Technical Report

Feasibility Study

Čoka Rakita Project

Eastern Serbia

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1 SUMMARY

1.1 Property Description, Ownership, Location and Access

The Čoka Rakita Project (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 approximately 48,000. The Project comprises one (1) exploration license – Čoka Rakita license (the License) – which was granted on 12 October 2022 to Crni Vrh Resources d.o.o., now DPM Crni Vrh d.o.o. (Crni Vrh), an indirect, wholly owned subsidiary of DPM Metals Inc. (DPM).

The license area is 14 km² and was 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 also 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 entities 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 299 drillholes for 141,505 m have been drilled since 2008, with the majority drilled since 2021. The drilling has been predominantly diamond (205 for 111,749 m) and diamond tail (46 for 21,797 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 Report.

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, Analysis and Security

Samples are transported to the DPM core shed in Bor for geological logging and preparation by DPM staff. Sampling procedures are appropriate, with adequate security and supervision to minimize contamination or sample mixing. Prepared pulps from SGS Bor are stored in a locked onsite warehouse. The core shed, sample preparation laboratory, and pulp library are located within a gated, key card–controlled compound equipped with alarm systems and CCTV surveillance.

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

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

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

1.7 Data Verification

The QP, Malcolm Titley, visited the site on May 13 to May 15, 2025, for the purposes of reviewing project geology, logging and sample preparation facilities, and sample data management. 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 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.

Table 1.1 summarises the results of the program indicating that a flowsheet comprising flotation followed by leaching of the float tails can yield gold recoveries exceeding 90%.

The 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. The testing culminated in a set of locked cycle tests (LCTs) which simulated the selected flowsheet and employed optimised test conditions.

Table 1.1 – Wardell Armstrong Test Program Summary

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

Source: Wardell Armstrong, 2021

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

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

Sample ID	Test Procedure	Head Au Grade (g/t)	Au Recovery (%)	Tails Au Grade (g/t)
MetCRA23-01	Gravity -> Flotation 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
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 (3) 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.

Sedimentation testing indicated that both the flotation concentrate, as well as the tailings samples, are amenable to dewatering by a conventional thickener. Dynamic tests showed that a 1 tph/m² loading rate can be used for thickener sizing and can be expected to produce a dense underflow with the solids content exceeding 60%w/w solids. The flocculant addition used during these tests were 50 g/t and 20 g/t for the tailings sample and concentrate samples respectively.

Preliminary filtration testing showed that the tailings sample can be dewatered to 23% moisture, by pressing only. 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 Preliminary Economic Assessment (PEA) in respect of the Project. 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 body of mineralised material. A total of 18 samples were selected but three (3) of these, representing the Monzonite and Marls lithologies were not considered when evaluating results because they represented less than 0.13% of the body of mineralised material.

Table 1.3 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 1.3 – 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%). For the PFS Phase, the gravity recovery model was adjusted to align with these variability samples results.

A set of tests which mimics the gravity-flotation flowsheet was performed on the variability samples. The test procedure comprised of grinding followed by two (2) stages of gravity concentration and two (2) 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. The gravity recovery is marginally lower than what was used in the PEA as is the overall recovery. For the PFS phase, the design criteria was adjusted to align with these variability test results.

A hydrofloat test was performed to determine whether most of the gold can be floated at a coarser primary grind (P_{80} = 350 µm), which would require only the concentrate from this “scalping” step to be ground to 53 µm. The test was not successful as the gold distribution to the hydrofloat tails was unacceptably high at 38.5%.

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.

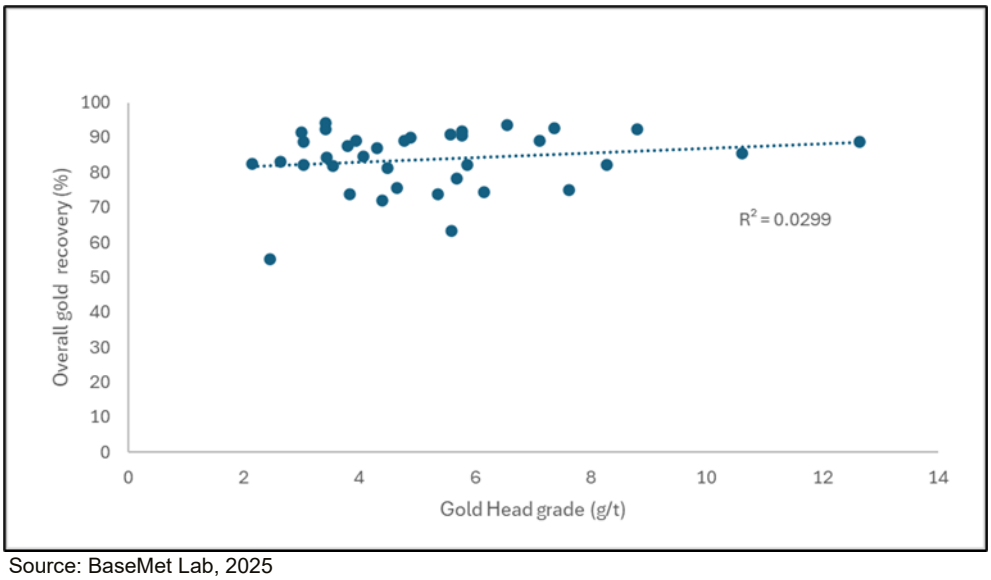
In 2025, BaseMet Labs conducted a FS testwork program. The primary objective was to assess how the metallurgical performance of the ores would vary over the mine’s life. Three (3) Life of Mine (LOM) composite samples were selected to represent the initial two (2) years of operation, the middle years (Years 3 to 5) and the final years (from Year 6 onwards). An additional master composite (MC), representing the entire LOM, was produced for comparison and for Jameson cell pilot testing, together with 36 new variability samples, which represent a wide range of the mine geological characteristics.

An Eliason and a Levin Tests were performed using the MC to determine grinding energy requirements for the Vertical Stirred Mill. The results confirmed that the design criteria adopted in the previous phase was appropriate.

Gravity recovery results for the 36 variability samples exhibited a large degree of variation and weak correlation to head grade. When appended to data from previous phases, the empirical correlation used to predict the gold recovered into the primary gravity concentrate adequately fit the entire dataset.

Rougher flotation tests on the 36 variability samples indicated a marginally positive correlation between overall recovery (gravity plus flotation) versus head grade as illustrated in Figure 1.1.

Figure 1.1 – Overall Gold Recovery vs. Head Grade



Source: BaseMet Lab, 2025

This leads to the recovery model adopting to a constant flotation stage recovery of 79%, while the gravity model remains described by the previous phase's gravity recovery. As a result of the positive correlation between gravity recovery and grade, the overall recovery versus grade shows a slight positive trend. The revised model yields an overall recovery of just above 88% at the LOM average grade of 6.4 g/t, which aligns with results from the various testwork campaigns.

Dilution tests conducted on the initial-year, middle-year and late-year composites showed no consistent trend in overall recovery, with results ranging from 83.1% to 89.4%. A decreasing trend would be anticipated as grade declines over the LOM. The absence of a clear trend is attributed to the nugget effect in these samples and is not considered a concern.

A sample of concentrate was subjected to dynamic settling tests to confirm the sedimentation behaviour for this stream. It was found that a solids loading of 1 tph/m² will result in a 67%w/w solids density underflow stream accompanied by an acceptably clear overflow. The design criteria was updated accordingly.

Analysis of a concentrate sample produced from composite samples during phase 2 testing showed that arsenic is present, but below the 2,000 ppm threshold for triggering penalties. However, some of the variability samples tested during phase 3 yielded concentrates that exceeded this threshold. A subsequent analysis of deliberate and inadvertent blending within the processing facilities indicated that this mechanism sufficiently negates this risk. All other deleterious elements analysed for in the concentrate samples were insignificant.

1.9 Mineral Resource Estimate

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 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 were characterised by moderate nuggets and short to moderate ranges. In-situ dry bulk density (Density) was estimated into mineralisation and geological domains using Ordinary Kriging.

Gold 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 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 Optimiser (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 ERM under the guidance and supervision of the QP, Mr. Malcolm Titley, MAIG, with an effective date of 17 January 2025. The MRE tabulation constrained within the DSO volumes is presented in Table 1.4. Summary assumptions used in RPEEE calculations are presented in the notes section of Table 1.4 and are discussed in more detail in Section 14.18.1.

Table 1.4 – Čoka Rakita MRE using Underground Mining Scenario

Čoka Rakita Mineral Resource Estimate Effective Date of January 17, 2025			
Category	**Tonnes** (Mt)	**Gold Grade** (g/t)	**Contained Gold** (koz)
Indicated	0.53	3.94	67
Inferred	0.09	3.60	11

Notes:

1. The cut-off grade value of 2 g/t assumes US$1,900/oz gold price, 86.75% gold recovery, 0% dilution, US$77.65/t operating cost (mining, process and G&A), US$11.20/t sustaining capital cost, as well as offsite and royalty costs.
2. 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 November 29, 2019.
3. The QP, Mr. Malcolm Titley, is not aware of any legal, political, environmental, or other risk factors that might materially affect the estimate of Mineral Resources, other than those specified in Table 1.5.
4. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
5. Mineral Resources are reported exclusive of Mineral Reserves.
6. 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 environmental, permitting, legal, socio-economic, marketing or political factors that could materially impact the MRE disclosed in this Report, other than those specified below and in Table 1.5:

- 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 1.5. The overall risk to the 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 1.5 – 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).
QAQC	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 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-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.

1.10 Mineral Reserve Estimate

The Mineral Reserves for the Čoka Rakita Project were estimated by WSP using the CIM 2019 Best Practices Guidelines and are classified using the 2014 CIM Definition Standards.

Mineral Reserves are based only on Indicated Mineral Resources identified in the block model provided by ERM and all Inferred Mineral Resources are set to waste.

Mineral Reserves are reported as diluted tonnes delivered to the process plant and have an effective date of January 17, 2025.

The Qualified Person (QP) responsible for the Mineral Reserve Estimate is Mr. Khalid Mounhir, P.Eng., a WSP employee. The Mineral Reserve Estimate is summarised in Table 1.6.

Table 1.6 – Mineral Reserve Estimate – Effective as of January 17, 2025

Classification	Tonnes (Mt)	Au (g/t)	Contained Gold (Moz)
Proven	-	-	-
Probable	7.344	6.44	1.520
Proven + Probable	**7.344**	**6.44**	**1.520**

1. As of the Reserve effective date, 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 and 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 January 17, 2025.
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$1,600/oz for gold.
7. Mineral Reserves are reported using variable cut-off grades (Refer to Subsection 15.3.1 for COG estimation details):
   - Stope full cost cut-off grade of 2.5 g/t (in-situ).
   - Stope marginal cut of grade of 2.0 g/t (in-situ).
   - Development COG of 1.0 g/t.
8. Mineral Reserves account for a:
   - Hanging wall (HW) and footwall (FW) ELOS external dilution of 1.0 m and 0.5 m, respectively applied to the stopes at matching Au grades of the BM
   - Back fill dilution of 6% applied to the stopes at zero Au grade
   - Mining recovery of 95 % applied to the stopes and 100% applied to development tonnes
9. Contained Metal (CM) is calculated as follows: Au Contained Metal, (oz) = Tonnage (Mt) * Grade (g/t) / 31.1035 (g/oz).
10. The Mineral Reserve Estimation was completed under the supervision of Mr. Khalid Mounhir, P.Eng., Principal Mining Engineer at WSP Canada Inc., who is a QP as defined under NI 43-101.
11. The QP, Mr. Khalid Mounhir, is not aware of any 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.

