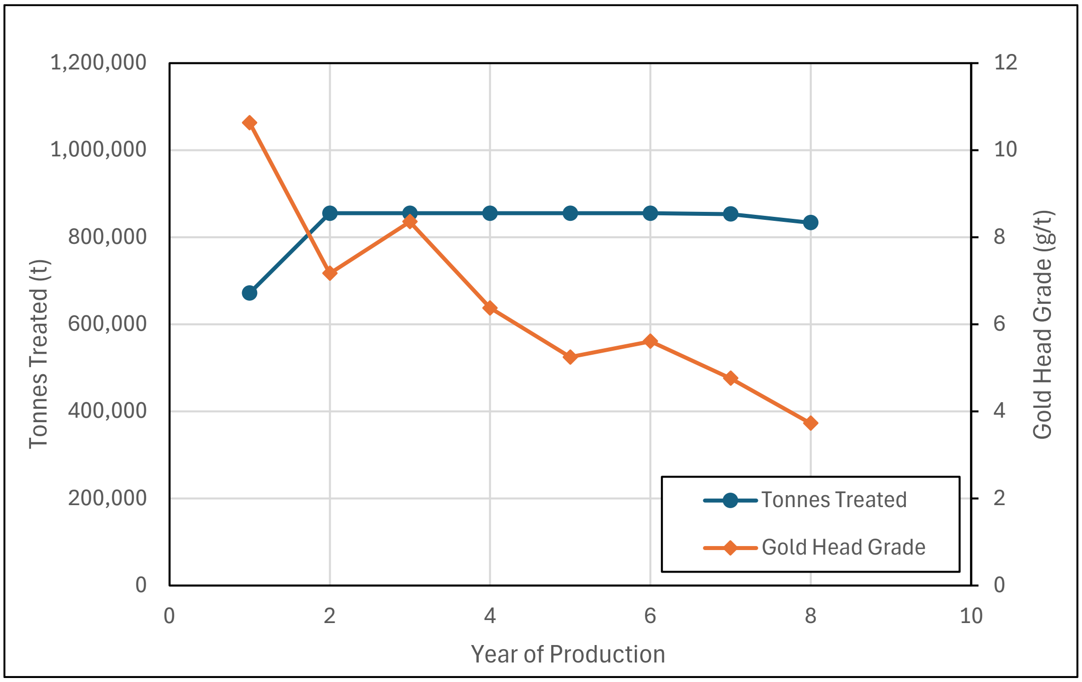
- Testwork indicated that there is a constant upgrade ratio from the gold head grade to the gravity concentrate grade. For this study a value of 600 was adopted as this aligns with typical industry practice.
- A 20% scale-up downgrade was applied to the gravity recovery. This is based on industry experience 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 = 35 x Gold Head Grade ^ 0.14.
- Gold stage recovery from primary to secondary gravity concentrates are fixed at 63% and the stage mass pull is 5%.
- Mass pull to final flotation concentrate = -0.2008 x Gold Head Grade + 5.860
- For the first year of operation (Year 1), recoveries 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 17.1.

Figure 17.1 depicts the ROM feed profiles over the LOM. It was assumed that all ore in Year 0 would be stockpiled and treatment would commence in Year 1 only.

Figure 17.1 – Plant Feed Tonnage and Gold Grade Profiles



Source: DRA, 2024

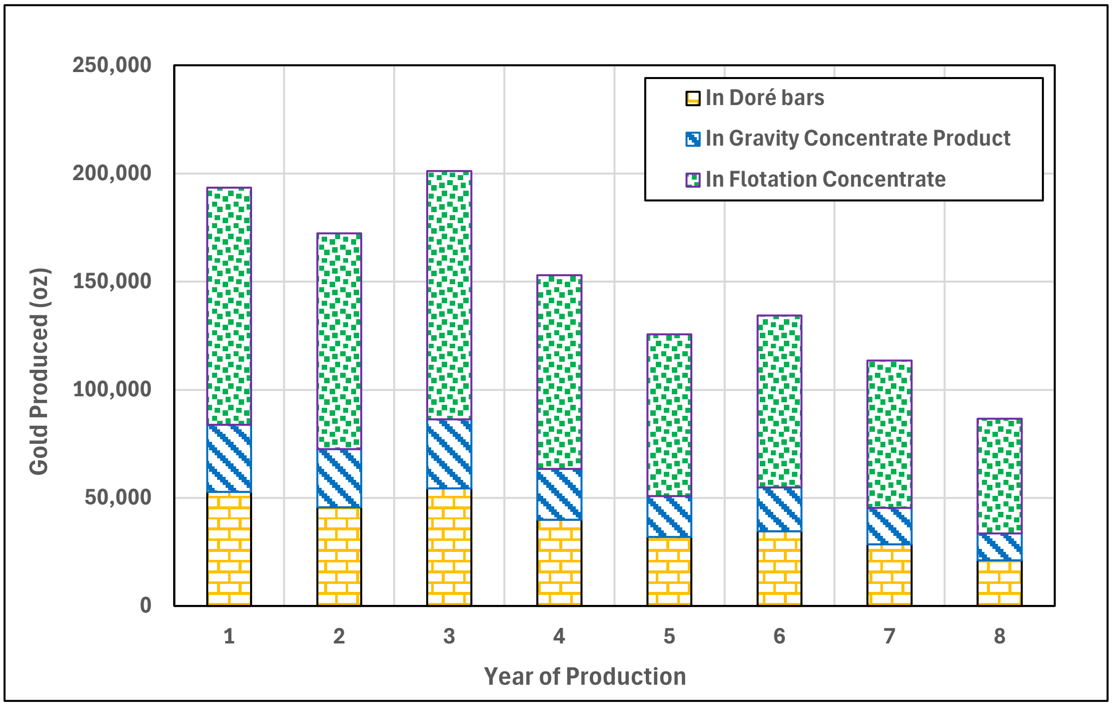
Table 17.1 – Gold Production Profile

Description	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Tonnes Treated	t	671,797	854,922	854,902	854,966	854,983	854,872	852,950	833,491
ROM Au Grade	g/t	10.63	7.17	8.35	6.38	5.25	5.61	4.76	3.73
Gravity Product Au Distribution	%	13.53	13.62	13.92	13.40	13.04	13.17	12.87	12.44
Gravity Product Au Content	oz	31,044	26,848	31,955	23,490	18,810	20,302	16,799	12,427
Doré Au Distribution	%	23.0	23.2	23.7	22.8	22.2	22.4	21.9	21.2
Doré Au Content	oz	52,858	45,714	54,410	39,996	32,028	34,568	28,604	21,160
Flotation Conc. Au Distribution	%	47.8	50.6	50.0	51.1	51.8	51.6	52.2	53.2
Flotation Conc. Au Content	oz	109,613	99,711	114,728	89,511	74,768	79,539	68,176	53,105
Overall Au Recovery	%	84.3	87.4	87.6	87.3	87.1	87.2	87.0	86.8
Overall Au Produced	oz	193,516	172,274	201,093	152,997	125,607	134,408	113,579	86,692

Note: Totals may not add due to rounding.
Source: DRA, 2024

Figure 17.2 illustrates the ounces of gold produced in total and the contributions by each of the products. Gold production will be lower in the first year than in the following two years due to the lower tonnage treated combined with the expected lower recovery due to operator inexperience. Production will peak in Year 3 and then decline steadily due to declining head grades. The flotation concentrate will contribute more than half of the total saleable ounces, followed by doré bars and then the secondary gravity tails product.

Figure 17.2 – Plant Feed Tonnage and Gold Grade Profiles

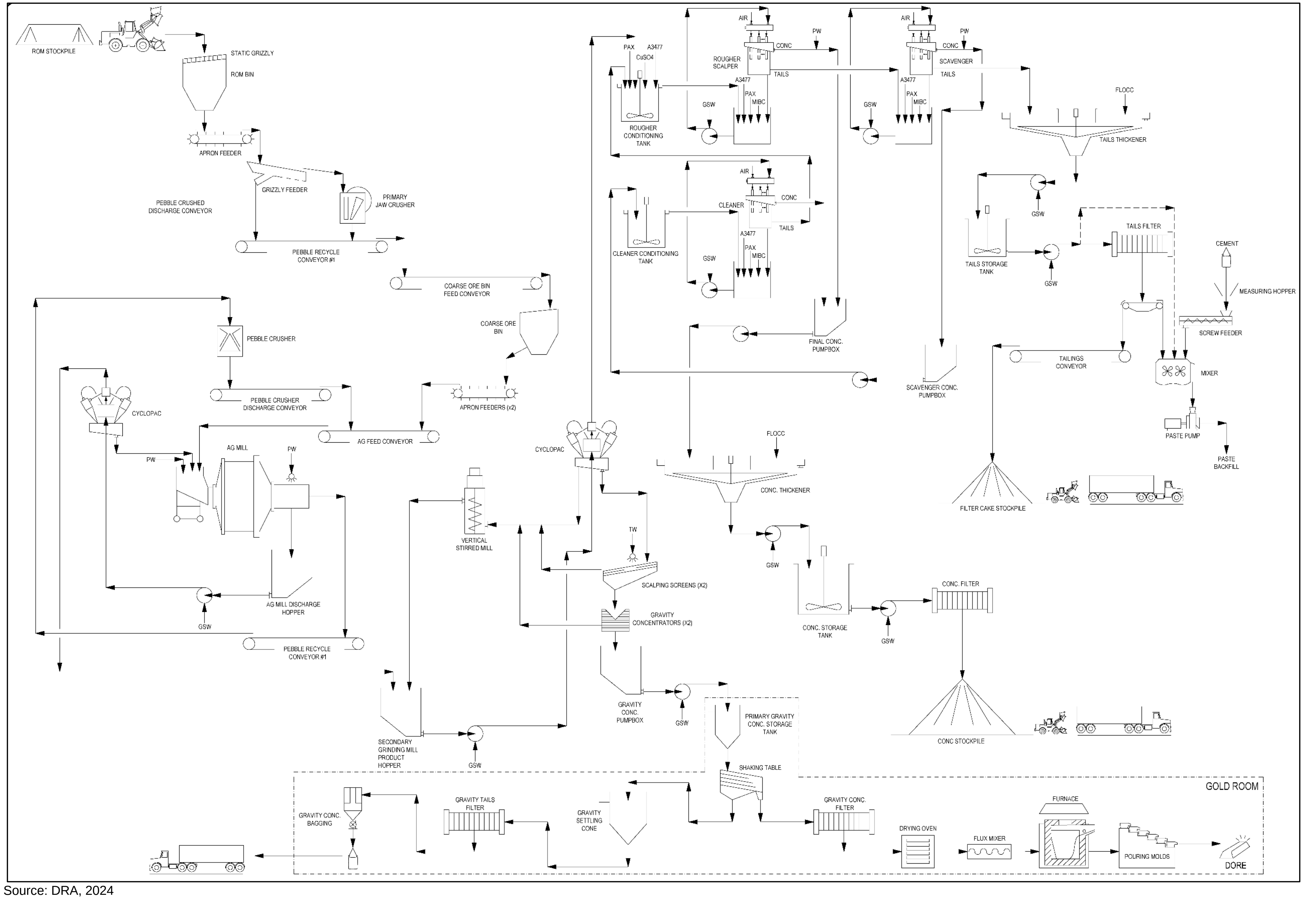


Source: DRA, 2024

17.3 Process Flowsheet

The Čoka Rakita deposit has high gold-grade, gold-copper skarn type mineralisation approximately 250 m below surface. Based on relatively modest copper grades, copper is not expected to contribute significantly to the revenue stream. Preliminary metallurgical test results indicate the mineralisation is amenable to flotation and gravity concentration, and has the potential to produce doré bars and clean gold concentrates with overall gold recoveries around 87%. The proposed process plant consists of surface facilities including crushing, grinding, gravity concentration, smelting, Jameson flotation cells, thickening, filtration, and paste backfill sections. The proposed process plant will produce doré bars, a gravity concentrate and a bulk flotation concentrate. Tailings will be disposed underground as paste backfill or above ground as filtered cake in a dedicated Dry Tailings Storage Facility (DTSF). 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 17.2 summarises the Process Design Criteria (PDC) to nominally process 850,000 dry tpa of ROM ore from the mine.

The proposed process plant consists of surface facilities including jaw crusher, AG mill, vertical stirred mill, gravity concentrators, flotation cells, thickeners, filtration, and paste backfill.

Table 17.2 – Process Design Criteria Summary

Description	Unit	Nominal	Design
Average ROM Throughput	dry t/h	97	111.6
ROM Throughput-Plant	dry t/a	850,000	977,500
Feed Grades			
ROM Material Head Grade	Au g/t	5.68	10.0
Material Hardness			
Bond Ball Mill Work Index (BWi)	kWh/t	13.7	15.2
Abrasion Index (Ai)	g	0.114	0.167
Operating Schedule			
Life of Mine	years	8	12
Crushing Plant Availability (Day shift only)	%	80	80
Milling Plant Availability	%	92	92
Crushing			
Feed Rate	dry t/h	242.6	279.0
Required Power	kW	51	56
Feed Size F_{80}	mm	320	
Product Size P_{80}	mm	92	
Primary Grinding			
Feed Rate	dry t/h	105.5	121.3
Installed Power	kW	3000	3000
Mill Type	-	Autogenous Grinding Tumbling Mill	
Mill Dimensions	-	6.71 m Dia. × 3.96 m EGL	

Description	Unit	Nominal	Design
Primary Grinding Cyclone Cluster			
Target Product Size P_{80}	µm	212	
Cyclone Diameter	mm	508	
Secondary Grinding			
Feed Rate	dry t/h	105.5	121.3
Installed Power	kW	1250	
Mill Type	-	1 x Stirred Vertical Mill	
Secondary Grinding Cyclone Cluster			
Target Product Size P_{80}	µm	53	
Cyclone Diameter	mm	254	
Gravity Concentration			
Scalping Screens Dimensions		1.83 m L × 0.92 m	
Slurry Flow to Screens (combined)	t slurry/h	168	202
# of Parallel Gravity Circuits	#	2	
Slurry Flow to Concentrators	m^3/h/unit	61	70
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 Screens(combined)	m^3/h	94	116
Flotation Concentrate Handling			
Thickener Fresh Feed Slurry Flow	m^3/h	28.7	32.7
Thickener Underflow	m^3/h	3.8	4.3
# of Flotation Concentrate Storage Tanks	-	1	
Concentrate Storage Tank - Storage Time	h	24	
Concentrate Storage Tank – Live Volume	m^3	91.0	103.6
Tailings Handling			
Tailings Thickener Fresh Feed Flow	m^3/h	344.6	396.3
Tailings Thickener Underflow	m^3/h	110.6	127.2

Description	Unit	Nominal	Design
# of Tailings Filter Feed Storage Tank	-	1	
Tailings Filter Feed Storage Tank-Live Volume	m^3	95.4	
Number of Filter Presses	-	2	

Source: DRA, 2024

17.5 Process Description

The Čoka Rakita process plant is designed to process 850,000 t of ore per annum. ROM ore is crushed by a primary FLSmidth jaw crusher and then stored in a coarse ore bin which provides 24 h surge capacity. The crushed ore undergoes two-stage grinding, first in an autogenous grinding (AG) mill (including a pebble crusher) and then a vertical stirred mill, achieving 80% passing 53 µm. Part of the secondary cyclone underflow feeds a two-stage gravity concentration circuit, producing high-grade concentrate for smelting, while second stage tailings are filtered, bagged and sold as a separate product. A bulk sulphide flotation plant generates a gold concentrate that is filtered and transported off-site. In total, 87% of the gold is recovered (23% as doré, 13% as gravity concentrate, and 51% in flotation concentrate). Tailings are disposed as either filtered cake for storage or paste backfill for underground. Key reagents include flocculant, potassium amyl xanthate (PAX), A3477, copper sulphate, and MIBC frother. Water is sourced from mine dewatering and precipitation, with clean water collected in dedicated ponds and contact water treated before discharge.

ROM ore is transported from the 3-day stockpile to the ROM bin using front-end loaders and fed into the primary jaw crusher through a vibrating grizzly feeder, with grizzly bar openings set at 64 mm. Undersized material from the grizzly goes directly to the product conveyor, while oversized material is crushed from an initial size of up to 500 mm down to 80% passing 92 mm. The crushed ore is then transported via a short sacrificial conveyor and a longer overland conveyor to the coarse ore bin. Crushing operations will run only during the 12-hour day shift. The coarse ore bin, designed with a 24-hour surge capacity, enables continuous operation of downstream processes and uses a mass-flow design to prevent freezing of ore in winter.

The Čoka Rakita ore is of medium hardness with minimal abrasiveness, as shown by test values of 13.7 kWh/t for the Bond Ball Mill Work Index and an abrasivity index of 0.114. The grinding operation is designed in two stages, each circuit closed-out by dedicated hydrocyclone clusters. The primary grinding circuit uses a 6.71 m diameter AG mill to grind the ore to 80% passing 212 µm. A pebble crusher avoids build-up of critical size pebbles within the AG milling circuit. Steel grinding media can be added to increase flexibility and control of grind size if needed.

The secondary grinding circuit features a vertical stirred mill (VTM-1250-WB) that further reduces the material to 80% passing 53 µm, drawing up to 850 kW for harder ore. Both grinding circuits are supported by recycled process water to achieve optimal slurry density, with water added at the discharge hopper to facilitate efficient classification by the cyclones. Hydrocyclones close each stage, with installed spare cyclones and unused cyclone ports to facilitate maintenance and efficiency adjustments.

Circulating loads are set at 146% for the primary circuit and 188% for the secondary, with additional cyclone capacity available to increase efficiency. This flexible grinding setup allows for capacity expansion and efficient processing, accommodating variations in ore hardness while maintaining consistent product quality and operational continuity.

The gravity concentration circuit at Čoka Rakita operates in two stages. In the primary stage, two parallel lines contain scalping screens and centrifugal concentrators, treat a portion of the underflow from the secondary grinding cyclones. Water is added to improve flow and screening efficiency. The scalping screens remove oversize particles (+3 mm), sending them back to the pump box. Screen undersized material proceeds to centrifugal concentrators, which captures dense and coarse particles in its grooves. Tails from this stage return to the vertical mill feed, while concentrate is flushed hourly to a holding tank in the secure gold room. The holding tank is sized to accommodate 1 day of arisings which allows for day-shift operations only.

Concentrate from the holding tank is processed on a shaking table for about five hours daily, producing around 75 kg of final concentrate at a grade of 43 kg/t gold. The table is sized to accommodate surges up to 105 kg/day at 76 kg/t. The final gravity concentrate is filtered, dried, and smelted to produce doré bars, while the table tailings are dewatered using a dewatering cone followed by a filter press, bagged, and shipped off-site as an intermediate-grade product, averaging 1,327 g/t gold in a mass of about 1.43 dry tonnes daily.

The overflow from the secondary grinding cyclone, at about 27 wt% solids, flows by gravity to the rougher conditioning tank, where flotation reagents are added. This mixture overflows to the rougher scalper flotation cell. The rougher scalper cell concentrate goes to the flotation thickener, while its tailings feeds to a scavenger cell to capture any remaining floatable particles. Tailings from the scavenger reports to a tailings thickener, with both cells recycling about 46% of their flow.

The scavenger concentrate is sent to a cleaner conditioning tank, which flows to a dedicated cleaner cell. Cleaner concentrate combines with scalper concentrate to produce the final flotation concentrate, while cleaner tailings are recycled to the rougher tank. This final concentrate contains 51.7% of the ROM gold at 63 g/t and represents 4.7% of the ROM feed by mass. Reagents used include 0.2 kg/t PAX, 0.09 kg/t MIBC, 0.25 kg/t copper sulphate, and 0.2 kg/t Aero3477.

The flotation concentrate produced is dewatered in a 4-m diameter thickener. The thickened underflow is then pumped to an agitated surge tank located by the plate-and-frame concentrate filter press. Filter cakes, containing 9% moisture, are stockpiled in compartments which provide a combined 7-days storage capacity, allowing for blending to meet required grade specifications for various elements. A front-end loader will transfer the filtered concentrate from this storage area into a bulk transport truck for shipment.

The tailings stream from the flotation scavenger cell is pumped to a 20 m diameter tailings thickener. The thickened underflow, with 57% solids by weight, is pumped to a storage tank near the Filter Plant at the first Mine Decline. Two (2) parallel plate-and-frame filters dewater the tailings, producing filter cakes with a 13% moisture content.

When needed, part of the tailings is sent to a paste mixer, where it is blended with water and approximately 5% cement (by dry solids mass) to create a paste for backfilling underground. Around 46% of the annual tailings are used as paste backfill underground, while the remaining 54% is transported and stacked in a designated DTSF.

17.6 Reagents and Consumables

The following reagents are used throughout the process plant:

- Flocculant – Concentrate and tailings thickeners;
- Copper sulphate – Flotation reagent;
- Potassium amyl xanthate (PAX) – Flotation collector;
- Cement – For paste plant;
- Methyl isobutyl carbinol (MIBC) – Flotation frother;
- A3477 – Flotation collector: and
- Borax, Silica sand and Lime – Smelting fluxes.

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 on site long-term reagent supply storage is provided at a safe distance away from the process plant.

Reagents solutions are produced by dilution with treated water obtained from the clarifier.

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 17.3.

Table 17.3 – Reagents and Consumables

Description	Unit	Consumption	
		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	200	250
Frother (MIBC)	g/t ROM	90	112.5
Grinding Steel			
VertiMill Media Consumption	kg/t ROM	0.4	0.4

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. This water will be used to replenish other water sources, when needed. 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 routed to the Treated Water System as make-up to minimise the volume that has to discharge to the environment.

17.7.2 Air Services

Compressors will provide compressed plant air for the filter presses and dried instrument air for pneumatic devices. The aboveground operations have two (2) separate plant air systems: one dedicated to the tailings filtration and paste backfill plant, including the primary crusher area, and another serving the grinding, flotation, and workshop areas.

17.7.3 Electrical Services

The total installed power for the site is estimated at 7.3 MW.

17.8 Process Control Strategy

The Project will have an adequate level of automation and remote-control facilities. Instrumentation will be provided within the plant to measure and control key process parameters and to provide a safe work environment.

Automated samplers feeding online analysers will be used to control key process parameters and for metal accounting purposes.

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 stations (OIS).

Control loops will have two modes of operation from the PCS: Manual or Automatic. All parameters for PID control loops will be available to the control room operator for adjustment. The Manual (MAN) mode for control loops is when a value like a percentage or speed is set by the Operator directly to valve or motor without PID controlling. The Automatic (AUT) mode has three sub-modes AUT INT, AUT EXT and AUT ACT. AUT INT is a mode where the set-point (SP) of the PID controller is set by the Operator and PID control is activated (ON). AUT EXT is a mode where the Operator cannot change the SP as the SP follows internal process control logic. AUT ACT is a mode where the PID control follows advanced control logic

In general, the plant process drives will, as a minimum, report their Ready, Run and Fault status to the PCS system and will be displayed on the OIS.

The OIS will allow drives/valves to be selected to any of the Auto, Manual and Maintenance modes via the drive/valve control popup on the OIS. Statutory interlocks such as emergency stops and overload protection are hardwired and will apply in all modes of operation.

The control of most of the vendor packages should be controlled remotely from PCS OIS and ethernet connection to control system network. General equipment fault alarms from each vendor package will be monitored by the PCS system and displayed on the OIS. 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 be able to perform trending of all measured data as well as system variables.

18 PROJECT INFRASTRUCTURE

18.1 General

This section describes site-wide mine and process plant supporting infrastructure, as follows:

- Roads and terraces;
- Site buildings;
- Filter cake and paste backfill system;
- Mine waste rock management;
- Dry tailings storage facility (DTSF);
- Water management;
- Power supply and distribution; and
- Automation and Communication.

No geotechnical investigations have been completed to date for the purpose of assessing ground conditions for construction.

The proposed site layout and access are shown in the Figure 18.1. and Figure 18.2 illustrates the proposed details of the plant layout.

18.2 Main Access Road

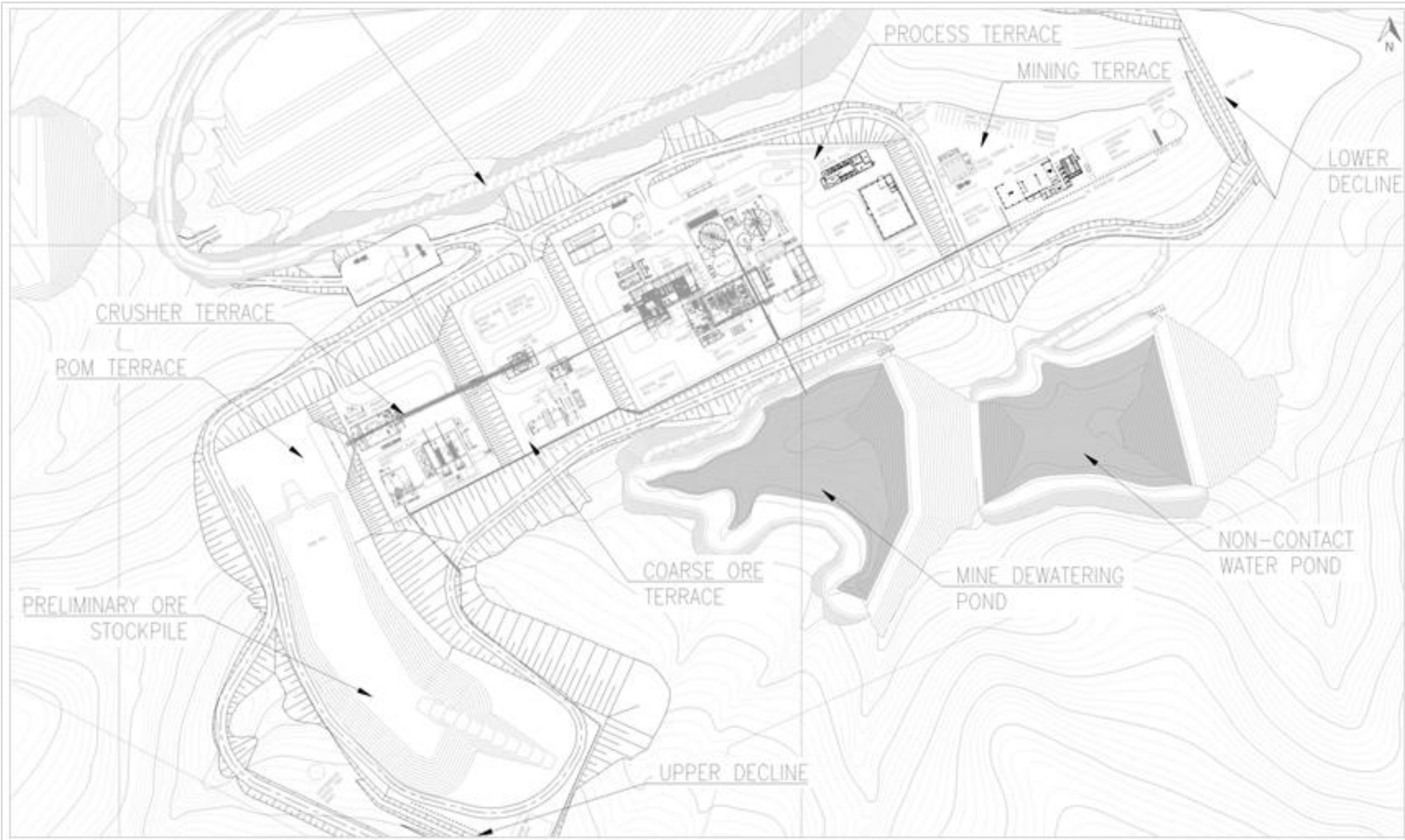
The Project site will be accessed via a new road to be constructed from an existing public road near the southeast corner of the site perimeter. This road will be orientated in a general north-south direction. This road will service the administration terrace near the highway turnoff and provide access as the main travel way to the mining and process plant areas.

The main access road will feature two (2) guard houses: the first will control access to the administration area, and the second will control access to the mining, process plant and DTSF areas.

Figure 18.1 – General Site Layout

Source: DRA, 2024

Figure 18.2 – Process Plant Layout



Source: DRA, 2024

18.3 Site Roads

A series of roads will provide access to the various Project site areas. All site roads, except for mine roads, will be 8 m wide. The haul roads will be sized for underground mining trucks exiting the mine portals and traveling to the ROM pad

New roads on the site will include:

- Main Site Access Road to the Process Plant Terraces.
- Access road between the process terraces – north of Process Plant Terraces.
- Haul Road – south of Process Plant Terraces.
- Dry Tailings Stockpile Access Road.
- Dry Tailings Facility Access roads.
- Tailings Material Transport Roads.
- Contact Water Pond Access Road.
- Non-Contact Water Pond Access Road.
- Reclaim Pond Access Road.
- Ventilation Facility Access Road.

The road on the south side of the process plant terrace will be used by mining trucks to haul between the upper and lower declines. Since this road will be managed and controlled by mining personnel, it will also be used by other mining vehicles (materials transport, personnel transport). The South Haul Road will not be used by process plant or general surface vehicles.

The road on the north side of the process plant terrace will be used as the access route to and from each of the process plant terraces, for general surface personnel, materials and equipment. It will be developed at full width, down to the lowest terrace level (Mining Truck Wash terrace).

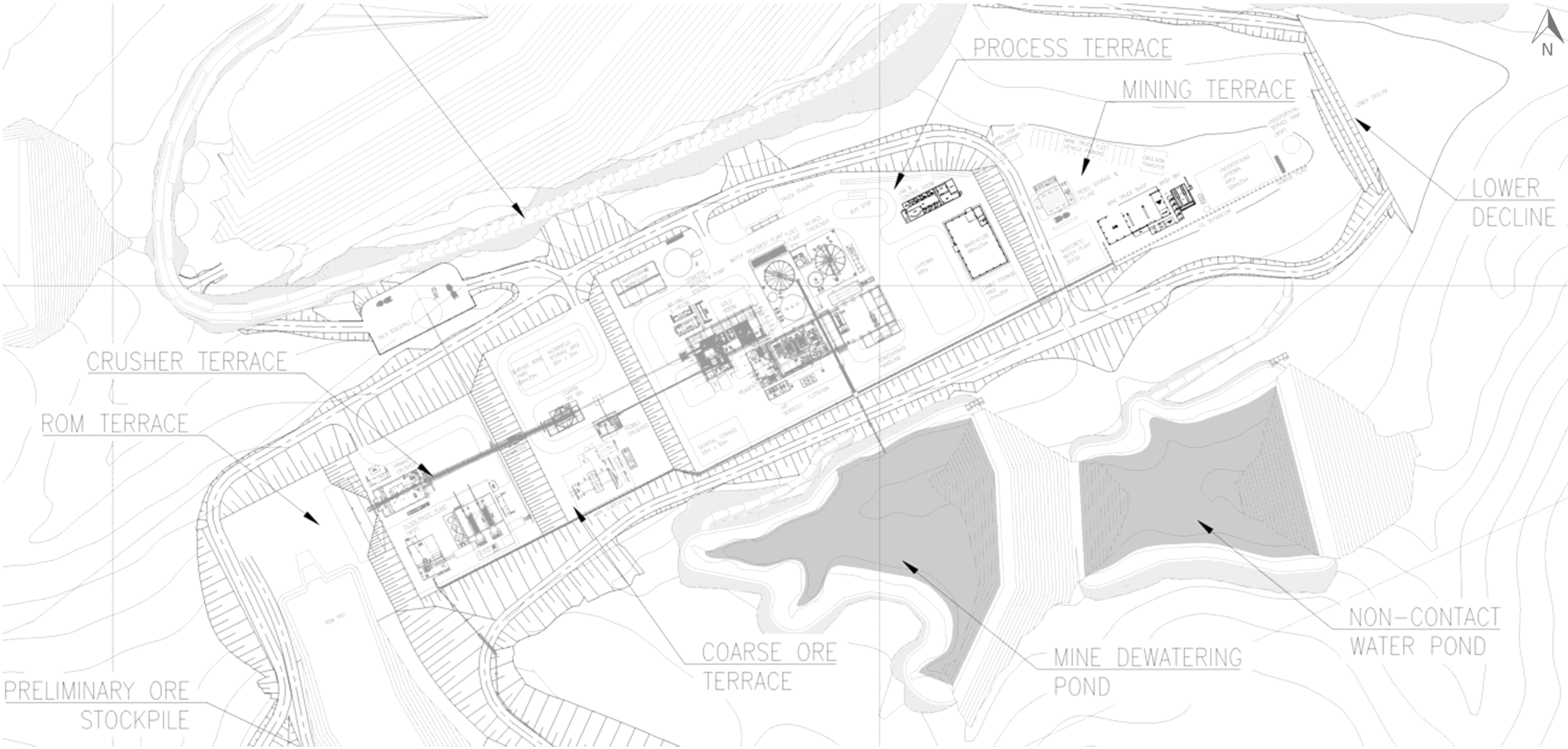
18.4 Terraces

The Project site requires terraced areas for Project infrastructure and equipment. Terraces are designed iteratively to accommodate both Project element layout and site topography / terrain.

The seven (7) terraces are designed with grading and drainage systems to allow the runoff water to reach the collection ponds and natural drains in accordance with the site water management requirements. Five (5) of the terraces are depicted in Figure 18.3.

The terraces will provide working levels for the various site facilities. The terraces, locations and facilities are further depicted and described in the following subsections.

Figure 18.3 – On-Site Terraces (5 of 7)

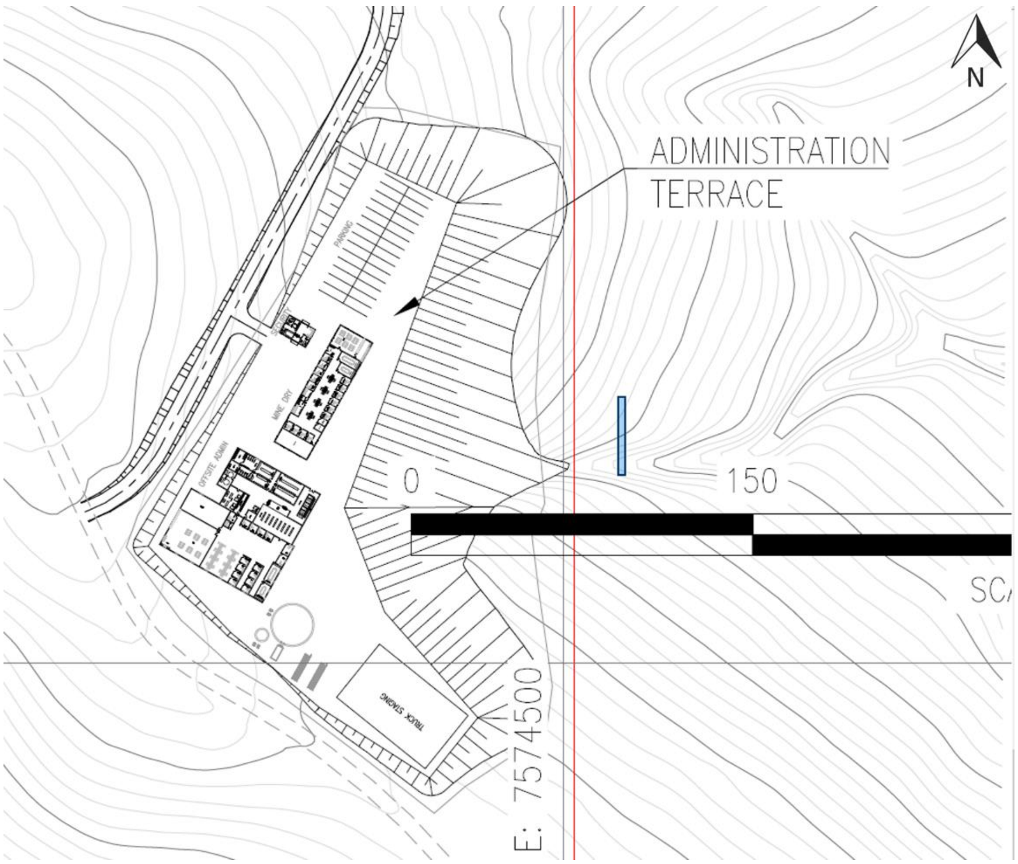


Source: DRA, 2024

18.4.1 ADMINISTRATION TERRACE

The Administration Terrace (depicted in Figure 18.4) is located at the entrance to the southwest access road, and it provides a platform for the main guard house, main administration building, mine administration and mine dry, designated parking areas, and utilities such water storage systems and sewage treatment facilities. The area is enclosed by fencing for security.

Figure 18.4 – Administration Terrace

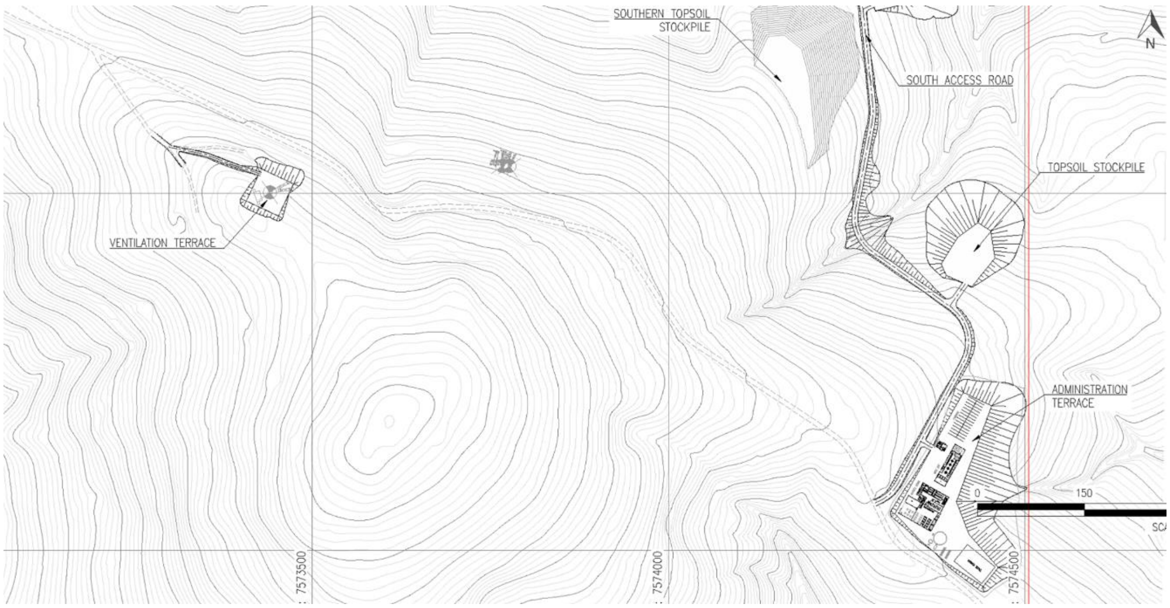


Source: DRA, 2024

18.4.2 VENTILATION TERRACE

The Ventilation Terrace (depicted in Figure 18.5) provides a platform for the fans and their associated and dedicated E-House all required for ventilating underground areas. The area is enclosed by fencing for security, and a security gate is provided for access.

Figure 18.5 – Ventilation Terrace



Source: DRA, 2024

18.4.3 ROM TERRACE

The ROM Terrace (depicted in Figure 18.3) is located adjacent to the crusher building and will provide a storage area for ROM material.

18.4.4 CRUSHER TERRACE

The Crusher Terrace (depicted in Figure 18.3) is located adjacent to the ROM stockpile. It provides a platform for the crusher building, the paste backfill plant and the filter plant as well as the filtered tailings discharge conveyor and the coarse ore bin feed conveyor.

18.4.5 COARSE ORE (CRUSHED ORE BIN) TERRACE

The Coarse Ore (Crushed Ore Bin) Terrace (depicted in Figure 18.3) is located next to the Crusher Terrace. It provides a platform for the crushed ore silo, AG mill feed conveyor, pebble crusher and discharge and feed conveyors, main sub-station, surface boneyard and hazardous storage area.

18.4.6 Process Terrace

The Process Terrace (depicted in Figure 18.3) is located adjacent to the Coarse Ore (Crushed Ore Bin) Terrace and will serve as the location for the main process and support facilities, including: AG mill and secondary regrind mill building, gold room, flotation building, concentrate handling building, reagents building and tailings thickener. The support facilities include walkways and a bus stop area, plant offices, laboratory and control room, warehouse and laydown area, cable storage yard, water treatment plant, fire and domestic tank, truck staging and weighbridge, and air compressor building.

18.4.7 Lower (Mining) Terrace

The Lower (Mining) Terrace (depicted in Figure 18.3) is located east of the Process Terrace and supports mining activities. The facilities located on this terrace include: mine truck-shop and wash bay, shotcrete batch plant, diesel storage facility, emulsion transfer area, mine truck fleet vehicle parking, treated water tank, underground laydown area and main sewage treatment plant.

18.4.8 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.

18.5 Buildings

18.5.1 Process Plant Configuration

The PFS describes the development of a crushing, grinding and flotation process and supporting infrastructure associated with treatment of sulphide minerals, which will require various site buildings to house the required Project elements. Design elements were progressed to establish a layout of buildings and other site elements for the Project, including:

- Crushing building – from Ada Tepe.
- Tailings Filtration building – new.
- Paste Backfill building -new.
- AG Mill building – from Ada Tepe.
- Flotation and Filtration building – from Ada Tepe.
- Reagent building – from Ada Tepe.
- Storage buildings – new.
- Truck Shop – new.
- Truck Wash – new.
- Administration building – new.

- Gate houses (2) – new.

18.5.2 Plant Location

The process plant site location was established considering various criteria, including:

- Proximity to DTSF.
- Proximity to mine portals.
- Proximity to land ownership boundaries.
- Condemnation drilling.

Available topographical areas were considered that will require a practical quantity of earthworks for the establishment of plant terraces and connecting of roads to minimise haulage.

18.5.3 Building Designs

All structures will be conventional type insulated structures. Concrete foundations will be designed for soil conditions, following geotechnical work, and site climatic conditions.

For the purposes of the PFS, DPM decided to utilise as many of the existing Ada Tepe facilities as possible for the new Čoka Rakita Project. It should be noted that only the steel structures and equipment will be dismantled and relocated to the Project site. The buildings to be relocated are:

- Primary crusher building.
- AG mill and secondary regrind mill building.
- Flotation building.
- Concentrate handling building.
- Reagents building.
- Tailings thickener.
- Plant workshop.

Other facilities will also be relocated, including: E-Houses, weighbridge and emergency generator.

Prefabricated facilities include the main administration building, mine dry facility, guard houses, plant offices, laboratory and control room, sewage treatment plants, and water treatment facility.

18.5.4 Process Facilities

18.5.4.1 Primary Crusher

The Primary Crusher area will consist of a new retaining wall to create the tip area, a Run of Mine (ROM) Tip with jockey slab and static grizzly. The Primary Crusher building structure will be relocated from Ada Tepe and adjusted to suit the conditions of the Project site. The Primary Crusher building is 37 m x 20 m and will receive new insulated cladding and roof sheeting.

18.5.4.2 Coarse Ore Bin

The Coarse Ore Bin will be a new steel structure located on the process plant terrace. It will receive ore from the Primary Crusher and feed the AG Mill. This bin will have a volume of 1,819 m^3 in accordance with the Process Design Criteria. The Coarse Ore Bin will also be fitted with the existing dust collector system from Ada Tepe.

18.5.4.3 Process Area

The process area facilities comprise the equipment and facilities required for processing ROM ore. These facilities will be conventional insulated structures equipped with 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.

18.5.5 Site Wide Infrastructure

18.5.5.1 ROM Pad

The ROM Pad is the terraced area where ore material will be stockpiled and handled from the upper decline to the ROM tip. Initially this will be covered by the larger Preliminary Ore Stockpile.

18.5.5.2 Preliminary Ore Stockpile

The upper decline terrace and ROM terrace is connected with engineered fill in order to create a laydown space for the Pre-Production Stockpile as required and sized by WSP. It is located close to the upper decline portal and the ROM Tip where the material needs to be fed to the plant once operational. Further details of this stockpile are given in Section 18.8.

18.5.5.3 Topsoil Stockpile

Topsoil will be stockpiled in two (2) locations on the western embankments of the DTSF, above the diversion channel levels as it is Non-Acid Generating (NAG). It will be accessed via the DTSF access roads and temporary construction roads.

Further details of these stockpiles are given in Section 18.8.

18.5.5.4 Additional Stockpiles

Three stockpile locations were identified along the main access road for NAG overburden material from cut and fill operations during construction, should the need to stockpile surplus material arise.

18.5.5.5 Ventilation Facility

A ventilation facility is required with a vent shaft diameter of approximately 4 m to meet the ventilation requirements for the underground workings. It is presumed at this stage that it will be a bi-furcate unit. In order to meet the locational needs of the underground, this facility is located off-site due south from the process plant, across the existing public road. The ventilation facility will require its own access road and terrace.

18.5.5.6 Main Administration Building

The Main Administration Building will be a 60 m x 17 m modular building structure featuring offices, boardrooms, a training centre and the main site canteen. The building will be located on the Administration Terrace near the main access road highway turn off from the existing public road.

18.5.5.7 Mine Administration and Mine Dry

The Mine Administration Building will be a 52 m x 29 m modular building structure featuring offices, boardrooms and a dining area. The Mine Dry will be a 42 m x 26 m modular building containing the lamp room and underground tag control room, boot-wash, laundry, showers, change rooms, and ablutions.

These buildings will be a combined facility and will be located on the Administration Terrace near the main access road highway turn off from the existing public road.

18.6 Filter Cake Production and Paste Backfill System

WSP completed the PFS Paste Backfill engineering portion of the Čoka Rakita Project based on information that was developed by WSP and other parties during the PEA phase of the Project.

The Filter and Paste Plants are located on the Crusher Terrace. Filter cake will be conveyed horizontally along the crusher terrace to the north side of the processing terraces and deposited in a stockpile. The filter cake will then be trucked to the DTSF.

The Paste Plant would pump paste from the plant on the crusher terrace via a surface pipeline to the upper decline portal from where it would be delivered into the paste Underground Distribution System (UDS).

Dewatered tailings are received as thickener underflow from the Mill and stored in an agitated filter feed tank, which delivers the thickened tailings primarily to the horizontal plate and frame pressure filters, and as required to the Paste Plant mixer as dilution slurry to create the proper CPF consistency, or slump.

There are two (2) pressure filters, one Duty and another Standby, each having a dedicated hydraulic power pack and air compressor. This redundancy is very important since uninterrupted tailings filtration is critical to allow for the continuous operation of the Mill.

Each pressure filter has a dedicated conveyor beneath it, which discharges onto a common reversable conveyor. The reversable conveyor directs filter cake to the Paste Plant for backfill production or to the DTSF for disposal, via dedicated inclined conveyors.

The inclined conveyor sending filter cake to the DTSF discharges onto a horizontal conveyor on the crusher terrace which in turn discharges onto a filter cake stockpile where a loader fills a truck that hauls the cake into the DTSF for final placement. This use of a conveying system to partially deliver filtered tailings to the DTSF is employed to reduce mobile equipment traffic in a busy part of the Mill complex and the potential of intersecting site traffic.

The pressure filter plate cleaning, core, and cloth wash processes consume a large amount of water that is collected in a central sump located between the two (2) filters and containing four (4) sump pumps. One (1) additional sump pump is also located in the below grade area housing the reversable conveyor.

The sump pumps discharge into an agitated wastewater tank. Should the filtrate quality be good, this may be diverted back to the process water tank.

The plant is equipped with an overhead crane above the filters to allow the installation and removal of filter plates as well as general maintenance activities. Storage tanks for water, dirty water, and thickened tailings are all located outdoors, and it is anticipated that any maintenance associated with the tanks will be completed from the outside. Their associated pumps are located inside the Filter Plant and are readily accessible by forklifts.

Figure 18.6 illustrates the position of the Filter and Paste Plants and the crusher terrace conveyor to the Filter Cake Stockpile to the north of the Crusher terrace.