Compared to 2024 the 2025 Mineral Reserve for the Čoka Rakita Deposit shows an increase in both tonnes and grade as presented in Table 1.7.

Table 1.7 – Čoka Rakita Mineral Reserve Estimate - Variation

Mineral Reserve Estimate	Tonnes	Au	Contained Gold
	Mt	g/t	Moz
2024	6.633	6.38	1.359
2025	7.344	6.44	1.52
Variation	**0.711**	**0.06**	**0.161**
Variation %	**10.7%**	**0.9%**	**11.8%**

The increase is primarily due to revisions in the engineering design parameters adopted for the 2025 Mineral Reserve estimate. Key changes include the incorporation of marginal material (following the

determination of a marginal cut-off grade), the inclusion of planned dilution in the stoping design, and manual adjustments to the DSO shapes to convert them to mineable stopes.

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

- Interpretations of mineralisation geometry and continuity of mineralisation zones.
- Changes in market conditions, commodity prices and COG estimation assumptions.
- Changes in rock quality and geotechnical assumptions.
- Changes in process plant recoveries.
- 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.

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 in retreating fashion. Each series of longitudinal stopes requires an ore sill on the upper sublevel for drilling and loading explosives and an ore sill on the lower sublevel for mucking blasted ore.

The mine will be divided into two (2) mining horizons: the upper horizon will be from RL440 to RL620, and the lower horizon from RL 360 to RL440. RL420 is considered as a sill pillar. The long-holes will be drilled in rings angled downwards and loaded with bulk emulsion explosives.

A diesel fleet will haul the material to surface via the spiral ramp through the upper decline. 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

Rock mass classification values (RMR76 and Q' values) were estimated for each geotechnical domain. The geotechnical domains are classified as exhibiting poor to good characteristics 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 the underground 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. Where possible, critical infrastructure was moved to avoid those fault zones observed to have the poorest ground characteristics. Ground support recommendations were developed for the range of conditions expected from the geotechnical domains and the faults zones.

Empirical and analytical stability assessments and 3D elastic and non-elastic numerical induced stress assessments were completed based on the interim mine plan and schedule available during the initial stages of the FS. The assessments were completed to identify stability concerns regarding the stope sizing, mine sequence, pillars, and standoff distances of the access drifts, spiral ramp, and surface ventilation raise. Ground support recommendations were also developed for each type of excavation.

1.11.4 HYDROGEOLOGY

As part of the FS approved scope of work, WSP has updated the groundwater inflow estimates developed during the Project PFS. The update to the PFS numerical groundwater flow model aimed to increase confidence in its predictive capability through the incorporation of new (FS) datasets to its structure and calibration. The objectives of the modelling update were to:

- Develop an updated conceptual and numerical hydrogeological model for the area of the Underground Development.
- Incorporate the regional and local lithology from the latest geological map and updated Leapfrog model.
- Incorporate selected geological structures relevant for groundwater movement.
- Incorporate the streams draining the catchments of Čoka Rakita based on the hydrographic map developed by WSP as part of the FS.
- Update the recharge rates used in the PFS model based on the hydrological analyses performed by WSP as part of the FS.
- Calibrate the model to up-to-date piezometric data, including hydraulic heads from Vibrating Wire Piezometers (VWPs) and groundwater levels from monitoring wells.
- Incorporate the latest available mine development plan (at the time of model update) at a quarterly level in the first three years and at an annual level from Year 4 until Year 10 (LOM).
- Simulate mine dewatering and estimate groundwater inflows for three scenarios: base case, low inflow and high inflow.

1.11.5 Mine Design

All stopes will have consistent dimensions of 20 m in height and 30 m in length (measured along strike). However, due to variations in rock quality and stress conditions with depth, the stope width (measured from footwall to hanging wall) will be 20m in the Upper Horizon (above RL440) and 15m in the Lower Horizon (below RL440).

Stopes will be mined in a retreating sequence. The stope void will be created by a conventional 2.5 m x 2.5 m drop raise and the production holes will be 89 mm in diameter and will be drilled in rings with a 2.5 m toe spacing and a 2.5 m burden.

The development declines will have a 5.5 m x 6.0 m (width x height) arched cross-section, and the ore sills 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 in profile.

1.11.6 Mine Portal 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. 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 RL510. It is not expected that the lower ramp to the final levels will be developed until Year 2 of the mine life. The spiral ramp will provide access to all levels of the mine, spaced at 20 m intervals and enable the transport of ore to surface throughout the LOM, and for waste during the years prior to the mill operation.

The South return air ventilation raises connecting the sublevels will be equipped with escapeway ladders, providing an emergency escape alternative to the spiral ramp. A maintenance shop and auxiliary infrastructure are designed on RL620 while additional supporting infrastructure are located throughout the mine, such as dewatering sumps, a main pumping station, electrical substations, explosive storage, emulsion bay, fuel bay and paste backfill stations. Lastly, in terms of mine safety infrastructure, there are three (3) portable mine rescue chambers and an escapeway ladder system in each of the Southern raises in case of emergency.

1.11.7 Life of Mine Plan

The Life of Mine Plan (LOMP) extends over nine (9) years, during which the mine will produce ore at full capacity for seven (7) years. At the end of Year-1, the spiral ramp will have broken through from the lower and upper horizons, and stope production will commence on the RL440, situated approximately two-thirds down the depth of the mine. The deposit will be mined in two (2) phases.

Initially, the upper horizon of the deposit will be mined to deliver ore to the processing plant per the planned schedule. From RL440, stoping operations will progressively advance to higher levels in an ascending sequence. The mine will operate at full output during Year 2, producing slightly over 855 kt of ore. In Year 5, sublevel stoping will commence on RL360. Production in the lower horizon will progressively supplement that of the upper horizon. The mine will continue operating at full capacity from Years 2 to 8. In Year 9, the final year of the LOMP, production will decrease to 578 kt of ore.

Mine development will progress at an average rate of approximately 4,970 m/year during Years -2 to 2, decreasing to 2,290 m in Years 3 to 6. From Years 7 to 9 will see a drop to 330 m and development will cease entirely for the final year. Years -2 and -1 will focus on driving the two (2) declines, the spiral ramp, and level access headings, with additional progress in developing ventilation drifts and infrastructure excavations. In Year 1, the emphasis will shift to include crosscuts and ore sills within the deposit. The spiral ramp will resume development in Year 2 below RL440. From Years 5 to 7, the focus will be development targets, crosscuts and ore sills required for stoping operations. Development activities will continue to the end of LOM as the mining sequence in the lower levels require delays some development headings.

1.11.8 MINING VENTILATION

The main ventilation system is driven by two (2) x 575 kW main fans, installed at the top of the surface ventilation raise, in a parallel horizontal arrangement. Ventilation is by forced exhaust through a single raise to surface, with intake from two separate ramp portals. Free-hanging jet fans will be installed in the portal intake segments to enable modulation of quantity of air flow from each portal. Once the air enters the decline, the air is directed to the two internal raise systems connecting to the main exhaust raise. Automated air flow regulators will be installed at the exhaust raise connections, to be controlled, in concert with primary fans, and jet fans, by a ventilation-on-demand (VOD) automation system. Air flow sensors and truck-detection system will also be essential equipment for ventilation control.

1.11.9 MINE BACKFILL

All paste backfill required by the mine is delivered via the mine's upper decline, by a positive displacement pump in the paste plant. 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.

Redundancy of the UDS system's components exists from the Paste Plant to the end of the upper decline piping where it reaches the orebody on RL650 level is facilitated through the use of a redundant surface pipeline between the Paste Plant and the upper decline portal and down the decline to RL650. There is no redundancy in the UDS within the orebody with the exception of all

underground inter-level boreholes (duty and standby) containing an internal carbon steel pipe casing to mitigate possible poor rock quality concerns.

The secondary backfill method is a cemented rock fill (CRF) system utilising a modular plant on surface to mix the cement slurry that is transported underground by the fleet of mobile transmixers. CRF accounts for approximately 8% of the required backfill volume over the LOM with the majority being used in the mine development period before Year 4.

1.11.10 MINE DEWATERING

All water leaving the mine will be pumped through the lower decline and via boreholes situated close to the process plant at surface. From here the water is either diverted to the clarifier or the Contact Water (CW) pond for further processing.

The Main Dewatering Station is located at RL450 and is positioned to maximize gravity draining at level and secondary sumps resulting in a reduction of submersible pumps in the overall system. Level Sumps at RL400 and RL380 gravity drain to Level Sump at RL370. Water from this sump is pumped up to a transfer station at RL410. The Transfer station collects all water below RL430 and pumps it up to the Main Dewatering Station.

1.11.11 ELECTRICAL AND COMMUNICATIONS

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 step down the site transmission voltage to the distribution voltage of 10 kV and delivers it to the portals. One (1) feeder cable will be routed down from the switchgear breaker on busbar A to the upper decline portal and the second feeder cable will be routed down from the switchgear second breaker on busbar B to the lower decline via borehole.

These two (2) mine power feeders will provide a redundant connection to ensure 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.

For emergency power, a dedicated 10kV supply will be sourced from the diesel generator at surface and brought down to the lower portal via borehole to provide backup power to the Main Dewatering Pumps. This will ensure continuous operation of the dewatering system through a site-wide power failure.

1.12 Processing and 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 of 87.5%, varying from 86.3% - 89.5%. 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 indicated an optimum grind of 80% passing 53 µm. A comminution trade-off study (ToS) was conducted during the PFS 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, pebble crusher and vertical stirred mill. The primary mill will be operated in fully autogenous grinding mode i.e., with no grinding media added. It is recommended to retain the same lifters and liners as currently used at Ada Tepe, so that media can be added if required.

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. The primary gravity concentrate will be upgraded through a second gravity concentration stage, followed by smelting of the final concentrate and filtering of the secondary gravity tails to produce an intermediate grade product for sale.

A ToS was conducted during the PFS to compare flotation technologies. Based on the outcomes of the ToS, Jameson cells were adopted for the FS.

The nameplate capacity of the treatment plant will be 850,000 dmtpa. The crushing section will be manned 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 grinding and flotation sections will operate 92% of the time. The gold room, which includes secondary gravity concentration, dewatering and smelting, will operate five (5) days each week during day shift only (i.e., 60 hours per week).

Cement will be added at 5% to the paste backfill and will represent the most expensive component. Other reagents used in the process will include flocculant, PAX, MIBC, collector A3477, copper sulphate and smelting fluxes. 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 to a clarifier from where 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 little water exits the site because all tailings will be disposed of either as filtered cake on surface or as paste backfill underground.

Mill Tailings Thickener underflow slurry will be pumped to the Tailings Pressure Filtration Plant, which will dewater the 57 wt% solids slurry to Filter Cake at 87 wt% solids (13 wt% moisture) for disposal to the Dry Tailings Storage Facility (DTSF) on surface, or for use in the Paste Backfill Plant at 85 wt% solids (15 wt% moisture) to produce paste backfill.

Paste backfill augmented with a 5 wt% blend of ground iron blast furnace slag/normal Portland cement binder (90/10 ratio) will be prepared in the Paste Backfill Plant and pumped underground using a 120 bar Positive Displacement Pump at sump consistence between 178 and 254 mm, which is a solids content of between 75.3 and 73.0 wt%.

1.13 Project Infrastructure

The main project on-site infrastructure components include:

- Roads.
- Process terraces.
- Mining services area terrace.
- Administration terrace.
- Mine and process plant supporting infrastructure.
- Administration buildings.
- Access control.
- Mine waste rock management.
- Overburden and topsoil management.
- Dry tailings storage facility (DTSF).
- Water management.
- Ventilation shaft and equipment.
- Power supply and distribution.
- Automation.
- Communication.
- Services and utilities.

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 portal. The underground development waste rock is considered to be 100% potentially acid generating (PAG) material, which will need to be stockpiled and contained within a segregated area

of the DTSF. As such the waste rock from the lower decline and the upper decline will be transported to the stockpile location after the initial sections of the DTSF are developed and lined. Minimal waste rock haulage is anticipated during the LOM.

Water from the CW pond (mine dewatering and surface contact run-off water) along with the DTSF reclaim CW will be piped to the clarifier and utilised as clarified water for the process, gland seal and mine usage water. Some of the clarified water will go through additional treatment phases so that it can be discharged to the environment or used as treated water and chlorinated water.

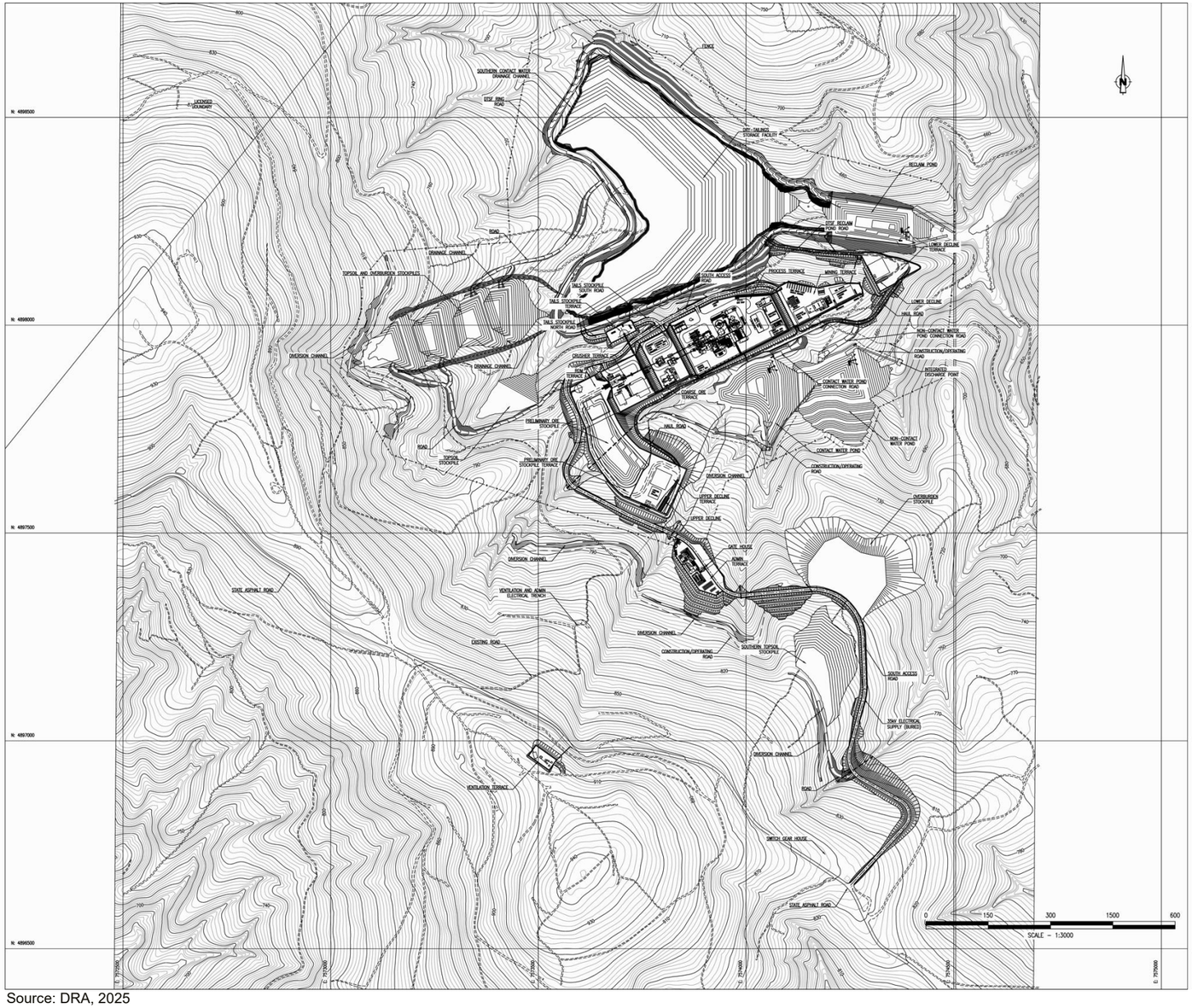
There will be three (3) ponds on-site:

- A CW pond - fed by the dewatering of the mine and water management of the ROM stockpile terrace area, process terraces mining services terrace, administration terrace and roads.
- A Non-Contact Water (NCW) pond - fed from surface locations that are considered to be non-contact sources and post water treatment (to environmental discharge standard). The intended volume of the NCW pond will provide fourteen (14) days of water storage for dry periods.
- A reclaim water pond – fed from seepage from the DTSF and surface CW sources that aren't able to be easily sent to the main CW pond. The reclaim pond water will be pumped to the clarifier, for particulate removal and then treatment through the Waste Water Treatment Plant (WWTP).

Support infrastructure is located on two (2) main terraces, namely the administration area terrace and process plant terraces (inclusive of the mine services area). The administration terrace is located on the main site access road, approximately half-way between the municipal road and the ROM pad, and will consist of the main administration building, mine offices and mine dry building, as well as the main gate house for access control to the administration building, the process areas and the mining operations.

The process plant terraces will house all process plant related infrastructure, including the tailings filtration plant, paste backfill plant, mining infrastructure, warehouse building, reagents storage, plant offices control room and laboratory, maintenance workshop, mine truck workshop and fuel depot. An overall site layout is provided in Figure 1.2, and details are further explained and illustrated in Section 18.

Figure 1.2 – General Site Layout



1.13.1 DRY TAILINGS STORAGE FACILITY

The FS for the 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 FS optimises the previous PFS design. A Dam Break Analysis (DBA) was conducted early in the FS, guided by the Canadian Dam Association (CDA) Technical Bulletin 2021 on Tailings Dam Breach Analysis (TDBA), to determine design requirements based on potential DTSF failure impacts. Consequence Classification Assessment (CCA) was performed classifying the DTSF as "Low" with minimal environmental impact and no effect on external infrastructure. The DTSF was designed to contain 2.23 Mm³ of filtered tailings and 368,000 m³ 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 FS 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. Trucks will end-tip tailings in the deposition zone, where a dozer will spread to be compacted in 300 mm layers 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 considered final slope gradients and capping options designed to limit seepage. The FS closure design of the DTSF has been updated for the engineered cover design and revised since the PFS to consider refined inputs and logic to better represent on-site conditions. The cover design was largely driven by the outcomes of the recharge and seepage assessments combined with inputs on material availability and cover thickness calculations. A vegetation assessment concluded that native grassland would form the dominant vegetation cover type on the rehabilitated DTSF, additionally, native shrub species should also be used to provide species diversity and aid with soil stability. During the active closure phase, it is envisaged that the Contact Water Pond and Non-Contact Water Pond (and their associated structures) will be decommissioned. As the DTSF is gradually remediated, the Reclaim Pond will be replaced by a Reclaim Sump to continue to collect seepage from the DTSF. In addition, the stockpiles and their associated water conveyance structures will be rehabilitated during active closure. Passive closure involves no additional treatment, with potential design modifications based on climate change scenarios.

1.13.2 SURFACE WATER MANAGEMENT

The water management strategy aims to intercept and divert non-contact surface water runoff downstream of the site, thereby minimizing the quantity of CW requiring treatment. Runoff and seepage from contact catchments is captured and directed to CW management ponds prior to treatment by the WWTP and reused or discharged to the environment. The water management ponds across the site are sized to not spill more frequently than the 1:100 annual exceedance probability (AEP). The ponds' operations have been optimised with the goal of minimizing the pump out rate required, thereby reducing flows to the Clarifier and WWTP and providing more consistent inflows for treatment. For the channel sizing, four main runoff events have been adopted for the design criteria: 1:10 AEP for roadside channels, 1:100 AEP for CW and stockpile channels, 1:200 AEP for major diversion channels and 1:1,000 AEP for spillways.

The water management of the DTSF consists of NCW diversion channels to carry non-contact runoff from upstream catchments around the facility. These discharge downstream of the site into a natural water course. Contact water runoff and seepage from the DTSF is captured in separate CW channels and directed to the lined Reclaim Pond, located downstream of the DTSF. The CW channels will be built in stages to align with the progressive construction of the DTSF. The Reclaim Pond has a total volume of 51,900 m³ and is equipped with an emergency spillway which discharges to the downstream environment. The required pump out rate form the Reclaim Pond to the Clarifier is 15 L/s.

Runoff from the Terrace Topsoil Stockpile is directed into the Reclaim Pond via the DTSF CW channel due to its continuous use, resulting in high runoff sediment loads. Runoff from the Ore Stockpile is considered to be contact and is directed to the CW Pond. Runoff from the Initial Topsoil Stockpile will be managed by local sediment control measures before being discharged to the environment. Runoff from upstream natural catchments will be diverted around the stockpiles.

Contact runoff from the upper terraces of the ROM Pad and Terrace area is captured in the Contact Water (CW) Pond. In the event the clarifier or water treatment plant shuts down, the CW Pond may receive additional pumped inflows form the Reclaim Pond, the underground mine dewatering and the water recirculated from the Processing Plant. The total pond volume is 62,000 m³ with a required pump out rate to the Clarifier of 15 L/s. The pond is equipped with an emergency spillway discharging into the NCW Pond. NCW diversion channels intercept and divert non-contact runoff from upstream catchments away from the CW Pond to reduce the volume of water requiring treatment.

The NCW Pond, located directly below the CW Pond, has been designed to satisfy the domestic water demand of the mine and act as an emergency water supply. The pond captures NCW runoff originating from the natural catchment located south-east of the ROM Pad and Terrace area and has a maximum capacity of 58,700 m³. Once the maximum capacity of the pond is reached, it is

allowed to overflow via the emergency spillway to the Integrated Discharge Facility. The site has a minimum environmental flow requirement, that under normal operating conditions would be satisfied by the regular discharge from the water treatment plant. In the case where the water treatment plant is not operating, the minimum environmental flows will be supplemented with water stored in the NCW Pond via a low-level outlet.

The water management for the closure design is based on the following concepts:

- The DTSF will be progressively rehabilitated during operations.
- The Reclaim Pond, CW Pond and NCW Pond and their associated spillways will be removed. All the stockpiles will be removed as well as the associated water management structures.
- All water management structures sized for closure will be designed to the 1:10,000 AEP storm event with climate change. This is to align with the Global Industry Standards on Tailings Management (GISTM) (ICMM, 2020).

The existing DTS NCW channels from the operation stage will be enlarged to convey the additional closure design peak flows. To minimise the need to maintain the channels in the long term it is proposed to line the channels with interlocked block stone, or mass concrete blocks if sufficient quantities and quality block stone cannot be sourced.

1.13.3 Site-Wide Water Balance

As part of the study, a site-wide stochastic Water Balance Model (WBM) and Water Quality Model (WQM) was developed using GoldSim. This FS WBM builds upon work completed in the preceding PFS. The model presented within this report forms an update to the previous WBM and WQM, utilising refined inputs and logic to better represent mine water transfers and site conditions.