Figure 18.6 – Crusher Terrace

Source: WSP, 2024

18.6.1 DESIGN CRITERIA

The PFS Design Criteria for the Filter and Paste Plants are summarised in Table 18.1.

Table 18.1 – Basic Design Criteria for Filter and Paste Plants

Description	Units	Quantity Value
Mine and Mill Operation		
Mine and Mill Operating Time – Days per Year	d/y	336
Production Years (Life of Mine Plan [LOMP])	y	10
Ore Production and Processing		
-Annual Ore Total	t/y	865,715
-Daily Ore	t/d	2,578
-Hourly Ore	t/h	107

Description	Units	Quantity Value
Tailings for Backfill Use or for Disposal		
-Annual Tailings Total	t/y	831,086
-Daily Tailings	t/d	2,475
-Hourly Tailings	t/h	103
Paste Plant Operation (same as Mill Operation)		
Available Operating Days per Year	d/y	336
Paste Plant Utilisation	% operating required per year	52.5
Tailings Disposal/Utilisation		
Tailings Reporting to DTSF	%	46.4
Tailings Reporting to Backfill	%	53.6
Ore Characteristic		
Ore Specific Gravity		3.17
Tailings Characteristic		
Tailings Specific Gravity		3.20
% passing 20 µm	%	29
Paste Characteristics		
Paste Backfill Solids Content at 178 mm (7") Slump	wt% solids	75.3
Paste Bulk Density at 178 mm (7") Slump	t/m^3	2.07
Consolidated (cured) Solids Content at 178 mm (7") Slump	wt% solids	77.8
Consolidated (cured) Bulk Density at 178 mm (7") Slump	t/m^3	2.14
Paste Backfill Solids Content at 254 mm (10") Slump	wt% solids	73.9
Paste Bulk Density at 254 mm (10") Slump	t/m^3	2.02
Consolidated (cured) Solids Content at 254 mm (10") Slump	wt% solids	77.2
Consolidated (cured) Bulk Density at 254 mm (10") Slump	t/m^3	2.12
Target Strength		
Sublevel Longhole Open Stoping (28-day)	MPa	0.5
Backfill Requirements (at 178 mm (7") Slump)		
In-situ Paste Fill Requirement	m^3/d	1,706
Fill Replacement Factor- 178 mm Slump (dry tons)	%	52.5
% of Voids Requiring Paste Backfill for Mining (1)	%	98.0
-Total Paste Backfill Volume Required Annually	m^3/y	267,634
Binder Addition		

Description	Units	Quantity Value
Open Stoping	wt%	5.0
Binder Consumption		
Open Stoping	t/y	23,452

Source: WSP, 2024

18.6.2 Filter Plant Process Description

Dewatered tailings are received as thickener underflow from the Mill and stored in an agitated filter feed tank, which delivers the thickened tailings to the horizontal plate and frame pressure filters. After the filtrate is removed from the filter presses, it is transported via conveyor to either the filtered tails stockpile or to the Paste Plant mixer as dilution slurry to create the proper CPF consistency, or slump.

Process water is received from the Mill and distributed throughout the Filter and Paste Plants for various purposes:

- Pressure filter plate cleaning, core and plate wash;
- Gland water for pump seals to the various pumps in the Filter and Paste Plants; and
- General use process water in the Filter and Paste Plants.

Compressors in the plant deliver compressed air to air receivers which in turn provide air for:

- Dry instrument air required throughout the Filter and Paste Plants;
- General use in both the Filter and Paste Plants; and
- Core and cake air blow to purge excess water from the pressure filter cake and to further dry the cake prior to discharge.

There are two (2) pressure filters, one duty and one standby, each having a dedicated hydraulic power pack and air compressor. This redundancy is very important since uninterrupted tailings filtration is critical to allow for the continuous operation of the Mill.

Each pressure filter has a dedicated conveyor beneath it, which discharges onto a common reversable conveyor. The reversable conveyor directs filter cake to the Paste Plant for backfill production or to the DTSF for disposal, via dedicated inclined conveyors.

The inclined conveyor sending filter cake to the DTSF discharges onto a horizontal conveyor on the crusher terrace, which in turn discharges onto a filter cake stockpile where a loader fills a truck that hauls the cake into the DTSF for final placement. This use of a conveying system to partially deliver filtered tailings to the DTSF is employed to reduce mobile equipment traffic in a busy part of the Mill complex and the potential of intersecting site traffic.

Storage tanks for incoming process water and outgoing discharge water, and thickened tailings are all located outdoors; and it is anticipated that any maintenance associated with the tanks will be completed from the outside. Their associated pumps are located inside the Filter Plant and are readily accessible by forklifts.

18.6.3 PASTE PLANT PROCESS DESCRIPTION

The Paste Plant operates in a campaign fashion whereby paste is only produced when backfilling an underground stope is required. Filter cake is delivered from the Filter Plant via an inclined conveyor and into a live bottom feeder which serves to break up any large pieces of filter cake prior to depositing it onto a conveyor. This conveyor has a weigh bridge and discharges the filter cake into the paste mixer.

Thickener underflow slurry from the filter feed tank at the Filter Plant is metered into the paste mixer to dilute the filter cake to the desired slump consistency.

A binder silo/bin delivers cement binder through a screw feeder to a loss in weight hopper which meters binder into a second screw feeder and into the paste mixer. The paste mixer continuously combines and homogenises measured amounts of filter cake, binder, tailings slurry, and water to a predetermined slump consistency.

There are two (2) positive displacement pumps, one (1) duty and another standby. The positive displacement pumps discharge paste into the surface pipeline to the upper decline portal and through the underground paste distribution system to stopes.

The surface pipeline has a positive gradient between the Paste Plant and the upper decline portal, so cleaning of this portion of pipeline requires an additional measure over simply flushing with water and compressed air. A pipeline Pig insertion system is installed in the paste pipeline before it leaves the plant building. A pig is inserted into the pipeline system at the pump discharge to allow the water pushing the pig to fill the pipe cross section as it cleans the pipe. This process cycle may be repeated. Then finally air is used as a final flush to clear the pipeline of residual water.

18.6.4 FILTER AND PASTE PLANT 2D AND 3D MODELS

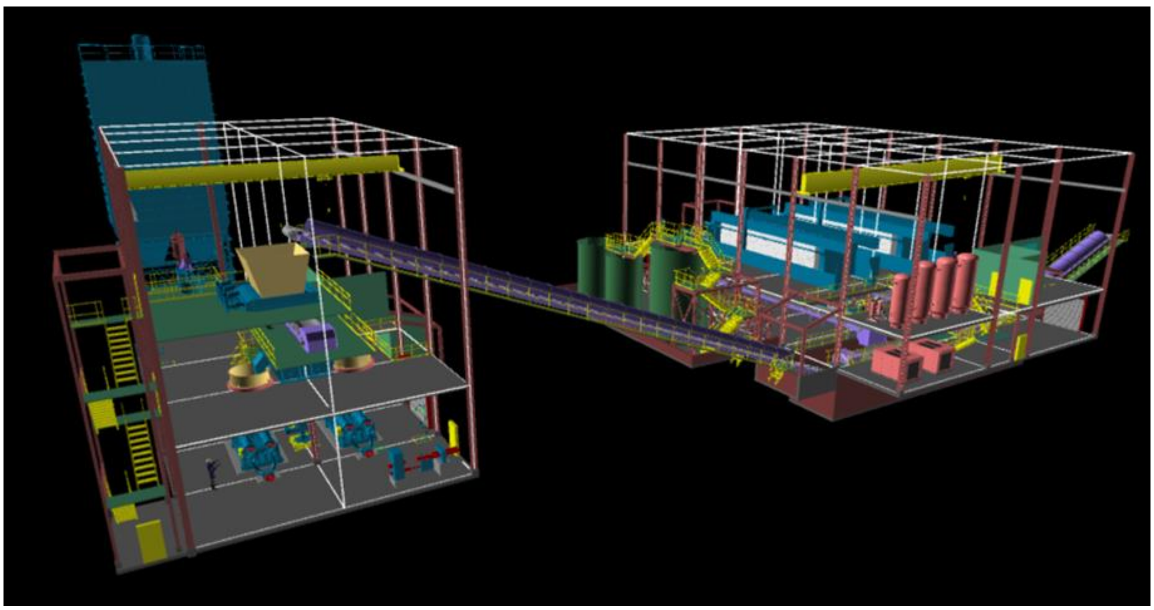
The plants are designed with thought given to such elements as:

- Clearance and accessibility for equipment operation, maintenance and housekeeping, e.g., area on the pressure filter floor to store cartridges of fully prepared filter plates, and spare plate frames and filter cloth or accessibility to the paste hoppers for regular cleaning;
- Proximity of facilities to where QA/QC activities are required, e.g., curing and UCS room on the same floor in close proximity to the paste mixer for UCS cylinder casting activities;
- Lifting well and exterior overhead door for ease of delivery and removal of equipment and materials from the plants;

- Optimising the footprint of the plants by minimising unnecessary area;
- Lowering the height of the Filter Plant by installing the reversible conveyor below the ground floor elevation;
- Centralising common services for both plants in the Filter Plant building;
- Providing adequate storage to allow uninterrupted operation of the Paste Plant such as weather events that could affect delivery of binder;
- Providing emergency egress stairs in case of an unexpected building evacuation; and
- Providing comfort facilities for the plant personnel in the form of washroom and breakrooms as well as an area to monitor record and transmit key operating parameters of the plants.

Figure 18.7 is a view looking northeast, with the Paste Plant on the Left and Filter Plant on the right.

Figure 18.7 – Filter Plant (right) and Paste Plant (left), Looking Northeast



Source: WSP, 2024

It should be noted that the equipment sizes used in the models are “typical” and will be refined in the FS phase of the Project, when the final equipment vendors have been selected.

18.6.5 ELECTRICAL

The estimated electrical Installed Power for all Filter and Paste Plant equipment is 2.72 MW.

For the Filter Plant the estimated equipment power is:

- Installed Power – 1.18 MW;

- Consumed Power for the Filter Plant equipment that which operates 100% of the time the Mill is operating – 727 kW or a consumption of 5.9million kW hours for 336 operating days (not including continuous operating equipment such as agitators);
- Connected Emergency Power – 490 kW; and
- Operating Emergency Power – 275 kW.

For the Paste Plant the estimated equipment power is:

- Installed Power – 1.54 MW;
- Consumed Power for the Filter Plant equipment that which operates approximately 52% of the time the Mill is operating – 936 kW or a consumption of 3.9 million kW hours for 52% of 336 operating days;
- Connected Emergency Power – 1,120 kW; and
- Operating Emergency Power – 560 kW.

The supply of power is accomplished via a standalone modular power distribution system on the east side of the Filter Plant that is based on the requirements presented in the MEL.

18.7 Mine Waste Rock Management

The PFS LOM Plan and Mining strategy indicate that 45% of mine waste will be stored on the surface, with the remaining waste stored underground via a paste backfill system. Mine waste Management at the Čoka Rakita Project will be managed via two (2) processes:

1. Potentially-Acid Generating (PAG) waste rock from initial development will be hauled to the surface and deposited within the DTSF footprint, enclosed in a composite lining system
2. Filtered tailings will be transported via conveyor to a loading area, then trucked to the deposition point, progressive covering the PAG waste rock.

Detailed design, transportation and construction information for the DTSF is present in Section 18.8.

18.7.2 Preliminary Ore Stockpile

The optimisation, design and stability analysis of the Preliminary Ore Stockpile as completed by WSP. The key design criteria are presented in Table 18.2 with the design summary details presented in Table 18.3.

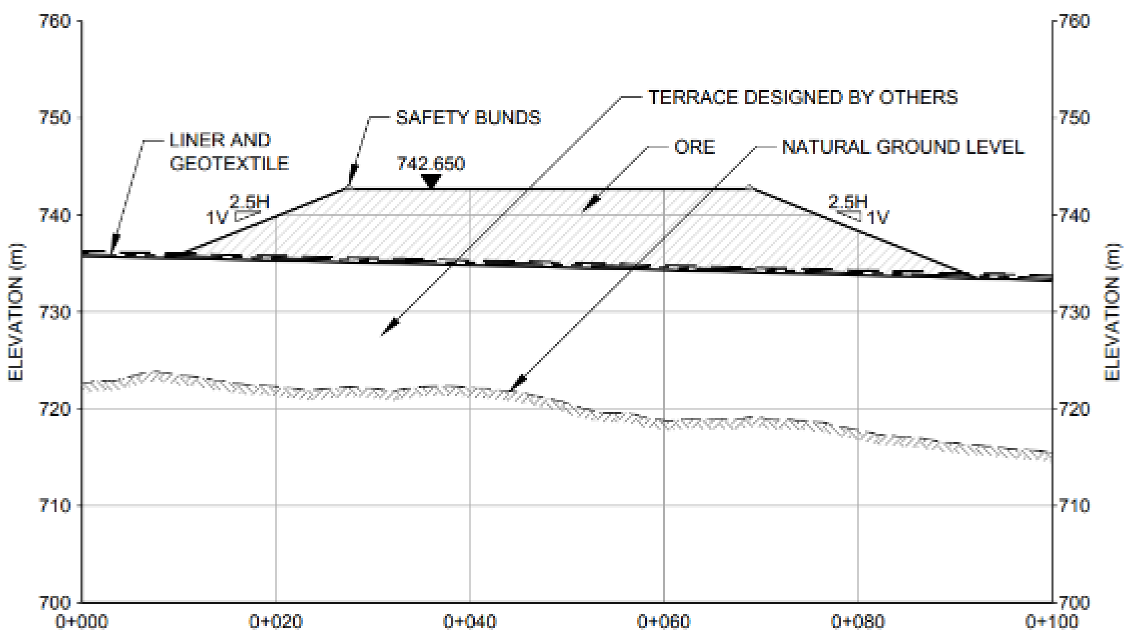
Table 18.2 – Preliminary Ore Stockpile Design Requirement

Description	Quantity (t)	Quantity (m^3)	Comment
Preliminary Ore Stockpile Capacity	185,000	92,500	Storage quantity requested by DPM, total storage required will likely reduce during FS following update to LOM plan.

Table 18.3 – Preliminary Ore Stockpile Design Summary

Element	Value
Ore Storage Volume Required (m³)	92,500
Total Stockpile Height (m)	11.5
Stockpile Footprint (m²)	18,802
Stockpile Crest Elevation (mRL)	742.65
Stockpile Base Elevation – North (mRL)	740
Stockpile Base Elevation – South (mRL)	732
Ore Stockpile Slope Gradient	1V in 2.5H
Ore Stockpile Ramp (North) Volume (m³)	421
Ore Stockpile Ramp (South) Volume (m³)	4,169
Ore Stockpile Ramp Embankment Slopes	1V in 2H
Ore Stockpile Ramp Gradient (%)	10

Figure 18.8 – Preliminary Ore Stockpile Design



Source: WSP, 2024

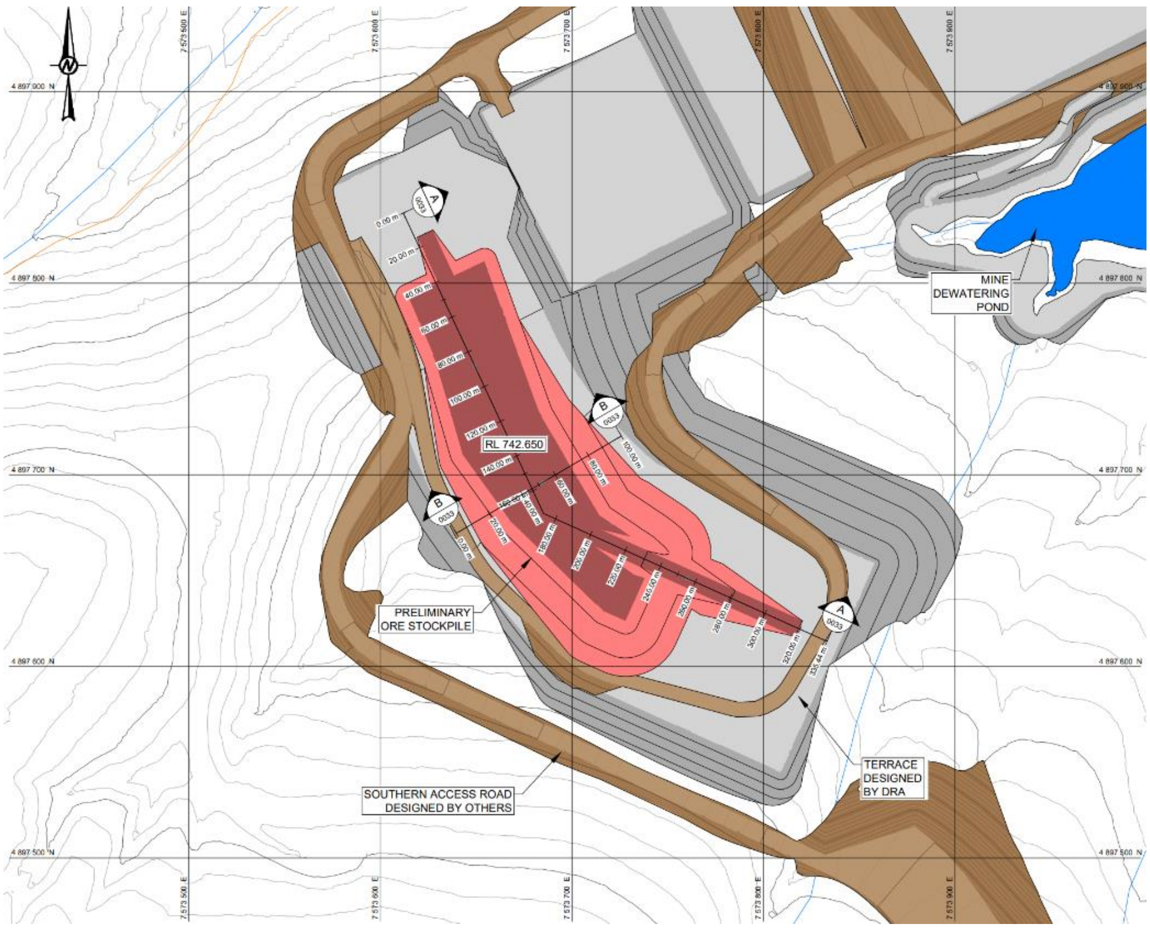
The process terrace that supports the preliminary ore stockpile has been designed by others as part of an overall terrace design package and is discussed elsewhere.

18.7.2.1 Operation

The preliminary ore stockpile temporarily stores ore excavated and transported to the surface via the decline system. Located adjacent to the upper portal and process terrace, it will be built in the early LOM years before processing begins.

The design shows a maximum footprint, but the actual size may vary based on throughput and processing start. A 10 m gap around the perimeter will allow for an access road and drainage channel, with access ramps on the north and south sides at a maximum 10% slope. Surface mobile machinery will be allocated by DPM during the FS stage.

Figure 18.9 – Preliminary Ore Stockpile Design



Source: WSP, 2024

18.7.2.2 Liner

The preliminary ore stockpile will have a 2 mm HDPE liner covering 19,000 m² to prevent acidic leachate from seeping into the ground. Perimeter drainage channels will capture runoff and filter hazardous materials into the terrace drainage system.

18.7.2.3 Stability Analysis

Stability was assessed using the Morgenstern-Price and Fellenius/Ordinary methods, along with seismic analysis. The design meets Serbian standards and CDA requirements. The stockpile terrace will be built from overburden material, with stability analysis parameters inferred from the Timok Gold Project Study. Sensitivity analysis was conducted to account for material strength uncertainties.

The findings of the stability assessment are presented in Table 18.4.

Table 18.4 – Preliminary Ore Stockpile Slope Stability Summary

Description		Scenario	Minimum Required FoS[1]	Calculated FoS (MorganStern-Price)	Calculated FoS (Fellenius/ Ordinary)
Section 1	Downstream Slope	Drained Static	1.5	1.8	1.6
		Undrained Seismic	1.0	1.5	1.3

1. FoS = Factor of Safety

The findings confirm that the design meets the requirement of the Serbian standards (SRPS U.C5.020, 1980) and CDA, (CDA, 2014) in both MorganStern-Price and Fellenius/Ordinary conditions.

The sensitivity assessment details a reduction in the effective friction angle of the recompacted overburden material and its associated Factor of Safety (FoS) value is presented in Table 18.5.

Table 18.5 – Preliminary Ore Stockpile Terrace Sensitivity Analysis

Description		Effective Friction Angle (°)	Minimum Required FoS	Calculated FoS (MorganStern-Price)	Calculated FoS (Fellenius/ Ordinary)
Section 1	Downstream Slope	33	1.5	1.5	1.5
		32	1.5	1.5	1.5
		31	1.5	1.5	1.4
		30	1.5	1.4	1.4
		29	1.5	1.4	1.3

Description		Effective Friction Angle (°)	Minimum Required FoS	Calculated FoS (MorganStern-Price)	Calculated FoS (Fellenius/ Ordinary)
		28	1.5	1.4	1.3
		27	1.5	1.3	1.3
		26	1.5	1.3	1.2
		25	1.5	1.2	1.2

18.7.2.4 *Water Management*

A 2 m wide v-drain drainage channel will be built next to the perimeter access road. Surface water runoff from the preliminary ore stockpile will flow into this channel and be directed to the dewatering pond, as it may have contacted ore material. This local drainage channel will connect to the larger terrace water management system designed by DRA.

18.7.3 TOPSOIL STOCKPILES

18.7.3.1 *Design Inputs*

Topsoil stockpiles are required to provide total storage for all site-won topsoil excavated during the LOM. Table 18.6 presents the general assumptions regarding topsoil generation as well as the basic design criteria for stockpile design as specified by DPM.

Table 18.6 – Topsoil Stockpile Design Criteria

Criteria	Value	Unit	Reference/Comment
Topsoil thickness in situ	0.3	m	From previous project experience and initial PFS GI Results
Max stockpile height	10	m	To reduce degradation of topsoil. Using limits in Bulgarian standards as guidance.
Slope	1 : 3	V : H	From previous project experience and results of stability analysis
Density	1.6	t/m³	From previous project experience and discussion with DPM

18.7.3.2 *Optimisation*

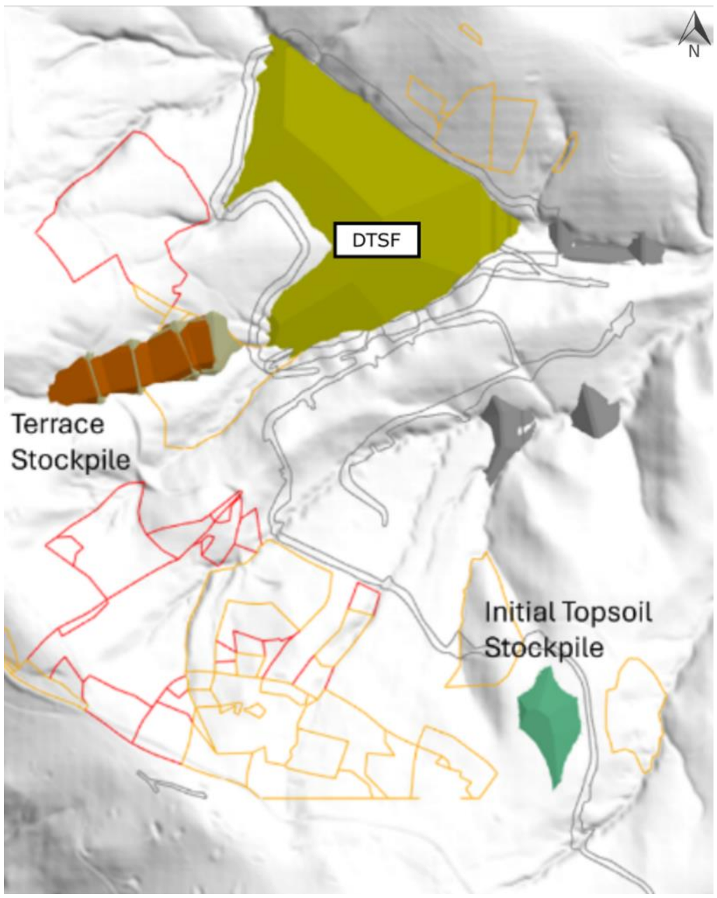
Table 18.7 presents a summary of the finalised PFS topsoil stockpile designs.

Table 18.7 – Čoka Rakita Topsoil Stockpile Quantities

Stockpile ID	Terrace Fill Volume (m³)	Topsoil Stockpile Volume (m³)
Terrace Stockpile	330,000	250,000
Initial Topsoil Stockpile	24,587	103,610

The locations of these stockpiles are presented in Figure 18.10.

Figure 18.10 – Optimised Topsoil Stockpile Locations



Source: WSP, 2024

18.7.3.3 Stability Analysis

The Serbian Standard SRPS U.C5.020, 1980) was not applicable to assess the stability of topsoil stockpiles. The stockpiles were assessed using CDA guidelines (CDA, 2014), with the Morgenstern-Price method analysis applied.

Stability analyses have been conducted to assess localised slip surfaces occurring in the topsoil stockpiles. Analyses have been performed using Entry and Exit methods.

The findings of the stability assessment are presented in Table 18.8.

Table 18.8 – Topsoil Stockpiles Slope Stability Summary

Description		Scenario	Minimum Required FoS	Calculated FoS (MorganStern-Price)
Section 1	Terrace Stockpile	Drained Static	1.5	2.2
Section 2	Initial Topsoil Stockpile	Drained Static	1.5	1.8

It is recommended that both sections be further assessed against other scenarios including undrained and seismic stability assessments in the next phase of the Project.

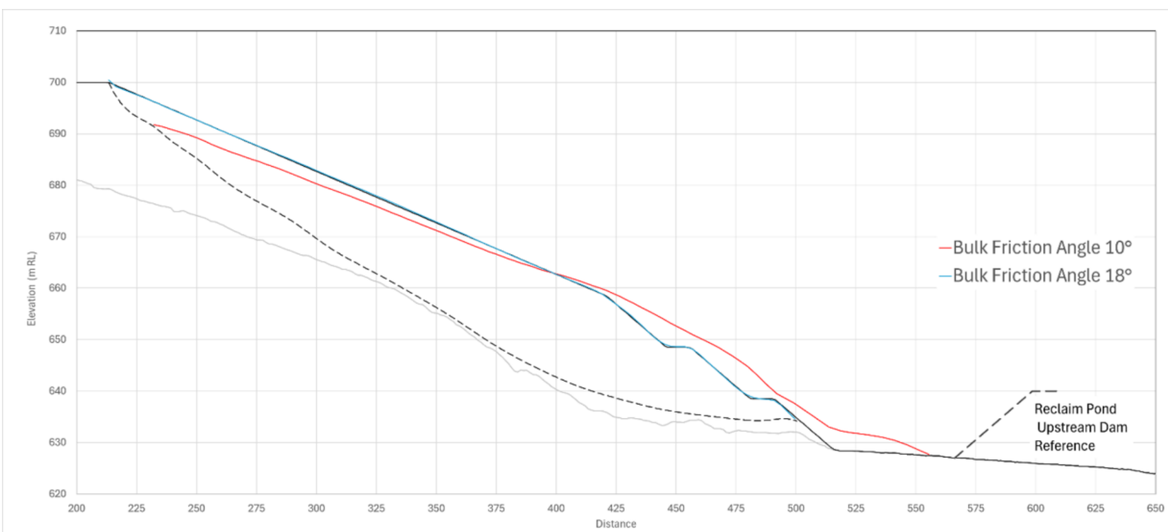
18.8 Dry Tailings Storage Facility

18.8.1 Dam Break Analysis Summary

Results show the runout distance will not overtop the reclaim pond proposed downstream of the DTSF. The maximum runout distance for Scenario 1 was estimated to be 39.5 m, with a maximum material height deposition downstream of the facility of 3.5 m. For Scenario 2, the maximum runout distance is 48.8 m with a negligible depth. Therefore, it is unlikely that the tailings runout will inundate any of the locations of interest downstream the DTSF.

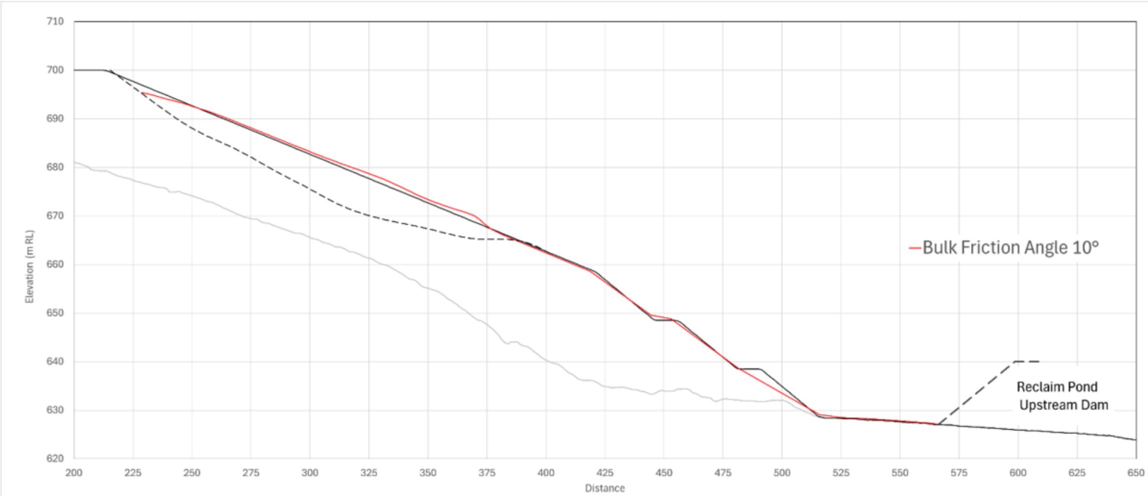
The runout profiles estimated from the 2D analysis are shown in Figures 18.11 and 18.12.

Figure 18.11 – Runout Profiles – Scenario 1 – Frictional Rheology



Source: WSP, 2024

Figure 18.12 – Runout Profiles – Scenario 2 – Frictional Rheology



Source: WSP, 2024

Runout profiles for sensitivity scenarios show a distance about 70 m, this just considers the natural ground without the upstream dam of the Reclaim Pond. By adding the upstream dam profile, it is determined that the sliding mass will not overtop the upstream dam, therefore a cascade failure is not plausible.

18.8.2 Dam Classification

Following the consequence category classification using GISTM and CDA, the Čoka Rakita DTSF has been determined as "Significant". The standards and guidelines are presented in Table 18.9.

Table 18.9 – Dam Classification

Description	Canadian Dam Association, (CDA, 2013)	Global Tailings Standards (GISTM), (GISTM, 2020)	Comments
PAR	Low	Low	PAR of zero was estimated for worst scenario
PLL	Significant	Significant	Unspecified. Conservatively it is assumed that Mine personnel cannot be ruled out from the inundation area
Environment	Low	Low	The total outflow is contained within the project site and there is minimal short-term loss and no long-term loss.
Health, Social and Cultural	-	Low	Minimal impacts on the surrounding community and no measurable effect on human health.
Infrastructure	Low	Low	No external infrastructure or transportation routes impacted.
Infrastructure	Low	Low	No external infrastructure or transportation routes were impacted.

Description	Canadian Dam Association, (CDA, 2013)	Global Tailings Standards (GISTM), (GISTM, 2020)	Comments
Overall DTSF Consequence Classification	Significant	Significant	Overall consequence category is based on the highest consequence rating given during the assessment
Design Seismic Event	Between 1/100 and 1/1,000	1/1,000-year event	
Flood Event (IDF)	Between 1/100 and 1/1,000 Note	1/1,000-year event	
Peak Ground Acceleration (Active)	0.262		Seismic Assessment (2021) PGA determined from Design classification and WSP evaluation (WSP 2024b), (WSP, 2024)
Peak Ground Acceleration (Closure)	0.7	0.7	Seismic Assessment (2021), GISTM, PGA determined from Design classification and WSP evaluation. (WSP 2024b), (WSP, 2024)

Note: Selected on basis of incremental flood analysis, exposure, and consequences of failure

18.8.3 Design Criteria

The design of the DTSF during the PFS has been initially assessed against the tailings throughput outlined within the PEA; these figures have since been refined by WSP over the course of the PFS. The newly establish throughput values are presented in Table 18.10 and will be implemented during the FS to refine the design.

Table 18.10 summarises the key design parameters for the design of DTSF.

Table 18.10 – Čoka Rakita Mine Throughput

Description	Quantity	Units	Remarks/Assumptions
Life of Mine	10	Years	PFS Data
Annual Throughput	850,000	tpa (Average)	RFI 2024 Full annual tailings throughput presented in Basis of Design.
Total Tailings	7.14	Mt	PFS Data (Received from DPM)
Tailings (Surface Disposal) Design Value	5.0	Mt	Value adopted from PEA mine plan, values refined through PFS and updated values to be adopted at the FS.
Tailings (surface Disposal) PFS Mine Plan Value	3.6	Mt	Approximately 50 % of total tailings to be stored at surface as per Mine plan from 1st October 2024 (DPM Email)

Description	Quantity	Units	Remarks/Assumptions
Tailings (Paste Backfill)	3.54	Mt	Approximately 50 % of total tailings to be stored at as paste backfill. Paste back fill to be created from filter cake and cement to infill underground primary mining stopes. As per Mine plan from 1st October 2024
Total Mine Waste Rock	0.53	Mm³	As per Mine plan from 1st October 2024
Surface Mine Rock Storage	0.225	Mm³	As per Mine plan from 1st October 2024. Remaining mine waste rock to be stored underground.

18.8.4 Tailings Characterisation

A summary of the geotechnical parameters derived from this suite of laboratory testing on the filtered tailings sample is presented in Table 18.11.

Table 18.11 – Summary of Geotechnical Parameters

Type	Parameter
Classification	Low plasticity SILT (ML)
Compaction	MDD = 1 963 kg/m³ OMC = 12.6 %
Consolidation	Min-Max c_v (over 21 – 2 470 kPa stress range) = 1.13E^{-01} – 3.07E^{-02} cm²/yr
Shear Strength	Effective Friction Angle = 36° Effective Cohesion = 0 kPa
Permeability	Average Permeability from Constant Head Test = 8.69E^{-08} m/s Min-Max k (over 21 – 2 470 kPa stress range) = 5.91E^{-09} – 2.48E^{-10} m/s

Laboratory testing was used to derive geotechnical parameters for the tailings mass, these are presented in Table 18.12.

Table 18.12 – Geotechnical Design Parameters - Tailings

Material	Unit Weight	Effective Friction Angle	Effective Cohesion	Undrained Strength Ratio	Reference
	(kN/m³)	(°)	(kPa)	Peak	
Tailings (Fine – Silt)	19.6	36	0	0.6	Geotechnical Tailings Test Work Review, (WSP, 2024)

Further geotechnical testing has been proposed on future tailings samples for use in the FS design. The next batch of tailing samples is anticipated to be produced at the end of 2024.

18.8.5 Ground Characterisation

Without site-specific testing, the PFS ground model and parameters were based on the Timok Report (2020). These geotechnical and hydraulic parameters are presented in Tables 18.13 and 18.14.

Table 18.13 – Čoka Rakita Ground Characterisation Model

Material	Unit Weight	Effective Friction Angle	Effective Cohesion	Undrained Strength Ratio	Reference
	(kN/m³)	(°)	(kPa)	Peak	
Weathered Bedrock (Volcaniclastic Rocks)	26	32	-	-	Values based on interpretation from the Final Interpretative Report on Geotechnical Investigations for Timok Gold Project for Feasibility Study, (University of Belgrade, 2022)
Weathered Bedrock (Dioritic Porphyry Intrusive Rocks)	26	35	-	-	
Weathered Bedrock (Monzonitic Intrusive Rocks)	26	35	-	-	

Table 18.14 – Timok Hydraulic Conductivity Lithological Units

Lithological Unit	Hydraulic Conductivity		
	Average (m/s)	Minimum (m/s)	Maximum (m/s)
Volcanics and Volcaniclastics	1.8E-06	2.9E-05	2.4E-08
Diorite	5.0E-07	2.2E-06	4.0E-08

18.8.6 Proposed Geotechnical Investigation

The investigative program includes boreholes, test pits and geophysics.

- Intrusive Investigation: Conducted by Technofor, with geotechnical logging, sampling and lab testing schedule by the University of Belgrade. Geomehanika laboratory performs all laboratory testing.
- Non-Intrusive Survey: Completed by the University of Belgrade, with an interpretive report pending.

18.8.7 Dry Tailings Storage Facility Design

18.8.7.1 Tailings Conveyance Method Summary

A semi-quantitative SWOT assessment ranked each tailings technology with weighted sub-category from 1 to 5. "Filtered Tailings" was identified as the most advantage due to easier permitting and higher technical scores.

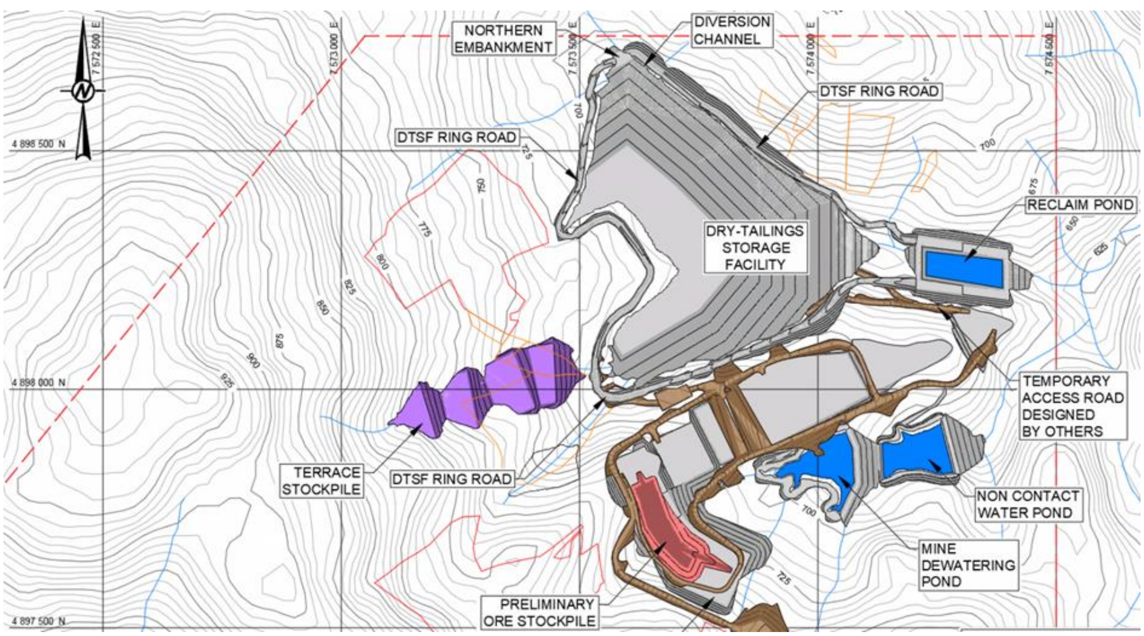
18.8.7.2 Optimisation Summary

The optimised DTSF storage capacity and overall design basis as per the PEA are summarised in Table 18.15. The DTSF plan is presented in Figure 18.13.

Table 18.15 – Čoka Rakita DTSF Optimisation Summary

Locations	Units	Site 1 Volume
Tailings Storage Volume Target	m^3	3,000,000
Total DTSF Height	m	73.5
Tailings Storage Volume Capacity	m^3	3,054,800
Filter Tailings Footprint	m^2	235,015

Figure 18.13 – DTSF Location Plan



Source: WSP, 2024

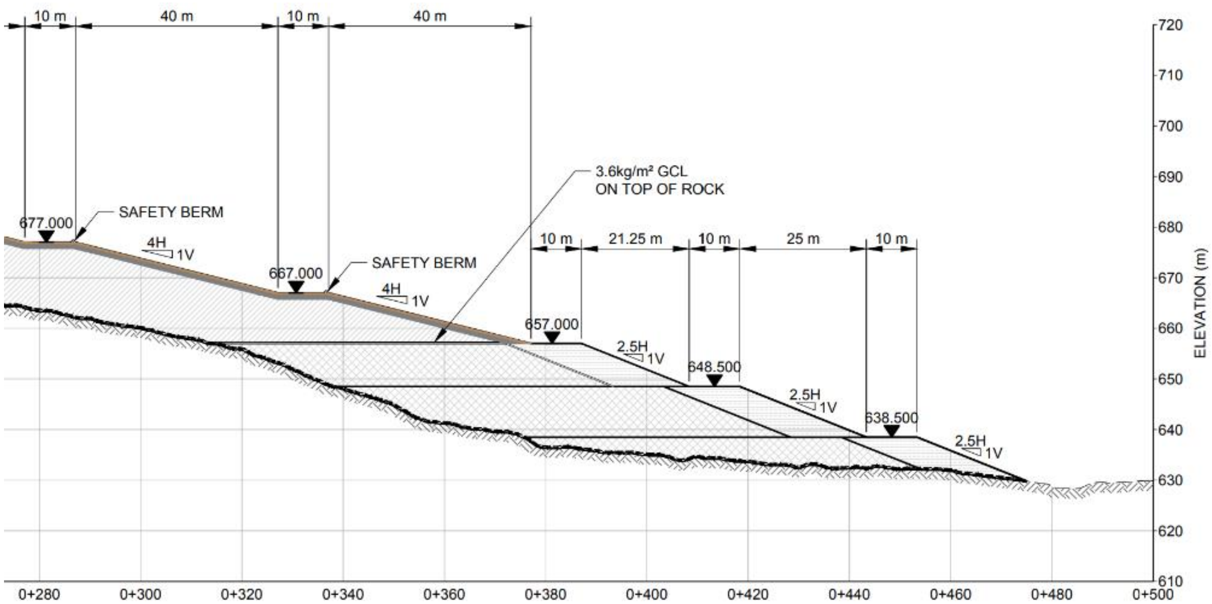
18.8.7.3 Potentially-Acid Generating Zone Design

During the initial years of mining, excavation will produce Mine Waste Rock, considered Potential Acid Generating (PAG) for this PFS. Since backfilling underground stopes isn't feasibly initially, this material will be stored at surface.

The PAG rock will be contained in three embankments constructed of "Clean" rock. These embankments will have a composite liner top prevent seepage. The liner will be installed between PAG and Non-Acid Generating (NAG) rock fills with PAG rock placed first, followed by NAG rock forming the outer embankment. PAG rock will be stored at the DTSF base, with a lining system across the valley base for containment. After placing all PAG rock within the first two years, filtered tailings will be placed above it.

Figure 18.14 presents the cross section PFS design of the DTSF PAG Foundation.

Figure 18.14 – DTSF PAG Foundation Design



Source: WSP, 2024

If geochemical testing confirmed the mine rock is PAG, it could cause acid rock drainage (ARD) when exposed to air. To mitigate this, WSP has developed an integrated storage approach from PAG Mine Waste rock and tailings within the DTSF. The goal is to keep the PAG rock saturated and oxygen-free to prevent ARD.

Two (2) design approaches are considered to progress:

1. Alluvial Soil Layer: placed between the PAG rock and filtered tailings to direct most seepage way while allowing some to maintain water levels in the PAG rock. Watet levels will be monitored to ensure saturation
2. GCL or HDPE layer: Minimised seepage from filtered tailings into the PAG zone. Water level will be monitored to ensure saturation and check for ARD conditions if containment is lost.

A seepage assessment will be conducted at the FS level to determine if constant saturation is feasible. The PFS design includes a GCL capping layer in top of the PAG rock to prevent seepage. This design will be reconsidered based on FS seepage analysis results.

Table 18.16 presents the required and designed PAG storage capacity, comparing the initial PEA and the revised LOM plans.

Table 18.16 – PAG Zone Design

Description	Quantity		Units	Remarks / Assumptions
Required Surface Mine Rock Storage	0.225		Mm^3	As per Mine plan from 1st October 2024 First two years, all waste rock will be stored on surface. Following which, all waste rock will be stored with Paste backfill.
Annual Surface Mine Rock Disposal	Year 1	120,184	m^3	Reduction in anticipated PAG material to be delivered to the DTSF footprint from updated mine plan, contingency for additional material to be stored in the PAG zone footprint. As per Mine plan from 1st October 2024
	Year 2	105,695		
Required Surface Mine Rock Storage (Design Values)	0.345		Mm^3	Anticipated PAG Volume as per PEA
	Year 1	153,000 + 40,000	m^3	
	Year 2	120,900		
	Year 3	63,165		
	Year 4	4,800		
	Year 5	0		
PAG Foundation Design				
Bench Height	10		m	
Bench Width	15		m	
Downstream Slope:	1:2.5		V:H	Defined in PFS design, considering stability, closure requirements, construction practicalities, sustainability of vegetation etc.

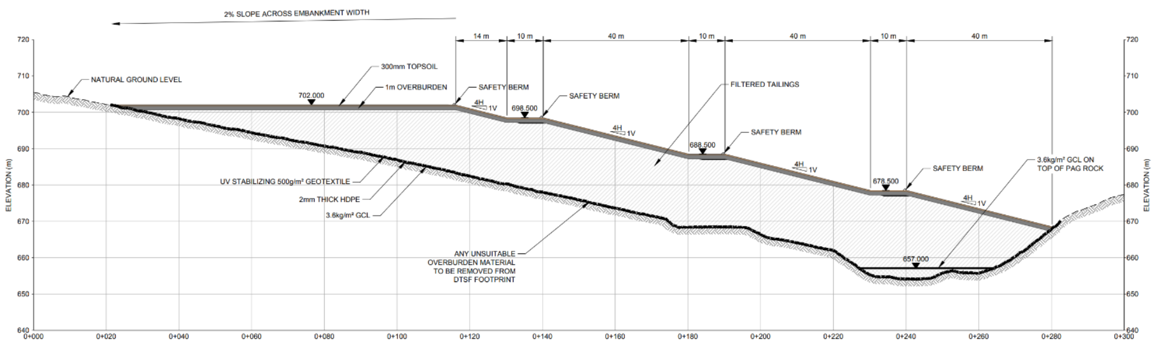
18.8.7.4 *Filtered Tailings Design*

Filtered tailings will be placed across the top of the PAG foundation to create four (4) benches, as detailed in Section 18.8.7.13. Table 18.17 provides information on the overall filtered tailings design and annual surface stacking rate.

Since this design, DPM has updated the LOM plan, reducing the total tailings storage required. It is recommended to reassess the design using the update mine plan values from October 1, 2024.

A cross section of the PFS filtered tailings design is presented in Figure 18.15.

Figure 18.15 – Filtered Tailings Design - Cross Section



Source: WSP, 2024

Table 18.17 – Filtered Tailings Design

Criteria	Input	Units	Notes and References
Required DTSF Capacity	3,000,000	m³	Designed to PEA Design tonnages
	4,950,000	t	
DTSF Construction			
Operating Hours Per Day (24 Hours)	8	h	DPM
Daily Rate of Tailings Placement	820	m³ per day	PFS Design
Crest Elevation	702	mRL	PFS Design
Total Height	72.5	m	PFS Design
Tailings Height	42.5	m	PFS Design
Rockfill Height	30	m	PFS Design
Bench Height	10	m	PFS Design
Bench Width	10	m	PFS Design
Bench Slope Length	40	m	PFS Design

Criteria	Input	Units	Notes and References
Downstream Slope: Overall Inter slope	 1:5 1:4	 V:H V:H	Defined in PFS Design, considering stability, closure requirements, construction practicalities, sustainability of vegetation etc.
Year		**Tailings for Surface Disposal (PFS Design Values)** (t)	
Year -2		-	PEA throughput values used for PFS Design
Year -1		-	
Year 1		490,645	
Year 2		574,069	
Year 3		574,539	
Year 4		581,761	
Year 5		574,538	
Year 6		572,482	
Year 7		573,848	
Year 8		573,961	
Year 9		321,923	
Year 10		162,336	
Total		**5,000,100**	

18.8.7.5 Liner Design

The liner system identified from this study is summarised in Table 18.18.

Table 18.18 – Čoka Rakita DTSF Liner Selection

Description	Quantity	Surface Preparation
Entire DTSF Footprint	Composite – GCL and 2 mm HDPE GCL (Bentonite weight 4 kg/m²) Further overlain by 1000 g/m² carbon rich protector geotextile	Unsuitable material will be removed to required elevation and gradient, material beneath the foundation level is to be compacted with roller, with GCL and HDPE and carbon rich protector geotextile placed above. Foundation beneath the embankment will have more stringent material quality requirements

18.8.7.6 Liner System Characterisation

Table 18.19 presents a summary of the recommended laboratory testing to be conducted.

Table 18.19 – Lining System Testing

Test Type	Standard	Sample Preparation	Effective Vertical Stress Confining Pressure (kPa)
Direct Simple Shear	ASTM D5321-12	GCL and HDPE interface	200
		GCL and HDPE interface	400
		GCL and HDPE interface	800
		GCL and HDPE interface	1,200

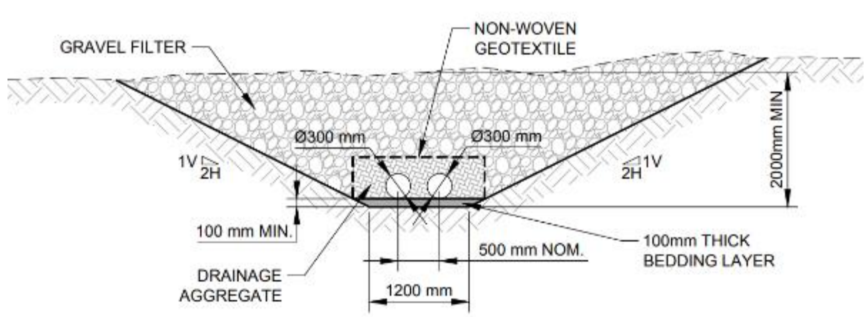
18.8.7.7 Basal Drain

A basal drain will be installed at the base of the northern and southern valleys where the DTSF will be constructed. It will run beneath the DTSF composite liner, channeling natural groundwater flow, and continues under the reclaim pond, discharging into the original stream bed. Access points at the DTSF's downstream toe will allow for water monitoring and sampling to detect any leaks. If contamination is found, water can be diverted to the reclaim pond for treatment or reuse. The estimated drainage material required is 28,354 m^3.

The basal drain design includes filling the valley base with appropriate 25 m^2 per meter of locally sourced rockfill (>50 mm aggregates) with <5% fines. Two perforated HDPE pipes will be placed on the valley floor, surrounded by finger drainage aggregate (<20 mm) encasing the pipes by 200 mm. A non-woven geotextile will wrap around the finer aggregate and pipes, overlapping by at least 300 mm, and then be covered with gravel filter material.

Figure 18.16 presents the cross section of the typical basal drainage design. Figure 18.17 presents a plan view of the basal drainage alignment beneath the DTSF facility.

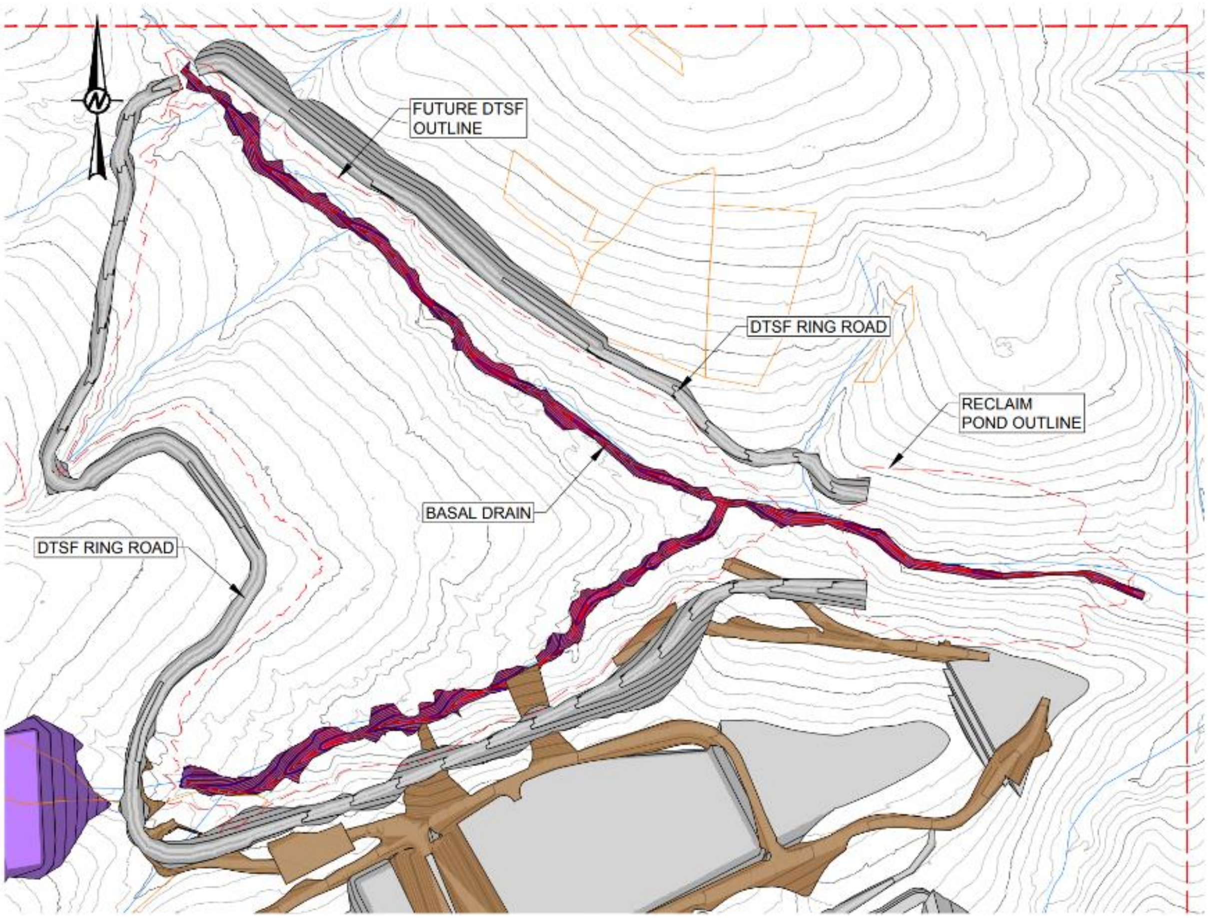
Figure 18.16 – Typical Basal Drainage Cross Section



Source: WSP, 2024

The HDPE pipe dimensions will be determined during the FS after completing the water balance study to establish the minimum size needed for expected flow. A filter compatibility assessment is recommended at FS to determine the gravel PSF requirements and if a finer sand filter layer is necessary. Further hydrogeological studies will also assess the required HDPE pipe dimensions and perforations.

Figure 18.17 – DTSF Basal Drainage – Plan View



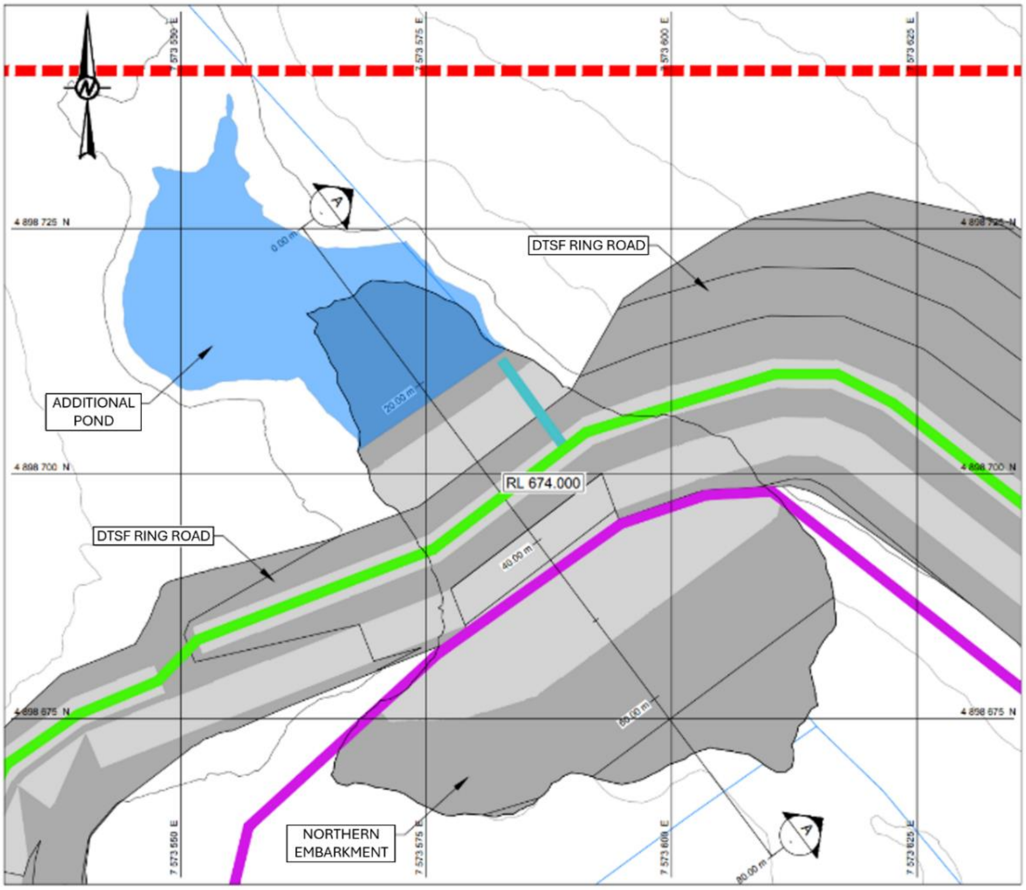
Source: WSP, 2024

18.8.7.8 North Embankment

The Northern Embankment will be built at the north toe of the Durmitru Valley to connect access roads and allow the DTSF footprint to extend higher on the northern side. It will contain locally sourced overburden material and be lined on both sides to prevent seepage. The catchment area where water collects will also be lined. The upstream face will collect water, causing ponding upstream of the facility. This water will drain by gravity through the spillway into the non-contact channel before reaching the project boundary.

A plan design of the northern embankment is presented in Figure 18.18.

Figure 18.18 – Northern Embankment Design



Source: WSP, 2024

Design details of the North Embankment are presented in Table 18.20.

Table 18.20 – North Embankment Design Summary

Description	Value
Fill Volume (m^3)	24,755
Dam Height (downstream) (m)	8.5
Slope Gradient (V:H)	1V:2.5H
Crest Width (m)	8
Crest Height (mRL)	674
Freeboard (m)	1.5
Pond Height (mRL)	672.5
Pond Capacity (m^3)	1975

18.8.7.9 Temporary Lay-Down Area

A multi-purpose storage area with the southern valley DTSF footprint will store tailings during wet weather and serve as parking for DTSF surface mobile machinery.

The design details are as follows:

- Storage Capacity: 5 days, 1.4 kg/m³ density, approximately 9,000 m³.
- Design: Covered to prevent rain wetting, using a stacker to create an 8 - 10 m high conical stockpile with a 1,200 m³ footprint.
- Parking Area: Adjacent to storage, 900 m² for surface mobile machinery
- Total Area: 2,200 m².

Figure 18.19 presents a plan view and location of the temporary lay-down area.

Table 18.21 – DTSF Construction Equipment List

Item	Size/Capacity/ Description	Quantity
Wheeled Front Loader	CAT 980	1
Articulated Dump Truck	CAT 745	3
Tracked Dozer	CAT D6 LPG Dozer	2
Tracked Excavator	CAT 323	2
Soil Compactor	CAT 20t Smooth Roller	1

Figure 18.19 – Temporary Lay-Down Area - Plan View

Source: WSP, 2024

18.8.7.10 DTSF Stability Analysis

Material parameters were derived from the Timok Report (University of Belgrade, 2020) due to a lack of site-specific testing. Filtered tailings parameters were based on geotechnical testing of the "BL-1291 Final tails" sample. Table 18.22 presents the assigned material parameters.

Table 18.22 – Stability Analysis Material Parameters

Material	**Unit Weight** (kN/m³)	**Effective Friction Angle** (°)	**Effective Cohesion**	**Undrained Strength Ratio**	**Reference**
			(kPa)	Peak	
Filtered Tailings	19.6	31	0	0.6	Geotechnical Tailings Test Work Review (WSP, 2024)
Weathered Volcanics and Volcaniclastics (Bedrock)	26	32	-	-	Final Interpretative Report on Geotechnical Investigations for Timok Gold Project for Feasibility Study (University of Belgrade, 2020)
Weathered Intrusive Dioritic (Bedrock)	26	35	-	-	Final Interpretative Report on Geotechnical Investigations for Timok Gold Project for Feasibility Study (University of Belgrade, 2020)
Geosynthetic Liner	20	25	-	-	Assuming a needle punched GCL. Previous project experience, to be confirmed through testing of geosynthetics at FS.
Mine Waste Rock (PAG)	20	40	-	-	WSP Engineering Experience
Clean Rock (Embankment)	20	35	-	-	
Bedrock	Impenetrable				

Two (2) cross sections were developed to assess the stability of the DTSF design. Each cross section was assessed against CDA guidelines (CDA, 2014) and Serbian Standards (SRPS U.C5.020, 1980). The findings of this stability assessment indicate that both cross sections exceed minimum required Factor of Safety (FoS) values. The achieved FoS for each scenario are presented in Table 18.23.

Table 18.23 – Slope Stability Summary

Description	Scenario	Minimum Required FoS (CDA, 2014)	Serbian Standard (SRPS U.C5.020, 1980)	Calculated FoS (MorganStern -Price)	Calculated FoS (Fellenius/ Ordinary)
Section 1	Drained Static	1.5	1.5	1.9	2.0
	Undrained Static	1.3	1.3	1.8	1.8
	Undrained Seismic	1.0	1.15	1.4	1.3
Section 2	Drained Static	1.5	1.5	3.0	2.9
	Undrained Static	1.3	1.3	3.1	3.1
	Undrained Seismic	1.0	1.15	1.5	1.5

A sensitivity analysis was conducted to evaluate the impact of porewater pressure seismic or construction loading on tailings material. Results in Table 18.24 indicate that stability of the DTSF is compromised when porewater pressure exceed 25% of the full tailings tack load.

Table 18.24 – Peak Undrained Seismic Sensitivity Assessment

Scenario	Minimum Required FoS	Peak Ground Acceleration (PGA)	Peak Undrained Strength Ratio	Bbar	Calculated FoS (Morganstern-Price)	Calculated FoS (Fellenius)
Peak Undrained - Seismic	1.15	0.262	0.6	0.0	1.5	1.5
				0.1	1.4	1.4
				0.2	1.2	1.2
				0.3	1.0	1.0
				0.4	0.9	0.9
				0.5	0.6	0.7

18.8.7.11 *Trade-Off between Truck and Conveyor*

A trade-off study (ToS) between trucks and conveyors for tailings transportation was conducted. The associated advantages and disadvantages are highlighted below;

Trucks

- Advantage
  - High flexibility in routes and deposition locations
  - Lower upfront investment
  - Easier to scale operations by adding more trucks
  - Potential use of trucks from existing DPM sites

  - Single method of transportation from the process terrace to deposition location
- Disadvantages
  - High fuel, maintenance and labour cost
  - Higher greenhouse gas emissions
  - Increased traffic and potential for accidents

Conveyors

- Advantages
  - Lower energy consumption and maintenance costs
  - Few emissions and smaller environmental footprint
  - Continuous and reliable material transport
- Disadvantages
  - Significant upfront investment
  - Less adaptable to route changes
  - Complex maintenance requiring specialised knowledge

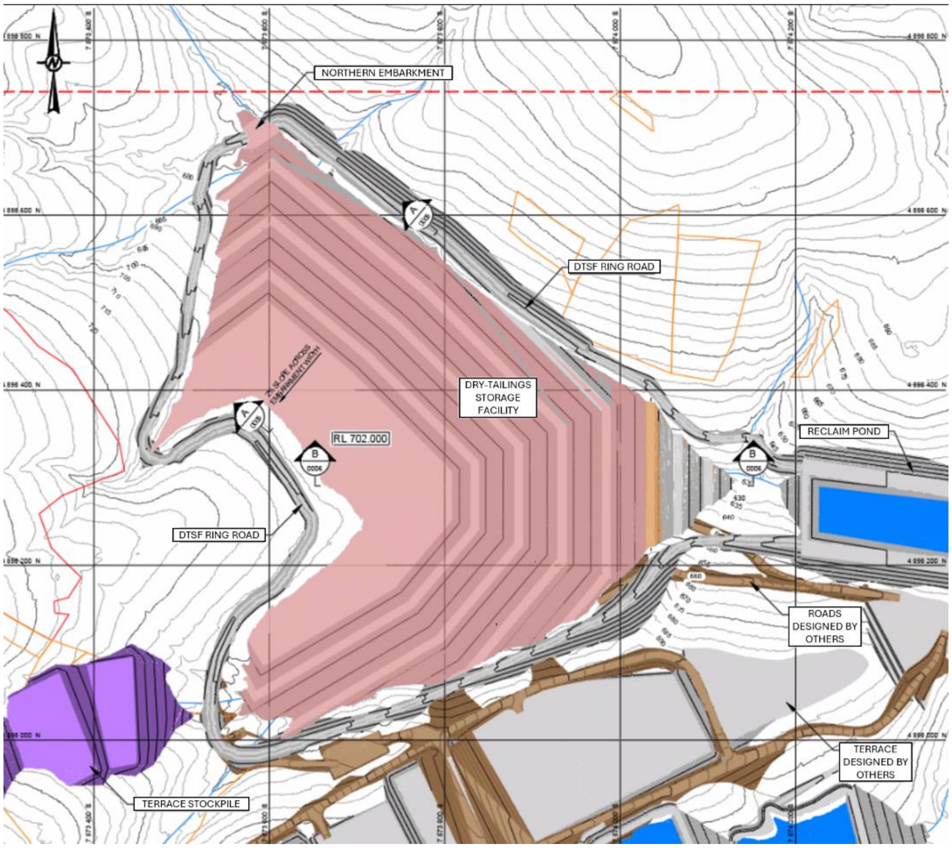
Considering the design basis, deposition complexity, and material variability, trucks were identified as the most practical and feasible method, supported by the available of mobile machinery at DPM's Ada Tepa Gold Mine.

18.8.7.12 *DTSF Ring Road*

The DTSF perimeter road spans 2,490 m (~2.5 km) with gradients ranging from 1% to 11%. Cut slopes are designed at a 1H:1V slope, and soil nails may be needed for additional support due to significant weathering profiles. The PFS ground investigation will provide better insight into the weathering profiles strength. Excavated material from the ring road construction will be repurposed for other earthwork structures.

Diversion drainage channels are included in the design for the operational phase and may need widening for closure phase. These channels are expected to serve as access roads during the passive closure phase. An access road around the process terrace perimeter is also included in the design. Figure 18.20 presents the DTSF ring road design.

Figure 18.20 – Čoka Rakita DTSF "Ring Road" Access



Source: WSP, 2024

18.8.7.13 Stage Storage Deposition

The sizing of the DTSF was competed using the PEA LOM Plan and when compared with the updated PFS quantities, it has additional capacity. Refinement of the DTSF sizing will be completed at the FS.

The stage storage plan for the Čoka Rakita DTSF is presented in Table 18.25. The table presents the PAG and tailings deposition volumes with associated dam height elevation (mRL).

Table 18.25 – Stage Storage Deposition Summary

Year (Operation)	**PAG Material (PEA Design Value)** (t)	**PAG Material (PFS Mine Plan) *** (t)	**Tailings Surface Disposal (PEA Design Value)** (t)	**Tailings Surface Disposal (PFS Mine Plan) *** (t)	**Elevation** (mRL)
Year -2	193,000	120,184	-	-	-
Year -1	120,900	105,695	-	-	-
Year 1	63,165	-	490,645	385,507	-

Year (Operation)	PAG Material (PEA Design Value) (t)	PAG Material (PFS Mine Plan) * (t)	Tailings Surface Disposal (PEA Design Value) (t)	Tailings Surface Disposal (PFS Mine Plan) * (t)	Elevation (mRL)
Year 2	4,800	-	574,069	451,054	657.0
Year 3	-	-	574,539	451,424	-
Year 4	-	-	581,761	457,098	675.5
Year 5	-	-	574,538	451,422	-
Year 6	-	-	572,482	499,807	-
Year 7	-	-	573,848	450,881	683.5
Year 8	-	-	573,961	450,969	-
Year 9	-	-	321,923	252,940	-
Year 10	-	-	162,336	127,549	702.0
Total		**225,879**	**5,000,100**	**3,928,650**	

* As per Mine plan from 1st October 2024

18.8.7.14 Construction Ground Preparation

Construction of the DTSF involves multiple excavation phases, including stripping unsuitable material, clearing vegetation, excavating overburden, compacting and preparing for the lining system. The sequence will vary based on the specific structure and design.

Table 18.26 presents an overview of on the step-by-step process required for constructing the DTSF.

Table 18.26 – Construction Sequencing Summary

Subgrade Category	Occurrence	Treatment
Topsoil	Across the site	To be removed entirely from the footprint including: Remove trees and stumps Remove any other unsuitable material Removal/ infilling of large rocks
Alluvium and Colluvium	Beneath tailings Embankment foundations (Clean Rock Embankment)	To be removed entirely from the footprint including: Removal/ infilling of large rocks Removal of any sharp objects Potential placement of compacted overburden material to provide level surface for construction.
	With the footprint (PAG and Filtered tailings – excluding embankment foundations)	To be excavated and graded to suitable gradient. Material to be compacted with a roller or hydraulic press prior forming compacted overburden prior to lining.

Subgrade Category	Occurrence	Treatment
Composite – GCL and HDPE	Entire DTSF Footprint	Alluvium material will be either completely removed or excavated to required elevation and gradient, any remaining material is to be compacted with roller, with GCL placed above.
	Above PAG Mine Waste Rock	Alluvium material will be either completely removed or excavated to required elevation and gradient, any remaining material is to be compacted with roller, with GCL and placed above.
HDPE Protective Geotextile	Entire DTSF Footprint	Protective geotextile will be placed across the GCL and HDPE liners

18.8.7.15 Construction Sequencing

The Čoka Rakita project construction required specific sequencing to manage water and environmental risks. The key priority structures include:

- Topsoil stockpiles and temporary access roads.
- Early-stage water management ponds from mine dewatering.
- Diversion water channels to reduce runoff into the DTSF basin.
- Basal drainage channel installation to convey upstream water.
- DTSF basal lining before placing any PAG foundation material.
- Progressive raising of contact water channels as DTSF elevation increases.

18.8.7.16 Operations

The Čoka Rakita DTSF will operate 8 hours a day, 336 days a year. Surface mobile equipment details are presented in Table 18.27. Filtered tailings, produced in 30-45 minute cycles via a filter press, will be transported to a storage terrace by a conveyor system operating 24/7. During operational hours, tailings will be loaded into trucks and hauled to the DTSF deposition point. During non-operational hours, tailings will be stockpiles until the next shift.

A wheeled front loaded will continuously load trucks with tailings from the storage terrace. In poor weather, dry tailings will be moved to a covered lay-down area within the DTSF to reduce saturation. Temporary access roads may be needed and will be design at FS, considering truck weight and number. Reduce type pressure is recommend for vehicles on the DTSF.

Trucks will revise into position with a banksman support, and tailings will be end-tipped into the deposition zone. A tracker dozer will spread tailings in 500 mm layer, and a soil compactor will compact them to MDD and OMC requirements. Additional equipment may be needed for moisture conditioning. Excavator will maintain drainage ditches and create temporary channels.

Contact water channels will be raised progressively with tailings deposition, and liner systems will be installed in four stages.

Table 18.27 – DTSF Operations Surface Mobile Equipment

Item	Duty	Size/Capacity/Description
Wheeled Front Loader	Will be continuously used while the filter plant is feeding tailings to the DTSF stockpile (~390 tph)	CAT 980
Articulated Dump Truck	Will be continuously used during the eight-hour DTSF shift when tailings is being hauled to the DTSF stockpile (~390 tph)	CAT 745
Tracked Dozer	Will be continuously used during the eight-hour DTSF shift to spread tailings within the DTSF and for internal roads construction	CAT D6 LPG Dozer
Tracked Excavator	General trenching and construction activities	CAT 323
Soil Compactor	Will be continuously used during the eight-hour DTSF shift to compact tailings layers within the cells.	CAT 20t Smooth Roller

18.8.8 Reclaim Pond

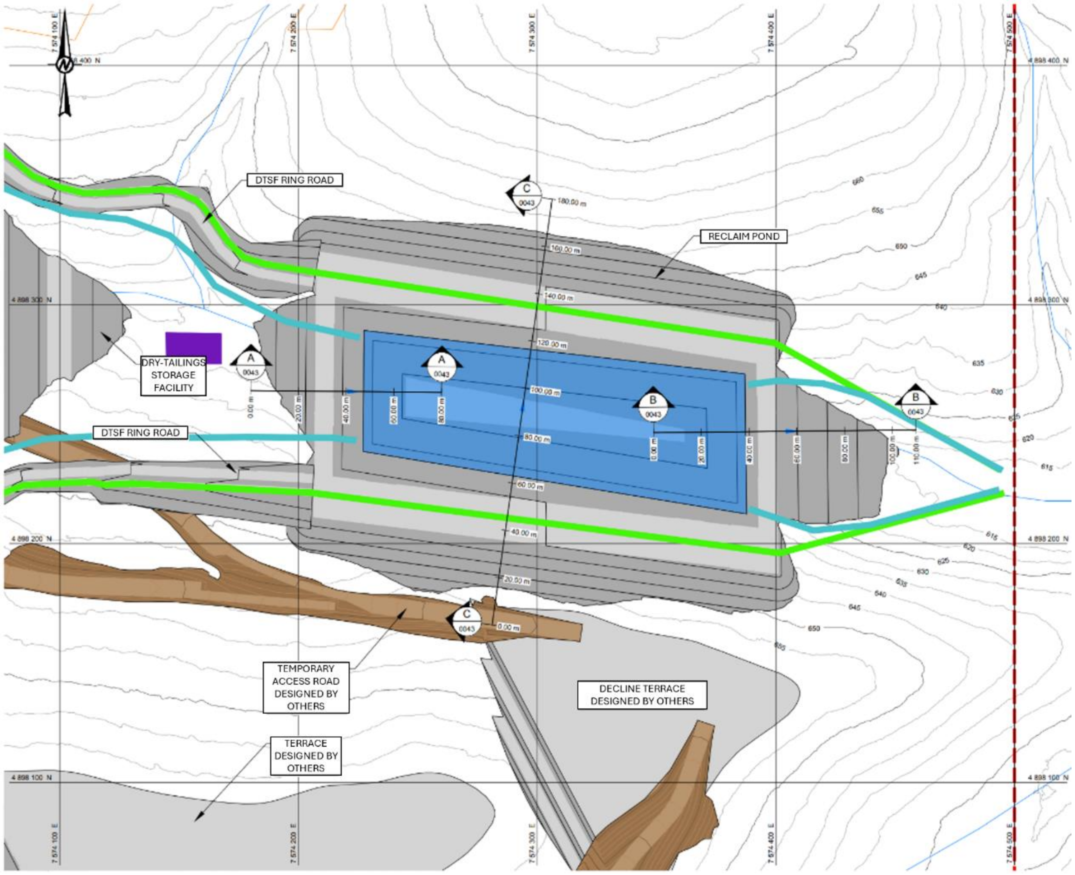
The Reclaim Pond at the DTSF toe will collect precipitation from the DTSF footprint and contact channels. It has a design capacity of 41,000 m³, including 10,000 m³ for operating volume Water will be pumped into the Mine Dewatering Pond to maintain the operating level. Du to contamination from tailings, the pond will have a composite lining system.

The Reclaim Pond is positioned to avoid damage from a potential DTSF dam breach. Valley slopes above the pond will be excavated and reprofiled, with slopes cut at 1V:1H gradients. Additional stabilisation may be needed to due to weathering profiles.

The basal drainage beneath the DTSF basis and foundation will extend under the reclaim pond, allowing groundwater to flow unobstructed downstream. No stability assessment has been conducted yet; it is recommended to assess the design against CDA and Sebian standard in FS Studies.

Figure 18.21 presents the plan view of the Reclaim Pond.

Figure 18.21 – Reclaim Pond – Plan View



Source: WSP. 2024

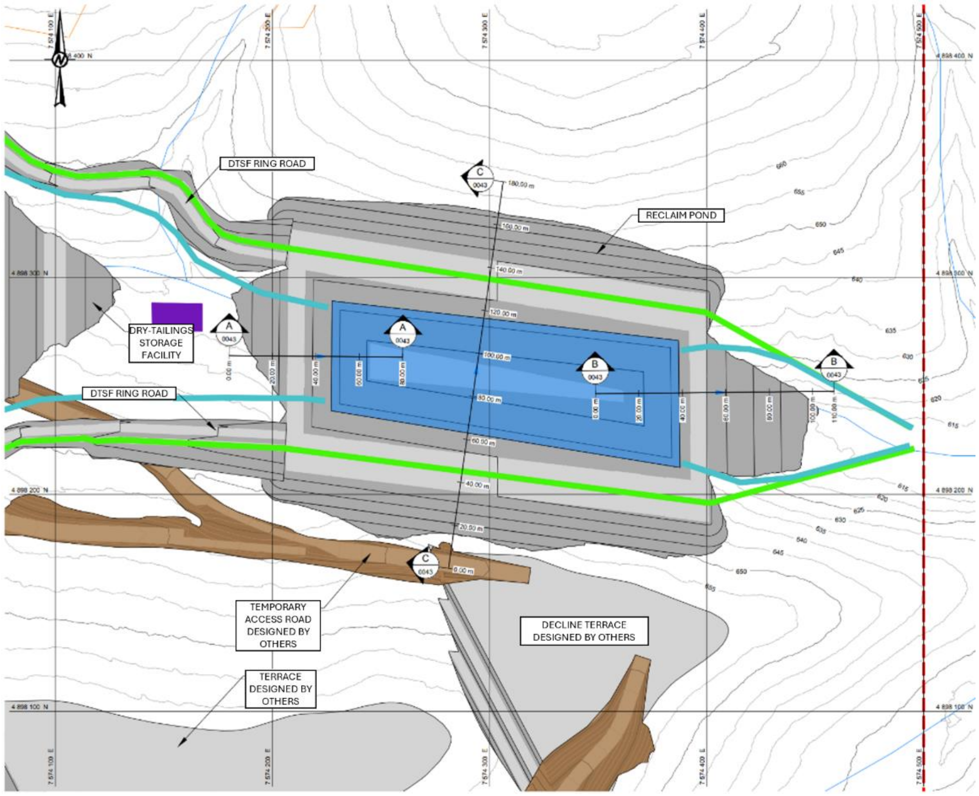
The Reclaim Pond at the DTSF toe will collect precipitation from the DTSF footprint and contact channels. It has a design capacity of 41,000 m³, including 10,000 m³ for operating volume Water will be pumped into the Mine Dewatering Pond to maintain the operating level. Du to contamination from tailings, the pond will have a composite lining system.

The Reclaim Pond is positioned to avoid damage from a potential DTSF dam breach. Valley slopes above the pond will be excavated and reprofiled, with slopes cut at 1V:1H gradients. Additional stabilisation may be needed to due to weathering profiles.

The basal drainage beneath the DTSF basis and foundation will extend under the reclaim pond, allowing groundwater to flow unobstructed downstream. No stability assessment has been conducted yet; it is recommended to assess the design against CDA and Sebian standard in FS Studies.

Figure 18.21 presents the plan view of the Reclaim Pond.

Figure 18.22 – Reclaim Pond – Plan View



Source: WSP. 2024

18.8.8.1 Reclaim Pond Design

The Reclaim Ponds design summary details are presented in Table 18.28.

Table 18.28 – Reclaim Pond Design Summary

Description	East Dam Value	West Dam Value
Fill Volume (m³)	24,755	15,752
Dam Height (downstream) (m)	20.5	10.5
Dam Height (upstream) (m)	14.75	13.5
Slope Gradient (V:H)	1V:2.5H	1V:2.5H
Crest Elevation (mRL)	633.5	636.5
Pond Maximum Level (mRL)	631.5	
Crest Width (m)	8	8
Minimum Freeboard Requirement (m)	1.5	2
Freeboard (m)	2	
Pond Capacity (m³)	41,000 (1)	

Note:
(1) The latest changes in the water management strategy would require a pond volume of 42,000 m³. The increase in dam elevation can be considered negligible, of approximately 10 cm, and will be addressed at FS stage.

18.8.8.2 Reclaim Pond Liner Design

The lining system will be anchored into the access road corridor either side of the channel. An anchor trench will be excavated, with the lining system placed into the trench and subsequently backfilled with aggregate, minimising any movement from drag forces down the embankment.

Table 18.29 presents a summary of the liner system selected for the DTSF Reclaim Pond.

Table 18.29 – Reclaim Pond Liner System

Feature	Lining System
Reclaim Pond	Composite – GCL and HDPE CGL Bentonite weight (3.6 kg/m² sodium bentonite)

18.8.9 MINE DEWATERING POND

The Mine Dewatering Pond, with a capacity of 48,500 m³, received water from the Reclaim Pond, upstream catchment runoff, and the ROM Pad and Terrace area runoff. It includes surge capacity for five days of maintenance, during which water from the underground mine will be pumped into the pond.

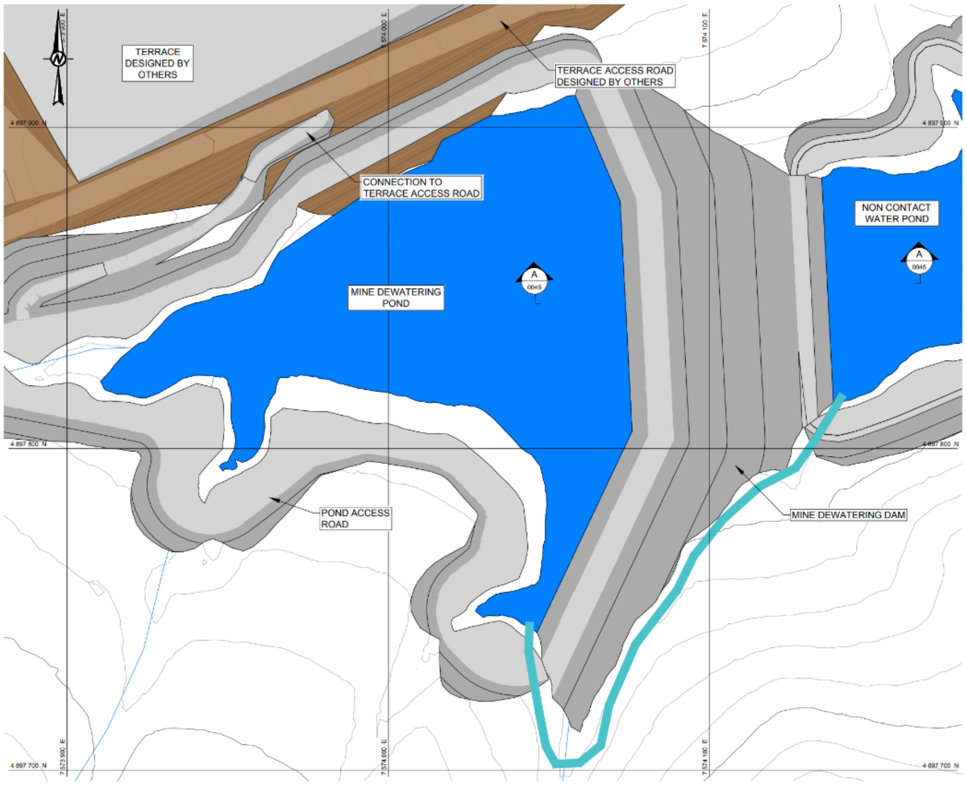
An access road will be built around the pond for monitoring and inspection, connecting to the main haul road south of the process plant terrace. Pumping and decant systems will be designed in alter stages.

No internal drainage is included in the PFS to handle potential seepage from lining defects or groundwater. If a leak occurs, a drainage system will be needed to manage seepage and mitigate stability risks, to be developed at FS stage.

No stability assessment has been conducted; it is recommended to assess the design against CDA and Serbian Standard in FS studies.

Figure 18.23 presents the plan view of the Mine Dewatering Pond.

Figure 18.23 – Mine Dewatering Pond – Plan View



Source: WSP, 2024

18.8.9.1 *Mine Dewatering Pond Design*

The Mine Dewatering Pond design summary details are presented in Table 18.30.

Table 18.30 – Mine Dewatering Pond Design Summary

Description	Value
Fill Volume (m^3)	90,496
Slope Gradient	1V:2.5H
Crest Elevation (mRL)	682.0
Crest Width (m)	8
Freeboard (m)	2
Pond Capacity (m^3)	53,000 (1)
Pond Maximum Level (mRL)	680

Note:
(1) The latest changes in the water management strategy would require a pond volume of 48,500 m^3. The required volume is achieved with a maximum pond level of 679.8 mRL. The reduction in dam elevation can be considered negligible, of approximately 20 cm, and will be addressed at FS stage.

18.8.9.2 *Mine Dewatering Pond Liner Design*

The Mine Dewatering Pond will receive contact water from the underground working and tailings leaching, requiring a composite lining system similar to the DTF and Reclaim Pond.

Table 18.31 presents a summary of the liner system selected for the Mine Dewatering Pond.

Table 18.31 – Mine Dewatering Pond Liner System

Feature	Lining System
Mine Dewatering Pond	Composite – GCL and HDPE CGL Bentonite weight (3.6 kg/m^2 sodium bentonite)

18.8.10 NON-CONTACT WATER POND

The Non-contact Water Pond captures the discharge from the WTP and the non-contact runoff from the southeast catchment of the ROM Terraces, including diverted runoff from the Mine Dewatering Pond Diversion Channel. With a capacity of 63,000 m^3, it can supply water for the process plant for about two months and serve as a domestic water source.

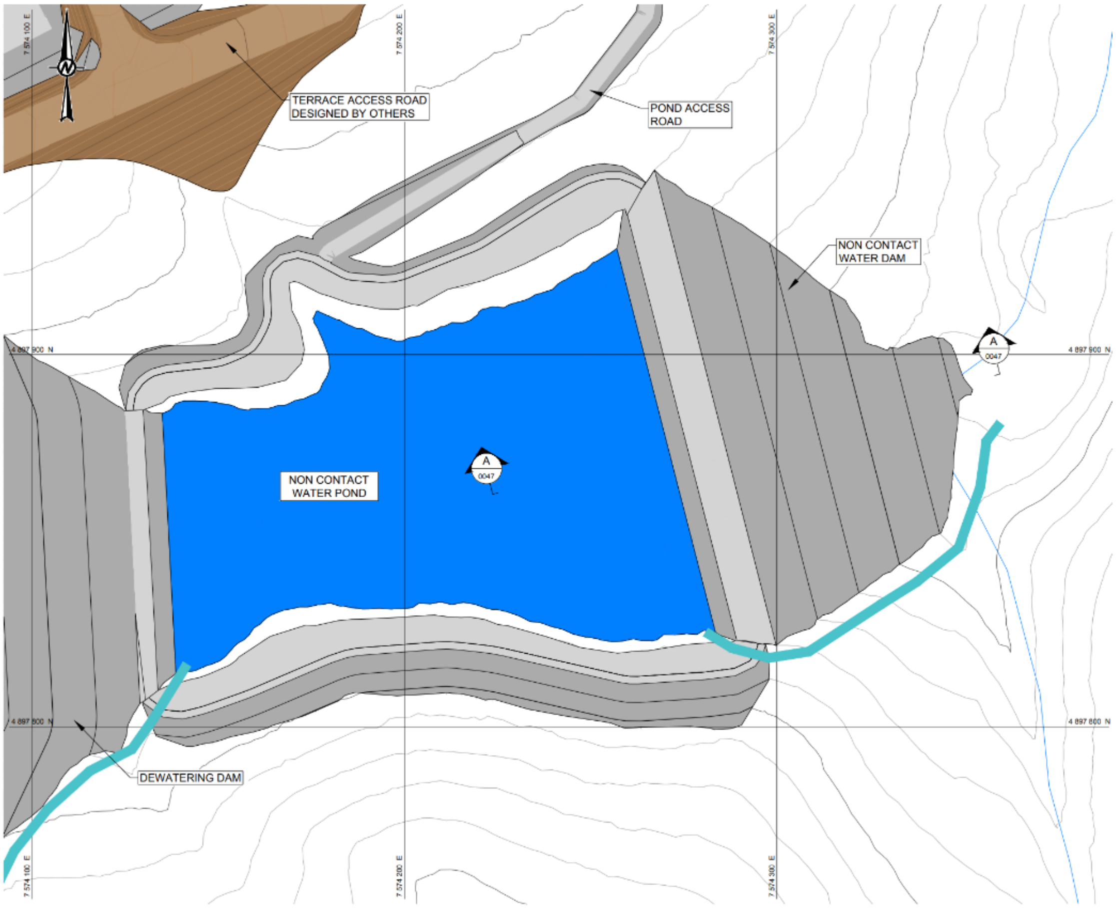
An access road will be built around the pond for monitoring and inspection, connecting to the main haul road south of the process plant terrace. Pumping and decant systems will be designed in alter stages.

No internal drainage is included in the PFS to handle potential seepage from lining defects or groundwater. If a leak occurs, a drainage system will be needed to manage seepage and mitigate stability risks, to be developed at FS stage.

No stability assessment has been conducted; it is recommended to assess the design against CDA and Serbian Standard in FS studies.

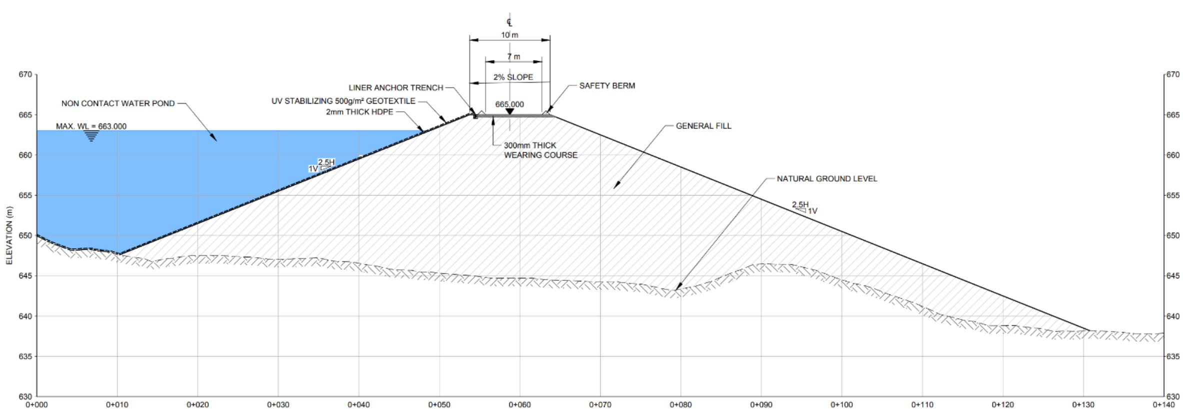
Figure 18.24 presents the plan view of the Non-Contact Water Pond while Figure 18.25 presents a cross-section.

Figure 18.24 – Non-Contact Water – Plan View



Source: WSP, 2024

Figure 18.25 – Non-Contact Pond – Cross Section



Source: WSP, 2024

18.8.10.1 Non-Contact Water Pond Liner Design

The Non-Contact Pond collected "clean" water, with no expected contamination from seepage through filtered tailings. Therefore, a single liner is initially considered for the lining system.

Table 18.32 presents a summary of the liner system selected for the Non-Contact Water Pond.

Table 18.32 – Non-Contact Water Pond Liner System

Feature	Lining System
Non-Contact Water Pond	Single Layer 2 mm HDPE

18.8.11 CONSTRUCTION MATERIALS

This section presents a summary detailing the quantities of the different construction materials required for the construction of the Čoka Rakita surface infrastructure projects.

18.8.11.1 Material Balance

The material balance for the Čoka Rakita Project is divided into four (4) key construction phases

- Stage 1 (End of Operational Year -2): Significant cleaning and de-vegetation, producing the highest amount of topsoil and overburden cut material. Much of the overburden will be reused for terraces and pond embankments
- End of Stage 2 (End of Operational Year 4): Minor cut and full activities as the DTSF elevation increases, with de-vegetation in higher valley regions.
- End of Stage 3 (End of Operational Year 7): Continued minor cut and fill activities, with overburden paces on outer slopes as the DTSF closes
- End of Stage 4 (End of Operational Year 10): Increased topsoil generation for the final DTSF prep, with 138,566 m^3 of overburden used as final closure capping layer.

Table 18.33 presents the total material balance for Topsoil and Overburden Material throughout each construction stage as per the LOM plan.

Table 18.33 – Total Earthworks Material Balance

Material Balance (Stages 1 – 4)	Topsoil		Overburden	
	Cut (m^3)	Fill (m^3)	Cut (m^3)	Fill (m^3)
DTSF Prep	70,505	-	23,502	-
DTSF Closure	-	-	-	-
DTSF Closure (Capping Layer)	-	63,612	168,845	212,040
Ring Road	21,249	-	-	48,976
Mine Dewatering Pond	7,096	-	-	91,536

Material Balance (Stages 1 – 4)	Topsoil		Overburden	
	Cut (m^3)	Fill (m^3)	Cut (m^3)	Fill (m^3)
Non-Contact Water Pond	5,440	-	-	73,298
Reclaim Pond	11,050	-	113,520	40,507
Initial Topsoil Stockpile	7,376	-	-	-
Terrace Stockpile	13,478	-	-	342,563
Access Roads	10,124	-	101,243	66,753
Terraces	91,732	-	917,327	425,585
Total	**238,050**	**63,612**	**1,324,437**	**1,301,258**
Total Earthwork Balance	**174,438**		**23,179**	

18.8.11.2 *Borrow Material*

The mine site construction requires locally sourced borrow material. Initial DTSF development needs 30,154 m^3 of clean rock for PAG Mine Waste Rock containment and 119,869 m^3 of overburden fill and wearing coarse material from the DTSF ring-road. The surface infrastructure has a positive material balance, minimising the need for borrow areas, expect for clean rock for road wearing course.

Initially, "clean" non-acid generating (NAG) rock was expected to be sources from the DTF footprint. however, ground investigation and geophysical surveys indicate a deeper weathering profile, up to 20 m. clean rock may be obtained from the decline excavation, pending geochemical testing to confirm its suitability. Additionally clean rock could come from terrace and access road excavation if good quality rock is found near surface.

Some rock material may need to be procured externally. After the PFS ground investigation, a review of geotechnical logs is recommended to identify suitable rock with the project footprint. Developing a digital twin using sequent leapfrog software can help determine material quantities available on-site.

18.8.12 CLOSURE

The key elements of closure are the following:

- Final Slope Gradient: 1V:5H, with lower slopes initially at 1V:2.5H, extended post-closure for a shallow gradient.
- Capping Options: Use of geosynthetic clay liner (GCL) or alluvial material to cover PAG material, limiting seepage. Progressive closure with low permeability overburden and topsoil capping.

- Active Closure Phase: No additional treatment or monitoring expected, Design modifications based on climate change scenarios.
- Reclaim Pond: Decommissioned and possible replaced by a wetland. Non-Contact. Non-Contact diversion channel widened for 1:10,000 AEP event.
- Stockpiles: No modification expected, rehabilitated during active closure phase.

Key activities include:

- Monitoring DTSF runoff geochemistry.
- Vegetation of slopes.
- Upgrading water management infrastructure.
- Assessing stability and potential settlement of the closure capping layer.

Further recommendation for additional studies with application to closure design is presented in the PFS closure plan. The main areas of study are summarised below.

- Geochemistry.
- Geotechnical Investigations.
- Geohydrology.
- Vegetation.
- Stakeholder Engagement.