The modelling results suggests that operational demands are consistently met, the study identifies a potential shortfall to satisfy the minimum environmental flow during the pre-operational period in case of a low groundwater inflow scenario. Based on the outcomes of the water balance the NCW Pond sizing should be revised in the next phase of the study. Based on the source terms derived from available geochemical data, exceedances with respect to Class 1 water quality limits are noted in the water management ponds. It is important to note that these results reflect data available during model development, and future iterations should incorporate evolving project data and system updates.

1.14 Market Studies and Contracts

Čoka Rakita will produce two (2) different concentrates, specifically a gold pyrite flotation concentrate and a high-grade gravity gold concentrate, and a gold doré. The flotation concentrate is clean from an impurity perspective and will have many options for sales to smelters globally as

well as trading companies. The market for the gravity concentrate and the doré is smaller, with precious metals refineries as the target market for both products in order to achieve a high payable gold level. DPM has extensive experience in selling both pyrite gold concentrates and high-grade concentrates from its Chelopech and Ada Tepe mines. Discussions regarding Čoka Rakita concentrates, with DPM's current buyers of Chelopech and Ada Tepe concentrates, have provided DPM with confidence that there is an available, long-term market. The doré market is even more commoditized with little variability in commercial terms. Long-term contracts for the products will be established in due course.

1.15 Environmental Studies, Permitting and Social or Community Impact

1.15.1 INTRODUCTION

The Čoka Rakita Project is located in Serbia, and as such 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, and 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 European Union 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 the Cukaru Peki copper-gold mine, nearby, is now four (4) years into operation. However, there are still 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 European Development Bank for Reconstruction and Development (EBRD) Environmental and Social Policy and the 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.

1.15.2 PERMITS

The Project has prepared a Permitting Strategy (DPM, 2025) and this document sets out the principal permits and approvals required for exploration, construction, operation and closure of the Project.

Crni Vrh, a Serbian corporate entity and wholly owned subsidiary of DPM, is the holder of all licenses and permits for further exploration on the Project.

1.15.3 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 Žagubica, on the mountain range between Bor to the southeast and Žagubica and Laznica to the west. Majdanpek 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. The Serbian government approved the development of the Special Purpose Spatial Plan on the 14th of November 2025. The Company's approach to the development of the Spatial Plan 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, local streams 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 hunting 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 June 2025 and periodically reviewed throughout the FS. These risks will be further investigated and assessed as part of the Environmental Impact Assessment (EIA) process. Social risks are further defined in the Socio-Economic Impact Assessment (ERM, 2025). The key risks are related to 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; small areas of habitats that may host protected species; adverse changes in laws and delays to the permitting process; increase in local community opposition to the project; and economic displacement associated with land acquisition.

1.15.3.1 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.

1.15.3.2 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, parts of which are located within the Crna Reka catchment. The Nature Park is currently in the process of being upgraded to National Park status. This area is a forested mountainous region which hosts a number of endangered 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 depth) and 5 km from the nearest project infrastructure on the surface.

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 and 2025, 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. 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 detailed 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.

1.15.3.3 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. A winter baseline survey was undertaken in March 2025 and included additional monitoring points to the northeast of the Project area. Both sets of campaigns were two (2) weeks long with attended and unattended measurements at identified receptor locations over and around the Project footprint.

Baseline vibration surveys are yet to take place.

Baseline air quality monitoring commenced in December 2021 at 11 locations in the adjacent area (associated with the Timok project: PM_{10} and airborne metals at six (6) locations, $PM^{2}_{.5}$ at five (5) locations). Results indicated that the area adjacent to the Project airshed is considered undegraded with regards to carbon monoxide (CO), SO_2, NO_2, Volatile Organic Compounds (VOCs), benzene and dust deposition, PM_{10} and $PM_{2.5}$. (particulate matter). Occasional exceedances of the daily limit were recorded in certain locations for PM_{10} and $PM_{2.5}$; therefore, project 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 continued at five (5) locations throughout 2024 and 2025.

The Project will generate greenhouse gas emissions that will need to be measured and managed in line with international good practice.

1.15.3.4 Soil and Geology

An assessment of the soil in the context of the baseline agricultural capability and chemical quality was completed in October 2024, the results indicated elevated levels of heavy metals across the Project area. The heavy metals are naturally occurring in soils, the disturbance and re-use or long-term storage of soil and subsoils from these areas will need to account for potential leaching of heavy metals. Consistent with the FS planned layout, where topsoils are to be disturbed, the

principle will be to conserve these in stockpiles for ultimate reuse as a rehabilitation medium where possible.

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. Geotechnical studies are on-going and further detailed investigations to address landslide and ground instability risks are planned to inform comprehensive risk assessments and mitigation measures.

1.15.3.5 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 have 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 EIA 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.

1.15.3.6 Social Setting, Land Acquisition and Livelihoods

A Social Impact Assessment (SIA) was conducted for the Project in 2024. Key socio-economic characteristics of the Project area are summarised as follows:

- Significant out migration of youth, from this region, for employment and education opportunities elsewhere in the country or abroad, leading to an aging population.
- Key land uses in the area include agriculture, livestock rearing, and forestry.

- The Project area has higher rates of youth unemployment compared to national averages, with key employers being the Mining and Quarrying sector and Wholesale Trade sector. The aging population and prevailing economic conditions have led to a locally depressed economy in Žagubica Municipality.
- Many households in the Project area rely on agriculture, livestock rearing and forestry for livelihoods and subsistence.
- Incomes are lower than national averages. Households, especially the elderly in the Project area, rely on remittances from family members living abroad to meet their basic need.
- Education at the elementary level is good in the Project area. However, there are plans to potentially close an elementary school due to the lack of students in the area. High school and tertiary education opportunities are limited. Consequently, educational attainment is lower in the Project area than the national average.
- Prominent health issues in the Project area are characteristic of an aging population and the demands for healthcare are high due to this. However, the healthcare services available in the Project area are limited and lack funding. Many people have to travel significant distances to receive specialised medical care. The costs associated with this can be detrimental to low income (vulnerable) households.
- Infrastructure and service delivery in the wider Project area of influence is mixed, with good electricity connections, water supply and good telecommunications within the main settlements of Žagubica and Laznica. The areas surrounding the settlements, including isolated homes and agricultural holdings which do not have access to grid electricity or a piped water supply. These households rely on natural water sources such as springs that may run dry during periods of drought. Some isolated areas are not covered by the telecommunications network, including the mobile network. There is a limited sewage network, and the transport network is not well connected to other parts of the country with certain roads requiring substantial upgrades.
- Anti-mining protests have, also, been taking place in Žagubica Municipality. Protests in the municipality in 2024 focused on the exploration activities of DPM and the alleged use of drinking water in the exploration drilling process. Activists from elsewhere in Serbia have travelled to Žagubica to protest against DPM.

The proposed Project is expected to generate positive impacts at local, regional and national levels through the generation of direct and indirect jobs - to different extents - during the different phases of the Project, creation of long-term benefits associated with capacity enhancement of local Serbian labour force through on-the-job and formal training, increased spending capacity, a growing market to cover supply chain needs, and tax and royalty payments.

At a much smaller scale, the Project is likely to result in economic displacement caused by land acquisition for the Project and access restrictions to ecosystem services used for subsistence and

livelihoods. At this stage, no physical displacement or resettlement is envisaged. However, economic displacement was taken in consideration in the land acquisition strategy and in the determination of the unit value of the properties to compensate landowners for it.

The presence of the Project could affect the health, safety and security of the communities in the Project area as a result of increased road accidents and road trespassing and environmental health impacts (e.g. noise, air quality and water quality). It was also found that stakeholders engaged with during the primary data collection noted road safety and environmental health as key concerns relating to the Project.

The decommissioning phase of the Project will be associated with a marked decrease in employment associated with the mine and wider supply chains. Given the limited alternative employment opportunities in the area, it can be expected that communities in the Project area will have become highly reliant on the mine as a source of income during the construction and operational phases. Likewise, Žagubica Municipality may have become dependent on contributions to the municipal budget as a result of the presence of the mine and the associated economic activity. As such, they are highly vulnerable to the mine's closure and the resultant loss of direct and indirect employment unless alternative economic activities are developed throughout the life of the mine. There is also a potential for an economic downturn in the region following mine closure.

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, and the siting of mine infrastructure has avoided those areas under dispute.

The Project has developed a Land Acquisition Strategy (DPM, 2024) 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.

1.15.4 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. The Project maintains an office in the town of Žagubica that employs local people to be an interface between the Project and the community. Community members are able to visit the office to ask questions, raise concerns, and access information about the Project.

During the SIA process, as part of the primary data collection, stakeholders ranging from local authorities, Non-governmental Organisation (NGO) representatives and local community members, were asked about their perception of the Project, including their concerns and views as to whether there may be any benefits.

It was found that stakeholders interviewed generally expressed a positive sentiment towards the Project and that the mining sector in the region is seen as a key source of employment. However, many stakeholders interviewed stated that they were not well informed about the Project. This has created growing concerns and confusion around potential impacts. Many concerns about the Project have stemmed from lack of information and confusion between various proposed mining projects in the area since 2004.

Stakeholders expressed that they have already seen benefits from the mining exploration activities for the Project and others in the area, particularly in the hospitality sector. The Project is seen as positive in that it will provide local employment, especially for youth, and may lead to a reduction in their emigration in search of education and employment opportunities elsewhere. It is believed by stakeholders that mining Projects such as Čoka Rakita can be a source of funding for communities (development/ social initiatives). Taxes and fees paid by mines are also seen as key to boosting municipal budgets and in doing so lead to improvements in service delivery and overall economic growth.

The Project's social license to operate is of critical importance. The stakeholder engagement plan, going forward, is a key tool which may need agility to ensure relationships are effectively managed. In addition, The key risks and mitigations for these different social aspects 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.

A Social Management Framework will be developed to define the social objectives and principles that guide the Project to achieve sound performance by minimising the most significant social risk impacts identified. Other management plans that will form part of the Framework include project level Code of Conduct, Human Resources Policy, Corporate Responsibility Policy, Human Rights Policy, Contractor and Supplier Management Plan, Retrenchment Plan, Recruitment and Procurement Procedure, Workers Grievance Mechanism, Occupational Health and Safety (OHS) Management Plan, Emergency Preparedness and Response Plan (EPRP), Traffic Management Plan, and Security Management Plan.

1.15.5 Mineral and Non-Mineral Waste

1.15.5.1 Mineral Waste

DPM has undertaken a comprehensive geochemical testing program to assess the potential for metal leaching (ML) and acid rock drainage (ARD) at Čoka Rakita. Static testing of 101 representative samples of waste rock and tailings was done to evaluate acid generation potential and leachability. Results indicate that certain lithologies, particularly mineralized skarn and volcanogenic units, are potentially acid generating, while tailings are classified as non-PAG due to high carbonate content.

A kinetic testing program is currently underway, involving long-term humidity cell tests on composite samples of the dominant waste rock types as well as tailings and overburden. These tests are designed to evaluate long-term leaching behaviour under atmospheric conditions. The program is aligned with Serbian environmental regulations and international best practices and will inform the development of a robust waste management plan

1.15.5.2 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.

1.15.6 Closure

The Project will advance closure planning and costing exercises periodically throughout the planning stages and operational life, with the next revision at the EIA stage.

1.16 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 1.8.