18.9 Water Management

18.9.1 Surface Water Management System

18.9.1.1 Introduction and Methodology

As part of the PFS, WSP developed the Surface Water Management design of the following areas:

- Dry Tailings Storage Facility (DTSF).
- Stockpile.
- Mine Dewatering Pond.
- Non-Contact Water Pond.
- Drainage channels and spillways, associated with the clean, surface water distribution.

Figure 18.26 depicts the conceptual water management for the site.

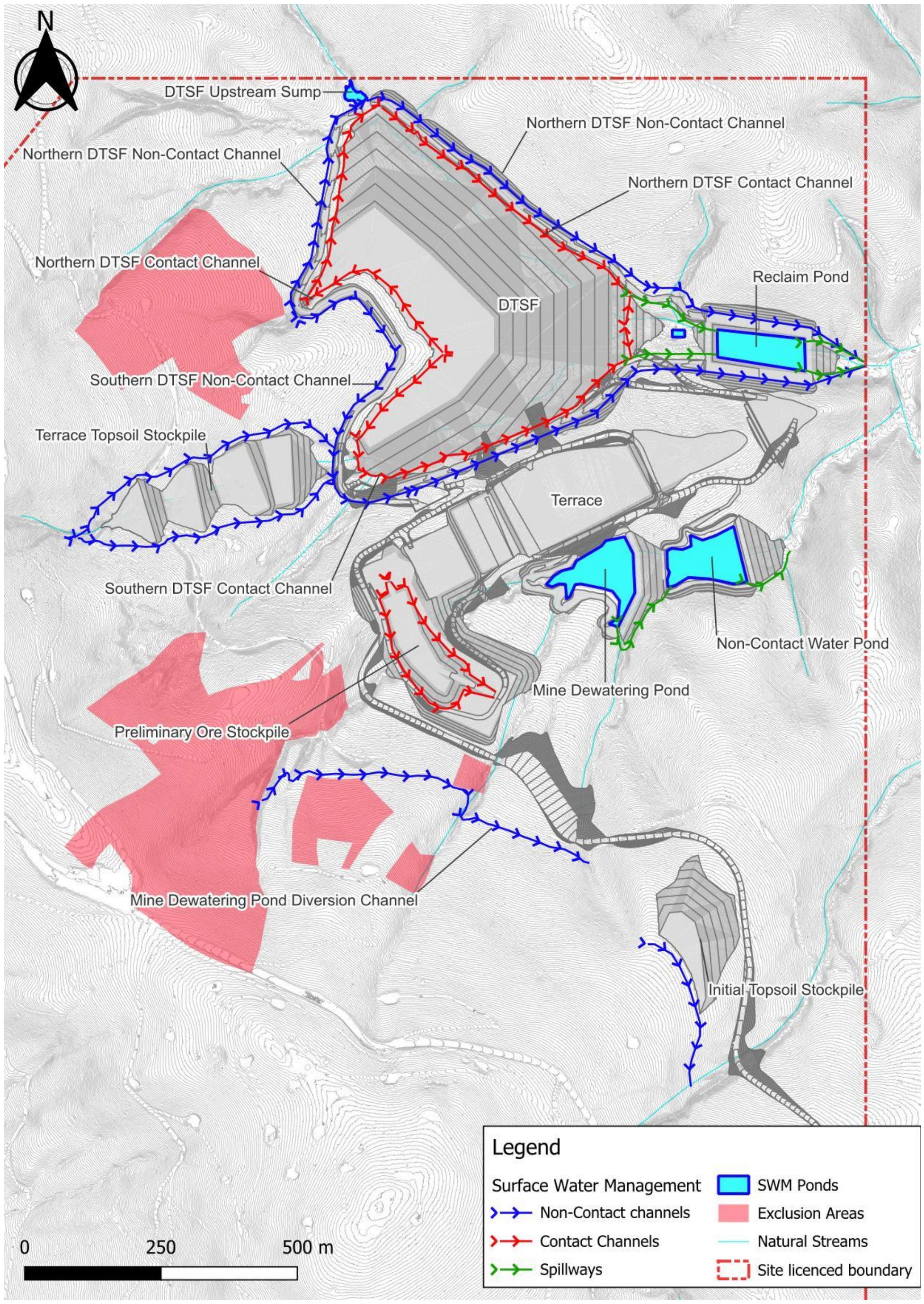
For the pond sizing, a spill frequency of 1:100 Annual Exceedance Probability (AEP) has been selected in consultation with DPM as design criteria, based on what is considered to be a reasonable pond size for the available space and a reasonable pump out rate from the pond. A daily time step rainfall runoff model with soil water budgeting was used to generate a daily time series of pond inflows together with different operating rules for the pump-out rates to simulate a daily pond water balance. The historical climate timeseries of excess water (rainfall plus snowmelt) has been used for the calculation.

Drainage channels and spillways have been sized for the peak runoff rate computed with the Rational Method based on the historical climate timeseries. For each channel, a different return period has been agreed with DPM for the sizing, with a freeboard allowance of 10% of the water depth for the channels and 0.5 m for the spillways. Emergency spillways have been designed to convey the Inflow Design Flood (IDF) assumed to be the 1:10,000 AEP event. A detailed dam break analysis will be completed during the FS to determine a more accurate dam classification, which will inform the review of the IDF assumptions adopted for the PFS.

Typically, the channels have a trapezoidal shape with 1V:2H side slopes, exception for the DTSF non-contact channels which have 1V:1.5H side slopes to reduce the channel footprint. Depending on the channels longitudinal slope, the lining system is composed by geocell filled with stones, for the steeper sections, or by rip-rap, for the mild slope sections.

The spillways are anticipated to be constructed from concrete, in order to resist high flow velocities; and will include stilling basins, to dissipate flow energy in a controlled manner.

Figure 18.26 – Conceptual Surface Water Management



Source: WSP, 2024

18.9.1.2 Dry Tailings Storage Facility Water Management

The surface water management system associated with the DTSF comprises:

- Non-Contact Water Channels, intercepting and conveying runoff from the natural catchment around the DTSF, to the downstream environment;
- Contact Water Channels, intercepting and conveying runoff from the DTSF area to the Reclaim Pond;
- Reclaim Pond; and
- DTSF and Reclaim Pond spillways.

In agreement with DPM, the Non-Contact Water Channels have been designed based on the 1:1,000 AEP peak flow, resulting in a top width generally varying between 2.5 m and 9.5 m.

Runoff from the catchments north of the DTFS report to a nominal sump upstream of the DTSF (DTSF Upstream Sump). The primary function of this sump is to direct flows and runoff into the downstream section of the northern Non-Contact Channel by gravity. It is not intended to store storm runoff volumes.

The Contact Water Channels have been designed based on the 1:100 AEP peak flow, resulting in a top width generally varying between 3.5 m and 8 m.

The Reclaim Pond is designed to contain runoff and seepage from the DTSF facility and runoff from natural catchments below the surrounding Non-Contact Water Channels. The Reclaim Pond has been sized adopting a spill frequency of 1:100 AEP. An additional operational pond volume of 10,000 m^3 has also been considered following discussion with DPM to be used as emergency water supply for the processing plant. Based on a maximum pump out rate of 30 L/s to the Mine Dewatering Pond, the total pond volume requirement is 42,000 m^3.

The DTSF and the Reclaim Pond are equipped with emergency spillways designed to convey the 1:10,000 AEP event. WSP has assumed that during this event, the Non-Contact Water Channels would be unable to convey the peak runoff flow; therefore, all non-contact runoff would discharge to the DTSF. Both, DTSF and Reclaim Pond, will be equipped with two (2) spillways, one on the northern and one on the southern side of the dam. The typical spillways cross-section has a top width of between 4 m and 4.5 m.

18.9.1.3 Stockpiles Water Management

The perimeter channels managing runoff from the three stockpiles have been sized to convey the 1:100 AEP peak flow.

Runoff originating from the Terrace Topsoil Stockpile is considered Non-Contact Water, with the perimeter channels discharging into the downstream DTSF Non-Contact Water Channels. The typical channel cross-section has a top width of 2.5 m, with gabions or rip-rap check dams dissipating the kinetic energy and reducing the risk of erosion.

Runoff originating from the Preliminary Ore Stockpile is considered contact water. As such, the stockpile perimeter channels will discharge to the terrace water management systems and ultimately flow to the Mine Dewatering Pond. The typical channel cross-section has a top width of 2 m.

Runoff from the Initial Topsoil Stockpile is considered Non-Contact Water, a drainage channel on the south-western side of the stockpile will intercept the runoff from the upstream catchment and divert it to the natural drainage towards the east of the stockpile. The typical channel cross-section has a top width of 2 m.

18.9.1.4 Mine Dewatering Pond Water Management

The Mine Dewatering Pond receives pumped water from the Reclaim Pond, with a maximum pumping rate of 30 L/s, along with runoff from the ROM Pad Terrace Area and the upstream natural catchment. From the Mine Dewatering Pond, water is pumped to the WTP via the Clarifier prior being discharged to the Non-Contact Water Pond and subsequently to the environment. The Mine Dewatering Pond has been sized for a 1:100 AEP spill frequency. An additional surge capacity for the pond has been considered to account for maintenance of the treatment plant, estimated to last for five (5) days, during which pumping will continue from the underground mine, and inflows from the external catchment will still be directed to the pond. Based on a maximum pumping rate to the WTP via the Clarifier of 70 L/s, the total pond volume requirement is 48,500 m^3.

The pond will be equipped with an emergency spillway discharging to the Non-Contact Water Pond on the southern side of the dam. The typical spillway cross-section has a top width of 4 m.

To reduce the volume of water requiring treatment, a non-contact diversion channel will be constructed on the southern side of the terrace to intercept and divert runoff away from the pond into the Non-Contact Water Pond catchment. The channel has been designed based on the 1:100 AEP peak flow, resulting in a typical top width of approximately 5 m.

18.9.1.5 Non-Contact Water Pond Water Management

The Non-Contact Water Pond captures the discharge from the WTP and the non-contact runoff originating from a catchment located southeast of the ROM Pad Terraces. The pond is used to supply domestic water and as emergency water supply for the processing plant, the pond is not designed for storing runoff originating from storm events. In the PEA study, the Non-Contact Water Pond was initially designed to store two (2) months of processing plant water supply, equating to 66,000 m^3. To accommodate a larger volume in the Mine Dewatering Pond, the dam embankment has been moved downstream, reducing the capacity of the Non-Contact Water Pond to 63,000 m^3. The water balance model indicates that the new pond volume would be sufficient to satisfy the site water demand during dry periods. Once full, the pond will release excess water to the downstream environment via the spillway. The typical spillway cross-section has a top width of 4 m.

18.9.1.6 Conclusions and Recommendations

The current assessment has identified key gaps that should be addressed as part of the FS. The understanding of climate and hydrology at the site should be improved by collecting and analysing additional data. To review the design criteria for the surface water infrastructure and the subsequent infrastructure sizing, it is recommended to progress the studies for defining site-specific discharge criteria both in terms of water quality and quantity. An updated groundwater model has been developed as part of the PFS; however, the results were not available to inform the current study. The sizing of the Clarifier, the WTP, Mine Dewatering Pond, Non-Contact Water Pond should be reviewed at FS stage, once the updated groundwater inflows are available.

18.9.2 WATER TREATMENT PLANT

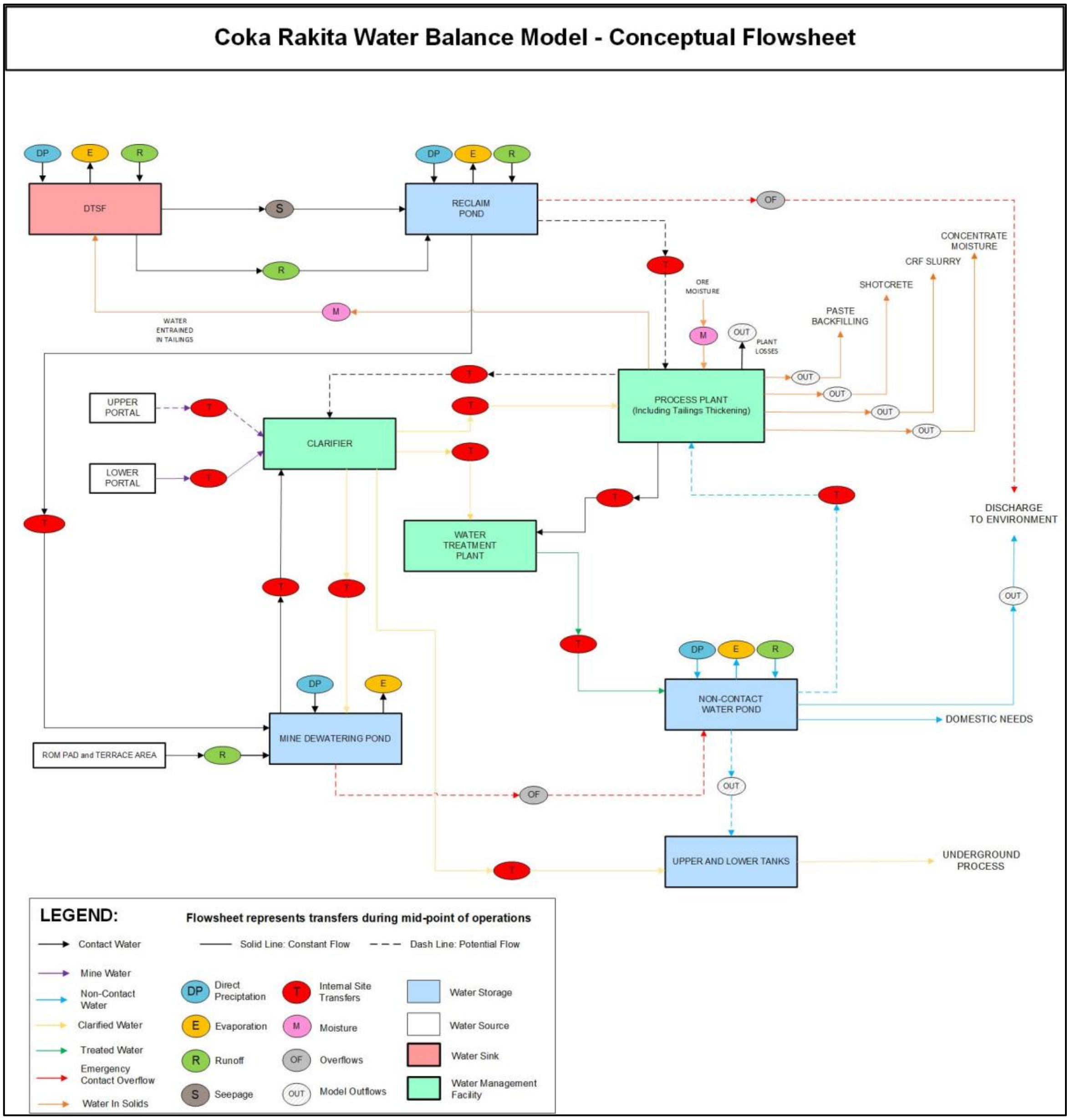
Based on the available information when developing the water balance, an allowance has been made to treat mine dewatering and/or tailings reclaim water. Further quantification of the mine dewatering capacity and quality are required to define the scope of the WTP.

18.9.3 SITE-WIDE WATER BALANCE

WSP has developed a site wide SWBM as part of the Čoka Rakita PFS using GoldSim modelling software. The SWBM covers the LOM from the start of decline construction to the end of operations. The development of the SWBM has allowed for the appraisal of internal transfer pump rates, the consumption of water by the mining process, storage pond sizing and environmental discharge rates. This was achieved by running the SWBM stochastically for 100 realisations, introducing variability into the results and allowing the SWBM to assess the Site water management under periods of both wet and dry conditions. The SWBM incorporates all on-site facilities, including the DTSF, and encompasses a contaminant transport component in order to assess site water qualities.

A Process Flow Diagram (PFD) was created to provide a visual representation of the model flows and transfers. The schematic shows each on-site facility, including the ponds, Process Plant, Clarifier and DTSF. As the schematic details the interconnections between each facility, it is able to offer a clear snapshot of the SWBM operational dynamics. Different coloured lines are also incorporated in order to categorise flows by water type.

Figure 18.27 – Site Water – Process Flow Diagram



Source: WSP, 2024

The key inflows into the model include precipitation and runoff (into each of the on-site ponds) and underground dewatering, which enters the Site water management system via the Lower Portal. The dewatering flows from the Lower Portal are routed through the Clarifier, where the water is directed to the Process Plant, WTP, and the Upper and Lower Tanks (which support underground processing requirements). Any excess water, if required, is directed to the Mine Dewatering Pond.

As well as receiving water from the Clarifier, a minor amount of water enters the Process Plant as ore moisture. The plant extracts and recovers minerals from the raw ore, utilising water throughout the process. Water exits the system within various outputs, including concentrate, tailings (paste backfill and surface tailings sent to the DTSF), shotcrete, and Cemented Rock Fill (CRF) slurry. Additionally, a nominal amount of water is lost due to inherent process inefficiencies. If ore moisture and water from the Clarifier are insufficient to meet process demands, additional water can be supplied from either the Reclaim Pond or the Non-Contact Water Pond. Any excess water is directed back to the WTP, or to the Clarifier whether it is characterised by high solids concentration.

The WTP receives water from the Clarifier and excess water from the Process Plant. The WTP discharges exclusively to the Non-Contact Water Pond. Along with precipitation and runoff, the Non-Contact Water Pond can also receive emergency overflow from the upstream Mine Dewatering Pond, however the water management systems have been designed to prevent overflows from the Mine Dewatering Pond where possible. Water for domestic needs is abstracted from the Non-Contact Water Pond. Additional supply for process plant demands and the Upper and Lower Tanks is also sourced from the pond if necessary. Any surplus water is freely discharged to the environment via a spillway.

The DTSF receives water entrained in tailings along with precipitation and runoff. Water from the DTSF enters the Reclaim Pond either as runoff or seepage. Water from the pond is primarily directed to the Mine Dewatering Pond, though it may also supply the Process Plant if necessary. Emergency overflows from the pond enter the downstream environment, however the water management systems have been designed to prevent overflows where possible.

Along with pumped water from the Reclaim Pond and excess water from the Clarifier, the Mine Dewatering Pond is subject to runoff from the ROM Pad and Terrace Area, and direct precipitation. Water is pumped to the Clarifier and subsequently treated in the WTP before being sent to the Non-Contact Water Pond. The pond emergency overflows are discharged into the Non-Contact Water Pond, however the water management systems have been designed to prevent this where possible.

The SWBM has been set-up to simulate both historical observed climate data and climate change projections in order to stress-test the Site resilience to future climate conditions. There are therefore three distinct climate scenarios which can be used: a historic climate scenario based off data from the nearby Crni Vrh weather station and two climate change scenarios. Several climate change projection models were acquired and statistical analysis was run on each to determine which two could be considered the wettest and the most arid. Therefore, the SWBM is able to run a 'wet' climate change scenario and a 'dry' climate change scenario. All three scenarios can be run stochastically. Additional analysis has also been conducted on the sensitivity of the WTP treatment rate and failures of the underground dewatering system.

Other than climate data a key inflow to the model comprises underground dewatering predictions, these have been based on groundwater modelling estimates by WSP (it is understood that an updated groundwater model is being developed for the site at the time of writing, but the results are not yet finalised to be incorporated in the current assessment). Process plant demands have been calculated using tailings parameters and ore throughput rates provided by DPM and pond geometries were agreed upon following discussions with the client regarding earlier trade-off studies. Pump rates within the model were similarly agreed upon with the client during the trade-off study discussions and further validated through use of the SWBM. Catchment areas used for the calculation of runoff were delineated based on the agreed Site layout following the trade-off studies. It should be noted that the results discussed here are representative of the data available at the time of model development. It is understood that new data will become available and that the proposed water management systems will be updated as the project progresses. Any new data or system changes will be incorporated in a future update to the model.

Modelling results indicate that the Site process demands are able to be met under all three climate scenarios (the historic climate scenario, the 'wet' climate change scenario and the 'dry' climate change scenario). This is because process demands, for both the Process Plant and the underground, are met using water from the Clarifier, which is in-turn supplied by underground dewatering. The underground dewatering rate exceed the Site process requirements across the LOM, suggesting that drought periods would have a limited impact on these demands. Based on the inputs adopted for the current modelling, no additional water would be required from the Site storage ponds and excess water stored in the ponds is eventually discharged to the environment after being routed through the WTP (which has a capacity of 80 L/s) and the Non-Contact Water Pond. The Non-Contact Water Pond water is allowed to spill freely once the pond capacity is exceeded, whereas the Reclaim Pond water is pumped to the Mine Dewatering Pond where it is then treated in the WTP and discharged to the Non-Contact Water Pond. As the modelling indicates that the Site can function adequately without a significant non-contact water supply, it is recommended that the Non-Contact Water Pond sizing is reviewed further.

The results also show that when run for 100 realisations under historical climate conditions, there is no overflow from either the Reclaim Pond or the Mine Dewatering Pond. The Reclaim Pond is designed to maintain a 10,000 m^3 emergency water supply, with its additional volume used for storm surge capacity to handle a spill frequency of less than 1:100 AEP. The Mine Dewatering Pond has also been sized to accommodate a spill frequency of less than 1:100 AEP. An additional surge capacity for the pond has been considered to account for maintenance of the treatment plant, estimated to last for 5 days, during which pumping will keep occurring from the underground mine, and inflows from the external catchment will be reporting to the pond. Based on the pond sizes considered, the model suggests that no overflows will occur over the LOM based on the historical climate data.

Under the climate change scenarios included in the SWBM, the Mine Dewatering Pond was largely unaffected, with no increased risk of overflow. However, there is a low probability of overflow from the Reclaim Pond under the wet climate change scenario. As a mitigation measure, future studies should consider pumping down the 10,000 m³ of water which is kept in the pond at all times as an emergency process supply buffer. Reducing this volume would increase the available capacity within the pond, allowing for more storm surge capacity. To balance the need for a greater storm surge capacity in the wet months, and a greater emergency process supply buffer during the dry months, future studies could consider only reducing the 10,000 m³ during the wet months and allowing it to build up again during the drier months.

A sensitivity analysis has been conducted on the WTP to assess the impacts of potential variation of the treatment rate on the surface water management. A WTP rate of 80 L/s was selected based on what is considered to be a reasonable pond size for the available space and a reasonable treatment rate in consultation with DPM. Analysis indicates that if the rate were to be lowered, a significantly larger Mine Dewatering Pond volume would be required, which due to topographical constraints would not be possible.

Dewatering inflows were identified as the key driver for the system ability to meet the Site process demands, these include Process Plant demands, underground process requirements and domestic needs. Further tests were conducted to evaluate the system ability to continue meeting these demands in the event of a sudden disruption to this source. During such a disruption, additional water can be sourced from the Site ponds. Results indicate that Site demands can be met for just over one (1) month, on average.

WSP proposes the following recommendations to enhance the accuracy of water resource management at the Čoka Rakita Site:

- Incorporate dynamic catchment areas.
- Include further climate scenarios.
- Further assess the impact of different system failures.
- Integrate future updates and data into the model.
- Optimise pond and pump sizing.
- Review Non-Contact Water Pond sizing.
- Fully model the closure phase within the SWBM.

18.9.4 SOLUTE TRANSPORT MODEL

A solute transport model was incorporated into the SWBM to calculate probabilistic estimates of the water quality in the different water management ponds on site (Mine Dewatering Pond, Reclaim Pond and Non-Contact Water Pond) and in the feed water for the WTP during the operational period. Source term water qualities were estimated based on available geochemical testing and environmental monitoring data for the following water types:

- Natural Catchment Runoff.

- Terrace Runoff.
- DTSF Contact Water.
- Underground Mine Dewatering.
- Ore Stockpile Contact Water.

The water quality estimate for Natural Catchment Runoff was estimated from environmental monitoring data for location SW06 as a proxy, which is closest to the Project area. The river water displayed non-compliant concentrations against RS Gazette Class I or Preliminary EDC Class I or II for sulphate, Al, Cd, Fe, Hg and NH4-.

The WTP feedwater quality was screened against project water quality guidelines to assess non-compliance risks and requirements for potential water treatment or other mitigation measures. The following parameters exceeded RS Gazette Class I at the 95th percentile (i.e. non-compliant concentrations in at least 5% of the realisations): As, Cd, Cl, Cr, Cu, Hg, Ni, Pb, Sulphates, B. The following parameters exceeded the Preliminary EDC Class II at the 95th percentile (i.e. non-compliant concentrations in at least 5% of the realisations): As, Cd, Cl, Hg, Ni, Pb, Sulphates.

18.10 Power Supply and Distribution

During the PFS, it was decided by DPM that a 35 kV line from Zagubica substation was the preferred solution for power supply the Čoka Rakita site.

The Zagubica substation will provide two (2) 35 kV feeders to deliver redundant power to the Čoka Rakita site. The 35 kV supply cables will be buried in trenches along the roadside for approximately 6 km terminating at the main Čoka Rakita substation.

Voltage regulation at the plant will be supported by an appropriately sized capacitor bank and harmonic filter. The specific size requirements will be confirmed with DPM in the next phase of the Project.

Two (2) standby 1.4 MW diesel generators will be provided for emergency backup power to the mine and critical processes within the plant.

18.10.1 Power Distribution

The 35kV power supply will feed a pair of transformers, each with a capacity of 16 MVA, 35/10.5 kV. These transformers, with a delta-star configuration, will provide all power requirement on-site. Based on the current load profile, the transformers are designed to operate with sufficient spare capacity, ensuring redundancy for the entire site.

Table 18.34 presents the nominal voltage levels used on-site.

Table 18.34 – Nominal Voltage Levels

Category	Voltage (V)	Phases	Fault Current Rating	Grounding
Main Primary Supply (High Voltage)	35,000	3PH, 3W	To be furnished by Utility	N/A
Primary Distribution (Surface and Underground)	10,500	3PH, 3W	16 kA*	IT System (ungrounded)
Medium Voltage Utilisation	6,000	3PH, 3W	16 kA	Resistance Grounded System
Low Voltage Utilisation (Surface)	400	3PH, 4W	50 kA	Solidly grounded to IEC 60364 earthing arrangement TN-S.
Low Voltage Utilisation (Underground)	1000 (690)	3PH, 3W	50 kA	IT System

18.10.2 Electrical Rooms

Containerised Electrical Rooms (E-Rooms) will be utilised on the Čoka Rakita site to accommodate all switchgear, motor control centres, variable speed drives, communications equipment, and other services.

18.10.3 Refurbishment of Ada Tepe Equipment

To minimise equipment costs across the site, DPM intends to reuse the existing Ada Tepe electrical equipment, wherever feasible. For the PFS, the equipment manufacturer was contacted and provided with a preliminary scope of work to refurbish this equipment for the new site.

The refurbishment scope will include, but is not limited to:

- Transporting existing container E Rooms and equipment to OEM factory.
- Refurbishing existing 20kV switchgears for new site and voltage.
- Refurbishing existing Motor Control Centres for new site.
- Re-installation and re-wiring or all equipment for new site.
- Performing all cleaning and testing to comply with the original manufacturer's standards.

18.10.4 Emergency Power

A centralised emergency power system will be established as a standby source to supply essential services (emergency and exit lighting, fire pumps, etc.) and critical process loads in the event of power loss from the grid. The system will consist of two (2) 1.4 MW, 400 V diesel generators and 10 kV step up transformers, which will supply power to the main substation 10 kV bus.

From the main bus, these generators are capable of supplying power to both the mine and surface plants simultaneously, with the load controlled by the PCS.

One (1) generator shall be refurbished from the Ada Tepe site, while the other will be newly procured for Čoka Rakita.

18.10.5 Site Load

The total power demand is estimated at 15.6 MVA with 10.1 MVA for the surface plants, 3.6 MVA for the underground mine and ventilation facility, and 1.9 MVA for the waste dewatering and non-process facilities.

All utilisation factors developed in the summary have been selected with DPM's operational experience.

The power required non-process facilities services the following: administration, offices, mechanical shop, laboratory, electric rooms, truck maintenance, guard house, as well as losses in transformers and feeders. The process power demand was estimated based on data from the Project MEL.

18.11 Communication

The Čoka Rakita site relies on a carefully designed network to keep its operations running smoothly and securely. The network backbone uses fibre optic cables to connect essential systems like business applications, process controls, CCTV, and communication across the entire site. Using Cisco-branded equipment and robust, redundant connections, this setup ensures that data and communication channels are always reliable, even in demanding conditions.

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.

The commercial terms for the sale of gold and freight cost estimates used in this analysis were provided by DPM in an Excel Spreadsheet. A summary of the key terms for the gravity concentrate, flotation concentrate, and doré is provided in Table 19.1. From a logistics perspective, the mine would need to absorb all 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 – Commercial Terms for the Sale of Gold Concentrates

Description	Unit	Gravity Gold Concentrate	Flotation Gold Concentrate	Doré
Treatment Charge	$/dmt	170	150	-
Refining Cost (Gold)	$/oz	6.0	7.5	0.4[2]
Penalty	$/dmt	-	-	-
Gold Payable	%	99.8	97.5[1]	99.9
Concentrate Freight, Inland	$/dmt	125.5	51.8	0
Concentrate Freight, Ocean	$/dmt	75	-	
Doré Freight				
Shipping Charge	$/oz			1.0
Handling Charge	$/lot[3]			350
Handling Additional Charge	$/lot			80

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.
2. Minimum payable is $500/lot
3. Lot size is expected to be 50-100 kg.

19.2 Commodity Price

The QP (Daniel Gagnon, P. Eng.) has adopted the following price projection for the PFS financial model base case as presented in Table 19.2. The selected gold price is conservative compared to three-year trailing average of US$ 2,025/oz or Q4 2024 average spot price of US$2,700/oz.

Table 19.2 – Commodity Pricing

Element	Unit	Financial Model
Au	US$/oz	1,900

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 Metal Exchange (LME).

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This Section summarises the work undertaken for social and environmental studies, permitting and community engagement. These aspects will be more completely assessed in the Project Environmental Impact Assessment (EIA) to be started in Q4 2025. This Section addresses the current Project footprint of the mine infrastructure and landforms described in this Report.

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. However, the permitting system is undergoing change as part of Serbia's planned accession to the EU.

Serbia was granted EU 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 good international practice, such as the performance requirements of the European Bank for Reconstruction and Development 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 its projects in compliance with EBRD Environmental and Social Policy. The permitting process, and progress through it at the current PFS stage is detailed in the sub-sections below.

20.2 Project Description

The Project is located in the District of Braničevo in Eastern Serbia; a rural, hilly area with steep valleys, characterised by seasonally grazed pastures, woodlands and isolated farms and houses. The Project surface infrastructure is located 1.2 km north of the Čoka Rakita deposit in the uppermost catchment of the Dumitrov and Ogasu Giljei streams that join and run eastwards to the Lipa River.

The Project site is sparsely and seasonally populated, with the closest population centre being the town of Žagubica, 10 km to the southwest. There are no designated protected areas for biodiversity or cultural heritage in the Project footprint.

The Project comprises an underground mine accessed via an upper and lower decline, the upper mine portal will be located adjacent to the ROM stockpile and the second portal will be located at a lower elevation in the valley below the processing facility. Primary crushing will take place at the portal site and ore will be conveyed to the process facility. Gold will be recovered using a gravity concentration step followed by flotation of the gravity tailings. Filtered tailings and waste rock will be produced and transported via trucks to a filter stack (the DTSF) and a paste backfill plant which will utilise dry filter cake tails and waste rock in the backfill paste production.

Contact water collected from mine dewatering and surface runoff from process areas, roads and the DTSF and will be routed to two contact water collection ponds (a DTSF reclaim pond and the mine dewatering pond). All the contact water collected will be sent to the ETP for treatment. Treated water will be returned to the process plant as freshwater or discharged to the environment when required. Non-contact water will be diverted around infrastructure into the environment. A non-contact water pond is proposed to provide water supply contingency for the process plant. There will be a soil storage facility (stockpile) located to the north-west of the processing facility.

The mine plan runs over a period of 10 years. The Project layout is provided in Figure 18.1.

20.3 Water Management

A mine water management plan (Section 18.7) has been developed for the PFS design outlined above, and defines the hydrological parameters to support engineering design, estimates the underground dewatering rates, and defines the site water management plan, site-wide water balance modelling, sizing of water management ponds and ditches, pump sizing, and construction material estimates. The DTSF will have a composite liner system to minimise seepage to groundwater. Contact water from the DTSF will be pumped to the ETP for treatment prior to discharge to the environment. Non-contact water diversions will be constructed to divert water away from the DTSF, the DTSF reclaim pond and south of the process plant terrace where the ROM pad is located.

The Project is expected to have a positive water balance resulting in a surplus of water. Sufficient water will be available from the DTSF reclaim pond and underground mine dewatering to meet the water demand to run the mill and process facility. The Project will not need to rely on collection of surface water runoff from precipitation.

The underground mine will require dewatering which will be carried out using sumps on each underground level and a system of interconnected drainage boreholes. Water from level sumps will be pumped via a mono pump station at the bottom of the spiral ramp and additional pump stations situated at approximately 80 m (vertical) intervals on the spiral ramp up to the main surface pump station located at the top of the spiral ramp. The pumps will be connected by a system of pumping boreholes. Water will be pumped from the main pump station to the surface via a pipeline situated along the decline. All water pumped from underground will be pumped to the clarifier and once this process is complete it will be pumped to the process plant and mine dewatering pond and further pumped to the ETP for treatment. Following treatment water will be pumped to upper and lower storage tanks for use in underground processes, any excess water will be discharged to the environment.

Water collected in the DTSF reclaim pond will be piped to the mine dewatering pond, from here water is pumped to the clarifier (when required) and the ETP for treatment. The majority of water inflow to the DTSF reclaim pond will be due to surface water runoff and precipitation (this is due to the large catchment area; freshwater diversions will be implemented to divert fresh non-contact water away from infrastructure). Freshwater make-up for the process plant will be supplied in the first instance from the clarifier, and non-contact water pond (when required). Treated water will be discharged from the ETP to the environment into an un-named stream which drains into the Lipa River. Figure 18.26 illustrates the surface water management approach.

Two (2) contact water ponds (mine dewatering and DTSF reclaim) have been designed and will be constructed using a similar liner system to the DTSF, to minimise seepage to groundwater. The two (2) contact water ponds have been sized adopting a spill frequency of 1:100 AEP. Both ponds will be equipped with emergency overflow spillways to prevent dam overtopping (normal discharge will be undertaken via pumping). Water discharged from the spillways will drain overland to an un-named stream which drains into the Ogašu Lu Gjori Stream, a tributary of the Lipa River.

The non-contact water pond will have a single liner system and will supply domestic and emergency water. The volume is based on contingency storage to provide two (2) months of freshwater supply to the process plant. Once the non-contact water pond is full, water will be directed via a spillway to the same un-named stream which drains into the Ogašu Lu Gjori Stream.

Basal drains will be constructed beneath the DTSF foundation and will extend beneath the reclaim pond. The purpose of the drains is to allow non-contact groundwater to be discharged into the environment.

Process water balance calculations and diagrams have been developed in the site-wide water balance (Section 18.9.4) to model water needs in average, dry and wet conditions (based on the 25-year dry, average and 25-year wet annual precipitation conditions).

A hydrological study of watercourses and details of planned water management and treatment will be submitted as part of the application for water conditions, a key stage in permitting requirements in Serbia.

20.4 Alternatives

A range of alternatives have been assessed as part of the PFS. The Project has considered environmental and social aspects in assessing the alternatives in line with EBRD Performance Requirements and the Serbian Rulebook on the Contents of the Environmental Impact Study.

20.4.1 DTSF Footprint Optimisation Trade-Off Study

WSP conducted a DTSF footprint optimisation trade-off study with the aim of assessing the suitability of alternatives derived from the locations established in the PEA. Three (3) sites were identified, all of which are situated within the Dumitru Valley. Site One (1) is located within the centre of the Durmitrov Valley. The DTSF would cover both valleys (North and South), with the tailings deposition tied-in together. Two (2) variations of this site have been considered, the first being the original PEA position (Site 1A) and the second is 100 m upstream (1B) to provide additional downstream space for the water management infrastructure. Site Two (2) is located solely in the northern Durmitrov valley. All tailings would be stored within this valley and therefore limit the impact of the DTSF footprint. Site Three (3) is located solely in the southern valley and all tailings will be stored within this valley, limiting the impact of the footprint and not impacting the Durmitrov valley at all.

When considering the constraints for the site location, several key components were considered:

- PFS Mine Layout Plan and access roads;
- Land Parcel Procurement, divided into easily purchased, delayed purchase process and not acquirable;
- Mine Permit Boundary;
- Water management infrastructure;
- Lower Decline position; and
- Access road and Diversion channel infrastructure.

WSP assessed that Site 2 cannot fulfill the LOM tailings storage capacity without encroaching onto land outside the mine facility boundary to the north. Site 3 has the capability to store the LOM tailings capacity, but requires the land take of a land parcel with a delayed purchase process, and also leaves very little space for downstream water management structures. Site 1 has two (2) options, the original PEA TSF footprint (1A) and the upstream movement of the downstream toe by 100 m (1B). Site 1B provides extra space for water treatment infrastructure both for operation and closure.

Both Site 1A and 1B underwent re-design optimisation with updated filtered tailings slope gradients of 1V:5H in order to accommodate for a tailings storage volume capacity target of 3 Mm^3. Site 1B was assessed as the preferred location as it provides additional space for the development of water management structures. Operationally, there is space for the construction of basal drain access points and the DTSF reclaim pond. During closure, it was assessed that the downstream valley can be developed into a wetland area. If the land parcels with delayed purchase processes are included in the DTSF footprint, then gravity fed water management structures can be implemented, which would reduce operational costs and easily be transitioned into closure. A possible expansion of the DTSF footprint for Site 1B was considered, which would expand the facility to accommodate an additional 100% of the LOM, increasing the total capacity to 6 Mm^3 of tailings.

The footprint of Site 1B allows for gravity drainage of both contact and non-contact water around the perimeter of the DTSF, reducing the need for pumping systems or penstocks which may be complex both in operation and during closure. An additional consideration is the transportation of tailings from the process terrace to the DTSF. The use of a conveyor system as proposed in the PEA will be a costly structure to implement given the engineering required to convey the tailings down the steep sided hill slope, however it is likely this option would have reduced greenhouse gases, air quality and noise impacts than would be associated with the favoured road haulage option. Given the throughput rates and the LOM of the Project, the option of using trucks for the transportation of tailings to the DTSF footprint should be considered as they will likely (subject to confirmation) have a reduced Capital expenditure and will likely be easier to maintain.

20.4.2 Tailings Deposition Trade-Off Study

WSP conducted a trade-off study on the preferred tailings management option, they assessed four (4) variations of tailings technology including:

1. Filtered tailings;
2. Paste tailings with amphiroller;
3. Conventional paste tailings; and
4. Paste tailings with drainage cells.

Each technology was weighted based on categories including Permitting, Environment, Socio-economics, Project Economics and Technical aspects, and a strength, weakness, opportunity and threat (SWOT) assessment was conducted.

The filtered tailings option was found to have the following advantages over the paste options:

- Permitting of a dry stack facility would require less time than a paste option.
- Filtered tailings are considered a proven technology, and are well documented in terms of safety, this option is therefore perceived as being more acceptable in Serbia.

- Public perception of filtered tailings is that the chance of outflow and dam break is significantly reduced.
- Filtered tailings have a lower moisture content, this will reduce the chances of water infiltration to groundwater.
- Filtered tailings can be shaped to form an optimum landscape, allowing more flexibility during closure.
- Filtering the tailings will recycle large volumes of make up water for re-use in the processing plant.
- Waste rock for construction purposes may need to be sourced, filtered tails require less construction material due to limited capping requirements for the facility and reduced requirements for retaining embankments.

The assessment concluded that Filtered Tailings was the most suitable technology. Filtered tailings scored higher as this option would be more favourable to Serbian permitting regulations, environmental and technical categories.

20.4.3 Paste Backfill Plant Location Options

The current project definition includes a paste backfill plant which will utilise filtered tailings in the paste backfill process. As part of the tailings management trade off study, a number of scenarios for the location of the paste plant and the dewatering arrangement were considered.

WSP was requested to conduct a further trade-off study associated with possible surface locations of a tailings filtering and paste plant for the Čoka Rakita Project. The PFS trade off studies established that the preferred tailings management method was filtered tailings.

WSP investigated five (5) possible locations and configurations for the paste plant which are described as follows:

- **Base Case (Option 1)**: The paste plant is located at the current upper portal ROM stockpile area (3) while the location for the base case is near the orebody of the Mine where a borehole would intersect the upper decline (5). Both options will utilise mill discharge tailings via pressure filtration with truck haulage of filter cake to the paste plant, which will be used as backfill for delivery underground and for cake transport to the DTSF via either truck or conveyer. The only difference between these options is the distance from the mill. The base case paste plant location was used in the analysis as the worst-case location scenario.
- **Option 2A**: The paste plant location at location (3) where paste is produced for underground backfill and cake is either trucked or conveyed to the DTSF. Mill discharge tailings slurry will be pumped to the paste plant for pressure filtration.

- **Option 2B**: The paste plant location is at location (3) where paste is produced for underground backfill or delivered to the DTSF. Mill discharge tailings slurry will be pumped to the paste plant for vacuum filtration.
- **Option 3**: The paste plant location is at location (3) where paste is produced for underground backfill. Mill discharge tailings will be dewatered via thickening and pressure filtration with cake transport to the DTSF via truck or conveyer.
- **Option 4**: Paste will be produced at the mill (1) using mill discharge tailings via thickening and vacuum filtration. Cemented paste is pumped to a booster station (3) where the paste is delivered as underground backfill or, uncemented paste is pumped from the Mill to the DTSF, i.e., similar to Option 2B.

In summary, in terms of cost and operability perspective, it was determined that Option 2A, with the paste plant located at the ROM portal, provides the most favourable approach to the overall management of tailings, with delivery of filter cake to the DTSF via overland haulage route and paste delivery to underground. The transport of filter cake to the DTSF via overland haulage will result in increased greenhouse gases, air and noise impacts.

20.5 Permitting

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. Key permits are summarised in Figure 20.1.

The permitting system is undergoing change as part of Serbia's planned accession to the EU. The Project is located in the Čoka Rakita exploration license. In total, DPM has four (4) exploration licenses (Potaj Čuka, Pešter Jug, Čoka Rakita and Umka). The Potaj Čuka license hosts the Timok Gold Project, which was advanced to a PFS level by DPM as of 2021. 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.

Few private-sector mining projects of this scale have passed through the entire Serbian permitting process in recent years (Cukaru Peki copper-gold mine operated by Zijin Mining is the most recently permitted project and has been in operation for two years) and there is still some uncertainty in the permitting process.

The main permits and regulatory procedures are managed by four (4) key ministries:

- Ministry of Mining and Energy (MME);
- Ministry of Construction, Transport and Infrastructure (MCTI);
- Ministry of Agriculture, Forestry and Water Management (MAFWM); and

- Ministry of Environmental Protection (MEP).

The current Project status is an Exploration and Reservation of Exploration Field Permit, under this permit the project owner can perform exploration activities for an initial three-year period. The permit can be renewed twice following the initial three-year period, the first renewal is for a further period of three years, the second renewal covers a period of two years, or a total potential term of 8 years. After expiration of these 8 years, the project owner can initiate the retention period, Reservation of the Exploration Field Permit, which can last up to 3 years. The MME decides on the duration of the retention period. No further exploration activities are allowed during this period. The project owner must apply for the approval of exploitation, before expiration of retention period. Crni Vrh is currently in the second year of exploration.

Obtaining the Approval of Exploitation of Minerals is a key milestone, setting out the area and terms for exploitation. This requires various precursor permits and permissions to be obtained, including:

- Confirmation of compliance with the Spatial Plan.
- A Certificate of Mineral Resources and Reserves.
- A Serbian Feasibility Study.
- A Decision on the Environmental Impact Assessment Screening and Scoping.
- Regulator Conditions for water, cultural heritage and nature protection.
- There are three (3) groups of permits required for the construction phase to progress:
    - Approval for Construction of Mine facilities.
    - Approval for Construction of Non-Mine objects (civil facilities).
    - Mine Waste Management Permit.
- Following the construction, and trial period of the mine, the Project will obtain an Approval for Use of Mine Objects (issued by MME) which is the Operating Permit for the mine and mining objects. The permit is issued when the facility successfully passes the Technical Examination.
- Due to shared permitting jurisdiction between two Ministries, MME and MCTI, non-mine objects will obtain construction and operating permits in separate permitting procedures, led by MCTI.

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 Project will require a Spatial Plan for Special Purpose Area (SPSPA). The SPSPA commencement process requires DPM to formally engage with the MCTI and request that they initiate the SPSPA process. Currently, DPM initiated the SPSPA process on April 2, 2024 with the MCTI. Following the initial request, the MCTI will prepare a draft decision for submission to the Government of Serbia, who will provide a decision on the commencement of preparation of the SPSPA at a Government session.

While a decision by the Serbian government to initiate the development of the SPSPA is currently pending, the Company's approach includes having all preparatory work completed and ready for submission while continuing to proactively engage with relevant stakeholders to mitigate the risk of administrative delays.

The SPSPA will be prepared by a Plan Developer (typically Institute of Architecture and Urban and Spatial Planning Serbia) who is then appointed, and the overall process is owned by MCTI. DPM's role will be limited to a stakeholder; however, project information and data will be provided by DPM.

The Spatial Plan will require a Strategic Environmental Assessment (SEA). The Spatial Plan and SEA processes will be developed in parallel by a spatial plan developer. Approval of the SEA Report is a key stage and constitutes a critical permit for the Project.

Environmental information currently being gathered will be shared with the plan developer to ensure that the SEA that will accompany the Spatial Plan be compatible with the Project-specific EIA. Close engagement with plan developer during SEA development is recommended to ensure proper and efficient guidance of the process. During SEA Report development, the land acquisition approach for the entire Project area should also be finalised.

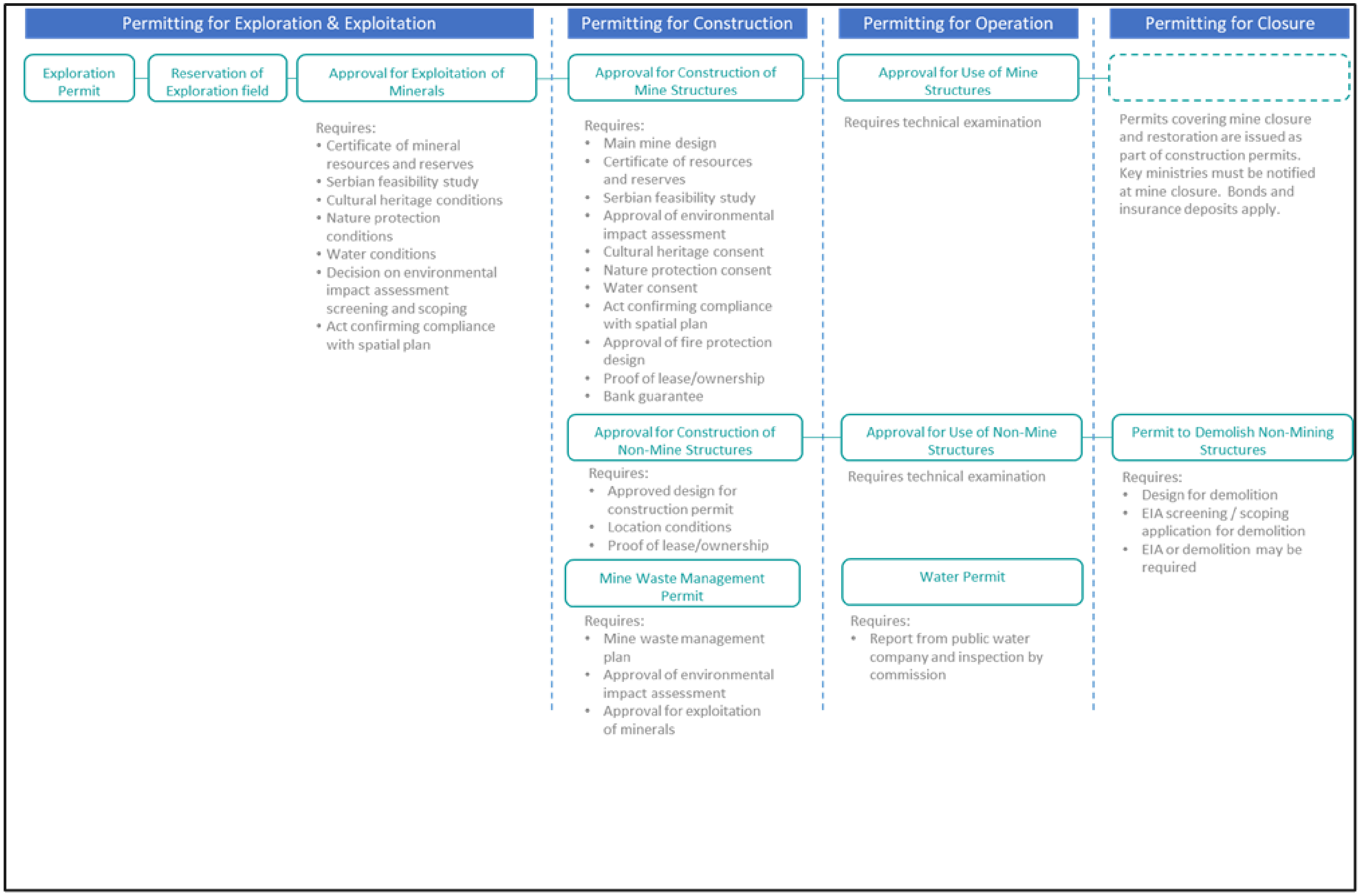
The process of development of a plan is lengthy, taking in the order of 18 to 24 months and up to 33 months in the case of a recent mining project in Serbia of a similar scale. This process is considered a critical path for permitting. Development of the plan is regulator-led and design information will be disclosed during the public hearings required for this process, including information on potentially sensitive topics.

The study to support the application for the Certificate of Mineral Resources and Reserves and the Serbian Feasibility Study will build on the studies developed as part of the PEA.

DPM has committed to undertake a Serbian EIA (required by Serbian authorities) in line with good international practice. The initial phase of this assessment will involve preparation of a scoping report, to be submitted as part of the request for a decision on Environmental Impact Assessment Screening. The assessment itself will cover impact of the Project design on air, water, biodiversity and soils, local communities, and other receptors in line with Serbian requirements. A range of baseline studies will be needed and given the long lead time of many of these studies, DPM has already embarked on planning and delivery of these studies. Relevant design information will be disclosed during the permitting and stakeholder engagement process.

Applications for regulator conditions on cultural heritage and nature protection are needed and will require background information and engagement with the regulators.

Figure 20.1 – Serbian Permitting Requirements



Source: ERM, 2024

Water conditions studies will also be required. These form the first part of a three-stage process to obtain the water permit, which covers abstraction, use and discharge of water for mining projects and the process plant operation, and the storage and discharge of hazardous and other substances that can pollute water (Law on Water, Article 122). It will define the quantity of water that can be abstracted and discharged, and the quality of the discharged water in line with the receiving water, or potential use of wastewater. Twelve months of baseline data is required for this permit and baseline data collection is underway.

The Spatial Plan process in Serbia is pending and other potential Project delays have been identified including potential changes to regulation to align with EU law, regulator delay, public challenge to the Spatial Plan or Serbian 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.

20.6 Environmental and Baseline Studies and Issues

20.6.1 Environmental and Social Studies

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. The baseline program is proposed to include:

- Air quality monitoring;
- Noise and vibration monitoring;
- Surface and groundwater monitoring and developing groundwater models;
- Further terrestrial and aquatic biodiversity assessments;
- Soil and geochemistry - including determining acid-generating potential of rocks and soils to be disturbed;

- Landscape baseline;
- Further cultural heritage baseline data collection (buried archaeology, historical farms, intangible heritage, particularly for certain minority ethnic groups);
- Identifying local users of water; and
- Conducting a social baseline and census and socio-economic survey for landowners and those who will be economically displaced by the Project.

Table 20.1 provides a summary of the environmental and social studies already completed for the Project and those planned in the coming 12 to 18 months. Baseline studies completed to date have provided an understanding of the key environmental and social sensitivities in the Project and wider area. The schedule of future work has been planned to meet Serbian permitting requirements, align with international good practice and to help further understand or close out environmental and social risks.

Table 20.1 – Environmental and Social Studies Completed and Planned

Environment	Work Completed	Work Planned
Groundwater and surface water	Surface water flow and quality data was collected on the Jagnjilo River, Ogašu Lu Gjori Stream and Lipa River. Surface water flow was measured monthly and water quality was measured every three months. Groundwater wells have been constructed and data collection for the Project area has commenced.	A hydrological study of watercourses, including an ongoing monitoring campaign, and details of planned water management and treatment will be submitted as part of the application for water conditions, a key stage in permitting requirements in Serbia. Fifteen (15) surface water locations will be monitored (13 on a monthly basis, two on a quarterly basis). Thirteen (13) groundwater monitoring wells will be monitored for groundwater quantity and quality on a monthly basis. DPM plans to construct two (2) further deep wells for monitoring purposes. In addition, six (6) groundwater monitoring wells in the adjacent Timok Project area will continue to be monitored on a quarterly basis. Twenty-six (26) springs will be monitored on a monthly and quarterly basis. A 3D numerical groundwater flow model for the Project area will be developed.
Terrestrial ecology - biodiversity	Habitat and species surveys were undertaken in 2024 in the Project area and a wider research area. Surveys included habitats and flora, mammals, birds, amphibians, invertebrates and reptiles. A Rapid Biodiversity Assessment of an area adjacent to the north of the Project area was undertaken in October 2019 and updated the 2013 work.	Biodiversity surveys will conclude in 2024, and a report of the study produced in Q1 2025. A biodiversity management plan will be developed which may identify requirements for additional surveys.

Environment	Work Completed	Work Planned
Aquatic ecology	Habitat and species surveys were undertaken in 2024 in the Project area and a wider research area. Surveys included phytobenthos, aquatic -invertebrates and fish,	. Further seasonal surveys for baseline covering phytobenthos, aquatic macrophytes, aquatic macro-invertebrates and fish design/development and EIA. eDNA sampling should be included as part of aquatic surveys in the project catchment area. Based on biodiversity surveys undertaken during 2024 a Biodiversity Management Plan will be developed, which may identify requirements for additional surveys. eDNA sampling should be included as part of aquatic surveys in the Project catchment area.
Air quality	Monthly baseline monitoring commenced in December 2021 to the north and west of the Project area. Monitoring included oxides of nitrogen, nitrogen dioxide, and sulphur dioxide, particulate matter (PM_{10} and $PM_{2.5}$) and dust deposition with associated metals analysis in line with good international best practice. This is part of an ongoing monitoring campaign to provide up to date information required for the EIA.	Monitoring for $PM_{2.5}$/PM_{10} is planned for three locations, monitoring for total suspended solids and gases (NO_x and SO_x) is planned for five (5) locations. Additional monitoring locations to the south and east of the project area should be included.
Noise and vibration	Summer monitoring campaign undertaken in 2022. Campaigns are 2 weeks long with attended and unattended measurements at 15 identified receptor locations over and round the project footprint. No vibration work undertaken to date.	Winter monitoring campaign planned for later Q1 2025. Additional monitoring locations to the northeast should be identified and included in future campaigns.
Social baseline	Desktop review of secondary publicly available data completed. A secondary data baseline chapter has been completed.	A socioeconomic baseline study is planned for Q4 2024 this will characterise the local context in the Social Area of Influence (AoI). The study will include desktop collection of public data and field work to gather primary data through a combination of key informant interviews and focus group discussions.

Environment	Work Completed	Work Planned
Land acquisition	An updated survey of land ownership is required.	A survey is required for land to be acquired. A census and socio-economic survey of landowners is proposed to be undertaken once the Project design has been completed to feed the development of a detailed Land Acquisition and Strategy, outlining eligibility and entitlements for compensation and livelihood restoration.
Landscape	No work undertaken to date	Observations and photographs at sensitive viewpoints. This information will be consolidated into a report which details baseline landscape and visual conditions.
Soils and geology	A project specific soil sampling programme (130 samples) has been completed over the project area. Geotechnical studies are underway, no results are currently available.	Soil descriptions and sampling for land capacity (agricultural) and chemical quality (pollution) required by Law on Soils Protection. Results of soil sampling programme expected Q4 2024.
Geochemistry	Sampling, static testing and analytical programme to characterise ARD/ML (ARD/ML) capacity of rocks to be disturbed has been conducted in 2023/2024.	Further studies on ARD/ML to be conducted during 2024/2025 on both rock and tailings.
Cultural heritage	No work undertaken to date.	Cultural heritage walkover surveys of the full project footprint are proposed. Rural and historical railway structures that are likely to be disturbed shall be recorded archeologically. Assess the need for further studies in areas to be disturbed that have high potential for Unidentified Archaeology and needs assessment for a LiDAR survey.

Source: ERM, 2024

20.6.2 Environmental Evaluation: Potential Environmental and Social Risks and Fatal Flaws

Environmental and social risks associated with the Project were identified through a risk review in August 2024, and periodically reviewed throughout the PFS. 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. Table 20.2 summarises the most important interactions between Project activities and the environment. Key aspects of the potential environmental and social risks are detailed in the sub-sections below.

20.6.3 Watercourses and Groundwater

A network of rivers and streams runs through the Project area (Dumitrov Stream, Ogašu Lu Gjori Stream, Valja Saka River and the Lipa River - Figure 20.2). 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 DTSF will be situated over the ephemeral Dumitrov Stream and an adjacent un-named stream, while the process plant, UG mine portals, 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 DTSF, the DTSF reclaim pond, the topsoil stockpile (located west of the DTSF), preliminary ore stockpile and the mine dewatering pond (Figure 18.9).

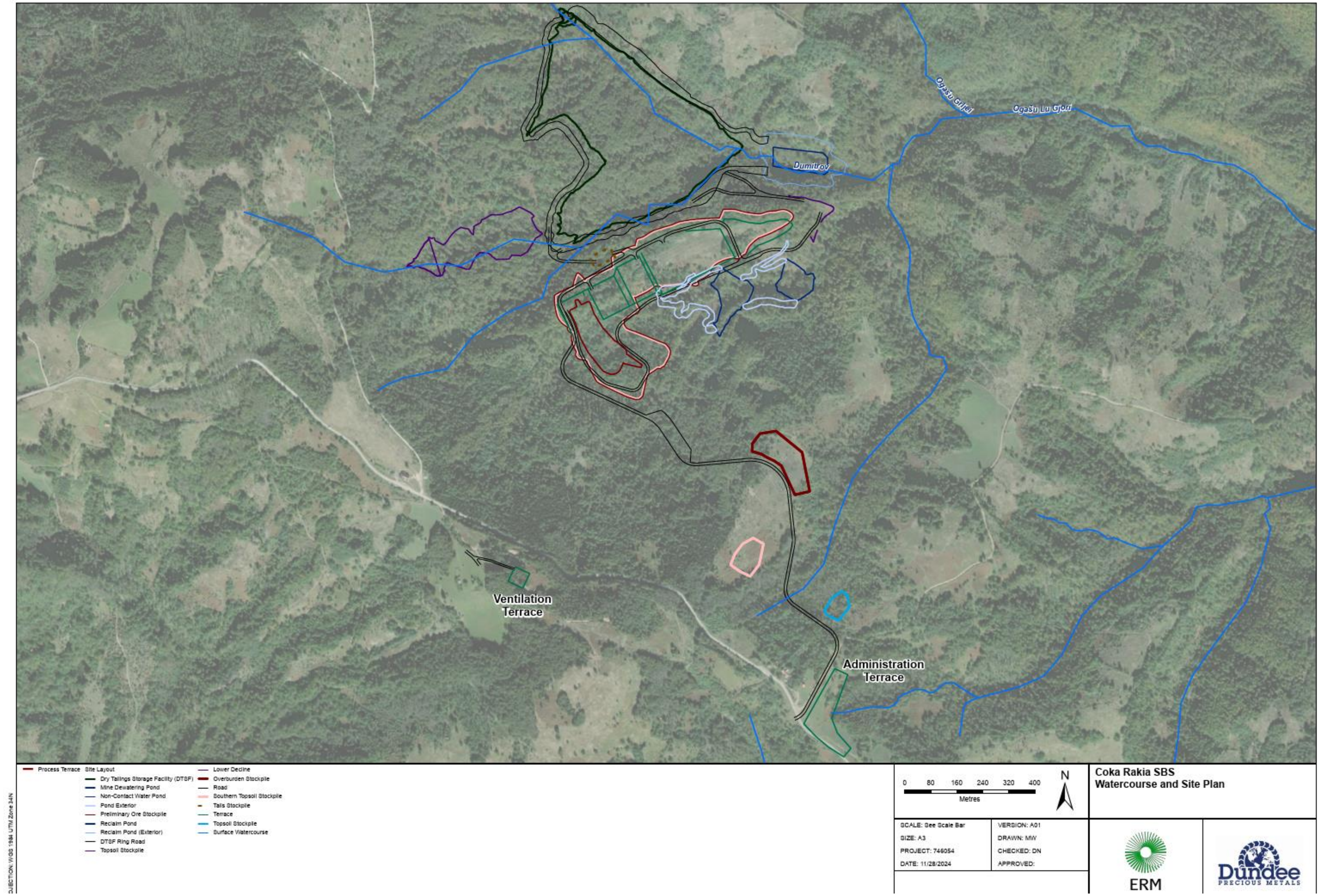
Discharge of hazardous substances into groundwater is prohibited (Law on Water, Article 97), and so the consequence of potential risks to surface water and groundwater quality are high. Events that could affect this include sedimentation, seepage of process chemicals including hydrocarbons, leachate or runoff from acid generating rock, reduction in baseflow due to underground mine dewatering, planned treated water discharges, and pollution during flood events, which typically take place during April snowmelt. Control of these risks is a design objective of the mine water management plan (Section 18.7).

Table 20.2 – Interactions Between Project Activities and Social/Environment

Item	Aspect								Receptor									
	Emissions to Air	Noise and Vibration	GHG Emissions	Water Usage	Discharges	Land Take	Light	Hazardous Materials	Air Quality	Soils and Geology	Water	Flora and Fauna	Land Use	Health and Safety	Fishing and Hunting	Livelihoods	Cultural Heritage	Landscape
Construction																		
Change in land use						✓			✓	✓	✓	✓	✓	✓	✓	✓	✓	✓
Decline construction	✓	✓	✓	✓	✓		✓	✓	✓	✓	✓	✓		✓	✓		✓	✓
Construction of infrastructure	✓	✓	✓	✓	✓		✓	✓	✓	✓	✓	✓		✓	✓		✓	✓
Road and Powerline Construction	✓	✓	✓		✓	✓		✓	✓	✓	✓	✓	✓	✓			✓	✓
Operation																		
Underground		✓		✓	✓			✓		✓	✓			✓				
Water supply				✓							✓	✓		✓	✓	✓		
Dewatering				✓	✓						✓	✓						
Discharges					✓					✓	✓	✓		✓	✓	✓		
Chemical / fuel storage								✓	✓	✓	✓	✓		✓				
Waste management	✓		✓		✓				✓	✓	✓	✓		✓		✓		✓
Employment														✓		✓		
Major accidents	✓	✓	✓		✓			✓	✓	✓	✓	✓		✓	✓	✓		
Closure																		
Removal of structures	✓	✓							✓	✓	✓			✓				
Rehabilitation	✓		✓	✓	✓					✓	✓	✓	✓	✓	✓	✓		

Source: ERM, 2024

Figure 20.2 – Drainage Map of Čoka Rakita PFS Area



Source: ERM, 2024

An underdrainage system will be designed beneath the DSTF liner allowing non-contact water to drain directly to the environment, discharge will be to the Ogašu Lu Gjori Stream.

All contact water from mine dewatering and surface runoff from process areas and the DTSF will be collected in lined ponds and ultimately directed to the ETP. Treated water will used in underground processes 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 a small part of the large Crna Reka catchment to the south. Groundwater models and a water balance are required to help identify potential impacts, and further engagement with the local community is needed to fully understand potential impacts to domestic wells.

20.6.4 BIODIVERSITY

The Project area is characterised by a high diversity of species and habitats, including some of conservation interest, which are typical for woodland habitats in the mountainous regions of Serbia where the Project is situated. Initial baseline surveys were conducted in 2024 and were undertaken by specialists from the University of Belgrade at the concession area and wider research area (shown in image below) and included surveys of terrestrial habitat, flora, fungi, aquatic surveys, invertebrates, mammals, birds, amphibians and reptiles.

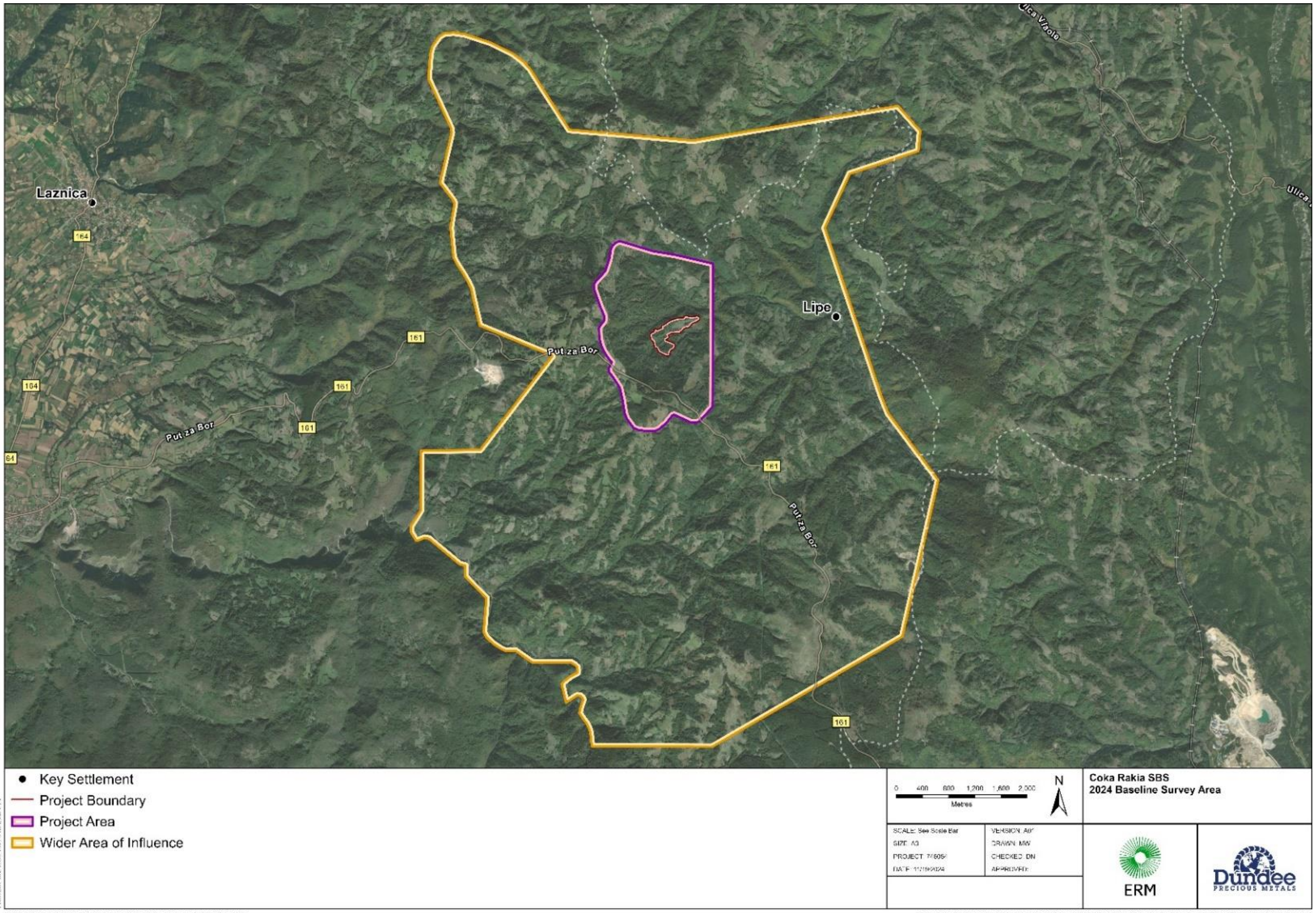
The dominant habitat found in the Čoka Rakita concession area is beech woodland with areas of herbaceous vegetation and transitional vegetation between herbaceous and forest cover. Steep sides of peaks are most often covered with a rock substrate and scanty vegetation cover. Meadow vegetation and conifer plantation habitats are also present within the surveyed area.

The 2024 baseline surveys recorded a total of 234 plant species, of which 55 are afforded a certain degree of protection. 24 of these plants are listed on national legislation as protected, 31 species are designated as Least Concern on the global IUCN Red List of Threatened Species. The species *Neottia nidus- avis* (L.) (Birds nest orchid) identified is on the Convention on International Trade in Endangered Species (CITES) list. No globally threatened species were recorded.

Two (2) species of fungi present in the area *(Boletus pinophilus* (Pine bolete) Pilat and Dermek and *Cantharellus cibarius* (chanterelle*)* Fr) are strictly protected under Serbian law. These surveys were not undertaken during the optimum period for fungi as autumn is the best time for the appearance of fruiting bodies.

Figure 20.3 illustrates the location of Project concession area and wider research area used for baseline surveys.

Figure 20.3 – 2024 Biodiversity Baseline Survey Area of Influence



Source: ERM, 2024

Aquatic surveys in the project area identified a total of 145 taxa of macroinvertebrates from 19 taxonomic groups in the study area, a significant diversity of species. The species *Epeorus youugoslavicus* (a species of slug) was recorded at the Valja Saka River, this species is listed as strictly protected under Serbian legislation. In addition, larvae of the genus *Cordulegaster* (a genus of dragonfly) were found in the Dumitrov potok, in the Ogašu Krloši and Vrkaluca Rivers and in Bigar, which include nationally protected species and those listed in Annexes II and IV of the EU habitat directive.

Aquatic surveys recorded a low abundance of fish species with only three (3) species being recorded. Two (2) of these species, bullhead (*Cottus gobio*) and the brook barbel (*Barbus balcanicus*), are protected under national legislation. During these surveys it was noted that the Lipa River water was acidic, a consequence of known mine-derived pollution from elsewhere in the Lipa catchment.

Aquatic surveys also identified the presence of *austropotamobius torrentium* a species of freshwater crayfish strictly protected by Serbian law and whose population status is declining by the International Union for Conservation of Nature Red List of Threatened Species. The crayfish is also listed in Annex II of the EU Habitats Directive (Council of Europe, 1992).

Surveys identified the presence of 64 species of butterfly. Among the identified species, seven are protected under national legislation. Four species identified are listed in Annex II and/or Annex IV. Surveys for dragonflies and damselflies recorded 12 taxa in the study area, while the presence of at least two more taxa is expected. One species is a protected species in Serbia, listed in Annexes II and IV of the EU Habitats Directive.

Surveys identified 38 species of longhorn beetles. More than 60% of the species recorded are of international or national importance. Three species recorded are listed as vulnerable and included in Annex IV of the Habitats Directive or Near Threatened on the IUCN Red List.

Bird surveys were undertaken in spring 2024. The surveys recorded a total of 60 species and of these 48 are strictly protected under Serbian law. One (1) species identified is listed as Vulnerable on the IUCN Red List *Steptopelia turtur* (European turtle dove).

The field surveys identified six (6) species of amphibians and six (6) species of reptiles of which five (5) amphibians and three (3) reptiles are strictly protected under Serbian law. One (1) species of amphibian is listed as vulnerable on the IUCN Red List and another species is listed in Annex II and IV on the Habitats directive. Both these species were recorded on the edge of the concession area. Four (4) species of reptile are listed on Annex IV of the Habitats Directive, only one (1) of which was recorded within the concession area.

All bats in Serbia are strictly protected. All 15 species recorded are listed in Annex IV of the Habitat Directive and seven (7) of these are also listed in Annex II of the Habitat Directive. Two (2) species are Near Threatened on the IUCN Red List. One species is listed as vulnerable on the IUCN Red List. Winter season (hibernation period) surveys have not yet been undertaken, but it is known from literature that the Radanova pećina cave (outside the project area) serves as a hibernation roost for at least two (2) bat species.

Field surveys identified 16 species of mammals (excluding bats) including three (3) species listed in Annex IV of the Habitats Directive: the Hazel dormouse (*Muscardinus avellanarius*), Grey wolf (*Canis lupus*) and European otter (*Lutra lutra*), the latter is also listed as near threatened on the IUCN Red List.

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.

Following baseline surveys undertaken in 2024, the presence of IUCN Red List Vulnerable species and species listed in Annex II and Annex IV of the Habitat Directive meets the current criteria for 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.

A number of nationally protected habitats and species were also identified, and these will require detail mitigation plans to avoid, reduce and restore impacts in line with the mitigation hierarchy approach and best practice guidelines.

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.

20.6.5 Noise, Vibration, Nuisance Dust and Greenhouse Gas Emissions

Noise

Baseline noise surveys began in 2022, two (2) seasonal campaigns were planned, and the summer survey was completed in 2022. Both campaigns were 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 in early 2025. Additional monitoring locations to the northeast of the Project area should be identified and included in future campaigns.

Vibration

Baseline vibration surveys are yet to take place, these should be planned for the next phase of the project.

Dust

Historically, the main source of industrial air pollution in area is the Bor smelter and mining complex, some 20 km to the southeast. The baseline environment in the region is also known to have naturally elevated concentrations of arsenic and cadmium, introduced through airfall dust from smelting and mining operations. The Project will generate noise and dust from construction and operational activities and in closure. This may affect the people living permanently or seasonally in the area, as well as the flora and fauna. A full assessment of noise, vibration and dust will be required as part of the EIA which will also need to take account of the existing background levels of arsenic, cadmium, and other historical pollution.

Air

Baseline air quality monitoring commenced in December 2021 at 11 locations in the adjacent area (associated with the Timok project) to monitor PM_{10} and airborne metals at six (6) locations and $PM_{2.5}$ 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.

Greenhouse Gas Emissions

The Project will generate greenhouse gas emissions that will need to be measured and managed in line with international good practice.

20.6.6 SOILS AND GEOLOGY

A soil sampling programme to assess the soil in the context of the baseline of agricultural capability and chemical quality was undertaken in Q3 2024, results from the sampling program will be provided in Q4 2024. Consistent with the PFS planned layout, where topsoils are to be disturbed, the principle will be to conserve these in stockpiles for ultimate reuse as a rehabilitation medium. For subsoils and bedrock that will be disturbed, the potential for Acid Rock Drainage/Metal Leaching (ARD/ML) has been assessed in a comprehensive screening program using industry-standard static testing. The results of the geochemical assessment are discussed further in Section 20.8.1.

Geotechnical studies are on-going, it is noted that many of the Project infrastructure locations, especially access roads and the DTSF, are on very steep terrain. Risks from landslides and other ground instability will need to be assessed by detailed geotechnical investigations in the future.

Detailed geological mapping by DPM and others has been available for the PFS, providing high resolution understanding of the lithologies and structural geology underlying the different components of the Project area.

20.6.7 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. Physical remains of past human activity on the site exist in the form of historical buildings, typically farmsteads that were seasonally or permanently occupied, and the wider historical landscape. No traces of buried archaeological remains have yet been found within the development area, although sites of ancient settlement are known in the wider region.

In terms of intangible heritage, the Project area is important to the Vlach community, an ethnic community in Serbia with its own language, dress, and culture. Religious customs are connected with the land, including beliefs in sprits of the woods (e.g., fairies) and celebration of sacred trees. In addition, 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. As material remains of a disappearing way of life, using the definitions set out in EBRD standards, these physical remains can also be considered tangible cultural heritage resources. Most of them have seen significant modernisation or rebuilding in recent decades.

The route of the historic Žagubica-Bor railway crosses the Project area. This was built (but never finished) by forced labour during the Second World War German occupation. This narrow-gauge line, preserved in cuttings, embankments and other concrete structures, runs along a meandering route to the west and south of the DTSF, 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 internment camps (each holding between 300-500 prisoners) located at various points along the route of the line, and at Bor. The railway, and the location of the labour camps, has international historical significance, notably to those communities whose ancestors were held there and forced to build the scheme. Small sections of the railway are likely to be disturbed by the Project infrastructure. The Project is also located close to the site of two (2) of the labour camps (one to the north called Westfalen-altabor, and one to the south called Tirol-altabor), whose exact location have yet to be determined.

No detailed cultural heritage baseline studies have been undertaken in the Project area to date. Cultural heritage walkover surveys of the full Project footprint should be undertaken including additional mapping to identify the location and extent of the Žagubica-Bor railway line and the internment camps. Rural and historical railway structures that are likely to be disturbed shall be recorded archeologically. Further assessment of the need for exploratory field studies in areas to be disturbed that have high potential for unidentified archaeology as part of the environmental and social impact assessment (e.g., flatter areas, close to water with direct sunlight, where human settlement is more likely to occur) are suggested. A LiDAR survey could also provide valuable insights in these areas.

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 will be recorded archaeologically.

20.6.8 Social Setting, Land Acquisition, and Livelihoods

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 community and relevant local government have expressed concern associated with economic displacement, particularly for those who seasonally utilise the Project area for summer season agriculture and grazing.

The Project plans to acquire land from local landowners on a willing-buyer willing-seller basis. At present, the Project footprint largely includes forests and smaller areas of summer grazing pastures and does not require resettlement of households. In cases where voluntary agreements cannot be reached, the Law on Expropriation allows government the right to acquire immovable property for projects that are demonstrated to be in the public interest.

The Law on Mining and Geological Explorations allows holders of rights to mineral explorations to be beneficiaries of an expropriation process. Expropriation for the development of mining projects is widely understood to be an application of the 'public interest' requirement though the process has yet to be tested for private sector mineral projects in Serbia.

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.

20.7 Social and Community Engagement

The Project will be subject to scrutiny by regulatory authorities and other stakeholders during the permitting process. There will be six (6) formal hearing and consultation periods included within the spatial plan, strategic environmental assessment and environmental impact assessment processes and a range of other points where the public and other interested parties could comment. The Project will supplement these with dedicated stakeholder engagement activities.

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.

Engagement activities include:

- Direct engagement with local community members on an individual basis;
- Engagement with community groups (schools, hunting groups, local institutions, and organisations);
- Engagement with regulators and local representatives;
- Corporate Social Responsibility actions including provision of flood relief and a medical vehicle, donations to local community cultural, sporting, and educational events and specific donations of equipment to the health sector; and
- A telephone survey of attitudes towards mining was undertaken in summer 2020.

DPM has also undertaken work to understand the likely economic impacts of the Project (Egzakta Advisory, June 2020), including contribution to direct and indirect employment, national economy through royalties and indirect impacts through the wider supply chain and on local and national businesses. Potential issues raised so far by various stakeholders include:

- Challenges associated with land acquisition and economic displacement. Currently a voluntary willing-buyer willing-seller approach is being pursued.
- An identified need for local employment opportunities, particularly within the Žagubica area, and across Eastern Serbia broadly where levels of unemployment are high.

- Environmental impacts, particularly associated with air pollution, soil contamination, access to safe water sources, increased landslide and flood risks. These are based on experiences regionally and in the wider area from the long history of mining.
- Compliance with local and international permitting processes, including the requirements for independent consultation with interested and affected parties.
- The need for corporate social responsibility initiatives/donations to be provided to directly affected communities, with relevant priority investment areas clearly identified.
- The need to respect human and labour rights.

It is likely that the potential presence of IUCN Red List Vulnerable species, and species listed on Annex IV of the Habitat Directive are present in the Project area, and if so, will meet the current criteria for EBRD priority biodiversity features and critical habitat. In this case, these will require an ecosystem services assessment to understand the potential social impact of the effects of the project on the critical habitat, and the development of dedicated social management plans to mitigate them. These assessments and management plans will be undertaken as part of the ESIA process.

A database of stakeholder contacts, engagement and feedback will be kept updated at all times. In terms of stakeholder feedback, any risks and opportunities will also be identified and recorded on the Stakeholder Register, for ongoing visibility and dissemination across the project team.

20.8 Mineral and Non-Mineral Wastes

20.8.1 Mineral Wastes

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. Waste rock, conservatively assumed to be PAG, excavated during initial mining operations, mainly from advancing the two declines, will be stored within the base of the DTSF within embankments constructed of NAG waste rock. The embankments will be composite lined to inhibit seepage of contaminated water, PAG waste rock will be emplaced prior to NAG waste rock and a liner will be paced between the two waste rock types. All PAG waste rock will be emplaced within the initial two years of mine development and prior to the placement of filtered tailings in stacks above the PAG waste rock.

To mitigate against the risk of the acid rock drainage the PAG waste rock will be saturated in perpetuity with a layer of GCL between the PAG waste rock and filtered tailings. This layer will inhibit seepage from filtered tailings from entering the PAG saturated zone, and potential regeneration in the case of basal liner failure. After two years of operations all subsequent waste rock will be crushed and returned underground within paste backfill.

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).

Samples were selected to be representative for each of those lithologies with respect to the Ca:S ratio and As, Sb, Cu, Pb and Zn contents as determined by four-acid ICP-MS drill-core assays. The 29 samples underwent static testing for the determination of acid potential and neutralisation (acid-base accounting – ABA; EN 15875), leachability tests (EN 12457-2), toxicity tests (EPA 1311) and mineralogical characterisation via semi-quantitative X-ray diffraction (XRD) analysis, and elemental analysis (C, S, As, Cd, Co, Cr, Cu, Hg, Ni, Pb, V, Zn) at the Mining and Metallurgy Institute in Bor, Serbia.

Based on the standard assumption that most of the Ca and S are hosted by carbonates and sulphides, respectively, DPM used the Ca:S ratio as a proxy for the neutralisation potential ratio (NPR) to pre-determine ML/ARD risk in the excavated rock. This indicated a potential risk for ML/ARD in the diorite samples due to low carbonate contents.

Most sandstone and marl samples were anticipated to be unlikely acid generating, whereas acid generation of the epiclastics was anticipated to be more variable, dependent on the sulphide content and secondary carbonates. This has been partially confirmed by the other ML/ARD test. The tested diorite and epiclastics samples should largely be classified as potentially acid generating (PAG) based on low NPR and net neutralisation potential (NNP) values as well as their elevated sulphur contents (>0.5%).

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.

While rock composition may be used to estimate ML/ARD characteristics of rocks at Čoka Rakita at this stage, additional experiments and data analysis provide further insight. The total carbon content for the 29 composite samples strongly correlates to the experimentally determined NP, while correlation of the Ca content and neutralising potential (NP) is much weaker. This is because of the presence of non-carbonate Ca in the epiclastics (SFD) and intrusives (EMPO), as well as in skarn-altered sedimentary rocks. Furthermore, the upper detection limit for Ca of 15% in the routine multi-element assays leads to an under-estimation of Ca in rocks with Ca contents larger than 15%. This highlights the limitations of Ca:S ratio as a reliable predictor of NPR and that implies that additional data may be needed for robust spatial predictive modelling of ARD characteristics of excavated rocks at Čoka Rakita.

A more accurate proxy to predict ARD potential of excavated rock may be developed from the available multi-element data, but these preliminary ARD testing results show that carbon data would be a useful additional input for estimating the NP of rocks at Čoka Rakita. DPM is considering the routine analysis of total carbon content of drill core samples.

At present, the ML/ARD potential of one composite sample of flotation tailings has undergone static testing and mineralogical analysis. The sample was determined to be non-PAG, attributable to considerable calcite content and no detectable sulphide content. In addition to that, cemented tailings that are to be backfilled are expected to have increased neutralisation potential due to the nature of the cement. However, further variability testing of tailings samples will be considered.

The currently available data provide initial insight that a proportion of the rock disturbed during mine development and operation will likely be classified as PAG or near-PAG. Additional static tests of rock samples will be part of the ongoing geochemical baseline program to fully characterise these materials and compose an appropriate ARD management plan. This will include the evaluation of the approximate volumes and relative proportions of each lithology that are likely to be disturbed by the mine development and base the total number of samples on those volumes. In addition to geological logging, geochemical data will be leveraged to distinguish rock types. The latter may especially help to discriminate PAG and non-PAG material within the S1/S2 unit.

A representative set of samples of rock and tailings materials will be subject to long-term kinetic testing. Materials to be used for construction will undergo humidity cell tests to determine geochemical behaviour under atmospheric conditions. Column tests to determine geochemical behaviour of those 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.

20.8.2 Non-Mineral Wastes

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.

20.9 Mine Closure, Aftercare and Remediation

A Conceptual Closure Plan has been completed for the Project (ERM, November 2024) that reviews and defines the closure objectives, vision, options, and activities of the Project in line with the International Council for Mining and Minerals (ICMM) Integrated Mine Closure: A Good Practice Guide.

The closure vision at Čoka Rakita must consider the limitation and minimisation of physical and biological impacts through mine design and in operations; engaging stakeholders throughout baseline studies, project life and mine closure; providing a beneficial post-closure land use and leaving a positive economic, environmental and social legacy for the region through efficient project development, operation and closure.

The preliminary closure objectives have been defined within the categories of Regulatory Compliance, Physical Stability, Geochemical Stability, Enhancing Biodiversity, Water Quality, Stakeholder Engagement and Optimisation of Financial Spending. These objectives will be further revised and aligned as the Project progresses and evolves.

The closure base case for post-closure land use will be to rehabilitate the mine site so that it is physically and chemically stable and compatible with the intended future land use, which includes restoring the site as far as practicable to pre-mining land use and status. Some of the options that might be considered as part of a post-mining land use include (but are not limited to); tourism, renewables, forestry, agriculture and mixed use.

The Conceptual Closure Plan defines the activities for a base case for the underground mine, DTSF, processing facility, water management infrastructure, miscellaneous infrastructure and roads. This includes an assessment of the closure risks, closure design criteria and future opportunities post-closure. All of these closure activities that have been defined are driven to align with the Project closure vision and objectives. Progressive closure options are also considered to rehabilitate mine hazards throughout the Project's lifespan with the aim of reducing the post-closure period and financial liability. At this stage, it is expected that progressive closure will be implemented at Čoka Rakita, with particular focus on the DTSF.

The Project's PEA estimated the closure and rehabilitation costs at US$28 M, based on Q1 2024 pricing. The estimate is within the range -30% to +60%. It is suggested that this cost should be revised in the following project stages once further assumptions can be made on the planned closure activities and success criteria.

Closure planning is a continuously evolving process involving regular review and planning during every stage of the Project life cycle. The approach to iterative closure planning, includes:

- PFS – PFS Conceptual Closure Plan;
- FS – FS Conceptual Closure Plan;

- ESIA – Conceptual Closure Plan;
- Operational Closure Plan Updates – Every 2 years;
- Detailed Closure Plan – 5 years prior to / from closure;
- Closure Execution Plan – 2 years prior to / from closure; and
- Mine Closure Completion Report – Once all proposed Closure Criteria are met.

The Project will advance closure planning and costing exercises throughout the planning stages and operational life, with the next revision at the FS and EIA stage.

20.10 Environmental, Social and Health and Safety Plans and Commitments

20.10.1 Environmental and Social Management Plans and Programs

In addition to the baseline data gathering listed in Table 20.2, the Project will continue to develop and enhance the following plans and programs:

- A Permitting Plan;
- A Stakeholder Engagement Plan and Consultation Strategy;
- A Mine Closure Plan;
- Environmental Design Criteria;
- An Environmental Protection Framework Plan including Monitoring Plan;
- A Change Management Procedure;
- An Occupational Health and Safety Plan; and
- An Emergency Response and Contingency Plan.

20.10.2 Environmental and Social Commitments

The Project has created a commitment register which will be updated as the Project progresses.

20.10.3 Health and Safety

An outline Occupational Health and Safety Management Plan (OHSMP) has been drafted as part of the PEA. The OHSMP includes a review of applicable legislation and is designed in line with the following Serbian requirements:

- ORSG 35/2023;
- ORSG 72/06, 84/06 corr., 30/10 and 102/15;
- ORSG 16/2018 and 5/2022;
- ORSG 111/2013, 57/2014, 126/2014, 111/2015, 113/2017 – oth., Law and 11/2019;
- ORSG 60/06,

- ORSG 111/13,
- ORSG 62/07, 102/15,
- ORSG 120/07, 93/08, 53/17; ORSG 72/06, 84/06 – corr., 4/16, 106/18, 14/2019;
- ORSG 15/2023;
- ORSG 23/09, 123/12, 102/15, 101/18 and 130/21;
- ORSG 14/09, 95/10, 98/18, 35/23;
- ORSG 21/09, 1/19;
- ORSG 92/08, 101/18;
- ORSG 106/09;
- ORSG 106/09, 93/13, 86/2019;
- ORSG 108/15, 106/09, 1117/17, 107/21;
- ORSG 61/10, 65/10, 159/2020;
- ORSG 101/12. 12/13;
- ORSG 96/10, 115/20;
- ORSG 93/11, 86/19;
- ORSG 96/11, 78/15, 93/19;
- ORSG 109/16;
- ORSG 96/11, 117/17;
- ORSG 94/20;
- ORSG 102/16;
- ORSG 102/16;
- ORSG 111/15, 130/21;
- ORSG 102/12, 29/13, 130/21; and ORSG 96/10, 115/20),
- International best practice (e.g., IFC Environmental, Health and Safety Guidelines and ISO45001, and
- DPM Internal Standards (Health and Safety Policy, Four Concepts and Health and Safety Model).

The OHSMP describes how occupational health and safety (H&S) will be managed during all stages of the Projects development to avoid and appropriately manage risks that could potentially affect Project personnel. Community safety, health and welfare is also considered where communities are envisaged to be affected by Project activities.

The OSHMP will:

- Provide a safety and health assessment and plan prior to commencement of scheduled work activities performed by DPM and contractor workforces within DPM properties.
- Provide a preliminary assessment and review with respect to the proposed Project construction activities within an operating area, and their effect on the operations personnel, within that area.
- Provide a preliminary assessment and review with respect to the facility operating activities, and their effect on construction personnel and related activities.
- Identify Serbian legal and best practice requirements relating to H&S management, which are applicable to the Project.
- Describe how this framework will continue to develop, in line with Project progress.

DPM has provided the appropriate levels of oversight through regular communications, project meetings and participation in risk assessment exercises. Serbian Law on Mining and Geological Exploration sets out the roles and responsibilities of a number of key personnel, during the operational phase DPM will appoint individuals to these roles to ensure ongoing appropriate management of H&S.

- Serbian regulations (Article 129 of the Law on Mining and Geological Exploration) require DPM to:
  - Arrange occupational H&S in accordance with the specificities and dangers that may arise.
  - Organise occupational H&S related activities in accordance with this Law and regulations governing the occupational H&S.
  - Provide personal protective agents and personal protective equipment for the employees.
  - Provide protection against fire, damages, accidents, and chemical and other accidents, and organise rescue operations.
  - Organise training for workers in the field of occupational H&S and rescue operations in the case of sudden danger threatening the lives of people and safety of the facilities, in accordance with the established schedule and program throughout the year and check personnel knowledge once a year.
- The following measures will be implemented to ensure on-going management of H&S risks:
  - A project wide risk assessment to develop a preliminary project risk register.
  - Day to day risk assessments undertaken as part of managing construction and operational risks and changes to the work environment.

20.11 Environmental Studies, Permitting and Social or Community Impact

DPM has undertaken permitting efforts, environmental studies and community engagement within the wider Project area since 2007. Environmental and social baseline work commenced in 2012 in the wider project area and continues to the present day, with a more extensive baseline collection programme planned.

Environmental and social factors have been considered in the assessment of the various alternatives, trade-off and options studies in the PFS design. Key environmental and social mitigation features include:

- Full design compliance with the Serbian regulatory requirements, within the context of the EIA permitting framework.
- 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, testing and treatment to the required permit limits prior to discharge and managed discharge flow rates.
- Critical habitat assessment and the development of biodiversity management and action plans, as well as ecosystems services assessment.

20.11.1 Key Risks

Key environmental and social risks are similar to those associated with other gold mining projects and as a result measures are required to safeguard rivers, groundwater, biodiversity, the local community and local heritage and mitigate permanent effects.

- The Spatial Planning process in Serbia is currently pending and there are risks to the project from other 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.
- During operations, streams and small rivers may be affected by dewatering, diversions and discharges, and permanent infrastructure will overlie several hundred metres of mostly ephemeral stream bed within the Dumitru and adjacent tributary valley in the upper headwaters of the Lipa tributaries.
- The Project may affect protected aquatic species, such as the listed and protected crayfish, requiring careful consideration as to the location of drainage outfalls and their discharge rates that will affect flow velocity. Groundwater models and water balance will be developed for the Project, and this will help to identify potential impacts and inform the design of the contact water discharge location, treatment and water quality monitoring system to minimise the risk of surface water and groundwater pollution and unacceptable flow velocities.
- Discharge of hazardous substances into groundwater is prohibited, and so consequence of potential risks to surface water and groundwater quality are high. Watercourses in this area drain to River Danube, which is internationally protected. Events that could affect this include sedimentation, seepage of process chemicals including hydrocarbons, leachate or runoff from

acid generating rock, escape of leachate from the saturated PAG waste rock area at the base of the DTSF, reduction in baseflow due to underground dewatering, planned discharges, and pollution during extreme flood events. Control of these risks is an objective of the in the Mine Water Management Plan (Section 18.7) which includes measures to avoid or mitigate these key risks. The Mine Water Management Plan (Section 18.7) includes measures to avoid or mitigate these key risks and includes embedded design mitigations such as non-contact water diversion, liners and secondary containment, contact water treatment and recycling, with discharge only of excess treated water to the environment.

- The Project may result in potential impact to habitat, including several hundred metres of mostly ephemeral riverine habitat, which may have an impact on biodiversity. This setting is considered likely to fall within the EBRD definition of critical habitats and if so, will require further assessment and development of management plans, such as a critical habitat assessment, biodiversity action and management plans and an offset strategy. This would also require an ecosystem services assessment to understand the potential social impact of the effects of the Project on the critical habitat, and the development of dedicated social management plans to mitigate them. These assessments and management plans would be undertaken as part of the EIA process.
- Dewatering could affect a potential source of municipal drinking water and water sources used seasonally for cattle. Further engagement with the local community is needed to fully understand potential impacts to domestic wells. These potential impacts and associated mitigation measures will be addressed through the EIA and engagement with regulators.
- There are geotechnical risks associated with the steep terrain in the Project area, a landslide would have the potential to damage the DTSF and water management infrastructure.
- The presence of the historic World War 2 German railway line and forced labour camps and other unidentified cultural heritage sites of importance pose a risk and should be further investigated.
- There are also risks associated with acquiring land, for which careful planning will be required. Acquisition through a voluntary willing-buyer willing-seller approach appears possible. Resorting to involuntary resettlement is permitted under Serbian Law but would have an effect on stakeholder relations.
- Access for seasonal farming, hunting and tourist amenity in the Project area will also be restricted lost during operations and certain cultural heritage buildings and features will be affected by the Project. This will be addressed through the Livelihood Restoration Plan and EIA. Closure planning could include the means to address some of these changes after operations cease.
- The short LOM, impacts to the environment as a result of mining, and expectations around employment may be key issues amongst stakeholders. The Project's social license to operate is of critical importance. DPM has built a good relationship with the local community over many

years. The key risks and mitigations for these different aspects that will be more completely covered in the Project EIA and Livelihood Restoration Plan to be undertaken in the near future, for which several baseline studies are already well advanced. A Project-specific Environmental and Social Management Plan will be developed in line with DPM's overarching environmental management system, to allow these mitigating procedures to be implemented as the Project goes forward to construction.