Table 1.8 – 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; with estimate support from DRA on bulk earthworks, building foundation concrete, outside wall structural steel, architectural, and outside the buildings electrical infrastructure requirements. Prepared initial direct Capex for Filter and Paste Plant Equipment, with estimate support from DRA on bulk earthworks, building foundation concrete, outside wall structural steel, architectural, and outside the buildings electrical infrastructure requirements. Prepared initial direct Capex for Shotcrete/CRF Slurry Batch Plant Equipment, with estimate support from DRA on modularized plant slabs and support foundation concrete. Prepared construction quantities for DTSF, Water Management and topsoil / over-burden / waste rock stockpiles (using same rates as DRA), to establish direct capital costs. WSP used DRA provided bids to establish costs for liners and geotechnical components required for construction of DTSF and site Water Management.
DPM	Provided Owner's General and Administration Costs, Operational Readiness, Taxes, Duties.
ERM	Provided the closure plan and costs with input from DPM

1.16.1 Capital Cost Estimate

The Capex was developed as an AACE Class 3 estimate to deliver an overall accuracy range of -10% to +20%. Ranges could exceed those shown, if there are unusual risks.

In this particular case, the Capex reflects an Engineering and procurement contract with a local engineering firm providing the detailed design and the Owner providing the construction management activities. 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 Q2 2025 pricing.

The Capex consists of direct and indirect costs, as well as contingency.

Table 1.9 presents a summary of the Initial Capex, and Sustaining Capex (Susex) distributed over the LOM indicated separately. Owner's costs, contingencies and risk amounts are included in this Capex.

Table 1.9 – Capital Cost Summary

Area	Description	Initial Capex (US$ M)	Sustaining Capex (US$ M)	Total (US$ M)
2000	Underground Mine	129.14	22.86	152.00
3000	Ore Handling	18.96	-	18.96
4000	Processing Plant	62.95	-	62.95
5000	Filtered Tailings / Water Treatment Facilities	52.38	6.07	58.45
6000	On-Site Infrastructure, Site-Wide General	65.30	-	65.30
7000	Incoming Power	2.41	-	2.41
8000	Operational Readiness	26.39	-	26.39
9000	Indirect Costs	32.96	2.89	35.85
9100	Owner's Costs	14.18	-	14.18
9900	Project Contingency	43.53	-	43.53
9900	Project Closure and Rehabilitation[1]	-	29.9	29.9
	Total Major Area Capex	**448.18**	**61.72**	**509.92**

- Closure costs include the non-recoverable VAT of approximately $2.6 M.
- All capital costs, including closure costs are assumed to be net of recoverable VAT.
- Numbers may not sum precisely due to rounding.

1.16.2 Operating Cost Estimate

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 7,344,769 tonnes milled provide the unit costs shown in Table 1.10.

Table 1.10 – Opex by Major Area

Area	LOM Total Opex (US$ M)	Average Opex (US$/t milled)
Mining	267.15	36.37
Processing and Tailings	204.84	27.89
G&A[1]	109.71	14.94
Total Opex	**581.70**	**79.20**

1 The operating costs presented above exclude property taxes of $5.8M., which are included in the economic analysis
2 Numbers may not add due to rounding.

1.17 Economic Analysis

The results of the economic analysis contain forward-looking information as defined by Canadian securities legislation. 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 detailed financial projection for the entire Project on an annual basis. The economic analysis is developed in terms of financial years with appropriate adjustments made to the production and cost schedules to convert the data from mine plan years. All prices and costs are in US dollars and accurate as of Q2 2025. No provisions have been made for the effects of inflation.

Table 1.11 provides a summary of the key parameters utilised in the economic analysis.

Table 1.11 – Economic Analysis Parameters

Description	Unit	Value
Macroeconomic Parameters		
Gold Price	US$ per oz	1,900
Project Parameters		
Discount Rate	%	5.0
Mine Life	years	10
Mineral Reserve (LOM)	Mt	7.3
Grade mined (LOM average)	g/t Au	6.44
Annual mill throughput	tpa	850,000
Gold recovery (LOM average)	%	87.9
Gold payability (LOM average)	%	98.5
Total gold produced (LOM)	Moz	1.34
Average annual gold production (LOM)[3]	oz	148,000
Average annual gold production (first five years)[4]	oz	189,000
Government royalty (NSR)	%	5.0

Description	Unit	Value
Capital Cost Estimate		
Initial Capital	US$ M	448
Sustaining Capital (LOM)	US$ M	32
Closure Costs[2]	US$ M	30
LOM Operating Unit Costs	**US$ M**	**US$/t ore processed**
Mining	267	36
Processing	205	28
General & Administrative[5]	110	15
Total Opex	***582***	***79***
Other Cash Costs	**US$ M**	**US$/t ore processed**
Property Taxes	6	1
Royalties	121	16
Cost Metrics (LOM Average)[6]		
Cash cost per Tonne of Ore Processed	US$/t ore processed	96
All-in Sustaining Cost per Ounce of Gold Sold	US$/oz payable gold	644

1. Numbers may not add due to rounding.
2. Closure costs include the non-recoverable VAT of approximately $2.6 M.
3. Average production from the 9 years with significant annual throughout (>50% of 850 ktpa).
4. Average production from first five full years of production.
5. The G&A operating costs presented above exclude property taxes of $5.8M, which are presented in a separate line.
6. Cash cost; Cash Cost per Tonne of Ore Processed; and All-in Sustaining Cost per Ounce of Gold Sold are non-GAAP financial measures or ratios. These measures have no standardised meaning under IFRS Accounting Standards (“IFRS”) and may not be comparable to similar measures used by other issuers. As the Project is not in production, DPM does not have historical non-GAAP financial measures nor historical comparable measures under IFRS, and therefore the foregoing prospective non-GAAP financial measures or ratios may not be reconciled to the nearest comparable measures under IFRS. 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.

DPM retained KPMG d.o.o. Beograd to review tax calculations for the after-tax economic evaluation of the Project. Current legislation in Serbia allows for income tax relief (‘tax holiday’) 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 holiday and that the tax relief will be declared by DPM once taxable profit is achieved. Based on a preliminary review of the project capital against the list of non-qualifying assets, the proportion of non-qualifying assets was assessed to be < 0.5%, resulting in a tax holiday percentage of around 99.5%. Thus, the effective income tax rate applied is close to 0% over life of the Project.

Based on the assumptions presented in this Report, the results of the economic analysis indicate a positive after-tax NPV of $782 M at a discount rate of 5%, an after-tax IRR of 35.6% and a payback period of 1.8 years, as shown in Table 1.12.

Table 1.12 – Economic Results Summary

Description	Unit	Pre-Tax	After-Tax
NPV @ 5%	US$ M	783	782
IRR	%	35.7	35.6
Payback Period	Years	1.8	1.8

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 1.13 and Figures 1.3 and 1.4. The NPV and IRR of the Project are most sensitive to variations in the gold price though the Project retains a positive NPV at the lower limit of the gold price interval tested. The IRR is more sensitive to variations in Capex than Opex, as evidenced by the steeper slope of the Capex line in Figure 1.4. 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 Report, the Project's potential after-tax viability does not seem significantly vulnerable to the under-estimation of capital and operating costs by up to 20%, when taken individually.

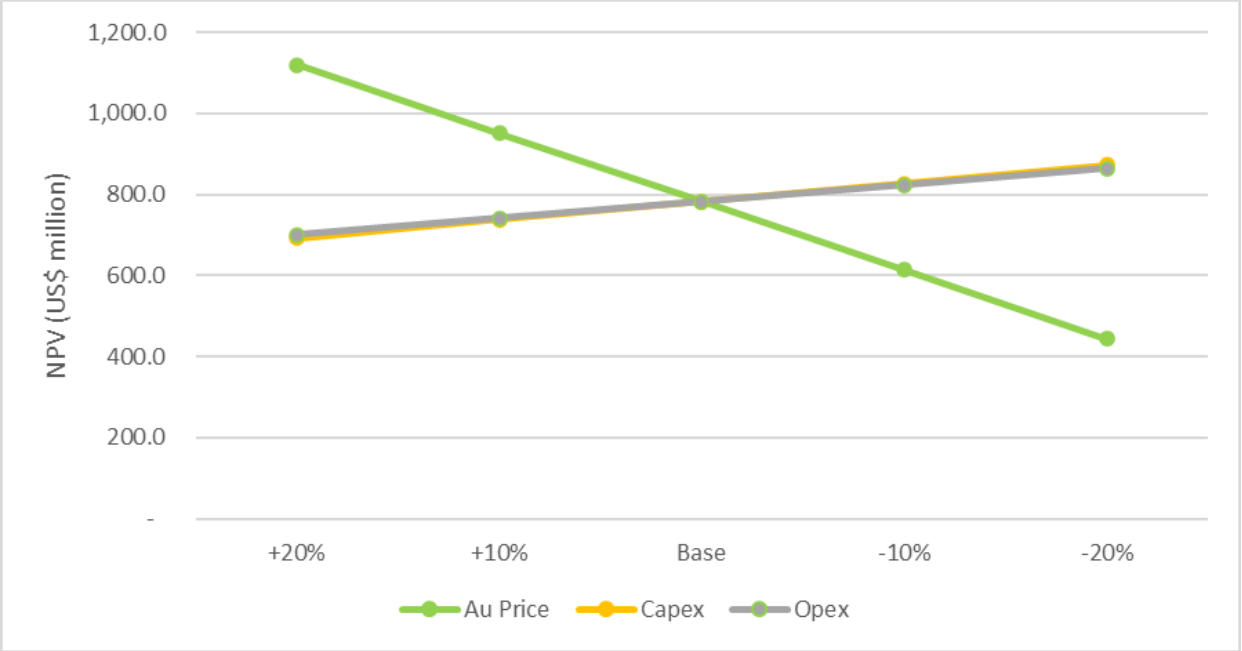
Table 1.13 – Economic Metrics Sensitivity to Variations in the Gold Price, Capex, and Opex

Au Price	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	1,120	951	782	613	444
IRR	%	44.7	40.3	35.6	30.5	24.9
Payback	years	1.5	1.7	1.8	2.0	2.3

Capex	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	693	737	782	827	872
IRR	%	29.2	32.2	35.6	39.6	44.2
Payback	years	2.1	1.9	1.8	1.7	1.5

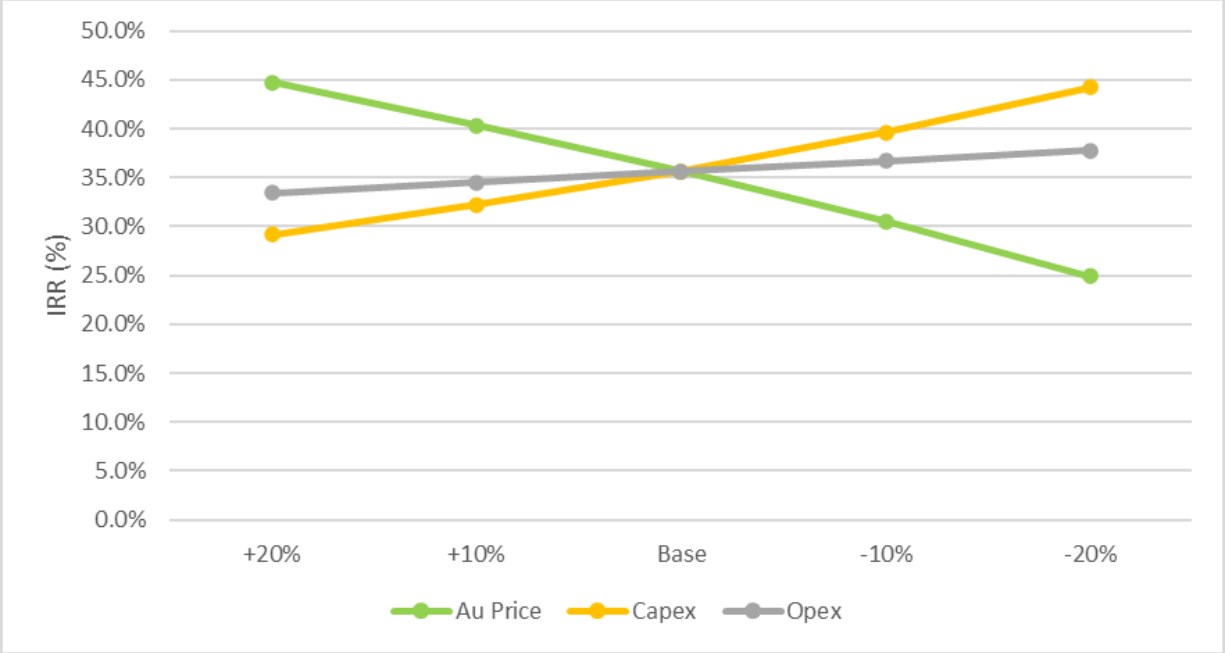
Opex	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	701	742	782	823	863
IRR	%	33.4	34.5	35.6	36.7	37.8
Payback	years	1.9	1.9	1.8	1.8	1.7

Figure 1.3 – After Tax NPV 5%: Sensitivity to Capex, Opex and Gold Price



Source: DRA, 2025

Figure 1.4 – After-Tax IRR: Sensitivity to Capex, Opex and Gold Price



Source: DRA, 2025

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 1.14 with the base case highlighted.