21 CAPITAL AND OPERATING COSTS

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

DRA under the supervision of QP, Ian Major, P. Eng., developed the Capex and Opex for the process plant, plant infrastructure and on-site infrastructure for the Project scope described in this Report. All contributors to the Capex and Opex are outlined in Table 21.1.

Table 21.1 – Capex/Opex Responsibilities

Contributor	Scope
DRA	Overall responsibility for Capex/Opex, Process Plant, Infrastructure, On-site infrastructure
WSP	Prepared underground mining and mining infrastructure direct costs for initial and sustaining Capex and Opex. Prepared initial direct Capex for Filter and Paste Plant Equipment; where DRA estimated bulk earthworks, concrete, structural steel, architectural, and electrical requirements. Prepared construction quantities for DTSF, Water Management and topsoil / over-burden / waste rock stockpiles; and DRA applied rates to establish direct capital costs.
DPM	Provided Owner's General and Administration Costs, Taxes, Duties; and Closure Costs

Ranges could exceed those shown if there are unusual risks.

The Capex was developed to deliver an overall accuracy range of -20% to +30%. Ranges could exceed those shown, if there are unusual risks.

In this particular case, the Capex reflects an EPCM-type execution model. Although some individual elements of the Capex may not achieve the target level of accuracy, the overall Capex should fall within the parameters of the intended accuracy.

All Capex and Opex costs are expressed in United States Dollars (US$ or $) and are based on Q4 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 dry tailings storage expansion. Amounts for the mine closure and rehabilitation of the site have been estimated as well.

Table 21.2 presents a summary of the Initial Capex, and Sustaining Capex distributed over the LOM indicated separately. Owner's costs, contingencies and risk amounts are included in this Capex.

Table 21.2 – Capital Cost Summary

Area	Description	Initial Capex (US$ M)	Sustaining Capex (US$ M)	Total (US$ M)
2000	Underground Mine	85.01	21.43	106.44
3000	Ore Handling	13.57	0.00	13.57
4000	Processing Plant	51.95	0.00	51.95
5000	Filtered Tailings / Water Treatment Facilities	34.49	7.60	42.09
6000	On-Site Infrastructure, Site-Wide General	54.66	0.00	54.66
7000	Incoming Power	1.42	0.00	1.42
8000	Operational Readiness	28.07	0.00	28.07
9000	Indirect Costs	46.38	0.00	46.38
9100	Owner's Costs	13.65	0.00	13.65
9900	Project Contingency	49.83	0.00	49.83
9900	Project Closure and Rehabilitation[1]	0.00	27.30	27.30
	Total Major Area Capex	**379.03**	**56.33**	**435.36**

Numbers may not sum precisely due to rounding.
1. Closure costs include the non-recoverable VAT of approximately $2.3 M.

21.1.2 Sustaining Capital

The Sustaining Capex commences upon production of concentrate and continues throughout the mine life. The Sustaining costs are included in the overall Project Capex.

These sustaining capital costs cover several areas, including mining and the DTSF as itemised in Table 21.2.

21.1.3 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 cost have been included in the sustaining capital. This includes the non-recoverable VAT of approximately $2.3 M.

21.1.4 Major Assumptions

The Capex is based on the following key assumptions:

- All relevant permits in a timely manner to meet the Project schedule.
- Quotes from Vendors for equipment and materials to be 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 twenty-eight (28) month construction and commissioning period from receipt of construction permit, including UG mine decline and infrastructure development followed by a 3-month process plant ramp-up.

21.1.5 Major Exclusions

The following items were 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;
- Insurance, bonding, permits and legal costs;
- Schedule recovery or acceleration; and
- Cost of financing, Property taxes, corporate and mining taxes, duties; and salvage values. However, they are considered in the Economic Analysis.
- There will be no construction camp and catering in this Capex. It is assumed that non-local contractors will source local accommodation. It is further assumed that accommodation is available in the surrounding areas for the EPCM personnel and vendor supervisors.

21.1.6 CURRENCIES

The base currency for the PFS is United States Dollars (US$). The currency exchange rates were determined by DPM and included in the Capex and Opex as shown 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 US$
US$	United States Dollar	1.0000	1.0000
CAD	Canadian Dollar	0.7143	1.3999
EUR	Euro	1.0583	0.9448
GBP	British Pound	1.2664	0.7895
AUD	Australian Dollar	0.6513	1.5314
ZAR	South African Rand	0.0555	18.0180
RSD	Serbian Dinar	0.0090	110.2927
BGN	Bulgarian Lev	0.5411	1.8479

* Not all presented currency conversions are necessarily used in the Capex
Source: OANDA Exchange Rate, 2024

21.1.7 UNDERGROUND MINE CAPEX

21.1.7.1 Summary of Estimated Costs

Table 21.4 presents WSP's Capex for the underground mine in United States Dollars (US$). The estimate's accuracy level aligns with DPM's required accuracy as stated in the introduction of this Section 21.

Table 21.4 – Underground Mine Capex

Description	Year-3 (US$ M)	Year -2 (US$ M)	Year -1 (US$ M)	Total (US$ M)
Personnel	0.0	6.20	6.45	**12.64**
Equipment Operation	0.0	3.35	5.00	**8.35**
Materials and Services	0.0	9.23	12.08	**21.31**
Capital Acquisitions	12.5	13.17	17.16	**42.86**
Total Capex	**12.5**	**31.94**	**40.69**	**85.16**

Numbers may not add due to rounding.

21.1.7.2 *Basis for the Capex for the Underground Mine*

The database of unit cost inputs for the estimate were derived from the following sources:

- Quotations from suppliers for new equipment.
- Costs from DPM's Chelopech Mine.
- Cost data from other mines and projects.
- Estimates based on experience from other projects.

Capex 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.

21.1.7.3 *Assumptions*

The following assumptions have been made in the preparation of the cost estimate for the underground mine:

- Quotations obtained from vendors for equipment and materials are provisional and are intended solely for budgetary purposes. WSP has reviewed and have deemed reasonable for the PFS.
- Tailings will be suitable for preparing paste backfill with the required quality and assumed cost.
- Waste rock generated from development will be suitable for preparing cemented rock 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 (ORP) will ensure that there are sufficient costs and list the required skills to start and operate the facility.
- Mining, mobile and other equipment reaching the end of their useful life will be replaced with new units; consequently, the cost estimate does not include overhauls.
- Decommissioned equipment has no salvage value.
- Except for activities related to raise boring, DPM employees carry out all tasks within the underground mine, including service hole drilling and casing, 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.

21.1.7.4 *Exclusions and Exceptions*

The mine capital estimate is a direct cost estimate and as such the following items are not included in the Capex for the underground mine as well as those stated in Section 21.1.5:

- Indirect Costs.
- Operational Readiness.
- Any work outside the defined battery limits for the underground mine.

21.1.8 Process Plant and Associated Infrastructure

21.1.8.1 *Quantity Development Basis*

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.
- Mechanical Equipment List (MEL).
- Electrical Equipment List (EEL).
- Process Flow Diagrams (PFDs).
- Preliminary Process Control Diagrams (PCDs).
- Preliminary Piping and instrumentation Diagrams (P&IDs).
- 3D Model.
- Layout Drawings.
- Basic Electrical Single Line Diagram.
- Sketches.

21.1.8.2 *Civil and Bulk Earthworks*

DRA has developed the MTOs for civil works using the following methodology. It should be noted that all available drilling information has been considered for assumptions in the determination of types of soil and depth of rock from surface. A percentage of rock excavation was applied to the MTOs to cover unexpected depth of encountered rock. The civil work assumes levelling the appropriate areas and does not include detailed excavation and backfill necessary for concrete foundations.

- Quantities of clearing and grubbing were established in m² from the site plot plans.
- Quantities for mass earthworks were developed using topographically integrated software to determine accurate quantities.

- Earthworks' take-offs were calculated using neat quantities, with no allowance for swell or compaction of materials. Industry-standard allowances for swell and compaction were incorporated into the quantities as depicted in the growth allowance shown in Table 21.5.
- Earthwork quantities for the ground water systems and tailings containment areas were developed by WSP based on site plot plans, contour development and Lidar.
- A cross section of the in-plant roads and the plant layout drawing were used to develop road quantities.
- Proper drainage was taken into account.
- Rock excavation for some areas.
- Underground piping for sanitary, fire water, tap water and fresh waters was quantified with material take-offs and priced using rates from suppliers and fabricators of piping systems.

The reference documents for the civil take-offs are the following:

- Surveying reports for area.
- Drawings and sketches.
- Previous geotechnical study, if any.
- Other applicable information.

The pricing of the site preparation and bulk earthworks quantities was based on contractor rates from four (4) contractors. The pricing included the direct and indirect costs, including supply of labour and materials, equipment costs, accommodation of contractor's personnel, temporary site shops, offices, and warehousing. Mobilisation and demobilisation costs are included in the Capex indirect costs.

Budget quotations were received from the following contractors all located in the surrounding areas. The quoted prices were all within 10% of each other.

- Belosavac.
- Matex.
- MVM.
- SET.

21.1.8.3 Concrete and Detail Earthworks

DRA developed the MTOs for concrete using the following methodology. The concrete work assumed detailed excavation, backfill and all concrete works, embedded steel, and concrete finishes.

- The existing layout drawings and MTOs for Ada Tepe Crushing and Grinding circuits formed the basis for the process areas, with adjustment to site conditions.

- For all other concrete, the preliminary design sketches were used to develop concrete quantities for new areas.
- Quantities were segregated into the following cast-in-place categories (DPM classification codes):
  - Building foundations.
  - Equipment base and piers.
  - Walls (categories for 0.3 to 2 m and higher than 2 m).
  - Elevated slabs.
  - Slab on grade.
  - Beams.
  - Lean concrete.

The pricing of the concrete quantities was based on contractor rates from two (2) contractors. The pricing included the direct and indirect costs, including supply of labour and materials, equipment costs, accommodation of contractor's personnel, temporary site shops, offices, and warehousing. Mobilisation and demobilisation costs are included in the Capex indirect costs.

Budget quotations were received from the following contractors all located in the surrounding areas. The quoted prices were all within 10% of each other.

- MVM.
- SET.

21.1.8.4 Structural Steel

DRA developed the MTOs for steelwork using the following methodology. The steelwork assumed building steel, support structures, platforms, handrailing, stairs, crane columns and rails.

- The existing layout drawings and MTOs for Ada Tepe, where appropriate, formed the basis for similar areas.
- Where possible, existing Ada Tepe steel members will be dismantled, refurbished and relocated to the Čoka Rakita Project site.
- New steelwork quantities will be estimated from General Arrangement and layout drawings, as well as the 3D model.
- Quantities were segregated into the following fabricated steel member categories (DPM classification codes):
  - Heavy Steel: > 75 kg/m.
  - Medium Steel: 25 – 75 kg/m.
  - Light Steel: < 25 kg/m.

- Bought out items: checker plate, grating, handrail, stairs, ladders, and floor deck.
- Miscellaneous steel such as bolts, connections, anchor bolts, shear lugs, walkways, metal plates clips, hardware, and any other material, as well as cuts from construction are considered as a percentage of the steel tonnage.

The pricing of the structural steel quantities were based on contractor rates from two (2) One (1) contractor had declined to bid. The pricing included the direct and indirect costs, including supply of labour and materials, equipment costs, accommodation of contractor's personnel, temporary site shops, offices, and warehousing. Mobilisation and demobilisation costs are included in the Capex indirect costs.

Budget quotations were received from the following contractors all located in the surrounding areas. The quoted prices were all within 10% of each other:

- Tehnopetrol.
- SET.

21.1.8.5 Architectural - Structural

The architectural work assumed insulated and uninsulated cladding and roofing, louvres, exterior windows, exterior doors and truck and vehicle doors. The prefabricated buildings were also included under this classification and include the following buildings:

- Main Administration building.
- Mine Dry building.
- Guard House(s).
- Laboratory, and Control Room.
- Pond pumphouses.

DRA developed the MTOs for architectural work using the following methodology.

- The existing layout drawings and MTOs for Ada Tepe, where appropriate, formed the basis for similar areas.
- The architectural building quantities were developed from preliminary layout drawings and the 3D model.

The pricing of the architectural quantities were based on the budget quotations received as a part the structural steel budget quotation. The architectural quantities were based on the following:

- Uninsulated and insulated roofing (budget quote).
- Uninsulated and insulated cladding (budget quote).
- Exterior doors and windows (DRA database).

- Truck and vehicle doors (DRA database).

21.1.8.6 *Architectural – Finishes*

The architectural finishes work included the interior windows, doors, finishes, lockers, and washrooms.

- The quantities were based on the area of the building and a unit price applied.
- The prefabricated building costs included all interior and exterior finishes including washrooms and showers, kitchen counters, sinks and finishes.
- No appliances were included in the prefabricated building costs.

The pricing of the architectural finishes was based on the budget quotations received as a part of the structural steel budget quotation.

21.1.8.7 *Mechanical Equipment*

The development of the mechanical equipment, including pumps, platework, dust collection systems, HVAC equipment and relocated and new equipment were based on the MEL and incorporated into the Capex as follows:

- The MEL, indicating size/capacity, dead weights and power requirements, along with the process flow diagrams was used to provide mechanical equipment quantities.
- The MEL specifically detailed the existing equipment, rehabilitation, and transport to the Project site. Equipment weight and power usage was also indicated.
- An allowance for miscellaneous construction installation materials, such as grout, shims etc. was included in installation costs.

The pricing of the mechanical equipment was based on the receipt of two to three quotations from qualified equipment suppliers and fabricators. The following equipment packages were issued to suppliers, with bids returned for evaluation:

- Conveyors and belt feeders.
- Centrifugal gravity concentrator.
- Scalping screens.
- Slurry pumps.
- Water pumps.
- Jameson flotation cells.
- Gold room equipment.
- Shaking table.
- Filter presses.

- Bagging system.
- Agitators.
- Air compressors.
- Thickeners.
- Jaw crusher.
- Diesel storage tanks and distribution.
- Platework fabrication and liners.

For other mechanical equipment required for the project, the equipment was priced from current in-house data, and allowances.

The pricing of the installation of the mechanical equipment was based on contractor rates from two (2) contractors. The pricing included the direct and indirect costs, including supply of labour and materials, equipment costs, accommodation of contractor's personnel, temporary site shops, offices, and warehousing. Mobilisation and demobilisation costs are included in the Capex indirect costs.

Budget quotations were received from the following contractors all located in the surrounding areas. The quoted prices were all within 10% of each other:

- Tehnopetrol.
- SET.

21.1.8.8 Piping

The development of the piping quantities was based on the following methodology.

- For Crushing and Grinding areas, the quantities were based on existing 3D models and P&ID drawings from the existing process plant, and piping quantities by type and size were used to form the quantity of piping and valves. A detailed MTO, for the Ada Tepe facility, was provided by DPM, to DRA.
- For other process areas not supported by Ada Tepe MTOs, the cost for piping was factored on the basis of the type of process and type of service.
- An allowance for building service piping was provided in each major building to cover the HVAC system, potable water, fire protection and compressed air distribution.

The pricing of the piping materials and valves was based on the receipt of two to three quotations from qualified equipment suppliers and fabricators. The following piping systems were issued to suppliers and bids returned for evaluation:

- Carbon steel piping.
- HDPE piping.

- Stainless steel piping.
- Rubber lined steel piping.
- Valves.

The pricing of the installation of the piping systems was based on contractor rates from two (2) contractors. The pricing included the direct and indirect costs, including supply of labour and materials, equipment costs, accommodation of contractor's personnel, temporary site shops, offices, and warehousing. Mobilisation and demobilisation costs are included in the Capex indirect costs.

Budget quotations were received from the following contractors all located in the surrounding areas. The quoted prices were all within 10% of each other:

- Tehnopetrol (Supply and installation of piping systems).
- SET (Installation the piping systems only).

21.1.8.9 Electrical

The development of the electrical equipment including transformers, MCCs, switchgear, generators, etc. and relocated and new equipment was based on the Electrical Equipment List (EEL) and incorporated into the Capex as follows:

- The electrical equipment list and the single line diagrams formed the basis for the electrical equipment estimate.
- Electrical bulks for Crushing and Grinding were based on the Ada Tepe actuals,
- Electrical bulks for the remainder of the plant were estimated by type of material and type of process.
- Bulk electrical quantities of raceway will be taken from the plot plan, equipment layouts and single line diagrams. Bulk electrical quantities of raceway were taken from the Čoka Rakita plot plan, equipment layouts and single line diagrams.
- All power was distributed to the various e-houses, substations and pumphouses via underground systems with the exception of power to the Refurbished Process Plant Substation, VFD Room, and Refurbished MCC Room which was distributed by cable trays within the buildings.
- Power cables were generated from the load list and average take-offs calculated.
- Quantities of electrical bulks were segregated into the following categories:
  - Complex facility raceway and cable.
  - Raceway and cable on pipe racks.
  - Conduit/cable in duct banks.

- Incoming power will be supplied via a 35 kV line from Zagubita. As an alternative, power could be supplied via a 110 kV line from Bor.
- On-site, main substation with fully redundant transformers steps down the 35 kV grid incoming voltage to 10 kV which is to be distributed to each area with isolated and armoured cable, buried below ground. Spread into the process and paste plant area, prefabricated type electrical rooms insure power distribution to each equipment. Each equipment to be locally controlled and de-energised with local disconnect switch.
- Permanent emergency gensets to be installed for critical load to be energised in case of power outage of main 35 kV and second liaison at 10 kV.
- For buried cable, lengths of typical duct bank were used, indicating sections and number of conduits.

The pricing of the electrical equipment was based on the receipt of two to three quotations from qualified equipment suppliers and fabricators. The following equipment packages were issued to suppliers and bids returned for evaluation:

- High Voltage Equipment.
- Power and distribution transformers.
- E-Rooms including refurbishment of the existing E-Rooms.
- MCCs.
- Generators.

For other electrical equipment required for the project, the equipment was priced from current in-house data, and allowances.

The pricing of the installation of the electrical equipment and the supply and installation of the cables, cable trays, interconnections, etc., was based on contractor rates from two (2) contractors. The pricing included the direct and indirect costs, including supply of labour and materials, equipment costs, accommodation of contractor's personnel, temporary site shops, offices, and warehousing. Mobilisation and demobilisation costs are included in the Capex indirect costs.

DPM furnished the budget price covering the incoming 35 kV power line.

Budget quotations were received from the following contractors all located in the surrounding areas. The quoted prices were all within 10% of each other:

- Rasina.
- Termoinzenjering.

21.1.8.10 *Instrumentation*

Where possible, the existing control system will be dismantled and relocated to the new plant. All associated instruments were reviewed and will be relocated with the existing equipment. New equipment and instruments were, also, incorporated into the Capex.

- The PFDs, PCDs and plot plan formed the basis of the instrumentation quantities. Processing equipment were mostly modularised with expected equipment and instrumentation installed. Control cables were generated from the load list and average take-offs calculated.
- Quantities of instrumentation bulks were segregated into the following categories:
  - Complex facility raceway and cables.
  - Raceway and cable on pipe racks.
  - Conduit/cable in duct banks.

The pricing of the supply of the DCS system was obtained from a European supplier and applied to the Capex.

The cost for Process Instrumentation & Controls was factored as a percentage of the mechanical equipment cost based on DRA's historical data and experience. In this case, the factor was 10%.

21.1.9 DESIGN AND GROWTH ALLOWANCES

All quantities provided by Engineering and/or contractors were taken off as neat quantities. During the preparation of Capex, it is a rare occasion when everything is known, specific and measurable.

Design growth allowances are:

- Considered as a "known unknown" requirement, when preparing the direct cost and more accurately reflect what the final design will require.
- Applied at the direct cost level, to compensate for the degree of engineering that is incomplete and subject to minor modifications.

Design growth allowances will be added to compensate for known items not identified in the estimate line items ("known unknowns").

This is not to be confused with contingency that is included below the line, to include for the "unknown unknowns." Contingency is a separate account. Design allowances are built around currently identified scope and do not make provision for scope changes. Scope changes are excluded and should be treated as variations to the capital cost of the project.

Allowances for bulk materials will be applied to quantities and include growth on both labour and material.

Design allowances and growth allowances were applied to the Capex by discipline by Level 4 WBS and is presented in Table 21.5.

Table 21.5 – Growth Allowances per Discipline

Discipline Code	Discipline Description	Design Growth
1200	Earthwork – Site Preparation	15%
1300	Earthwork – Terraces	15%
1400	Earthwork – Roads	15%
1500	Earthwork – Fencing	15%
1700	Earthwork – Liners	10%
2100	Detail Earthworks	10%
2200	Concrete	10%
2300	Structural Steel	10%
1120	Dismantled Steel	10%
2400	Architectural Building	15%
2500	Architectural Finishes	15%
2800	Building Mechanical Services	10%
2900	Building Electrical Services	0%
4000	New Mechanical Equipment	5%
1140	Dismantled Ada Tepe Equipment	0%
5000	Piping (with quantities)	0%
5100	Piping as a %	0%
6000	Electrical Ada Tepe E-Houses	5%
6100	Electrical Bulks	10%
7000	Instrumentation	0%
	Underground Excavations	5%

21.1.10 Capex Pricing Basis

The input cost sources are outlined in Table 21.6.

Table 21.6 – Input Cost Source

Description	Budget Quotes	Database	Factored
Mechanical Equipment	80%	20%	
Electrical Equipment	75%	20%	5%
Steel Works	90%	5%	5%
Prefabricated Buildings	85%		15%
Earthworks	90%	5%	5%
SMPP	85%		15%
Architectural	40%	40%	20%

21.1.11 Labour

The Capex is based on applying unit labour hours and crew rates for mechanical, electrical and instrumentation equipment and by factoring labour costs as a percentage of the equipment cost.

21.1.11.1 Labour Rates

The labour rates used in the Capex were obtained from quotations from local contractors. For this Project, budget quotations were received for the following contractual packages:

- Earthworks.
- Fencing.
- Liners.
- Concrete works.
- Structural Steel.
- Mechanical Equipment.
- Piping.
- Electrical and Instrumentation.

The labour rates obtained is further assumed to be a blended composite crew rate (all-inclusive of lodging, meals and incidentals) for various discipline activities.

The labour pricing included the direct and indirect costs, including supply of labour and materials, equipment costs, accommodation of contractor’s personnel, temporary site shops, offices, and warehousing. Mobilisation and demobilisation costs are included in the Capex indirect costs.

21.1.12 Indirect Costs

21.1.12.1 Contractor Indirect Costs

Contractor indirect costs cover only mobilisation and demobilisation. All other indirect costs are included in the contractor rates provided for the direct cost.

21.1.12.2 Capital Spares and Inventory

Capital Spares and Inventory include:

- Capital Spares.
- Operational Spares.
- Commissioning Spares.
- Initial Fills.

21.1.12.3 Project External Consultants

Project External Consultants include:

- Consultant Engineering Procurement.
- Consultant Geology.
- Consultant Geotechnical.
- Consultant Project Management.
- Consultant EP support.
- Consultant Expenses.
- Vendor Representatives.

The requirements for vendor representatives to supervise the installation of equipment or to conduct a checkout of the equipment prior to start-up of the equipment is deemed necessary for equipment performance warranties; and was calculated and included in the estimate.

21.1.12.4 Commissioning

Commissioning includes the cost for a commissioning team required for preparing the Process plant for the commissioning and ramping up of the 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.
- Corrections.

The estimate includes the cost of support crews for commissioning among others, including:

- Tools and equipment.
- Safety equipment and services.
- Radios.
- Consumables.
- Transportation, housing and meals.

Pre-Commissioning Teams include:

- Start up and Commissioning-Owner's Team.
- Start up and Commissioning-EPCM Team.
- Start up and Commissioning-Contractor Support.
- Start up and Commissioning-Vendor Support.

Commissioning costs are included in the estimate.

21.1.12.5 Freight and Logistics

Freight and Logistics costs include allowances for the following items. The allowances are estimated on a percentage of equipment and material costs.

- Ocean freight.
- Land freight.
- Air freight.
- Brokerage and Agent Fees.

21.1.13 OWNER'S COST

The Owner's Costs was compiled by DRA with assistance from DPM. The following represents a list of items that DRA considers included in the Owner's costs:

- Definition drilling, assaying and related reports and models.
- Owner's Project administration team (allowance).
- Owner's pre-production and mine development.
- Health safety and security.
- Closure and rehabilitation.

- Sunk costs (covered in the economic evaluation as supplied).
- Process rights, royalties, license fees, technology fees (addressed in financial model).
- Project financing and interest charges (covered in the economic evaluation).
- Training and recruiting of plant operating personnel (Operational Readiness).
- Working capital (covered in the economic evaluation) (Operational Readiness).
- Start-up and commissioning costs (client commissioning team).
- All production costs prior to commercial production.
- All costs associated with the permitting process.
- Builder’s risk insurance.
- Cost of this or any other study.
- Community relations.
- Emergency response.
- Environmental / permitting / government relations.
- Finance and general administration.
- Human resources.
- Information technology.
- Insurances.
- Legal.

21.1.14 CONTINGENCY

Contingency was calculated on a discipline-by-discipline basis, considering items that were quoted, estimated or factored. Contingency was 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.

Contingency, as defined herein, is not intended to cover such items as labour disputes, change in scope, or price escalation.

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 operating costs include in-house databases and outside sources particularly for materials, services and consumables. All amounts are in United States Dollars (US$), unless specified otherwise.

21.2.1 Summary Opex

The Opex costs presented in the table exclude pre-production Opex allowances for mining, process and G&A. These are covered in the Capex. Labour rates for this Report were provided by DPM. Direct employment during operations will total approximately 532 people including mine, concentrator, tailings, maintenance, management, and infrastructure. A summary of the overall LOM Project Opex The total LOM Opex and average Opex estimate, given as dollar per tonne milled (6,632,883 tonnes milled provide the unit costs), is summarised in Table 21.7.

Table 21.7 – Opex by Major Area

Area	LOM Total Opex (US$ M)	Average Opex (US$/t milled)
Mining	251.1	37.85
Processing and Tailings	165.7	24.98
G&A	98.3	14.82
Royalties	107.1	16.15
Offsite Costs	69.2	10.43
Total Opex	**691.3**	**104.23**
Numbers may not add due to rounding.		

21.2.2 Summary of Personnel Requirements

Table 21.8 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 personnel will come from the area.

Table 21.8 – Total Personnel Requirement (Year 2)

Description	Number
Mining[1]	239[2]
Processing[3]	88
Maintenance[4]	123
Administration[5]	82
Total Personnel	**532**

Note:
1 Mining personnel number includes 42 technical staff.
2 One (1) extra for each of cable bolting, haulage, and drilling that peak later in plan.
3 Processing includes 5 paste plant operators for mine backfill,
4 Maintenance includes 73 underground mechanics and electricians
5 Administration includes 16 HSE staff for mining

21.2.3 Mining Opex

The mining Opex consist of Personnel, Equipment Operation, and Materials and Services. The mine Opex was estimated based on Owner furnished equipment, materials and mine operators and based on the annual mine plans. Table 21.9 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 21.9 – Summary of Mining LOM Average Opex

Description	LOM Total Opex (US$ M)	Average Opex (US$/t mined)
Labour	76.2	11.48
Equipment Operation	69.0	10.40
Materials and Services	39.0	5.88
Utilities	66.9	10.09
Total	**251.1**	**37.85**

Numbers may not add due to rounding.

The mining Opex estimate generally excludes:

- Pre-production operating costs.
- Costs for surface activities such as ore processing, tailings, paste plant operation, and administrative works.
- Surface labour, including health and safety personnel.
- Capitalised underground development and construction activities.
- Any factors for contingency or escalation.

21.2.4 MINERAL PROCESSING OPEX

The estimated mineral processing LOM operating costs for the plant production are summarised in Table 21.10 which shows the breakdown by six (6) major components: labour, electrical power, reagents and consumables, laboratory, contract maintenance, and spares and miscellaneous. These costs are derived from testwork, supplier information, DPM inputs and / or DRA experience.

Table 21.10 – Summary of Processing LOM Average Opex

Description	LOM Total Opex (US$ M)	Average Opex (US$/t milled)
Labour	35.7	5.38
Power	45.9	6.92
Reagents and Consumables	66.4	10.01
Laboratory	1.9	0.29
Contract Maintenance	2.6	0.40
Spares and Miscellaneous	13.2	1.98
Total	**165.7**	**24.98**

Numbers may not add due to rounding.

21.2.4.1 Electrical Power Costs

Electrical power is required to drive Process Plant equipment such as crushers, mills, conveyors, screens, pumps, agitators, screens, compressors, etc. The energy required to operate each piece of equipment was calculated from basic principles or vendor information and combined with an expected annual utilisation to estimate energy consumption. The unit cost of electricity was established at $0.143 per kWh. The total annual operational electrical energy consumption is estimated at 41,230,423 kWh which results in a unit energy consumption rate of 48.51 kWh per tonne.

21.2.4.2 Reagent Consumption Costs

Reagents consist of flotation reagents (eg: MIBC, PAX, A3477 and Copper Sulphate), flocculant (for thickeners), smelting fluxes and reagents used for water treatment. The total annual cost of these reagents is $2,822,749 or $3.32 per tonne milled.

Cement is required for the paste backfill operation. Total cement requirement is 18,445 tpa based on a 5% addition rate and represents $2,766,750 annually or $3.26 per tonne milled.

The total annual cost of above reagents is 5,589,499 or $6.58 per tonne milled.

21.2.4.3 *Grinding Media*

The vertical stirred mill will need regular addition of balls to replace worn media. Media consumption is estimated based on steel consumption observed in similar operations and the abrasion indices and power consumption. Vertical mill media would cost $ 581,400 or $0.68 per tonne milled. The primary mill will be operated without any grinding media so there is no media cost estimated for this mill.

21.2.4.4 *Consumables Costs*

The consumptions and costs for crusher liners, screen deck panels, grinding mill liners, cyclone 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, FELs, trucking filter cakes to the DTSF and the dozer and graders used on the DTSF are also included. The costs of consumables and wear parts are estimated at $2,918,662 per annum or $3.43 per tonne milled.

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 LOM summary for G&A is estimated at $98 M or $14.82 per tonne milled.

22 ECONOMIC ANALYSIS

22.1 General

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

The economic analysis is based on the discounted cash flow (DCF) method on a pre-tax and after-tax basis. Current Sebian tax regulations were used to assess corporate tax liabilities. The key metrics determined in the analysis are the Net Present Value (NPV) at a discount rate of 5%, the Internal Rate of Return (IRR), and the Payback Period. A sensitivity analysis was carried out to assess the impact of variations in gold price, Capex, and Opex on the financial metrics.

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 economic analysis is developed in terms of financial years with appropriate adjustments made to the production schedules to convert the data from mine plan years. The capital and operating costs are taken from the estimates detailed in Section 21.

All costs and pricing are in Q4 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.
- Geotechnical or hydrogeological considerations during mining.
- Process plant production plan.
- Recovery rates of gold in the processing plant.

- Ability of plant, equipment, processes to operate as anticipated.
- Development capital costs.
- Sustaining and operating costs.
- Closure costs and unforeseen reclamation expenses.
- Environmental, social, and licensing risks.
- Taxation policy and tax rate.
- Royalty agreements.
- Cost inflation.
- Ability to maintain social license to operate. and
- Unrecognised environmental risks.

22.3 Economic Criteria

22.3.1 Construction Decision

Based on the project schedule, DPM expects to make a construction decision in the first half of 2026, followed by a ~24-month construction period (covering Year -3, Year -2, and part of Year-1) before commencing and ramping up production in Year -1. The project economics are evaluated from the time of the construction decision, which is assumed to be January 2026 for the purposes of this analysis, with production starting in Q3 2028 (Year -1).

22.3.2 Metal Price

Project revenues consist of gold concentrate and doré sales. No other product is considered as part of the economic analysis. A long-term gold price of US$1,900 per oz is used for the economic analysis throughout the LOM.

Gold price sensitives are tabulated at $1,500/oz, $1,700/oz, $1,900/oz, and $2,300/oz in Table 22.8.

22.3.3 Production Schedule

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 economic analysis is developed in terms of financial years by making appropriate adjustments to the production schedule to transform the data from mine plan years to financial years.

22.3.4 Commercial Terms and Freight

The commercial terms for the sale of gold and freight cost estimates used in this analysis were provided by DPM. A summary of the key terms for the gravity concentrate, flotation concentrate, and doré is provided in Table 22.1. Estimates of the cost of product freight are also provided in this Table.

Transport losses of 0.2% on a dry metric tonne basis were applied to each concentrate. No transport losses are considered for doré.

The concentrate terms provided also included payables for silver and copper. However, based on current information, the content of these by-product metals does not meet the minimum payability thresholds specified in the terms (30 g/t Ag and 5% Cu) and these metals were therefore not included in revenue.

Table 22.1 – Commercial Terms for the Sale of Gold Concentrates

Description	Unit	Gravity Gold Concentrate	Flotation Gold Concentrate	Doré
Treatment Charge	$/dmt	170	150	0
Refining Cost (Gold)	$/oz	6.0	7.5	0.4[2]
Penalty	$/dmt	0	0	0
Gold Payable	%	99.8	97.5[1]	99.9
Concentrate Freight, Inland	$/dmt	125.5	51.8	0
Concentrate Freight, Ocean	$/dmt	75	0	
Doré Freight				
Shipping Charge	$/oz			1.0
Handling Charge	$/lot[3]			350
Handling Additional Charge	$/lot			80

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.
2. Minimum payable is $500/lot
3. Lot size is expected to be 50-100 kg.

22.3.5 Mineral Royalties

The Government of Serbia imposes a mineral royalty fee of 5% on the net revenue value.

22.3.6 Capital and Operating Costs

The Capital Costs used in this analysis are taken from the estimates detailed in Section 21. The Development Capex is spread over three annual periods, namely 20% in Year -3, 40% in Year -2, and 40% in Year -1, which aligns with DPM's expected project spend profile. The schedule of sustaining capital expenditures for mining was provided from the mine design exercise outlined in Section 16. The schedule for remaining sustaining capital expenditures was determined using assumptions based on the type of expenditure. An assumption has been made regarding the Closure Cost with the majority scheduled in the year following the completion of operations.

The Operating Costs used in this analysis are taken from the estimates detailed in Section 21. A schedule of annual operating costs for mining was provided from the mine design exercise outlined in Section 16. The schedule of annual operating costs in each remaining category was determined on a fixed $/tonne ore milled basis.

No consideration has been made in this model for salvage value from the sale of mechanical equipment.

22.3.7 Taxes

Corporate tax liabilities were calculated under the Serbian tax regime, based on public information and information provided by DPM. The tax calculations were reviewed by DPM's financial team. The following taxes are applicable:

22.3.7.1 Land Acquisition and Usage Taxes

It is assumed that the land acquisition tax will be paid prior to the construction decision and, as such, is not included in this analysis.

Annual land usage taxes are calculated using a 0.4% tax rate on an assumed land acquisition cost estimate of US$11.9 M.

22.3.7.2 Income Tax

The standard corporate tax rate in Serbia is 15% and was applied to calculate income taxes. Serbian resident companies are subject to tax on their worldwide income. The income tax basis is determined based on accounting profit as calculated in accordance with IFRS Accounting Standards, adjusted for local tax legislation. All deductions and rates are based on currently enacted legislation and are subject to change in the future.

Operating losses generated in the course of business activities can be carried forward for up to five years and used to offset taxable income.

Current legislation allows for relief from income tax for large investments for a maximum period of 10 years subject to the fulfilment of the following conditions:

- Investment of more than 1 billion Serbian Dinar (RSD) or US$9.1 M (at an RSD:US$ exchange rate of 0.0091) in qualified fixed assets for use in registered business activities.
- Employment of 100 new employees for an indefinite period of time during the entire period of the tax relief.

DPM assumed that the Project is eligible for this tax relief. The tax relief period runs uninterrupted for 10 years and it is assumed that the tax holiday will be declared once the eligibility criteria is met and taxable profit is achieved. The effective income tax rate applied in this analysis is 0% over the 10-year life of the Project.

22.3.7.3 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 Value-Added Tax.

A detailed review of the Capex items has not been conducted at this stage to precisely determine the VAT payable for the Project. Given the early-stage nature of this Study, the proportion of each of the Development Capital, Sustaining Capital, and Closure Capital exempt from VAT has been taken from a benchmark project in Serbia. All labour costs in the Opex are considered exempt from VAT.

DPM will claim the VAT for the Project in the same period in which it is paid – except in the case of Closure Cost VAT, which is not claimed.

22.4 Base Case Cash Flow Analysis and Economic Results

At an assumed long-term gold price of $1,900 per ounce, the financial results indicate a positive pre-tax NPV of $735.0 M, at a discount rate of 5%, a pre-tax (and after-tax) IRR of 41.4% and a payback period of 1.7 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
NPV @ 5%	US$ M	735.0	735.0
IRR	%	41.4	41.4
Payback Period	Years	1.7	1.7
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.

Table 22.3 – Key Technical Assumptions and Cost Inputs

Description	Unit	Value
Macroeconomic Parameters		

Description	Unit	Value
Gold Price	US$ per oz	1,900
Discount Rate	%	5.0
Project Parameters		
Mine Life	years	8
Mineable Mineral Resource (LOM)	Mt	6.6
Grade Mined (LOM average)	g/t	6.38
Annual Mill Throughput	tpa	850,000
Gold Recovery (LOM average)	%	86.8
Gold Payability (LOM average)	%	98.5
Total Gold Produced (LOM)	Moz	1.2
Average Annual Gold Production (LOM)	oz	147,000
Average Annual Gold Production (first five years)	oz	170,000
Government Royalty (NSR)	%	5.0
Capital Cost Estimate		
Initial Capital	US$ M	379
Sustaining Capital (LOM)	US$ M	29
Closure Costs[2]	US$ M	27
LOM Operating Unit Costs		
Mining	US$ per tonne processed	38
Processing	US$ per tonne processed	25
General & Administrative	US$ per tonne processed	15
Royalties	US$ per tonne processed	16
Offsite Cost	US$ per tonne processed	10
Total Opex	*US$ per tonne processed*	***104***
LOM Average All-in Sustaining Cost [2]	US$ per oz gold	644

1. Numbers may not add due to rounding.
2. Closure costs include the non-recoverable VAT of approximately $2.3 M.

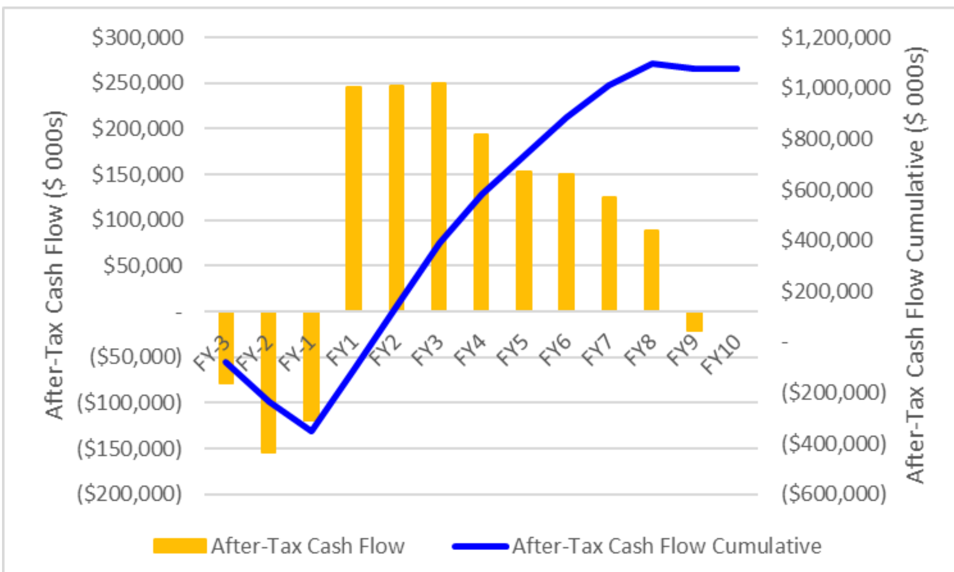
[2] All-in sustaining costs are non-GAAP financial measure or ratio and has no standardised meaning under IFRS Accounting Standards (IFRS) and may not be comparable to similar measures used by other issuers. As the Project is not in production, DPM does not have historical non-GAAP financial measures 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. Refer to the "Non-GAAP Financial Measures" in Section 2.7 of this Report for more information, including a detailed description of each of these measures.

Figure 22.1 presents the annual and cumulative cash flows of the project on an after-tax basis. An overview of the estimated cash flows of the Project is presented in Table 22.7 This table indicates a LOM net revenue of $2,134.3 M. The total operating cost (i.e., the sum of mining, process, and G&A costs) is estimated at $515.0 M for the life of the mine, amounting to $77.65/t of ore. Royalties have been calculated at $106.7 M and Property Taxes at $0.6 M, resulting in LOM Operating Earnings of $1,509.6 M.

The pre-production Capex was estimated at $379.0 M, the total sustaining capital requirement was estimated at $29.0 M and the closure cost was estimated $24.4 M, for a total capital expenditure over the Project life of $432.4 M. The non-recoverable VAT associated with the Closure Cost was estimated at an additional $2.3 M. The cash flow has been developed on the basis that the initial Capex carries through the first 274 days of Year -1 with the balance of the 90 days as production.

The resulting cash flow estimated for the Project over the LOM was $1,077.2 M.

Figure 22.1 – After-Tax Annual and Cumulative Cash Flows



Source: DRA, 2024

22.5 Sensitivity Analysis

A sensitivity analysis was carried out, using the base case described above as a starting point, to assess the impact of changes in the price of gold, total Capex and Opex on the Project's NPV at a 5% discount rate and IRR. The impact of each variable is examined individually with an interval of ±20% and increments of 10% applied. It is to be noted that the margin of error for cost estimates at the PFS study level is typically -20% +30%. However, the uncertainty in metal price forecasts usually remains significantly higher, and is a function of price volatility.

The after-tax results of the sensitivity analysis are shown in Table 22.4 to Table 22.6 and Figures 22.2 and 22.3. 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. The IRR is more sensitive to variations in Capex than Opex, as evidenced by the steeper slope of the Capex curve in Figure 22.3.. The NPV appears to be equally sensitive to variations in Capex and Opex. Overall, within the limits of accuracy of the cost estimates in this study, the Project's potential after-tax viability does not seem significantly vulnerable to the under-estimation of capital and operating costs up to 20%, when taken individually.