Table 1.14 – After-Tax NPV and IRR Sensitivity to Varying Gold Price

Gold Price	Unit	US$ 1,500/oz	US$ 1,700/oz	US$ 1,900/oz	US$ 2,300/oz	US$ 2,500/oz	US$ 3,500/oz
NPV at 5%	US$ M	427	605	782	1,139	1,317	2,207
IRR	%	24.3	30.3	35.6	45.2	49.5	67.8
Payback Period	years	2.3	2.0	1.8	1.5	1.4	1.0

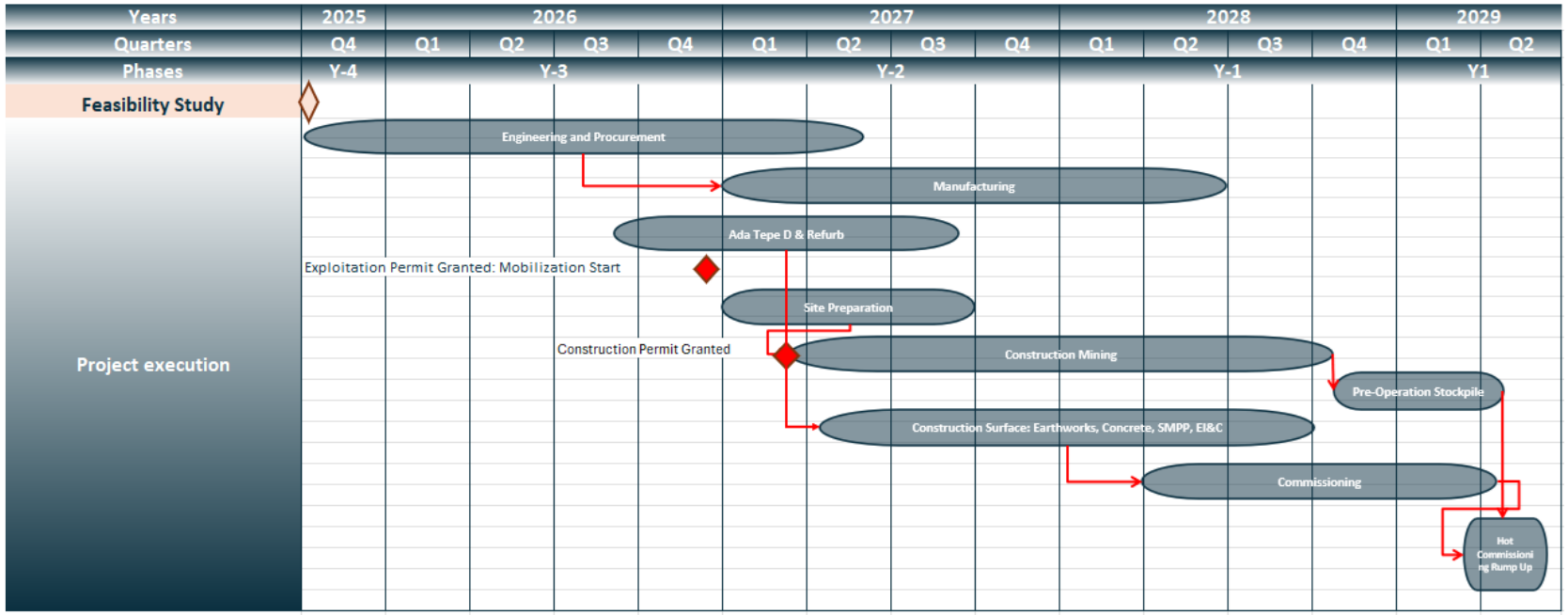
1.18 Other Relevant Information

1.18.1 Project Implementation Schedule

The Project Implementation Schedule, adopted for the FS, has been 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 FS by September 2025, followed by the completion of the basic engineering activities by the end of 2025. The schedule identifies the detailed engineering, procurement and execution phases in 2026, followed by project execution with production forecast for H1 2029.

A high-level Project Development and Implementation Schedule is provided in Figure 1.5, based upon the Project information available to date.

Figure 1.5 – Overall Key Date Schedule



Source: DPM, 2025

1.18.2 PROJECT EXECUTION PLAN

The Project will be executed as a green field project and will involve the engineering, detailed design, procurement, construction, and commissioning of a 70 kt per month (ktpm) concentrator plant, with feed being delivered from the ROM terrace. A number of existing equipment components and infrastructure will be re-purposed from DPM's Ada Tepe facility, with the remainder of the equipment and bulk materials being purchased new for the Čoka Rakita process facility.

The Project will be executed on the basis of one lead engineering consultant, who coordinates and integrates the efforts of other consultants contracted by DPM, to provide the detailed engineering requirements for the Project. In addition, the engineering consultant will provide procurement technical support for DPM-authorised purchase orders and contracts. The engineering activities will be managed by the engineering consultants, while leveraging DPM's Serbian and Bulgarian team(s) expertise, for the development and implementation of the procurement activities.

The engineering consultants will prepare the technical packages, specifications and tender documents. During the Feasibility Study, procurement packages were issued for firm bid tenders and budgetary quotes, with the local packages issued by the local DPM procurement team, and quotes received by DPM and provided to DRA and WSP, for technical analysis. Technical evaluations and recommendations were prepared by the engineering consultants and provided to DPM for final review and selection of preferred vendors. DPM will issue PO(s), during the basic engineering phase to acquire Vendor preliminary information and Vendor certified information.

The detailed engineering activities for the Project will be executed primarily by local Serbian engineering institutes, in accordance with national regulatory requirements. Their work will be supported and overseen by selected international engineering firms, who will provide subject-matter expertise, quality assurance, and independent technical reviews - particularly in specialized areas such as tailings, geotechnical, and mining engineering. The DPM project team will coordinate all engineering interfaces, ensuring alignment between local and international contributors, resolving design interactions early, and maintaining compliance with project standards. DPM will also lead the integration of engineering and procurement activities, to ensure that design deliverables, vendor data, and equipment specifications are fully aligned with the project schedule and contracting strategy.

DPM will self-perform the Construction Management program, as well as the Operations Readiness (OR), Commissioning and ramp-up phases. Technical assistance will be provided by the engineering consultant(s), other consultants and required vendors, during these phases.

1.18.3 OPERATIONAL READINESS DEVELOPMENT

The Operational Readiness Development 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.

1.19 Interpretation and Conclusions

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

1.19.1 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, Malcolm Titley, MAIG, conducted a personal inspection of the Project between May 13 to 15, 2025 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 January 17, 2025. A total of 299 drillholes totalling 141,505 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 approximate 1 g/t Au cut-off value. Detailed lithology and structural models were developed and used to constrain domain extents, as well as to

incorporate 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 (which is reported here), silver, 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 optimized mineable shapes using Deswik's Stope Optimiser (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).

1.19.2 Mineral Processing and Metallurgical Testing

Testwork results support the selected flowsheet and is adequate to size equipment with confidence. Variability results shows that there is little variability within the mineralised material. The only exception is with the arsenic content in concentrates produced from distinct variability samples; this prompted the introduction of concentrate storage bunkers to allow blending of the final product prior to shipment.

Confirmatory testwork performed during the FS phase validates the design decisions made during the PFS and adds confidence to the design criteria selected for sizing equipment. Pilot testing of the Jameson cell confirms both the recoveries and mass pull numbers that were previously adopted based on bench scale testing.

1.19.3 Mineral Reserve Estimate

Under the assumptions described in this Report, the proposed LOMP is achievable, and the economic analysis supports declaration of Mineral Reserves.

The Mineral Reserve Estimates are based on the most current knowledge, permit status, and engineering constraints. The QP is of the opinion that the Mineral Reserves were estimated using industry-accepted practices.

Under the assumptions and parameters discussed in this Report, results demonstrate the Čoka Rakita Project to be both technically and economically feasible. On this basis, it is recommended

DPM pursue a program of further investment and development to build upon the FS results presented.

1.19.4 MINING METHODS

Conclusions on mining methods are set out in Section 25.5 for the following areas:

- Geotechnical.
- Hydrogeology.
- Mining Methods.
- Mine Portal and Underground Facilities:
  - Mine dewatering.
  - Electrical and communications.
  - Mine backfill.
- Mine Ventilation.

1.19.5 FILTER AND PASTE BACKFILL PLANT

The laboratory testing completed by Jenike and Johanson (J&J) on the transportability of the 85 wt% solids Filter Cake confirmed that the current design inclination of the Paste Plant Incline Conveyor that transports Filter Cake from the Filter Plant to the Paste Plant for the production of paste backfill, was acceptable thereby validating the current design.

Similarly, the J&J testing also concluded the acceptable flow characteristics of the 85 wt% solids Filter Cake in the Paste Plant through the current configuration of the Live Bottom Feed Hopper into the Live Bottom Feeder and ultimately into the paste preparation process, thereby also validating the current design.

The testing conducted by The Metso Dewatering Technology Center (DTC) concluded that a Metso Larox FFP (62/62) could not dewater the 58 wt% solids tailings slurry to targeted 13 wt% moisture content (87 wt% solids) Filter Cake, however other Pressure Filter models provided by other Vendors were not investigated.

During the latter stages of the study, proctor testing showed that the optimum moisture content of the Filter Cake is 11 wt% or 89 wt% solids. As the Metso testing revealed a challenge in reaching 87 wt% solids, this remains a risk to be addressed. A DTSF design to accommodate a higher moisture content Filter Cake may be possible with implications to the operation explored in the Detail Design phase.

1.19.6 SHOTCRETE/CRF SLURRY BATCH PLANT:

A modular, integrated Batch Plant design will facilitate rapid construction during the mine development phase of the Project and offers flexibility in combining Shotcrete and CRF Slurry production capabilities into one plant.

The individual module's design proposed by two (2) Vendors provides the type of plant that is envisioned to meet DPM's needs.

1.19.7 DRY TAILINGS STORAGE FACILITY

Dry tailings storage facility's interpretations and conclusions are detailed in Section 25.9.

1.19.8 SURFACE WATER MANAGEMENT

Surface water management's Interpretations and conclusions on are contained in Section 25.10.

1.19.9 SITE-WIDE WATER BALANCE

Conclusions on site-wide salt and water balance are set out in Section 25.11.

1.19.10 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

Conclusions reached for the Project are contained in Section 25.14.

1.19.11 CAPITAL AND OPERATING COST ESTIMATES

The estimated capital and operating costs are reasonable for an AACE Class 3 estimate with a -10% / +20% level of accuracy as part of a FS level analysis.

The costs as presented are suitable for use as "the budget" for the project implementation.

1.19.12 ECONOMIC ANALYSIS

Based on the assumptions presented in this Report, the results of the economic analysis indicate a positive after-tax NPV of $782 M at a discount rate of 5% and an after-tax IRR of 35.6%. The sensitivity analysis indicates that the Project economics are most sensitive to the price of gold. Significant sensitivity is also observed to changes in the estimates of Capex and Opex. The Project maintains a positive after-tax NPV even at a gold price 20% below the base case of US$1,900/oz.

1.19.13 OTHER RELEVANT INFORMATION

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

The Project Execution Plan and Schedule will continue to evolve in the basic and detailed engineering phases of the Project; with commissioning and startup of the Project currently planned for Q2 2028 to Q2 2029, as per the DPM execution schedule presented in Figure 1.5.

DPM will finalise the ORP during the detailed engineering stage.

1.19.14 RISKS

The Project team (DPM, DRA, ERM, WSP) conducted a multi-day Project risk review workshop during the week of June 16, 2025, 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 FS risk review workshop utilised the risk register established during the PFS as the basis for the FS workshop.

The total number of risks identified in accordance with the DPM ERM Framework was 41 risks, which comprised: 4 “High” risks, 21 “Moderate” risks, and 16 “Low” risks.

During the detailed engineering and execution phases, additional work will be conducted to further mitigate the remaining High-risk items.

For details related to the Project Risk Review, refer to Section 25.17.

1.20 Recommendations

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

1.20.1 GEOLOGY

Recommendations on Geology are contained in Section 26.1 for the following items:

- Exploration.
- Drilling.
- Database.
- Assay QA/QC.

1.20.2 MINERAL PROCESSING AND METALLURGICAL TESTING

1.20.2.1 DRA Recommendations

All recommended testing has been completed. There are no further recommendations pertaining to future testwork.

1.20.2.2 Filter and Paste Plant Related Testing

Additional Pressure Filter testing should be conducted by preferably two (2) additional Vendors to finalise the selection of the appropriate Pressure Filter capable of producing Filter Cake at the target moisture content. The target moisture content of the Filter Cake has been revised from 13 wt% to approximately 11 wt% based on the latest Dry Tailings Storage Facility (DTSF) tailings testing information received following completion of testing in this phase of the Project.

The final selection of the applicable Pressure Filter model will also allow the finalisation of the Filter Plant facility footprint and its design in the next phase of the Project.

Hydraulic flow loop testing should be conducted to determine the paste backfill delivery system's friction loss values and finalise the design of the paste backfill underground distribution system (UDS) and positive displacement Paste Pump design pressure.

Unconfined compressive strength (UCS) testing should be conducted using a Serbian sourced blend of ground iron blast furnace slag and normal Portland cement (90/10), to meet the target paste UCS requirements.

1.20.2.3 Shotcrete/CRF Slurry Batch Plant

In order to advance the design of the Batch Plant both Shotcrete and CRF testing should be conducted to:

- Finalise and verify the selection of the Shotcrete recipe; and
- Determine the binder addition rates for a Serbian sourced blend of ground iron blast furnace slag and normal Portland cement (90/10), to meet the target UCS requirements of the Shotcrete and CRF.

1.20.3 Mineral Resource Estimate

It is recommended that the MRE be updated on the basis of new drilling and screen fire assays where possible. Infill drilling to derisk the portion of the mine plan that supports the first year of mining / the payback period of the project has been conducted, further studies to prepare for grade control drilling and geology support to mine production should be considered. Consideration of how to predict where higher and lower grades are present during mining should be made, investigating possible grade control scenarios and optimal drill spacing. This should be investigated by underground drilling to better investigate the close spaced grade and geological variability.

1.20.4 Mineral Reserve Estimate

DPM should prioritise underground infill drilling within the deposit areas defined in the FS, with the primary goal of converting Probable Reserves to Proven Reserves and upgrading Inferred Mineral Resources to the Indicated category. Simultaneously, an exploration drilling program should be

implemented to expand the mineral resource base, thereby enhancing the overall value of the Project.

1.20.5 Mining Methods

Detailed recommendations for mining methods are provided in Section 26.5 for the following items:

- Mining.
- Infrastructure.
- Geotechnical.
- Hydrogeology.
- Ventilation.
- Mine Backfill.

1.20.6 Recovery Methods

No additional ToS were performed during the FS. The flowsheet and main equipment remained unchanged from the PFS.

1.20.7 Project Infrastructure

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

- General site infrastructure development and optimisation.
- Filter Cake Production and Paste Backfill.
- Shotcrete/CRF Slurry Batch Plant.
- Dry Tailings Storage Facility.
- Surface Water Management.
- Site-Wide Water Balance.

1.20.8 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, and the Spatial Plan for the Project has been recently granted in November 2025. 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.

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 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.

1.20.9 Capital and Operating Cost Estimates

Recommendations with respect to Capital and Operating Cost Estimates are provided in Section 26.8.

1.20.10 Economic Analysis

Recommendations with respect to Economic Analysis are provided in Section 26.9.

1.20.11 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

DPM Metals Inc. (formerly 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, Bosnia and Herzegovina, Serbia, and Ecuador. DPM is listed on the Toronto Stock Exchange (TSX) (TSX:DPM) and the Australian Stock Exchange as a Foreign Exempt Listing (ASX:DPM) (ARBN: 689370894), with headquarters at 150 King Street West, Suite 902, Toronto, Ontario M5H 1J9.

This NI 43-101 Technical Report (the Report), for the Feasibility Study (FS) phase has been prepared on behalf of DPM, by the Contributors 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 (the Project).

Following the completion of the Mineral Resource Estimate (MRE), DPM has commissioned DRA Americas Inc (DRA) to lead a team of specialists to develop a FS, 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 FS is to build upon the effort of the preceding PFS and PEA, to 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. This FS 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 Terms of Reference Scope of Study

DPM retained various Consultants to participate in and prepare this Report. More specifically, the Consultants listed below provided Qualified Persons (QP (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 for process plant and related plant infrastructure; as well as to compile the Report.
- **Environmental Resources Management Ltd.** (ERM): develop Mineral Resource Estimate (MRE) as well as environmental studies, permitting, and social impact.
- **WSP Global 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 filtered tailings and paste backfill process plant designs.

2.2 List of Qualified Persons

As described above, DRA and other specialised consultants prepared the work in a collaborative manner. The QPs roles and responsibilities for this Report, 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	Malcolm Titley	ERM
5	Accessibility, Climate, Local Resources, Infrastructure, and Physiography	Malcolm Titley	ERM
6	History	Malcolm Titley	ERM
7	Geological Setting and Mineralisation	Malcolm Titley	ERM
8	Deposit Types	Malcolm Titley	ERM
9	Exploration	Malcolm Titley	ERM
10	Drilling	Malcolm Titley	ERM
11	Sample Preparation, Analyses and Security	Malcolm Titley	ERM
12	Data Verification	Malcolm Titley	ERM
13	Mineral Processing and Metallurgical Testing	Daniel (Niel) Morrison	DRA
14	Mineral Resource Estimates	Malcolm Titley	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	Darlene Nelson	WSP
	Section 16.3	Michal Dobr	WSP
	Sections 16.12, 16.13, and 16.15	Isaac Ahmed	WSP
17	Recovery Methods	Daniel (Niel) Morrison	DRA
18	Project Infrastructure		
	Sections 18.1 to 18.6 and 18.12 to 18.14	Ian Major	DRA
	Sections 18.7 and 18.8	Isaac Ahmed	WSP
	Section 18.9 and Section 18.10	Peter Corrigan	WSP

Section	Title of Section	Qualified Person(s)	Company
	Section 18.11	Ryan Sweetman	WSP
19	Market Studies and Contracts	Daniel Gagnon	DRA
20	Environmental Studies, Permitting and Social or Community Impact	Kevin Leahy	ERM
21	Capital and Operating Costs		All Parties
	All sections except for 21.1.7 and 21.2.3	Ian Major	DRA
	Sections 21.1.7 (Capex) and 21.2.3 (Opex)	William Richard McBride	WSP
22	Economic Analysis	Daniel Gagnon	DRA
23	Adjacent Properties	Malcolm Titley	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
Malcolm Titley	ERM	May 13 to May 15, 2025
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 11 to 13, 2024
Isaac Ahmed	WSP	June 11 to 13, 2024
Peter Corrigan	WSP	June 17 to 19, 2024 March 3 to 5, 2025
Darlene Nelson	WSP	December 5, 2024
Ryan Sweetman	WSP	March 3, 2025

2.4 Effective Date

The effective date of the Report is November 26, 2025. 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

The QP, Ian Major, P. Eng., from DRA was the lead author for the FS 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 IFRS and are referred to as non-GAAP financial measures or ratios. These measures have no standardised meaning under IFRS and may not be comparable to similar measures presented by other companies. These measures are intended to provide additional information and should not be considered in isolation or as a substitute for measures prepared in accordance with IFRS.

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

- Cash cost and cash cost per tonne of ore processed: Cash cost consists of all production related expenses including mining, processing, services, filtered tailings and paste fill, royalties and general and administrative. Cash cost per tonne of ore processed is calculated as cash cost divided by the tonnes of ore processed.
- All-in sustaining cost and all-in sustaining cost per ounce of gold sold: All-in sustaining cost consists of cash cost, plus treatment charges, penalties, transportation and other selling costs, as well as 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.

Cash cost per tonne of ore processed 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.

2.8 Forward-Looking Statements

This technical report contains "forward looking statements" or "forward looking information" (collectively, "Forward Looking Statements") that involve a number of risks and uncertainties. Forward Looking Statements are statements that are not historical facts and are generally, but not always, identified by the use of forward looking terminology such as "plans", "targets", "expects", "is expected", "budget", "scheduled", "estimates", "forecasts", "outlook", "intends", "anticipates", "believes", or variations of such words and phrases or that state that certain actions, events or results "may", "could", "would", "might" or "will" be taken, occur or be achieved, or the negative of any of these terms or similar expressions. The Forward Looking Statements in this technical report relate to, among other things; the estimation of mineral resources and mineral reserves and the realization of such mineral estimates; and expected next steps in the development of the Project, timing of permitting activities and other governmental approvals; potential gold recoveries; the price of gold, copper, silver, and other commodities; and the economic evaluation of the Project, including IRR, life of mine and NPV. Forward Looking Statements are based on certain key assumptions and the opinions and estimates of management and the QPs, as of the date such statements are made, and they involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements of DPM to be materially different from any other future results, performance or achievements expressed or implied by the Forward Looking Statements. In addition to factors already discussed in this technical report, such factors include, among others, risks relating to DPM's business, including possible variations in mineralised grade and recovery rates;

changes in project parameters; uncertainties with respect to actual results of current exploration activities; uncertainties inherent to the estimation of the MRE; uncertainties inherent with conducting business in foreign jurisdictions where corruption, civil unrest, political instability and uncertainties with the rule of law may impact the DPM's activities; the impact of the conflicts in Ukraine and the Middle East, including resulting changes to the DPM's supply chain and costs of supplies; product shortages; delivery and shipping issues; additional delays in the advancement of the project, including with respect to the commencement of drilling activities; closures and/or failure of plant, equipment or processes to operate as anticipated; labour force shortages; fluctuations in metal and acid prices and foreign exchange rates; limitation on insurance coverage; accidents, labour disputes and other risks of the mining industry; the ability of DPM, stakeholders and local communities to realize the anticipated benefits of the project; delays in obtaining governmental approvals or in the completion of exploration or development activities; opposition by social and non-government organizations to mining projects; unanticipated title disputes; claims or litigation; cyber-attacks and other cybersecurity risks; changes to tax regimes in the jurisdictions in which DPM operates; as well as those risk factors discussed or referred to in any other documents (including without limitation DPM's most recent Annual Information Form) filed from time to time with the securities regulatory authorities in all provinces and territories of Canada and available on SEDAR+ at www.sedarplus.ca. The reader has been cautioned that the foregoing list is not exhaustive of all factors which may have been used. Although the QPs have attempted to identify important factors that could cause actual actions, events or results to differ materially from those described in Forward Looking Statements, there may be other factors that cause actions, events or results not to be anticipated, estimated or intended. There can be no assurance that Forward Looking Statements will prove to be accurate, as actual results and future events could differ materially from those anticipated in such statements. The Forward-Looking Statements reflect current expectations regarding future events and speak only as of the date hereof. Unless required by securities laws, DPM and the QPs undertake no obligation to update Forward Looking Statements if circumstances or management’s or the QPs' estimates or opinions should change. Accordingly, readers are cautioned not to place undue reliance on Forward Looking Statements.