Table 22.4 – Economic Metrics Sensitivity to Variations in the Gold Price

Au Price	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	1,043.3	889.2	735.0	580.9	426.7
IRR	%	52.0%	46.8%	41.4%	35.5%	29.0%
Payback	Years	1.4	1.5	1.7	1.9	2.2

Table 22.5 – Economic Metrics Sensitivity to Variations in the Capex

Capex	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	657.6	696.3	735.0	773.7	812.5
IRR	%	33.9%	37.4%	41.4%	46.0%	51.4%
Payback	Years	2.0	1.8	1.7	1.5	1.4

Table 22.6 – Economic Metrics Sensitivity to Variations in the Opex

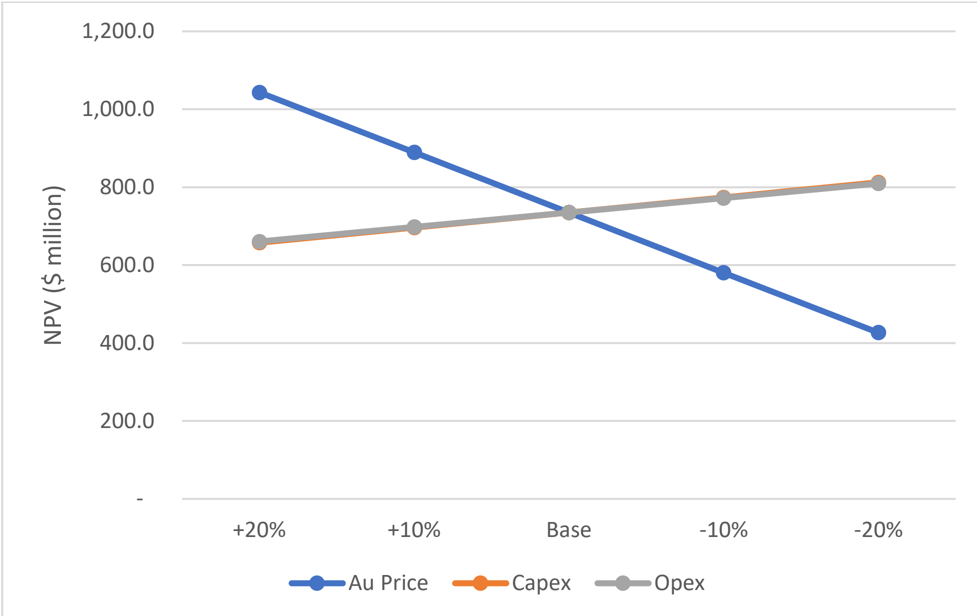
Opex	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	660.6	697.8	735.0	772.2	809.4
IRR	%	38.8%	40.1%	41.4%	42.6%	43.8%
Payback	Years	1.8	1.7	1.7	1.6	1.6

Table 22.7 – Summary of Production Schedule and Cash Flows

Period #			-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Period			FY-3	FY-2	FY-1	FY1	FY2	FY3	FY4	FY5	FY6	FY7	FY8	FY9	FY10	FY11	FY12
INPUTS		*(Tot. / Avg.)*															
Feed																	
Ore Tonnes Milled	*(ktonnes)*	**6632.9**	-	-	131.3	754.3	854.9	854.9	855.0	855.0	854.4	848.1	625.1	-	-	-	-
Feed Grades - Gold	*(g/t)*	**6.38**	-	-	8.6	10.0	7.5	7.9	6.1	5.3	5.4	4.5	3.7	-	-	-	-
Contained Metal - Gold	*(koz)*	**1359**	-	-	36.4	242.4	205.0	215.8	167.4	146.6	148.1	122.8	74.9	-	-	-	-
Recovery Schedule																	
Gold Recovery to Flotation Concentrate	*(%)*	**50.7**	-	-	48.5	48.3	50.4	50.2	51.3	51.8	51.7	52.4	53.2	-	-	-	-
Gold Recovery to Gravity Concentrate	*(%)*	**13.3**	-	-	13.1	13.6	13.7	13.8	13.3	13.1	13.1	12.8	12.4	-	-	-	-
Gold Recovery to Dore	*(%)*	**22.7**	-	-	22.4	23.1	23.3	23.5	22.7	22.3	22.3	21.7	21.2	-	-	-	-
Flotation Concentrate Production																	
Gold	*(g/t)*	**70.6**	-	-	101.3	125.2	86.2	92.1	67.3	57.7	58.4	47.7	38.8	-	-	-	-
Gold	*(koz)*	**689**	-	-	17.7	117.0	103.4	108.4	85.8	75.9	76.6	64.4	39.8	-	-	-	-
Concentrate Tonnage Produced	*('000s dmt)*	**303.8**	-	-	5.4	29.1	37.3	36.6	39.6	40.9	40.8	42.0	31.9	-	-	-	-
Gravity Concentrate Production																	
Gold	*(g/t)*	**1501.3**	-	-	2,013.4	2,335.6	1,742.7	1,835.1	1,423.4	1,246.5	1,260.2	1,052.5	871.2	-	-	-	-
Gold	*(koz)*	**181**	-	-	4.8	32.9	28.1	29.8	22.3	19.2	19.4	15.7	9.3	-	-	-	-
Concentrate Tonnage Produced	*('000s dmt)*	**3.8**	-	-	0.07	0.44	0.50	0.50	0.49	0.48	0.48	0.46	0.33	-	-	-	-
Dore Production																	
Gold	*(koz)*	**309**	-	-	8.1	56.1	47.8	50.7	38.0	32.6	33.0	26.7	15.9	-	-	-	-
Total Gold Production																	
Contained Metal - Gold (Total)	*(koz)*	**1,179**	-	-	30.6	205.9	179.3	188.9	146.1	127.7	129.1	106.8	65.0	-	-	-	-
Gold	*(koz)*	**1,160**	-	-	30.1	202.6	176.3	185.8	143.6	125.6	126.9	105.0	63.9	-	-	-	-
Period #			-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
Period			FY-3	FY-2	FY-1	FY1	FY2	FY3	FY4	FY5	FY6	FY7	FY8	FY9	FY10	FY11	FY12
PRE-TAX CASH FLOW		*(Tot. / Avg.)*															
Net Smelter Revenue	*($'000s)*	**$2,134,304**	-	-	$55,823	$377,791	$326,312	$344,392	$263,944	$229,466	$231,977	$190,195	$114,405	-	-	-	-
Less: Government Royalties	*($'000s)*	**($106,715)**	-	-	($2,791)	($18,890)	($16,316)	($17,220)	($13,197)	($11,473)	($11,599)	($9,510)	($5,720)	-	-	-	-
Less: Total Operating Costs	*($'000s)*	**($515,032)**	-	-	($12,380)	($59,966)	($67,608)	($66,676)	($67,200)	($67,929)	($66,651)	($62,184)	($44,438)	-	-	-	-
Less: VAT Payable after Credit Applied	*($'000s)*	**($2,344)**	-	-	-	-	-	-	-	-	-	-	($469)	($1,875)	-	-	-
Less: VAT Refund Account	*0*	**-**	($2,991)	($2,991)	($421)	$4,099	($198)	($44)	$61	($36)	$44	$249	$2,228	-	-	-	-
Less: Property Taxes	*($'000s)*	**($572)**	($48)	($48)	($48)	($48)	($48)	($48)	($48)	($48)	($48)	($48)	($48)	($48)	-	-	-
Operating Earnings	*($'000s)*	**$1,509,642**	($3,038)	($3,038)	$40,183	$302,987	$242,143	$260,405	$183,560	$149,980	$153,723	$118,702	$65,958	($1,923)	-	-	-
Capital Expenditures																	
Pre-Construction & Early Works Capital	*($'000s)*	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Development Capital exc. Capitalized Amounts	*($'000s)*	**$379,037**	$75,807	$151,615	$151,615	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	*($'000s)*	**$28,990**	-	-	-	$7,589	$3,639	$7,162	$3,464	$3,140	$3,604	$335	$56	-	-	-	-
Closure Capital	*($'000s)*	**$24,415**	-	-	-	-	-	-	-	-	-	-	$4,883	$19,532	-	-	-
Changes in Working Capital	*($'000s)*	**($0)**	-	-	($8,306)	($49,690)	$8,986	($3,036)	$13,366	$5,612	($500)	$6,562	$27,006	-	-	-	-
Pre-Tax Cash Flow	*($'000s)*	**$1,077,199**	**($78,846)**	**($154,653)**	**($119,738)**	**$245,708**	**$247,490**	**$250,207**	**$193,462**	**$152,451**	**$149,619**	**$124,929**	**$88,025**	**($21,454)**	**-**	**-**	**-**
Discounted Pre-Tax Cash Flow	*($'000s)*	**$735,017**	**($76,946)**	**($143,739)**	**($105,988)**	**$207,137**	**$198,704**	**$191,319**	**$140,885**	**$105,733**	**$98,828**	**$78,590**	**$52,737**	**($12,242)**	**-**	**-**	**-**
Pre-Tax IRR	*(%)*	**41.4%**															
Payback Period	*(years)*	**1.7**															
AFTER-TAX CASH FLOW		*(Tot. / Avg.)*															
Pre-Tax Cash Flow	*($'000s)*	**$1,077,199**	($78,846)	($154,653)	($119,738)	$245,708	$247,490	$250,207	$193,462	$152,451	$149,619	$124,929	$88,025	($21,454)	-	-	-
Less: Income Tax Paid	*($'000s)*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
After-Tax Cash Flow	*($'000s)*	**$1,077,199**	**($78,846)**	**($154,653)**	**($119,738)**	**$245,708**	**$247,490**	**$250,207**	**$193,462**	**$152,451**	**$149,619**	**$124,929**	**$88,025**	**($21,454)**	**-**	**-**	**-**
Discounted After-Tax Cash Flow	*($'000s)*	**$735,017**	**($76,946)**	**($143,739)**	**($105,988)**	**$207,137**	**$198,704**	**$191,319**	**$140,885**	**$105,733**	**$98,828**	**$78,590**	**$52,737**	**($12,242)**	**-**	**-**	**-**
After-Tax IRR	*(%)*	**41.4%**															
Payback Period	*(years)*	**1.7**															

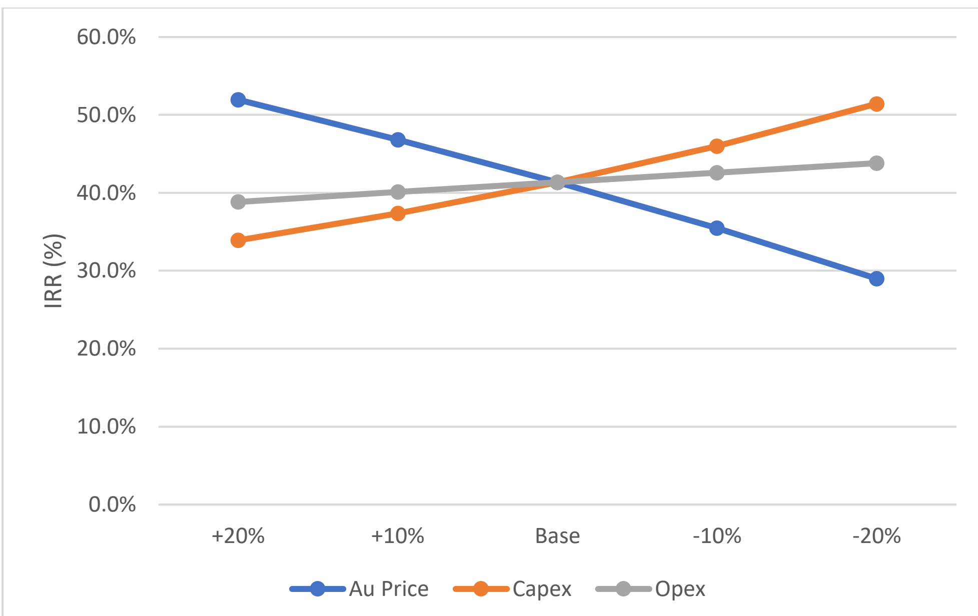
Source: DRA, 2024 Closure costs referenced in this table do not include the non-recoverable VAT of approximately $2.3 M.

Figure 22.2 – After Tax NPV 5%: Sensitivity to Capex, Opex and Price



Source: DRA, 2024

Figure 22.3 – 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 22.8 with the base case highlighted.

Table 22.8 – Gold Price NPV Sensitivity After-Tax

Gold Price	Unit	US$ 1,500/oz	US$ 1,700/oz	US$ 1,900/oz	US$ 2,300/oz
IRR	%	28.3%	35.1%	41.4%	52.5%
NPV at 5%	US$ M	$410.5	$572.7	$735.0	$1,059.6

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 km². 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).

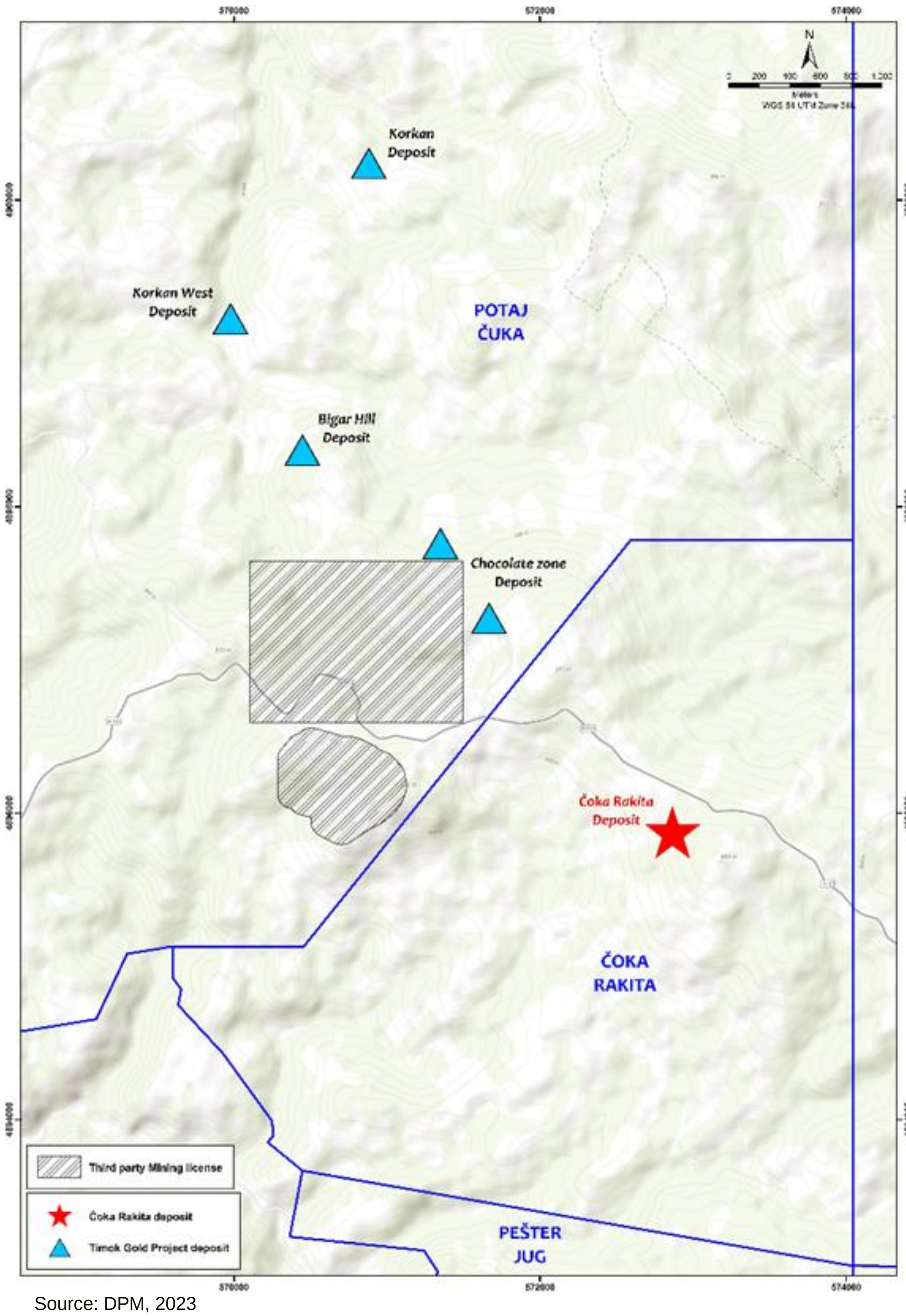
DPM completed a Mining PFS for the Timok Project in 2021 (De Weerdt et al., 2021) which is available on SEDAR+ at www.sedarplus.ca. The MRE used as a basis for the study (effective date of 29 May 2020) includes 32.3 Mt of Indicated Mineral Resource with an average grade of 1.27 g/t Au and 1,319 koz of 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.

It is noted that 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

23.2 Dumitru Potok and Frasen (DPM)

On September 11, 2024, DPM announced the discovery of high-grade copper gold mineralisation one kilometre to the North of Čoka Rakita (Link - Dundee Precious Metals Inc. | Dundee Precious Metals Reports High-Grade Copper-Gold Discoveries within One Kilometre of the Čoka Rakita Project. Results include 63 metres at 1.74% Cu, 2.18 g/t Au and 9.04 g/t Ag).

The high-grade copper-gold mineralisation encountered is found as stratabound mineralised zones developed within reactive sedimentary units which occur in proximity to fertile diorite porphyries that also exhibit weak to moderate copper-gold mineralisation. DPM's exploration teams have observed features that are characteristic of a transition between porphyry copper-gold-molybdenum mineralisation and IOCG mineral systems that would imply camp scale potential for stratabound skarn mineralisation when evaluating these traits against the current geological architecture.

Figures 23.2 and 23.3 illustrate a plan and a section view highlighting the location of Dumitru Potok and Frasen prospects relative to Čoka Rakita and recent drilling on those prospects.

At the Dumitru Potok prospect, located approximately one kilometre northeast of Čoka Rakita, a directional drilling program with multiple daughter holes outlined a subvertical fertile monzodiorite intrusive body, with proximal high-grade stratabound copper-gold-silver mineralisation within the conglomerate-marble contact zone. Mineralisation is found at both the eastern and western sides of the intrusion.

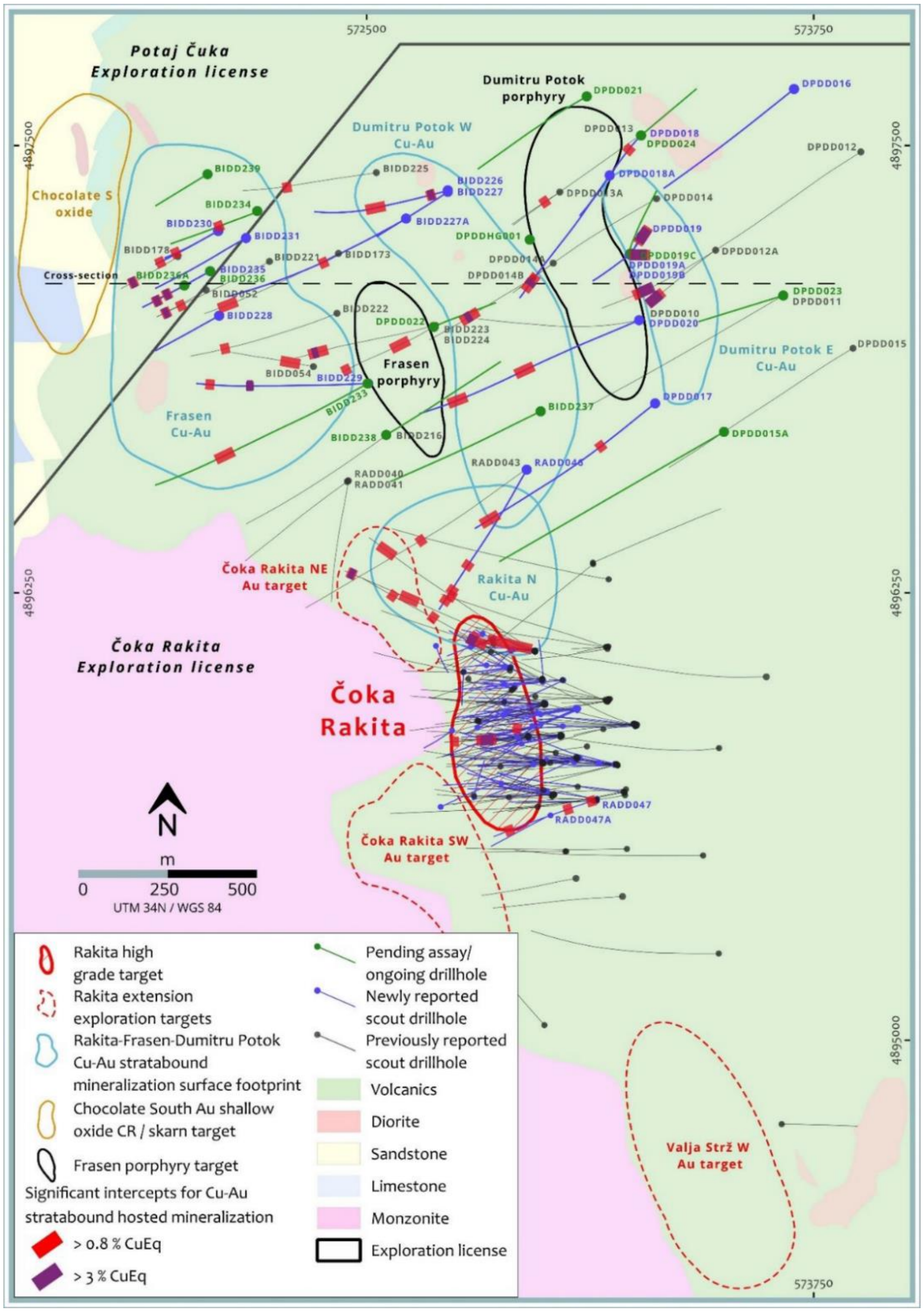
At the Frasen prospect, located approximately one kilometre north-west of Čoka Rakita, systematic drilling has confirmed manto-like carbonate-hosted replacement and skarn mineralisation at the conglomerate-marble contact over an area of 700 m by 500 m. Drilling aimed to test for mineralisation up-stratigraphy and westward from the Frasen-porphyry that was previously intersected in a drillhole and is considered as the causative intrusion for high-grade manto-like copper-gold replacement mineralisation.

The interpretation of the results suggests that high-grade stratabound copper-gold skarn mineralisation extends below Čoka Rakita and is open to south. Further drilling is planned to follow-up on these results, as well as to test the larger footprint of coincident surface geochemical anomaly and to test the target skarn stratigraphy, which is still open to the south and southwest from Čoka Rakita.

DPM intends to continue testing the Dumitru Potok and Frasen prospects to ascertain the potential size, shape and grades.

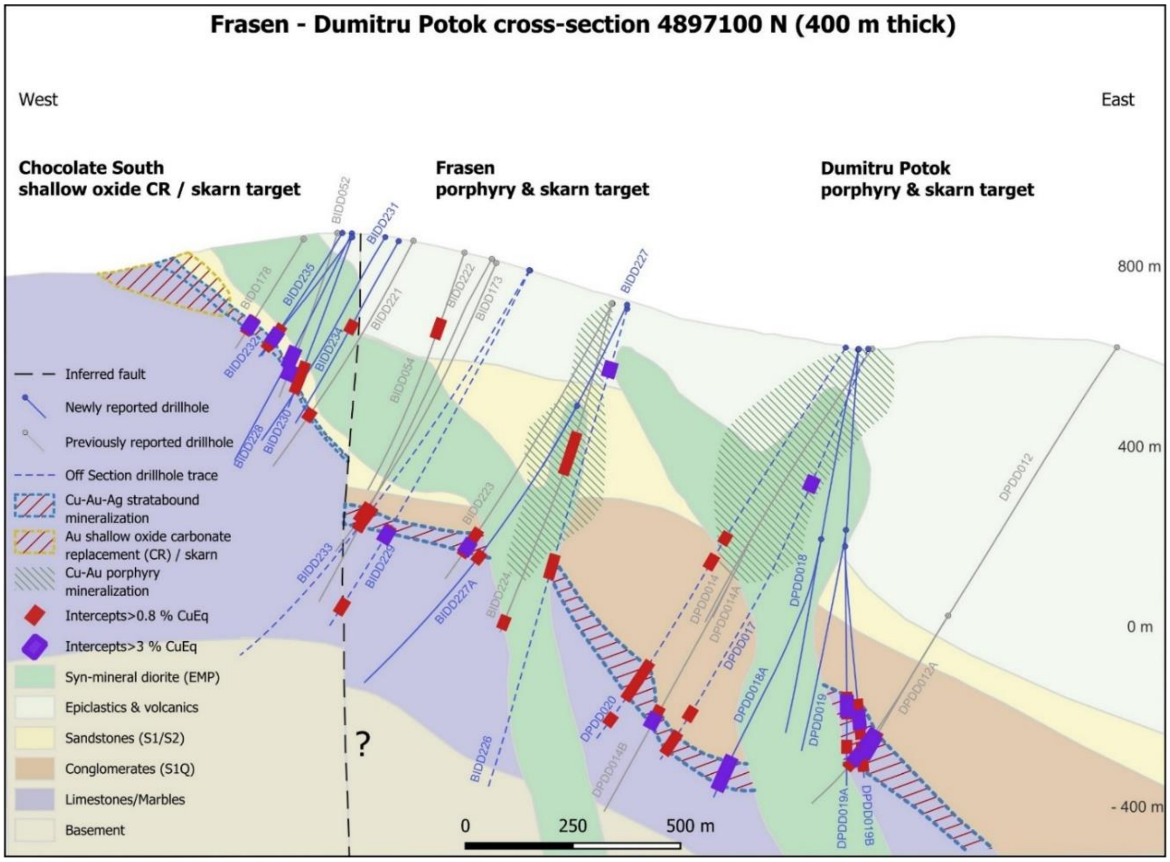
It is noted that the QP (Maria O'Connor, MAIG) has been unable to verify the scientific and technical information disclosed above on the Dumitru Potok and Frasen prospects and this information is not necessarily indicative of the mineralisation and resource potential of the Čoka Rakita Project that is subject of this Report.

Figure 23.2 – Project Scale Map of Frasen and Dumitru Potok Targets and Results from the Ongoing Scout Drilling Program



Source: DPM, 2024

Figure 23.3 – Cross Section Looking North at the Frasen and Dumitru Potok Targets



Source: DPM, 2024

24 OTHER RELEVANT DATA AND INFORMATION

24.1 Project Implementation Schedule

The Project Implementation Schedule includes the main engineering, procurement and construction activities. The information contained in this schedule is derived from information taken from supplier's quotes or in-house database. and it presents the total duration of the Project, considering Project Financing is available and environmental authorisations for construction are available, as identified.

The Project Implementation Schedule has been developed to include a ten (10) month FS and basic design phase. Time has been allocated for Project approval and Project financing. It is assumed that the environmental approvals process is continuing and in place prior to Project start, based on the latest permitting timeline provided by DPM.

24.1.1 Feasibility Study Phase

During the FS phase, the main focus will be to further develop the Project and reduce overall risks. and realise opportunities to further improve the project economics, while supporting the permitting and environmental approval process.

The activities will mainly focus on the following:

- Complete final borehole drilling, to better delineate the underground mine, to convert the mineral resources into mineral reserves.
- Update resource estimate and mine-plan.
- Complete final geotechnical borehole and test pit drilling, to establish an accurate geotechnical model by the geotechnical engineering company.
- Incorporate geotechnical, geochemical and hydrogeological report information into the FS designs.
- Obtain representative drilling samples for metallurgical testwork.
- Confirm metallurgical testwork, from the PFS, to improve recoveries, equipment sizing and throughputs.
- Receive the results of DPM's detailed analysis of the two (2) potential electrical supply options. and determine the optimal solution for the Project.
- Derive a more accurate capital and operating cost estimate and acceptable financial evaluation. and establish final recommendations.
- Obtain a greater level of engineering and design definition.
- Progress permitting and environmental approvals.

- Establish a Project Execution Plan (PEP) and Operational Readiness Plan (ORP). which are relevant to constructing the Čoka Rakita Project.

24.1.2 EPCM Phase

The Engineering, Procurement and Construction Management (EPCM) implementation schedule has been developed based on long lead delivery of the new process equipment required for the Čoka Rakita Project.

To reduce the schedule risk and costs of the Čoka Rakita Project, a significant amount of the Ada Tepe equipment and infrastructure has been identified in a ToS developed by DRA, for re-use at the Čoka Rakita site. All of the re-purposed equipment has been deemed to be off the critical path, as DPM has confirmed that the Ada Tepe facility will be shut-down and decommissioned, well in advance of the construction phase for Čoka Rakita. All activities associated with the shut-down and dismantling of the Ada Facility are planned to be executed by DPM's operation's team, with costs and durations established by DPM.

DPM has, also, identified that a significant amount of the surface mobile equipment will be relocated from DPM's Ada Tepe facility.

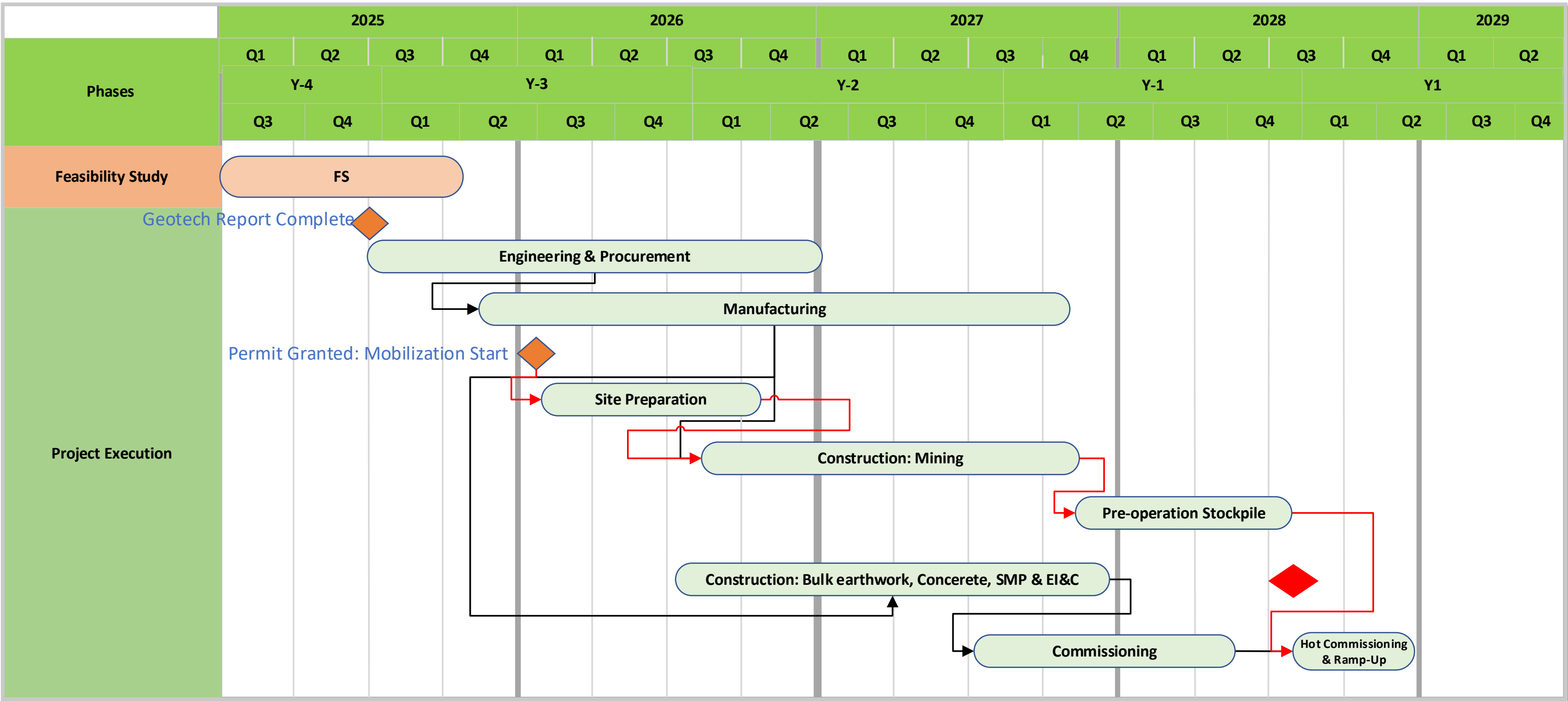
During the FS and basic design phases, prior to the EPCM phase, emphasis will be made to:

- Define the EPCM phase. Most likely that these will be two (2) distinct phases (EP and CM).
- Advance procurement of long lead process equipment items.
- Develop the site engineering and design required to establish the infrastructure and site preparation to adequately prepare for the underground preparation and construction phases.
- Establish the detailed design of the temporary and permanent access to the Čoka Rakita site.
- Determine logistics for equipment and material(s) deliveries, laydown areas and security.
- Establish procurement for the equipment and bulks, including bid preparation and evaluation, organisation of site visits, contract preparation and contract administration.
- Mobilise the construction management team to site, providing site assistance when needed. and supervising dry and wet commissioning and ramp-up.

24.1.3 Project Implementation Schedule

A high-level Project Development and Implementation Schedule is provided in Figure 24.1, based upon the Project information available to date.

Figure 24.1 – Overall Key Date Schedule



Site Preparation: Trees Removal, Temporary Access Road, Terrace and Box Cut

Source: DRA, 2024

24.2 Project Execution Plan

The Project Execution Plan (PEP) defines the methods and project management elements that will be used to manage the execution of the Project. The PEP establishes the execution philosophy and defines the organisation, work processes and systems necessary for management of the Project. The PEP will be further developed by the DPM project team during the FS and basic design phases.

It is critical to ensure all stakeholders are aligned and that the project controls are in place. The following list summarises the main items to be addressed and developed in the PEP, to align DPM and all stakeholders on the intended execution strategy.

Project Goals and Objectives:

- Project Location.
- Scope of Facilities and Overview.
- Project Organisation.
- Health, Safety, and Environment Execution Plan.
- Risk Management.
- Engineering Execution Plan.
- Procurement and Contracting Execution Plan.
- Construction Execution Plan.
- Commissioning Execution Plan.
- Project Controls Strategy / Objectives.
- Electronic Document Management Systems.
- Project Communications. and
- Public Relations

24.3 Operational Readiness Development

An Operational Readiness Plan (ORP) has been developed by DPM’s Operations Readiness team.

This ORP defines the methods and management elements that will be used to prepare the Project for successful handover to the DPM operations and maintenance teams.

24.3.1 ORP Process Overview

The ORP is focussed on readying operations and maintenance departments to effectively accept handover and operate newly constructed infrastructure and equipment with minimal destruction of business value during the initial ramp-up phase of the Project.

24.3.2 PURPOSE OF THE ORP

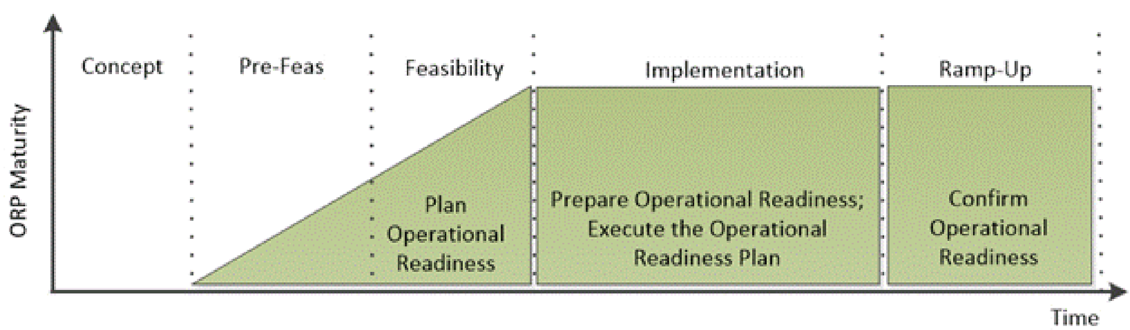
The aim of the ORP is to:

- Ensure that the DPM operations and maintenance teams' for Čoka Rakita are trained and prepared for a successful handover of the facility, from the construction and commissioning teams. That the correct people, are effectively trained and prepared to receive the handover of the plant, at the correct time. are prepared to work with the equipment, systems and technology, in a safety, environmental, reliable and efficient manner.
- Identify and address all significant operational issues, from FS through to Project handover.
- Ensure appropriate operational procedures, systems and people are in place to achieve and sustain the project objectives.
- Ensure consistent and coordinated design of all enterprise systems, to ensure a smooth transition from construction to commissioning to startup of operations and attaining ramp-up production targets. Further, embedded risk management and compliance activities will enable the business to achieve sustained delivery of business values, objectives and competitive advantages throughout the business cycle.
- Ensure infrastructure is safely and efficiently operated, aligned with business value targets.
- Ensure stakeholders are provided with a high level of satisfaction, pertaining the new assets.
- Ensure business risks, associated with new assets, eliminated and/or mitigated.
- Ensure compliance with statutory environmental and legislative requirements

24.3.3 ORP PHASES

The ORP will commence during the PFS stage with an initial study, by the DPM OR team. The plan and scope will be matured during the FS phase, followed by implementation during the execution phase. The ORP, also, extends beyond the handover period, into the ramp-up and steady state phases, as an overarching plan linking the phases together.

Figure 24.2 – ORP Process



Source: DPM, 2024

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 30 August 2024. A total of 271 drillholes totalling 126,495 m were included in the estimation of the MRE. The current drillhole spacing within the mineralised domains is predominantly at least 30 m x 30 m. 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 20% in the main mineralisation domain. A further strategy of capping within the estimate at distances greater than 30 m was also applied.

Mineral resource domains were created within volumes of moderate to intense skarn alteration and guided by grade composites generated at an approximate1 g/t Au cut-off value. Detailed lithology and structural models were developed and used to constrain domain extents, as well as to incorporate mafic sills which can either cut across the mineralisation or are capping structures to trap the mineralisation as well as continuous internal waste units. 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 5 mZ parent block size with sub-celling to honour domain volumes.

The Mineral Resource was reported exclusive of Mineral Reserves. A breakeven cut-off value of 2 g/t Au and a minimum width constraint of 3 m was used to define optimised mineable shapes using Deswik's Stope Optimizer (DSO) process to support Reasonable Prospects for Eventual Economic Extraction (RPEEE).

Material within the reporting constraints was classified as Indicated and Inferred Mineral Resources according to Mineral Resource confidence definitions 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. A Drill Hole Spacing Study (DHSS) was completed in 2024 which supported drill spacing of 30 m x 30 m for the classification of Indicated Mineral Resources, based on achieving a ±15% tolerance (with 90% confidence) on annual production volumes (Parker et al, 2014)

25.2 Mineral Processing and Metallurgical Testing

Testing demonstrated the amenability of the samples to treatment through the grind-gravity-float flowsheet and provided adequate data for the sizing of all major equipment for a PFS.

In the QP's (Daniel (Niel) Morrison, P. Eng.) opinion:

- Metallurgical testwork completed to date has been appropriate to establish appropriate 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.

The QP is not aware of any processing factors or deleterious elements, other than arsenic, that could have a significant impact on potential economic extraction. Blending of the final concentrates will be required to minimise penalties for exceeding the arsenic grade threshold.

25.3 Mineral Reserve Estimate

The Mineral Reserve estimation for the Project adheres to industry best practices and complies with the 2019 CIM Definition Standards. Mineral Resources were converted to Mineral Reserves based on a detailed mine plan, engineering analysis, and modifying factors such as dilution, mining widths, ore and extraction losses, suitable mining methods, metallurgical recoveries, permitting, and infrastructure needs.

Factors that may affect the Mineral Reserve estimates are geological complexity impacting grade. geotechnical and design parameters changes impacting dilution and mining recovery factors. lower mill recovery in new mining areas, fluctuations in commodity price and exchange rate as well as mining costs assumptions.

25.4 Mining Methods

25.4.1 Underground Geotechnical

The underground geotechnical assessments for the Čoka Rakita PFS were collected as part of the geotechnical and hydrogeological investigations program completed by DPM and their local contactors (Faculty of Mining and Geology at the University of Belgrade). The WSP underground geotechnical team worked closely with both DPM and the University of Belgrade staff during the collection of the data in 2024.

A total of 15 boreholes (8751 m), primarily located in the hanging wall, footwall and ore zones, were drilled, logged geotechnically, sampled, and partially surveyed with downhole geophysics to support the various PFS underground geotechnical assessments. Some of the data was not yet captured when the rock mechanics assessments were completed, this was intentional to meet the PFS schedule. Geotechnical data for the upper and lower declines, spiral ramp and surface vent raise are planned to be collected in Q1 2025.

Rock mass classification assessments (RMR_{76} and Q' values) were completed based on the data available and divided into seven (7) main lithologies. The main rock masses are classified as exhibiting fair to good average character and have similar classification averages. DPM identified numerous fault planes/structures located in the hanging wall, footwall and ore zones that also intersect some of the underground infrastructure (e.g. underground maintenance area). These fault zones create lower rock mass quality areas approximately 0.5 m to several meters in width (poor to very poor) that can have an impact on local stability and increase stope dilution. Ground support requirements for these fault zones will be increased (type of support and density of support) compared to the standard support in the ramp, access and development drifts.

An empirical stope stability assessment and 3D numerical modelling assessment was completed to review the PFS stope configuration and standoff distances for the access drifts, spiral ramp and surface vent raise to identify potential instability issues. The access of the deposit from the hanging wall area and the main mining method selected as longitudinal open stope with backfill were also supported by the underground geotechnical assessments completed. The temporary sill pillar was assessed using 3D numerical model and was determined not to have fatal flaws but the mining sequence could be refined during the next stage of study to further limit induced stress challenges anticipated when mining the sill pillar. Additional geotechnical assessments (empirical and numerical) were completed on the key infrastructure including surface vent raise, upper and lower declines to define minimum ground support requirements.

The available underground geotechnical data collected, and empirical and numerical assessments completed confirmed the key assumptions for the PFS mine design (e.g. stope size, stope mining general sequence, location of key infrastructure, ground support). Geotechnical data collected in Q4 2024 and Q1 2025 will need to be reviewed and applied where required to confirm and/or update the PFS mine design including the key infrastructure excavations.

25.4.2 HYDROGEOLOGY

The hydrogeological data for the Čoka Rakita PFS were collected as part of the geotechnical and hydrogeological investigations carried out in 2024 in support of the surface and underground mine infrastructure design. The objectives of the hydrogeological investigations were to assess the groundwater conditions within the areas of the proposed mine development. Selected boreholes across the site were instrumented to allow collection of additional hydrogeological data and to establish baseline conditions prior to mine construction.

Following completion of the field program, WSP compiled the available data as part of a conceptual hydrogeological assessment of the project area. The conceptual hydrogeological model formed the bases for development of a 3D numerical groundwater flow model that was used to predict groundwater inflows into the underground development during the LOM in support of the underground water balance and design of the dewatering infrastructure. Total predicted annual groundwater inflows to the mine reached a peak of approximately 20 L/s during the Mine Year 1. After that the inflows remained relatively steady until the end of the simulation with inflows varying from about 17.5 L/s to 18.5 L/s.

The available information provides an understanding of the hydrogeological conditions within the proposed project and surrounding areas required for the current stage of the project. However, additional data collection is recommended to advance the understanding of the hydrogeological conditions at the site as summarised in Section 26.6.3 (Recommendations).

25.4.3 MINING METHODS

Multiple aspects of the project have been investigated to develop a suitable mining method for the Čoka Rakita deposit, here are the conclusions and recommendations in that regard:

- Developing the mine from the hanging wall side of the deposit is the correct approach, given the current understanding of the ground conditions.
- Sublevel long-hole open 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.
- In order to reach the targeted development rates, it is recommended to investigate the potential use of contractors, improved equipment performance, revised schedules and other incentives to complete the planned development.

25.4.4 Electrical and Communications

The scope of the PFS is limited to the underground power distribution system and above ground main exhaust fans E-House including 690 V gear, VSDs and auxiliary power. 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.

25.4.5 Mine Backfill

The design and operation of the paste backfill and cemented rock fill systems at Čoka Rakita are typical of most mines and their implementation is not foreseen to be overly changing at this time and are worthy of refinement in future phase of this project.

It is believed that the biggest factor in reducing the operating cost of backfill on the project will be refining the possible cost benefits using a binder formulated with a blend of ground iron blast furnace slag and normal Portland cement.

25.4.6 Mine Dewatering

Ground water inflow is a significant contributing component to the total water pumped from the Mine.

The geotechnical and hydrogeological field program is still in progress and when completed and the results assessed, the mine wide dewatering requirement will be better defined in the FS phase of the Project.

Refinements in other contributing factors to the mine dewatering requirement such as the changes in the mining rate, paste backfill demand, or mine mobile equipment fleet, will also serve to further refine the design of the dewatering network and the size and quantity of pumping equipment required.

The current design of the Mine dewatering system utilises a good combination of gravity flow and pumping and while not introducing design complexities.

25.5 Recovery Methods

The main conclusions arrived at during the trade-off studies are:

- For the treatment of gravity concentrate the logical solution to the analytical challenges posed by the presence of coarse gold is to concentrate it into a high-grade secondary gravity concentrate which can then be smelted and sold as doré bars.
- The most economical and most flexible flowsheet is to re-use the equipment at Ada Tepe with the tumbling mill operated in autogenous grinding mode, which eliminates the operating cost associated with grinding media. The Ada Tepe pebble crusher will be re-used to avoid build-up of pebbles in the primary circuit. One of the vertical stirred mills from Ada Tepe will perform secondary grinding duties. Both circuits will be closed out by hydrocyclones.
- Jameson cells will be used instead of the refurbished SFR cells ex-Ada Tepe. The Jameson circuit is significantly simpler and less expensive. and its adoption avoids the complexity and expense of de-commissioning and dismantling the larger number of SFR cells at Ada Tepe.

25.6 Filter and Paste Backfill Plant

The PFS design of the Filter and Paste Plants is based on information provided by DPM and laboratory testing by RMS Corp. and BaseMet Labs. The level of engineering and design is commensurate with that expected for a PFS design.

The design and capacity of the Paste Plant is solely focused on the use of cemented paste fill (CPF) for the mine since although cemented rock fill (CRF) will offset the quantity of CPF required in certain years, there will be years near the end of mine life which will solely rely on CPF, since cemented rock fill will no longer be available.

25.7 Environmental Studies, Permitting and Social or Community Impact

DPM has been actively working in the Čoka Rakita and adjacent Timok area since 2007, 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 were considered in the assessment of the various alternatives, trade-off and options studies in the PFS design. Key environmental and social mitigation features include:

- A relatively small footprint, which comprises DTSF, process plant, portal and ROM pad site and paste backfill plant which minimises the area of habitats impacted by land take.
- Concurrently using the potentially-acid generating rock for underground backfilling and non-acid generating tailings to surface disposal and paste backfill in order to mitigate environmental impact.
- Use of liners where appropriate to manage potential acid and metal leaching risks to groundwater and surface water, such as for the saturated PAG waste rock at the base of the DTSF.
- 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.

25.8 Capital and Operating Cost Estimates

The capital and operating costs were estimated using methods described in this Report, which are typical of PFS-level cost estimation exercises.

25.9 Economic Analysis

Based on the available information, the project has an after-tax NPV of $735.0M at a discount rate of 5% and an IRR of 41.4%. The sensitivity analysis indicates that the Project economics are most sensitive to the gold price. Even with a gold price 20% below the base case of $1,900/oz, the Project maintains a positive after-tax NPV.

25.10 Other Relevant Data and Information

The Project is greenfield and involves construction of a 70 kt per month concentrator plant. The Project Implementation Schedule assumes a continuous flow of work from the completion of the PFS to the commissioning and startup of the Project in early 2028. The FS and Basic engineering design of the Project will be executed by DRA and WSP, with support from ERM and DPM's project management team. The Operational Readiness Development Plan (ORP) is an essential component of the Project, which is focused on readying the business to accept the newly constructed infrastructure and equipment. Preparation of the ORP starts during the PFS stage, requiring further definition of strategies and actions to be developed during the FS phase.

An overview of the operational phase strategy and assumptions is required, including an analysis of the business units, operational strategies and workforce quantification. A clear understanding of cost allocations for operating and support personnel is also critical.

The Project Execution Plan and Schedule will continue to evolve in the next phases of the Project. Effective execution of these plans require the cooperation and effort of all stakeholders, to ensure successful handover and ramp-up phases after the commission and startup of the Project in Q3 2028.

25.11 Opportunities

25.11.1 Mineral Processing and Metallurgical Testing

- Using cyanide to leach residual gold from tailings or from the whole ore remains an opportunity to improve recoveries but this would necessitate a change to the permitting strategy. The possibility of improved gold recovery in the finer size ranges due to the use of Jameson cells should be explored in future phases. Additional testing could also result in optimisation of reagent requirements.

25.11.2 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.
- Assess the possibility of adding incremental stops to reserves where possible.
- 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 ore or adjacent to the hanging wall.
- Optimising various auxiliary infrastructure and shortening up lateral development can assist by cutting down on auxiliary ventilation. Anything longer than 6m requires auxiliary ventilation and as such it may be worth exploring different layouts / arrangement profiles to reduce additional ventilation and power requirements.
- Explore the possibility of cutting back on development by studying the possibility, if geotechnical conditions and drilling performances allow, of increasing the height between sublevels by 10m and cutting back every other sublevel.
- Explore the possibility of switching to a working schedule of 2 shifts of 12 hours per day and examine whether if it would improve productivity.

25.11.3 Electrical and Communications

- Evaluating the potential to route cables down internal ventilation raises can save on service borehole development.

25.11.4 Recovery Methods

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.

The smaller flotation concentrate flocculant mixing and storage system was used to provide flocculant to the flotation thickener in the PFS, but it can be eliminated if the thickeners are in close proximity to each other. The tailings thickener flocculant system is much larger and has sufficient spare capacity to accommodate the extra end-user.

The two 30” centrifugal gravity concentrators used in the PFS are significantly oversized for the duty and can be replaced with 20” models. Apart from the Capex and Opex savings, the smaller models requires significant less fluidisation water which reduces the dilution of slurry in the secondary grinding circuit.

25.11.5 Filter and Paste Backfill Plant

The footprint area available for the Filter and Paste Plants is limiting in the design of the inclined conveyor between the Filter and Paste Plant. The inclined conveyor is somewhat steeper then ideal. however, it is still acceptable over the short distance between the plants.

Refinements in the Crusher Terrace such as the use of a retaining wall in place of the slope from the terrace above, or to the other plants on the crusher terrace may provide a larger available footprint for the two plants and a decrease in the angle of the inclined conveyor in the FS phase.

25.12 Risks

The Project team (DPM, DRA, ERM, WSP) conducted a multi-day project risk review workshop during the week of August 12, 2024, with the expectation of identifying and evaluating potential risks that could impact the overall success of the Project.

During this workshop, potential risks were identified, with each item's cause, likelihood, and potential impact evaluated across different inherent risk categories. The PFS risk review workshop utilised the risk register established during the PEA as the basis for the PFS workshop.

At completion of the PFS risk workshop, additional activities were conducted. The first activity grouped similar risks together, as per DPM corporate risk procedure and then applied the DPM corporate 5x5 risk matrix to establish the individual risk ratings.

After the DPM ratings were applied, a second activity determined the potential impact reductions which could be assumed, if effective mitigation measures are implemented into the design and/or procedures.

The total number of risks identified using the DPM corporate methodologies was 90 risks. which comprised: 36 “High” risks, 48 “Moderate” risks, and 6 “Low” risks.

Once the mitigation activity was completed, it was identified that several of the risks could be potentially re-rated to comprise: 11 “High” risks, 60 “Moderate” risks, and 19 “Low” risks.

A number of the identified risks can be successfully mitigated if the following activities are completed:

Mining and Geology

All mining designs, with proper mitigations, need to be completed during the FS.

Process

For the process designs to be completed and the risks mitigated effectively, all testwork needs to be concluded early in the FS phase, to ensure that the level of risk is correctly identified. and the appropriate mitigations are established in the design. The Geotech is one of the greatest concerns associated with the effective layout of the process equipment. Once the Geotech has been confirmed, DRA can ensure that the layout considers all necessary mitigations.

The risk of attracting penalties for exceeding the arsenic threshold should be addressed through a comprehensive risk mitigation strategy that encompasses all of the mine planning, provision for blending of ROM feed as well as blending of the final product.

Dry Tailings Storage Facility

The top five (5) risks identified within the Čoka Rakita PFS DTSF design are as follows:

1. Dam Failure – mitigated by use of filtered tailings technology to reduce impact of DTSF failures.
2. Potentially Acid Generating – mitigated by lining the PAG rock zone with composite lining systems and inclusion of the material within the DTSF footprint.
3. Seepage to environment – mitigated by use of composite lining system accompanied by basal underdrainage channel with access / monitoring point downstream of the DTSF.
4. Instability caused by inadequate construction material – mitigated by Construction Quality Assurance (CQA) and frequency of laboratory testing. and

5. Instability of access road cuts in weathered material – mitigated by reducing slope angle which may involve generating additional excavation material or through slope reinforcements.

Further risks associated with the DTSF were also identified and these, as well as the key risks above, will be evaluated in greater detail during the FS design.

Environment

All environmental mitigations, identified during the PFS, need to be addressed during the FS, to ensure a robust design which eliminates all potential for environmental Extreme-risk and High-risk items.

Economic Analysis

The Project economic performance is highly sensitive to the price of gold, as demonstrated in the sensitivity analysis. A key risk is the possibility of a significant decline in the price of gold during the life of the Project, which would negatively impact the Project economics. This risk is somewhat mitigated by the fact that the selected gold price used in the analysis is below the current spot gold price.

26 RECOMMENDATIONS

26.1 Proposed Work Program

The following activities should be undertaken in the next phase of the Project, leading up to the completion of the FS. 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)
Geology and Exploration	0.2
Geotechnical / Hydrological	1.6
Mining	1.3
Metallurgical Processing	0.4
Environmental Studies	0.7
Permitting	1.6
Feasibility Study	11.1
Engineering	3.4
Subtotal	*20.3*
Contingency	2.3
Total	**22.6**

Excluding operational readiness, land acquisition, and Owner's cost.

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

Infill drilling needs to be completed by end of 2024 to comply with Serbian law. Drilling to at least 20 x 20 m has been identified as achieving quarterly production volume tolerances of ±15% moisture based on a DHSS conducted in 2024, which can be considered appropriate for classification

Measured Mineral Resources, Priority is recommended to be given to drill holes covering the first year of production.

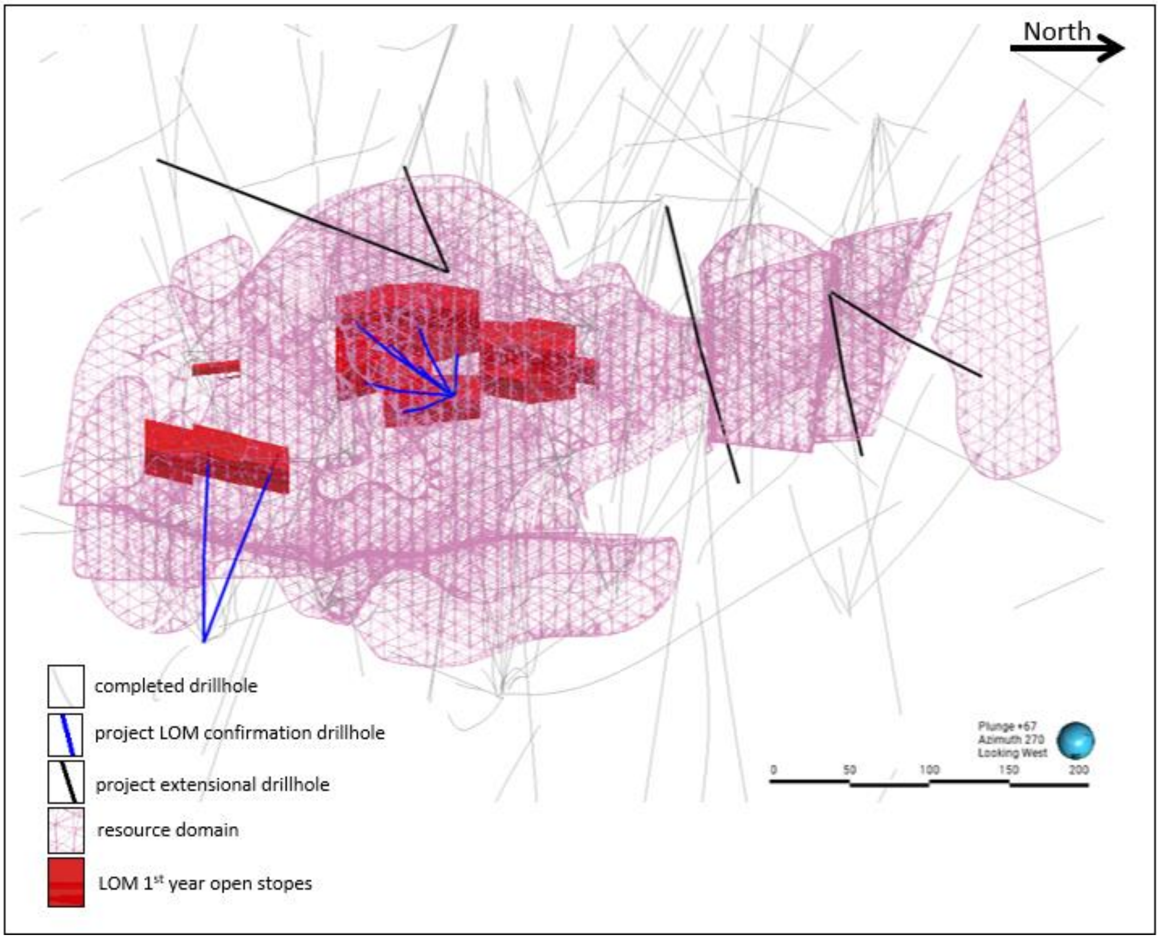
DPM is planning a drilling program by the end of 2025 to support further technical studies and ensure alignment between production and the FS model (Table 26.2). Current drilling plans include:

- Approximately 5,000 m of drilling will be conducted as part of the FS study for the Čoka Rakita project. This phase includes close-spaced infill drilling (15-20 m drill grid), with a primary goal of better constraining mineralisation and minimising grade uncertainty within the volumes planned for mining during the first year of the PFS LOM. In addition, approximately 4,000 m of drilling are planned in the peripheral parts of the mineralised domains. These drillholes are designed to test for extensions to the current mineralisation domains (Figure 26.1). Additionally, about 3,600 m of near-surface geotechnical and hydrogeology drilling is planned for site infrastructure investigations.
- Additionally, DPM has plans to complete approximately 40,000 m of additional exploration drilling at existing skarn targets and to test for manto-like copper-gold skarn identified across the Čoka Rakita licence and neighbouring licences held by DPM, including Potaj Čuka and Pešter Jug.

Table 26.2 – Čoka Rakita Licence – FS Planned Drilling Metres and Budget

Phase	Drilling category	Planned Metres	Budget (US$)
FS	LOM 1st year confirmation drilling	5,000	900,000
FS	Extensional drilling	4,000	700,000
FS	Geotechnical/hydrogeology drilling	3,600	600,000
2025 Exploration	Exploration Drilling	44,000	7,900,000
Total		**56,600**	**10,100,000**

Figure 26.1 – Confirmation and Extensional Drilling Plan for FS at Čoka Rakita



Source: DPM, 2024

26.2.3 DATABASE

DPM is using a reliable and well-known solution to capture and manage the data (acQuire). However, the database and data management practice are still evolving. To ensure that 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 sulphide and vein table, meaning, e.g. missing Lithology 1, Sulphide 1, Vein 1 respectively. Generally, this would lead to a validation error in relational databases, meaning a non-standard approach may have been used and this should be modified.
- Hole size information is present in the Geotech table for the diamond holes but no information was present for other hole types. Best practise suggests that this be stored in a table for all

holes and should also contain information such as start/end dates for each particular hole size, drill contractor and rig as a minimum.

- There are nine diamond holes that are missing recovery and RQD information with 0.03% of data having a percentage value outside the recommended tolerance of 110%, this data should be reviewed.
- Structure data should be presented in two separate tables, one for interval data and another for point data.
- 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 exported with the CRM data.
- 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 Gold 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 123 umpire samples were selected and assayed during 2024 at ALS_BO laboratory, however, these were not analysed for Gold. The Silver, Arsenic, Copper and Sulphide results reviewed demonstrate overall strong consistency and reliability between the original and umpire assay data, suggesting confidence in the data quality. Minor outliers and discrepancies should be reviewed, especially for lower concentration samples.

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 were obtained and inserted for this purpose in 2024 e.g. OREAS 993.

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.

For the QC data (Duplicates, Repeats and Splits) the overall quality of the data appears to be good, but attention should be paid to the outliers at higher concentrations, particularly for gold and silver.

26.3 Mineral Processing and Metallurgical Testing

The QP (Daniel (Niel) Morrison, P. Eng.) recommends the following:

1. Proposed testing program for further process development

- Flotation reagent optimisation tests should be performed. These tests should be performed with combinations of a range of PAX and A3477 additions.
- A small-scale Jameson cell pilot plant test run should be performed to confirm the design proposed by Glencore and to establish reagent requirements.
- 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.

2. Additional 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 and were 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.

26.4 Mineral Resource Estimate

It is recommended that the MRE be updated, based on new drilling and screen fire assays, where possible. This will form the basis of the FS, due to commence in 2025.

26.5 Mineral Reserve Estimate

1. DPM should prioritise infill drilling within the deposit areas defined in the PFS, to rapidly convert Inferred Mineral Resources to the Indicated category. Simultaneously, an exploration expansion drilling program should be conducted to grow the mineral resource base and enhance the Project's overall value.
2. Incorporating unplanned mining dilution into the stope optimisation process before running the DSO.
3. Investigating the possibility of using an incremental COG to increase reserves by including material with lower economic margins (when it's economically feasible) into the mine plan.

26.6 Mining Methods

26.6.1 MINING

1. Investigation to optimise all infrastructure excavations – potential for lateral development reduction.
2. Expand maintenance strategies and knowledge to build fit for purpose maintenance shop and associated auxiliary infrastructure.
3. For future studies, refine the estimates of the unplanned dilution and mining recovery factors based on the mining method, geotechnical conditions, and backfill properties.
4. For future studies, incorporate passing bays and safety cuddies into the spiral ramp design to enhance operational efficiency and safety.
5. Conduct underground haulage and material handling simulation to determine the ability of the base design to achieve target throughputs for ore, waste, personnel and material movement. fleet size optimisation. test potential impacts of traffic congestion.
6. Serbian law does not require the installation of mine rescue chambers but rather this is managed according to best practices. Distances between portable mine refuge chambers are calculated based on the usage time of self-rescuers. Based on the above, it may be worth consideration to implement portable mine rescue chambers in the upper and lower declines as an additional safeguard.
7. Consider implementing dedicated escapeway raises in the mine rather than installing ladderways in return air ventilation raises.
8. 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.
9. The pre-production period is very important for the project, and special attention must be accorded to it, in terms of purchasing equipment, recruiting and training personnel, and scheduling surface construction work.
10. Evaluate potential advantages of engaging a mining contractor to drive the lower and upper 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.

26.6.2 GEOTECHNICAL

1. Conduct a geotechnical drilling investigation along the proposed upper and lower decline alignments and in the surface ventilation raise.
2. 3D plastic numerical induced stress modelling for the life of mine infrastructure and sequence.

3. Optimise mine sequencing in the temporary sill pillar to best manage induced stress (avoid diminishing pillars).

4. In situ stress testing within the mining horizon. This can be completed during the FS investigation program or during initial develop of the declines. Testing should follow industry recognised standard practices (e.g., over coring, mini-frac, etc.).

5. Update the engineering geology model and geotechnical assessments and recommendations considering the full data suite from the 2024 geotechnical investigation (which was not available, in its entirety, during completion the PFS assessments) and the data collected from the upper and lower decline and the surface ventilation raise.

6. Develop a geotechnical block model considering all available geotechnical investigation datasets.

7. Further investigate the differences in geotechnical characteristics between the logged lithologies and the geological formations. If necessary, produce wireframes of logged lithologies for the FS assessments.

8. Continue to update the fault model and character matrix.

9. Further examine longevity of split sets in permanent headings based on empirical data collected from Chelopech and from groundwater chemistry data (pH) at Čoka Rakita.

10. Examine potential for the development of a discrete fracture network model of the structure at Čoka Rakita with the intention of better understanding the statistical distribution of possible wedge sizes to better optimise ground support design as the critical failure mechanism in most of Čoka Rakita is anticipated to be kinematics.

11. Optimise critical infrastructure (maintenance shop, spiral ramp, etc.) placement based on the most recent fault model update (October 2024) which suggests some faults may intersect the shop and run parallel within portions of the ramp.

12. Update the portal design when site specific geotechnical data is available. Further examine the trade-off between the use of drill and blast and a road header.

26.6.3 HYDROGEOLOGY

Based on the results of the hydrogeological assessment carried out as part of the PFS, the following presents a summary of recommendations for additional data collection:

1. Continue monitoring of groundwater levels and hydraulic heads within the Čoka Rakita project area including installed monitoring wells and vibrating wire piezometers.

2. Continue collection of background groundwater and surface water quality samples from monitoring wells and catchment areas prior to construction of mine facilities.

3. Continue monitoring of surface water receptors.

4. Additional hydrogeological testing in the study area to obtain additional information to confirm effectiveness of dewatering.

5. Additional investigation into the limestone unit underlying the ore body to obtain additional information required to identify potential connection between the units.

6. Additional testing of major faults to further assess their role in the groundwater regime within the project area.

7. Update of the conceptual model and recalibration of the numerical groundwater model as additional data becomes available.

8. Updated inflow assessments using the recalibrated numerical groundwater flow model.

26.6.4 Ventilation

1. To prevent the formation of dust, WSP recommends regularly spraying water at the tapping points and installing water spray systems in both declines.

2. During the FS, internal ventilation raises sizing should be revised to optimise Capex and Opex, a simulation of the distribution of equipment between the various active production levels is required.

3. A ToS should be conducted to have the ventilation raises closer to the sills to reduce ventilation duct requirements.

4. A ToS should be conducted to determine the optimal Ventilation-On-Demand level of control to optimise Capex and Opex.

5. Following a blast, ventilation curtains will need to be installed to avoid stope leakage. Personnel need to be provisioned to install and maintain the ventilation curtains to ensure adequate auxiliary ventilation on the production levels

26.6.5 Mine Backfill

1. Additional CPF laboratory testing is recommended to determine the binder addition rates required for Serbian supplied mixtures of blast furnace slag and normal Portland cement binder (90/10), needed across the paste slump range of 178 mm to 254 mm, and to develop a graphical relationship between binder addition rates and unconfined compression strengths at various curing days.

2. The additional testing would allow refinement in the backfill operating cost and aid in the future backfill recipe selection when mining individual levels in the Mine.
3. Additional laboratory testing in the form of flow loop testing is recommended to validate the empirical pipeline friction factors determined in this phase of the study and better define the hydraulic flow characteristics of the paste in the distribution system.
4. Based on the results of the laboratory testing above, it is recommended to conduct a trade-off study to determine if increasing the paste distribution systems design pressure, and associate increase in design ratings and cost of its components, can be justified through a reduction in binder consumption cost associate with delivering a lower slump paste underground.
5. Since CRF will be used in conjunction with paste backfill, and no laboratory cemented rock fill testing has been done to date, a cemented rock fill testing program should be conducted to determine the relationship between binder addition rate and unconfined compressive strength to achieve the target strengths required.
6. The CPF system design has been developed to a PFS level and the designs and concepts are suitable for and should continue to be developed in the subsequent FS phase.

26.6.6 Mine Dewatering

1. Additional information gathered in support of the groundwater inflow estimates should be evaluated and incorporated in the dewatering system design along with other possible changes such as the mining rate or the size of the mobile equipment fleet.
2. The PFS design presents a solid basis to continue its refinement in the FS phase of the project.

26.7 Recovery Methods

Additional strategies for ensuring that the flotation concentrate product avoids attracting penalties for arsenic content should be explored in future phases of the Project.

26.8 Project Infrastructure

At the next stage of the Project, the follow areas are recommended for further investigation:

- Additional exploration and testing to complete detailed designs in the site infrastructure areas where no direct exploration was done.
- Complete 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 trade off studies to determine the optimised foundation systems for buildings and equipment foundations.

- Layout optimisation. Together with point above, review alternate layout options by moving the mine portal closer to AG Mill Feed Coarse Ore Bin and potentially move the paste backfill facility closer to portal or move underground. This would eliminate a terrace and reduce the footprint of the Project.
- Discussion the closure plan and the 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.

26.8.1 Filter and Paste Backfill Plant

At the next stage of the Project, the follow areas are recommended to be investigated further:

- To better develop the paste backfill underground distribution system (UDS) flow model, actual laboratory scale flow loop testing is recommended whose output would generate friction loss versus velocity curves for 150mm diameter:
  - Sch 120 CS pipe,
  - Sch 80 CS pipe,
  - Sch 40 CS pipe, and
  - HDPE DR9.
- The RMS unconfined compressive strength (UCS) testing was conducted with Ontario, Canada sourced ground iron blast furnace slag (BFS) and the results indicate better UCS results over those of normal Portland cement. Since BFS quality and composition is specific to a source location, it is recommended to conduct a UCS testing program using Serbian sourced BFS.
- To quantify the effect of additional fines on the UCS additional fines may be added to the paste. The fines which may be generated through the use of the pressure filter filtrate and wash water in the process, it is recommended to conduct UCS testing on control cylinders (no added fines) and compare the results to those with the enhanced fines added to the paste matrix with a set binder addition rate and at two (2) slump consistencies (178 and 254 mm), and analyse the results over at least 28 days of curing. The fines generated from the filter charging cycle (those that pass through the cloth) will largely be in the 10 µm (or at least sub 20 µm) range. The best fines sample will be obtained from actual pressure filtration testing.
- Vendor pressure filtration testing for at least two selected vendors should be conducted to determine equipment sizing and the establishment of a performance guarantee from the selected vendor.
- Proctor testing should be conducted to determine the required moisture content of the filter cake to satisfy deposition criteria in the TSF. This will also determine the pressure filter design requirements to meet the desired final filter cake consistency.

- Since cemented rock fill (CRF) is also used, and no laboratory testing has been conducted to determine UCS with associated binder addition rates. Testing should be completed to determine the requirements to reach target UCS. Laboratory testing should be conducted across a seven (7) to 28 day CRF cylinder curing period at various binder addition rates.

The following are recommendations to be considered for inclusion in the FS phase of the Project and to be investigated further:

- The selection of cement binder type should be finalised, be it either 100% normal Portland cement, or a 90/10% blend of Ground Iron Blast Furnace Slag / normal Portland cement.
- Since cement binder consumption is a major contributing factor to the Mine backfill costs, additional laboratory testing should be conducted to define the binder addition to UCS relationship over the range of paste slump consistencies from 178 mm to 254 mm.
- Refinements in the LOM Plan will ultimately determining the maximum annual paste backfill production requirements and therefore, the overall paste backfill plant design capacity.
  - The paste backfill plant design criteria and equipment sizing should be re-examined, and a determination made if revisions are required based on the FS LOM Plan.
  - Upon selection of the major equipment vendors, their equipment dimensions should be incorporated into the design of the plants and based on the equipment, the plant functionality refined, as required.

26.8.2 Dry Tailings Storage Facility

At the next stage of the Project, 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 DTSF 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 the design of structural zone and develop strength parameters for stability modelling.
- 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.
- Refine the sizing of ditches and ponds as new information and data become available (e.g. precipitation data, streamflow data, and mine dewatering rates from groundwater numerical modelling).
- Design of DTSF structural zone.
- Perform Seepage analysis of the DTSF to determine the liner requirements and characteristics of the PAG rock zone.
- Create an Annual staged earthworks balance.
- Create an Annual staged stacking plan.
- Optimisation of the DTSF footprint with PFS LOM plan.
- Development of a FS Closure plan. and
- Complete a Dam Breach Analysis (DBA) for DTSF and mine site ponds.
- Perform a risk assessment and develop a scope of work to address data gaps and intolerable risks.

26.8.3 SURFACE WATER MANAGEMENT

The following recommendations have been included to fill the gaps identified in the limitations of the current PFS assessment and should be addressed as part of the next phases of studies:

- Gather additional climate data to assess the impacts of elevation on average precipitation and on storm events.
- Gather additional long duration flow discharge timeseries from nearby surface water monitoring locations to calibrate rainfall-runoff estimates.
- Progress the studies leading to the determination of site-specific discharge criteria both in terms of water quality and water quantity, if any, that would be used to review the design criteria for the surface water infrastructure and the subsequent infrastructure sizing.
- Review of the water infrastructure sizing based on the updated climate analysis and design criteria.
- An updated groundwater model is currently being developed as part of the PFS, but the results to inform the current study are not available at the time of this report. The sizing of the Clarifier, the WTP, Mine Dewatering Pond, Non-Contact Water Pond should be reviewed once the updated groundwater inflows are available.

26.8.4 SITE-WIDE WATER BALANCE

Based on the outcomes of the SWBM, WSP proposes the following recommendations:

- **Catchment progression:** The current model represents a static facility layout. Future model updates should incorporate this temporal evolution to further increase the accuracy of the simulation results.
- **Climate modelling:** Climatic variability is accounted for within the SWBM through the inclusion of a stochastic climate generator. This allows the model to test the Site resilience under a range of climate scenarios. Future model iterations could further test the Site resilience by artificially inserting drought periods or specific storms of a given return period.
- **Incorporate system failures:** The sizing of the Mine Dewatering Pond considered in the model accounts for five (5) days of maintenance of the WTP. The probability of failures of the individual components of the water management system could be incorporated into the modelling to assess their impacts on the water management.
- **Incorporate future updates into model:** Incorporate new data and management designs into the model as the project advances. In particular, it is recognised that dewatering forms a key input to the model. At the time of this Report, an updated groundwater model is being developed for the site. however, the results are not yet finalised to be incorporated in the current assessment. It is vital that these updates are incorporated into the SWBM at the next phase of the Project.
- **Optimise pond and pump sizing:** The initial sizing of ponds and pumps was determined based on the information available at the time of model development. Considering the integration of new data into the model, it is imperative to optimise the current sizing parameters.
- **Review Non-Contact Water Pond sizing:** The Non-Contact Water Pond requires specific attention and optimisation. The pond was initially sized based on the PEA studies. However, subsequent modelling indicates that the Site can function adequately without a significant non-contact water supply under primary operating conditions. Nevertheless, additional model scenarios demonstrate the value of the pond for emergency water storage in case of sudden dewatering inflow disruptions. Further analysis is recommended to refine the optimal pond size.
- **Closure Phase:** The current SWBM does not incorporate a closure phase. Closure phase results have only been provided at a conceptual level. It is recommended that the demolition and rehabilitation of the mine facilities be encompassed within a future iteration of the SWBM.

These recommendations aim to enhance the accuracy of water resource management at the Čoka Rakita Site and ensure the efficient and sustainable use of water resources over the LOM.

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, although nationally the Spatial Planning procedure is pending. 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 have been presented in Table 26.3. It is essential that these are completed so that impacts are properly identified. and 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.

Table 26.3 – Interactions Between Project Activities and Social/Environment

Item	Aspect									Receptor									
	Emissions to Air	Noise and Vibration	GHG Emissions	Water Usage	Discharges	Land Take	Light	Hazardous Materials		Air Quality	Soils and Geology	Water	Flora and Fauna	Land Use	Health and Safety	Fishing and Hunting	Livelihoods	Cultural Heritage	Landscape
Construction																			
Change in land use						✓				✓	✓	✓	✓	✓	✓	✓	✓	✓	✓
Decline construction	✓	✓	✓	✓	✓		✓	✓		✓	✓	✓	✓		✓	✓		✓	✓
Construction of infrastructure	✓	✓	✓	✓	✓		✓	✓		✓	✓	✓	✓		✓	✓		✓	✓
Road and Powerline Construction	✓	✓	✓		✓	✓		✓		✓	✓	✓	✓	✓	✓			✓	✓
Operation																			
Underground		✓		✓	✓			✓			✓	✓			✓				
Water supply				✓								✓	✓		✓	✓	✓		
Dewatering				✓	✓							✓	✓						
Discharges					✓						✓	✓	✓		✓	✓	✓		
Chemical / fuel storage								✓		✓	✓	✓	✓		✓				
Waste management	✓		✓		✓					✓	✓	✓	✓		✓		✓		✓
Employment															✓		✓		
Major accidents	✓	✓	✓		✓			✓		✓	✓	✓	✓		✓	✓	✓		
Closure																			
Removal of structures	✓	✓								✓	✓	✓			✓				
Rehabilitation	✓		✓	✓	✓						✓	✓	✓	✓	✓	✓	✓		

Source: ERM, 2024

26.10 Capital Cost Estimates

- 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 measure would improve the accuracy of cost estimates related to mobilisation, site setup, project logistics, equipment procurement, personnel recruitment, and training.
- For equipment procurement, establish details beyond the basic machine acquisition such as, but not limited to, OEM to provide remote radio control for operation of LHDs and for operation of Production Drills, rims for tire exchanges, drill steel and drill string, assembly supervision, operational and maintenance training specific to the equipment.
- Incorporate known growth factors related to ore outlines moving with definition diamond drilling (elevating ore reserves from possible to proven), excavation beyond “neat” outlines as part of drilling practices to avoid bootlegs, added excavations / slashes for operator positioning as part of radio remote operations, floor slashes for CRF mixing, etc.
- Establish relationships with local contractors and service providers. and
- Prepare organisational charts to support construction management and operating costs.

26.11 Other Relevant Data and Information

It is recommended that all efforts be placed in the obtaining of the permits required to develop the Project as permitting is one of the critical areas that could affect the overall delivery of the Project on the proposed schedule.

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28 ABBREVIATIONS

Abbreviation	Unit or Term
#	Number
%	Percent
% w/w	Percent Weight per Total Weight
€	Euro
€/km^2	Euro per Square Kilometre
<	Inferior
>	Superior
±	Plus-Minus
°	Degree
°C	Degree Celsius
µm	Micron
3D	Three-Dimensional
AAS	Atomic Absorption Spectrometer
ABTSB	Apuseni-Banat-Timok-Srednogorie Belt
AEP	Annual Exceedance Probability
AG	Autogenous Grinding
Ai	Abrasion Index
AISC	All-in Sustaining Cost
Al	Aluminum
ALS_VA	ALS Vancouver
AP	Acid Potential
ARD	Acid Rock Drainage
As	Arsenic
Au	Gold
BaseMet Labs	Base Metallurgical Laboratories
BAT	Best Available Technologies
BBWi	Bond Ball Mill Work Index
BD	Bulk Density
BFS	Blast Furnace Slag
BHP	Big Hornblende Porphyry

Abbreviation	Unit or Term
CAD	Canadian Dollars
Capex	Capital Expenditure, Capital Cost Estimate
CCA	Consequence Classification Assessment
CCLAS	Comlabs Computerised Laboratory Automation System
CCTV	Closed-Circuit Television
CDA	Canadian Dam Association
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carriage and Insurance Paid To
CITES	Convention on International Trade in Endangered Species
CLR	Centre-Log Ratio
CM	Construction Management
Co	Cobalt
CO	Carbon Monoxide
COG	Cut-off Grade
CPF	Cemented Paste Backfill
CRF	Cemented Rock Fill
CRM	Certified Reference Material
CS	Carbon Steel
Cu	Copper
$CuSO_4$	Copper (II) Sulphate
CV	Coefficient of Variation
cvs	Comma-Separated Value
DA	Dynamic Anisotropy
DBA	Dam Break Analysis
DCF	Discounted Cash Flow
DEM	Digital Elevation Model
DGPS	Differential Global Positioning System
DHSS	Drill Hole Spacing Study
dmtpa	Dry Metric Ton per Annum
DOL	Direct On-Line
DPM	Dundee Precious Metals Inc.
DRA	DRA Americas Inc.
DSO	Deswik's Stope Optimizer

Abbreviation	Unit or Term
DTM	Digital Terrain Model
DTSF	Dry Tailings Storage Facility
ea	Each
EBRD	European Bank for Reconstruction and Development
EEL	Electrical Equipment List
EGM	Engineering Geology Model
EGRG	Extended Gravity Recoverable Gold
E-House	Electrical-House
EIA	Environmental Impact Assessment
EMP	Early Mineralised Porphyry
EMPO	Diorite
EPCM	Engineering, Procurement and Construction Management
ERM	Environmental Resources Management Ltd.
EU	European Union
EW	Electrowinning
EWS	Engineering Workstation
FA	Fire Assay
FAAS	Flame Atomic Absorption Spectrometry
Fe	Iron
FoS	Factor of Safety
FPT	Full Plant Tailings
FRS	Fibre Reinforced Shotcrete
FS	Feasibility Study
FSE	Fundamental Sampling Error
g	Gram
G&A	General and Administrative
g/t	Gram per Tonne
GAAP	Generally Accepted Accounting Principles
GCL	Geosynthetic Clay Liner
GEN_PE	Genalysis Perth
GGIBFS	Ground Granulated Blast Furnace Slag

Abbreviation	Unit or Term
GIMS	Geological Information Management Systems
GISTM	Global Industry Tailings Standard of Tailings Management
GPS	Global Positioning System
GRG	Gravity Recoverable Gold
H&S	Health and Safety
H_2O_2	Hydrogen Peroxide
HDPE	High-Density Polyethylene
Hg	Mercury
HQ	Diamond Drill Size
HQ3	HQ Triple Tube
HR	Hydraulic Radius
h	Hour
HVAC	Heating Ventilation, and Air Conditioning
ICP	Inductively Coupled Plasma
ICP-AES	Inductively Coupled Plasma - Atomic Emission Spectrometry
ICP-MS	Inductively Coupled Plasma - Mass Spectrometry
IDF	Inflow Design Flood
IDW2	Inverse Distance Weighting Squared
IFRS	International Financial Reporting Standards
IOCG	Iron Oxide Copper Gold
IP	Induced Polarisation
IRR	Internal Rate of Return
IUCN	International Union for Conservation of Nature
JLS	Jurassic Limestone
K	Potassium
kg $CaCO_3$/t	Kilogram of Calcium Carbonate per Tonne
kg H_2SO_4/t	Kilogram of Sulphuric Acid per Tonne
kg/m	Kilogram per Metre
kg/t	Kilogram per Tonne
KLS	Lower Cretaceous

Abbreviation	Unit or Term
km	Kilometre
km^2	Square Kilometre
KNA	Kriging Neighbourhood Analysis
kPa	Kilopascal
kt	Kilotonne
kV	Kilovolt
kVA	Kilovolt-ampere
kW	Kilowatt
kW/t	Kilowatt per Tonne
L	Litre
L/s	Litre per Second
LCT	Locked Cycle Test
LHD	Load-Haul-Dump
LME	London Metal Exchange
LOM	Life-of-Mine
LOMP	Life of Mine Plan
m	Metre
m/h	Metre per Hour
m/s	Metre per Second
m^2	Square Metre
m^3	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
MEL	Mechanical Equipment List
MEP	Ministry of Environmental Protection
Mg	Magnesium
mg/L	Milligram per Litre
MI	Ministry of Interior
MIBC	Methyl Isobutyl Carbinol

Abbreviation	Unit or Term
min	Minute
ML/ARD	Metal Leaching and Acid Rock Drainage
mm	Millimetre
Mm^3	Million Cubic Metres
MME	Ministry of Mining and Energy
Mn	Manganese
MoM&E	Serbian Ministry of Mining and Energy
Moz	Million Ounces
MPa	Megapascal
MRE	Mineral Resource Estimate
ms	Millisecond
Mt	Million Tonne
MTO	Material Take-Off
MVA	Megavolt Ampere
MW	Megawatt
Na	Sodium
NaCN	Sodium Cyanide
NAG	Non-Acid Generating
Ni	Nickel
NI 43-101	National Instrument 43-101 Standards of Disclosure for Mineral Projects
NNP	Net Neutralising Potential
NO_2	Nitrogen Dioxide
NP	Neutralising Potential
NPC	Net Present Cost
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
OEM	Original Equipment Manufacturer
OF	Overflow
OHSMP	Occupational Health and Safety Management Plan
OIS	Operator Interface Station

Abbreviation	Unit or Term
Opex	Operating Expenditure, Operating Cost Estimate
OR	Operations Readiness
ORP	Operational Readiness Plan
P&ID	Piping and Instrumentation Diagram
Pa	Pascal
PAG	Potentially Acid Generating
PAX	Potassium Amyl Xanthate
Pb	Lead
PCS	Process Control System
PDC	Process Design Criteria
PEA	Preliminary Economic Assessment
PEP	Project Execution Plan
PFD	Process Flow Diagram
PFS	Pre-Feasibility Study
pH	Power of Hydrogen (measurement of acidity)
PoF	Probability of Failure
PLL	Potential Loss of Life
ppm	Parts per Million
PVC	Polyvinyl Chloride
PXP	Pyroxene Porphyry
Q1, Q2, Q3, Q4	Quarter 1, Quarter 2, Quarter 3, Quarter 4
QA/QC	Quality Assurance / Quality Control
SQL	Simple Query Language
RAR	Return Air Raise
RC	Reverse Circulation
RF	Rock Fill
ROM	Run-of-Mine
RPEEE	Reasonable Prospect for Eventual Economic Extraction
RQD	Rock Quality Designation
RSD	Serbian Dinar

Abbreviation	Unit or Term
S	Sulphur
SAB	SAG and Ball (Milling)
SAG	Semi-Autogenous Grinding
Sb	Antimony
SCADA	Supervisory Control and Data Acquisition
SEA	Strategic Environmental Assessment
SEM-EDS	Scanning Electron Microscopy/Energy-Dispersive X-Ray Spectrometry
SFA	Screen Fire Assay
SFD	Sequential Felsic Debris Flow (Epiclastic Unit)
SFR	Staged Flotation Reactor
SGS_BO	SGS Bor
SGS_BUR	SGS Burgas
SGS_CH	SGS Chelopech
SI	Système international d'unités
SM	Screened Metallics
SMR	Marls
SO_2	Sulphur Dioxide
SP	Set-Point
SPSPA	Spatial Plan for Special Purpose Area
SWBM	Salt and Water Balance Model
SWOT	Strength, Weakness, Opportunity and Threat
t	Tonne
Th	Thorium
TMC	Timok Magmatic Complex
TMI	Total Magnetic Intensity
ToS	Trade-off Study
tpa	Tonne per Annum
tpd	Tonne per Day
tph	Tonne per Hour
tph/m²	Tonne per Hour per Square Metre
UCS	Unconfined Compressive Strength
UDS	Underground Distribution System

Abbreviation	Unit or Term
UG	Underground
US$ or $	United States of America Dollars
US$ M	Million of 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
VAT	Value Added Tax
VFD / VSD	Variable Frequency Drive / Variable Speed Drive
VHG	Very High Grade
VoiP	Voice over Internet Protocol
VSD	Variable Speed Drive
VTEM	Versatile Time Domain Electromagnetic
VWP	Vibrating Wire Piezometers
WGS	World Geodetic System
WSB	Work Breakdown Structure
WSP	WSP Global Inc.
wt%	Percentage per Weight
WTP	Water Treatment Plant
WWM	Welded Wire Mesh
XRD	X-Ray Diffraction

29 QP CERTIFICATES

20 Queen Street West / 29th floor / Toronto / Ontario / Canada / M5H 3R3
T +1 416 800-8797 / **E** info@draglobal.com / https://www.draglobal.com/

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia*" dated January 30, 2025 with an effective date of December 18, 2024 (the "Technical Report"), prepared for Dundee Precious Metals Inc. (the "Company").

I, *Ian L. Major, P. Eng., MBA* of Toronto, Ontario, Canada, do hereby certify:

1. I am Senior 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 Queen's University, Canada with a Bachelor of Mechanical Engineering in 1991.
3. I am registered member of the Professional Engineers Ontario (Reg. 90372921).
4. I have worked as an Engineer in the Oil, Mining, and Metals industry continuously since my graduation from university. I have worked on similar projects to the Čoka Rakita Project; my experience for the purpose of the Technical Report includes:
   - Over 30 years of experience in project management, study management, operations, maintenance, supply, distribution, optimization, integration, and engineering design.
   - Management of numerous studies and projects of varying complexity, involving multi-disciplinary engineering teams for projects in gold, base metals, and other 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 preparing or supervising the preparation of Sections 2, 3, 18 (except for 18.6 to 18.9), 21.1 (except for 21.1.7), and 24; and 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 May 13 to 17, 2024 and December 4 to 5, 2024.

9. I have had no prior involvement with the property that is the subject of the Technical Report.

10. I have read the 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 30 day of January 2025, Toronto, Ontario.

"Original Signed and Sealed on file"

Ian L. Major, P. Eng.
Senior Project Manager
DRA Americas Inc.

Springfield House | Suite 2 First Floor | Horsham | West Sussex RH12 2RG

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia*" dated January 30, 2025 with an effective date of December 18, 2024 (the "Technical Report"), prepared for Dundee Precious Metals Inc. (the "Company").

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 (formerly 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 20 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 have participated in the preparation of this Technical Report and am responsible for Sections 4 to 12, 14, and 23, and 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 – "NI 43-101 Technical Report – Preliminary Economic Assessment – Čoka Rakita Gold Project, Eastern Serbia*" prepared by DRA Americas Inc., with an effective date of November 16, 2023 (issued Report Date June 11, 2024, prepared for DPM);
   - *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 30 day of January, 2025, Horsham, United Kingdom.

"Original Signed on file"

Maria O'Connor, B.Sc. (Hons), MAIG
Technical Director Mineral Resources
Environmental Resources Management (ERM)

20 Queen Street West / 29th floor / Toronto / Ontario / Canada / M5H 3R3
T +1 416 800-8797 / **E** info@draglobal.com / https://www.draglobal.com/

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia*" dated January 30, 2025 with an effective date of December 18, 2024 (the "Technical Report"), prepared for Dundee Precious Metals Inc. (the "Company").

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 the 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, 17, and 21.2 (except for 21.2.3) and 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 May 13 to 17, 2024.

9. I have had prior involvement with the property that is the subject of the Technical Report:
    - *Served as QP for – "NI 43-101 Technical Report – Preliminary Economic Assessment – Čoka Rakita Project, Eastern Serbia*" prepared by DRA Americas Inc., with an effective date of November 16, 2023 (issued Report Date June 11, 2024, prepared for DPM);
    - *Served as QP for - "Technical Report on the Maiden Mineral Resource Estimate for the Čoka Rakita 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 30 day of January 2025, Toronto, Ontario.

"Original Signed and Sealed on file"
Daniel (Niel) Morrison, P. Eng.
Principal Process Engineer
DRA Americas Inc.

CERTIFICATE OF QUALIFIED PERSON KHALID MOUNHIR

I, Khalid Mounhir, state that:

(a) I am a Principal Mining Engineer at:
WSP Canada Inc.
1600, René-Lévesque West Blvd.
Montreal, QC, Canada, H3H 1P9

(b) This certificate applies to the technical report titled Technical Report Pre-Feasibility Study Čoka Rakita Project, Eastern Serbia; dated January 30, 2025 with an effective date of December 18, 2024 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Ecole Polytechnique of Montreal with a Bachelor of Science degree in Mining Engineering granted in 2006. I am a member in good standing of the Professional Engineers of Quebec "Ordre des Ingénieurs du Québec" License Number 5047938. My relevant experience after graduation, for the purpose of the Technical Report, includes over 18 years of experience of working as a mining engineer on projects involving mineral reserves estimation, mining design and planning, including 13 years in gold mining operations located in Canada and 5 years of consulting experience involving multiple mineral projects nationally and internationally.

(d) I did not complete a personal inspection of the property described in the Technical Report.

(e) I am responsible for Item(s) 15, 16.1, 16.4, 16.5, 16.6, 16.7, 16.8, 16.9, 16.10,16.11, 16.14 and portions of Items 1, 25 to 27 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had any prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Montreal, QC, Canada this January 30, 2025.

Original Signed and Sealed on file

Khalid Mounhir; P.Eng.

CERTIFICATE OF QUALIFIED PERSON PAUL PALMER

I, Paul Palmer, state that:

(a) I am a Principal Geological Engineer at:
WSP Canada Inc.
36 Pippy Place, Suite 100
St. John's, Newfoundland, A1B 3X4

(b) This certificate applies to the technical report titled Technical Report Pre-Feasibility Study Čoka Rakita Project, Eastern Serbia dated January 30, 2025 with an effective date of December 18, 2024 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of University of Toronto with a B.ASc. in Geological Engineering from 1994, I am a member in good standing of the Professional Engineers of Ontario (#100050189) and Professional Engineers and Geoscientists Newfoundland and Labrador (#11387). My relevant experience after graduation, for the purpose of the Technical Report, includes over 30 years of experience in geological engineering in the areas of mine geology, mineral resource evaluation and underground geotechnical engineering of mineral projects nationally and internationally in a variety of commodities including 2.5 years of direct working experience in gold mining operations located in northern Manitoba, 2.5 years of experience in base metals operation in Northern Manitoba, and 24 years of consulting experience with a strong focus on gold and base metals related projects.

(d) I did complete a personal inspection of the property (from June 17 to 20, 2024) described in the Technical Report.

(e) I am responsible for Item 16.2 and portions of Items 1, 25 to 27 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had any prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at St. John's, Newfoundland this January 30, 2025.

"Original Signed and Sealed on file by Paul Palmer"

Paul Palmer; P.Eng.

CERTIFICATE OF QUALIFIED PERSON MICHAL DOBR

I, Michal Dobr, state that:

(a) I am a Principal Hydrogeologist at:
WSP Canada Inc.
840 Howe Street, 1000 Robson Court
Vancouver, British Columbia V6Z 2L2

(b) This certificate applies to the technical report titled Technical Report Pre-Feasibility Study Čoka Rakita Project, Eastern Serbia dated January 30, 2025, with an effective date of December 18, 2024 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of Charles University in Prague with a B.Sc. in Engineering Geology and Hydrogeology from 1982, I am a member in good standing of the Engineers and Geoscientists BC (# 20794). My relevant experience after graduation, for the purpose of the Technical Report, includes over 40 years of experience in detailed hydrogeological and geological investigations related to both open pit and underground mining development projects in North and South America, Europe, Australia, and Asia.

(d) I did complete a personal inspection of the property (from June 10 to 13, 2024) described in the Technical Report.

(e) I am responsible for the preparation of Item 16.3, portions of Items 1, and 25 to 27, and supervision of Item 18.9 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had any prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Vancouver, British Columbia this January 30, 2025.

"Original Signed and Sealed on file"

Michal Dobr; P.Geo.

CERTIFICATE OF QUALIFIED PERSON ISAAC AHMED

I, Isaac Ahmed, state that:

(a) I am a Director Process and Mine Infrastructure Design:
WSP Canada Inc.
6925 Century Avenue,
Mississauga, Ontario L5N 7K2

(b) This certificate applies to the technical report titled Technical Report Pre-Feasibility Study Čoka Rakita Project, Eastern Serbia dated January 30, 2025 with an effective date of December 18, 2024 (the "Technical Report").

(c) I am a "qualified person" for the purposes of National Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of University of Toronto with a Bachelor and a Master of Applied Science Degrees in Chemical Engineering and Applied Chemistry. I am a member in good standing of the Professional Engineers of Ontario (#100056490). My relevant experience after graduation and over 16 years for the purpose of the Technical Report specializing in the design of mine infrastructure which includes tailings dewatering, paste backfill and mine dewatering systems for projects in different parts of the globe. Specifically, I lead the Process and Mine Infrastructure Design group in Canada, a national team focused on the processing of tailings (transport, dewater, backfill) and the design of mine underground infrastructure.

(d) I did complete a personal inspection of the property (June 11 to 13, 2024) described in the Technical Report.

(e) I am responsible for preparing or supervising Items 16.12, 16.13, 16.15, 16.16, 18.7, 18.8, and portions of Items 1, 25, 26, 27 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had any prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Richmond Hill, Ontario this January 30, 2025.

"Original Signed and Sealed on file"

Isaac Ahmed; P.Eng.

CERTIFICATE OF QUALIFIED PERSON BRUNO MANDL

I, Bruno O. Mandl, state that:

(a) I am a Senior Principal Mining Engineer at:
WSP Canada Inc.
33 Mackenzie Street, Suite 100
Sudbury, Ontario
P3C 4Y1 Canada

(b) This certificate applies to the technical report titled Technical Report Pre-Feasibility Study Čoka Rakita Project, Eastern Serbia dated January 30, 2025 with an effective date of December 18, 2024 (the “Technical Report”).

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of Queen’s University at Kingston, Ontario Canada with a Bachelor of Applied Science Degree in Mining Engineering from 1979. I am a member in good standing of the Professional Engineers of Ontario (License Number 28874501). My relevant experience after graduation, for the purpose of the Technical Report, includes over 45 years of experience at new and existing mines globally, in the design, development, and operation of mine backfill systems, dewatering systems, and supporting infrastructure for underground mines, as well as the surface disposal of mill tailings. In my current role as the Senior Principal Mining Engineer of the Process and Mine Infrastructure group I am responsible for the technical design of backfill preparation plants and underground mine backfill systems.

(d) I did not complete a personal inspection of the property described in the Technical Report.

(e) I am responsible for Item 18.6 and portions of Items 1 and 25 to 27 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had any prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Sudbury, Ontario Canada this January 30, 2025.

“Original Signed and Sealed on file”

Bruno O. Mandl; P.Eng.

CERTIFICATE OF QUALIFIED PERSON (William Richard McBride)

I, William Richard McBride, state that:

(a) I am a Senior Principal Mining Engineer at:
WSP Canada Inc.
33 Mackenzie Street
Sudbury, ON, Canada, P3C 4Y1

(b) This certificate applies to the technical report titled Technical Report Pre-Feasibility Study Čoka Rakita Project, Eastern Serbia dated January 30, 2025, with an effective date of December 18, 2024 (the “Technical Report”).

(c) I am a “qualified person” for the purposes of National Instrument 43-101 (“NI 43-101”). My qualifications as a qualified person are as follows. I am a graduate of Queen’s University (Kingston) with a Bachelor of Science degree in Mining Engineering granted in 1973. I am a Registered Member of the Professional Engineers of Ontario (PEO), License Number29888013. My relevant experience after graduation for the purpose of the Technical Report includes over 50 years of working as a mining engineer and consultant working on projects involving multiple commodities such as copper, gold, and nickel and projects involving public disclosure reporting.

(d) I did not complete a personal inspection of the property described in the Technical Report.

(e) I am responsible for Items 21.1.7 and 21.2.3 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) I have not had any prior involvement with the property that is the subject of the Technical Report.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Sudbury, ON, Canada this 30th day of January 2025.

Original Signed and Sealed on file

<u>William Richard McBride</u>

William Richard McBride, P. Eng.

PEO Number: 29888013

555 René-Lévesque Blvd West / 6th floor / Montréal / Quebec / Canada / H2Z 1B1
T +1 514 288-5211 / **E** info@draglobal.com / https://www.draglobal.com/

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia*" dated January 30, 2025 with an effective date of December 18, 2024 (the "Technical Report"), prepared for Dundee Precious Metals Inc. (the "Company").

I, *Daniel M. Gagnon, P. Eng.,* do hereby certify:

1. I am Senior Vice President East Canada and Mining, with DRA Americas Inc., located 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's degree in Mining Engineering.
3. I am registered as a Professional Engineer in the Province of Quebec (Reg. #118521).
4. I have worked as a Mining Engineer for a total of 28 years continuously since my graduation.
5. I have worked on similar projects to the Čoka Rakita Project; my experience for the purpose of the Technical Report includes:
    - 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, the USA, and Morocco.
    - Participation and author of several NI 43-101 Technical Reports.
6. 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.
7. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.

8. I am responsible for preparing or supervising the preparation of Sections 19 and 22, and portions of Sections 1, 25 to 28 of the Technical Report.

9. I did not visit the Property that is the subject of the Technical Report

10. I have had no prior involvement with the property that is the subject of the Technical Report.

11. 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.

12. 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 30 day of January 2025, Montreal, Quebec

"Original Signed and Sealed on file"

Daniel M. Gagnon, P. Eng.
Senior Vice President East Canada and Mining
DRA Americas Inc.

ERM
2nd Floor | Exchequer Court | 33 St Mary Axe | London EC3A 8AA

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia*" dated January 30, 2025 with an effective date of December 18, 2024 (the "Technical Report"), prepared for Dundee Precious Metals Inc. (the "Company").

I, *Kevin Leahy, B.Sc. (Hons), PhD, C.Geol., SiLC,* of London, United Kingdom, do hereby certify:

1. I am a Geologist and Associate Partner with Environmental 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 have participated in the preparation of this Technical Report and am responsible for Section 20, and portions of Sections 1, 25, 26 and 27 of the Technical Report.

8. I visited the Property that is the subject of the Technical Report on October 9, 2023.

9. I have had prior involvement with the property that is the subject of the Technical Report.
   - *Served as QP for – "NI 43-101 Technical Report – Preliminary Economic Assessment – Čoka Rakita Gold Project, Eastern Serbia*" prepared by DRA Americas Inc., with an effective date of November 16, 2023 (issued Report Date June 11, 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 30th day of January 2025, London, UK.

"Original Signed on file"

Kevin Leahy, B.Sc. (Hons), PhD, C.Geol., SiLC
Associate Partner
Environmental Resource Management Ltd