3 RELIANCE ON OTHER EXPERTS

The QPs have reviewed and analysed 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, Malcolm Titley (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.

Malcolm Titley (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 the summary of the commercial terms included in Section 19, Daniel Gagnon (DRA) fully relied upon:

- Commercial terms memo provided by DPM via email dated October 16, 2025.

For the royalty and tax information included in Section 22, Daniel Gagnon (DRA) fully relied upon:

- Royalty and Taxation memo provided by DPM (*DPM retained KPMG d.o.o. Beograd to review the tax calculations included in the financial model*) via email dated October 16, 2025.

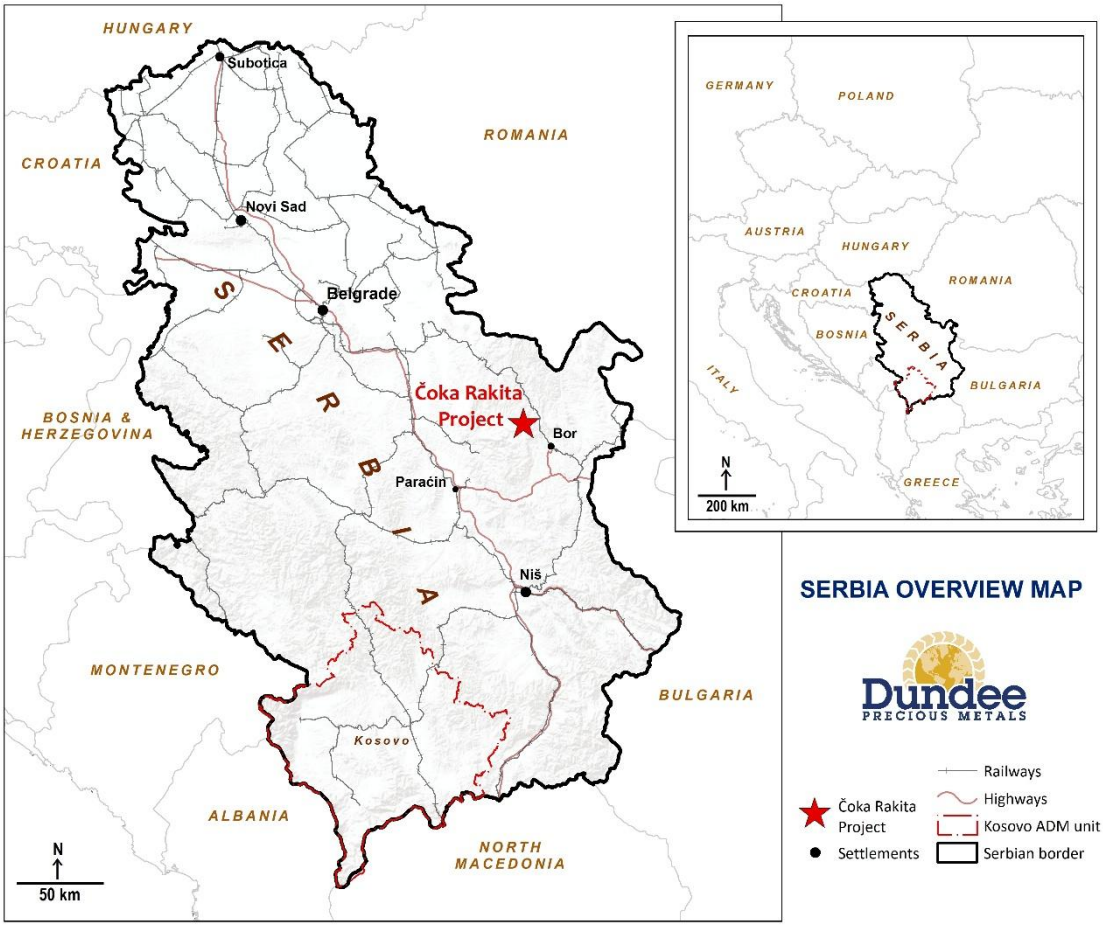
The QPs have assumed, and relied on the fact, that all the information and existing technical documents listed in the References Section 27 of this Report are accurate and complete in all material aspects, but have taken all appropriate steps in their professional judgement to confirm same and are not 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, 2025

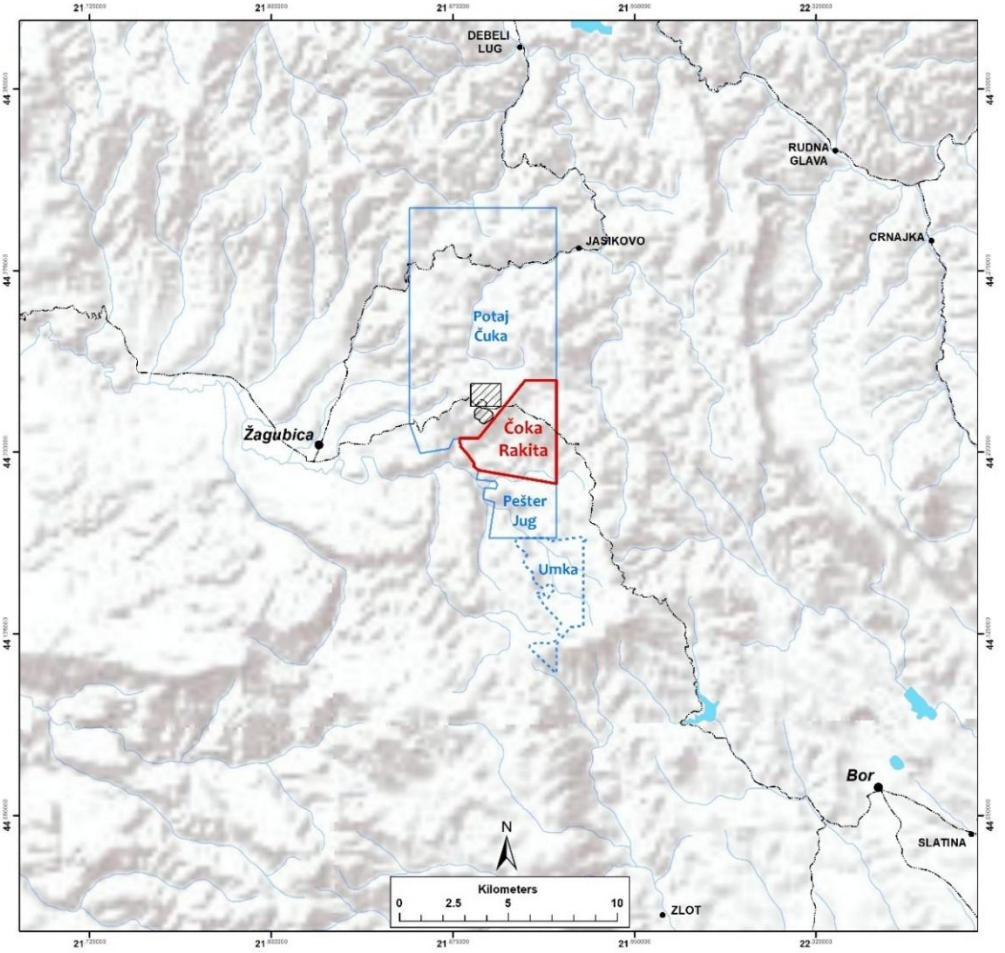
4.2 Mineral Tenure and Surface Rights

The Project is located in the Čoka Rakita exploration license which has an area of 13.81 km². In total, DPM has three (3) exploration licenses (Potaj Čuka, Pešter Jug and Čoka Rakita) and one (1) exploration license under approval process (Umka), covering an aggregate area of 95.71 km² in the

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

During 2022, the Potaj Čuka Tisnica license area was decreased, and a portion of the relinquished land was re-applied for by "DPM Crni Vrh d.o.o. (formerly Crni Vrh Resources d.o.o.) (Crni Vrh), 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 on 12 October 2023. The Umka exploration license was re-applied on 6 March 2025. The application process is ongoing. The exploration licenses and their boundaries are shown in Figure 4.2.

Figure 4.2 – Čoka Rakita Project Exploration License



Plan view showing the location of the Čoka Rakita exploration license highlighted in Red as well as the other exploration licenses owned by DPM shown in blue. The Umka License, which is currently under application, is shown in blue with a dashed outline. Additionally, third-party mining licenses are shown (black with crosshatch). Grid values in WGS 84 Grid System.
Source: DPM, 2025

There are no known material encumbrances affecting surface access to the Project. Where surface rights are held by private individuals, landowner consent must be obtained prior to the commencement of any surface exploration or development activities.

4.3 Exploration Licenses in Serbia

Exploration licenses in Serbia are currently granted by the Ministry of Mining and Energy (MME) 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 (2) years (second renewal), for a total potential term of eight (8) 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 DPM Crni Vrh d.o.o. which is a Serbian corporate entity and an indirect, 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 initial three-year period expired on 12 October 2025. DPM has submitted the application for the license extension for the first renewal period of three years and expects it to be granted in ordinary course in early 2026.

The Potaj Čuka, and Pešter Jug exploration licenses are currently within the first three-year phase while the renewal process for the Umka exploration license – which expired on October 19, 2024 (held by DPM Avala) - is ongoing. DPM fully expects the license approval as a new exploration license held by DPM Crni Vrh d.o.o., for the first three-year phase to be granted.

Upon the expiration of the exploration licenses, DPM is entitled to secure mineral rights to allow for permitting activities.

Table 4.1 – Summary of the Čoka Rakita Exploration License

License	License Number	Holder	Initial Grant Date	Expiry[1] Date	Area (km²)	Expenditure Commitment[2] (€)
Čoka Rakita	310-02-00980	DPM Crni Vrh d.o.o.	12 Oct 2022	12 Oct 2025	13.81	40,229,787

1 DPM has submitted the application for the license extension for the first renewal period of three years and expects it to be granted in ordinary course in early 2026.

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

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 €102 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, Malcolm Tilley, is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform work on the Project.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

5.1 Accessibility

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

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

Figure 5.1 – Project Location and Surrounding Towns



Source: DPM, 2025

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 approximately 40,000).

6 HISTORY

6.1 Prior and Current Ownership

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

No other private entities 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 Dundee Plemeniti Metali d.o.o.

In July 2010, Avala Resources Ltd. acquired certain exploration licenses in Serbia 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.

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 the previous operator Dundee Plemeniti Metali d.o.o. (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 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.

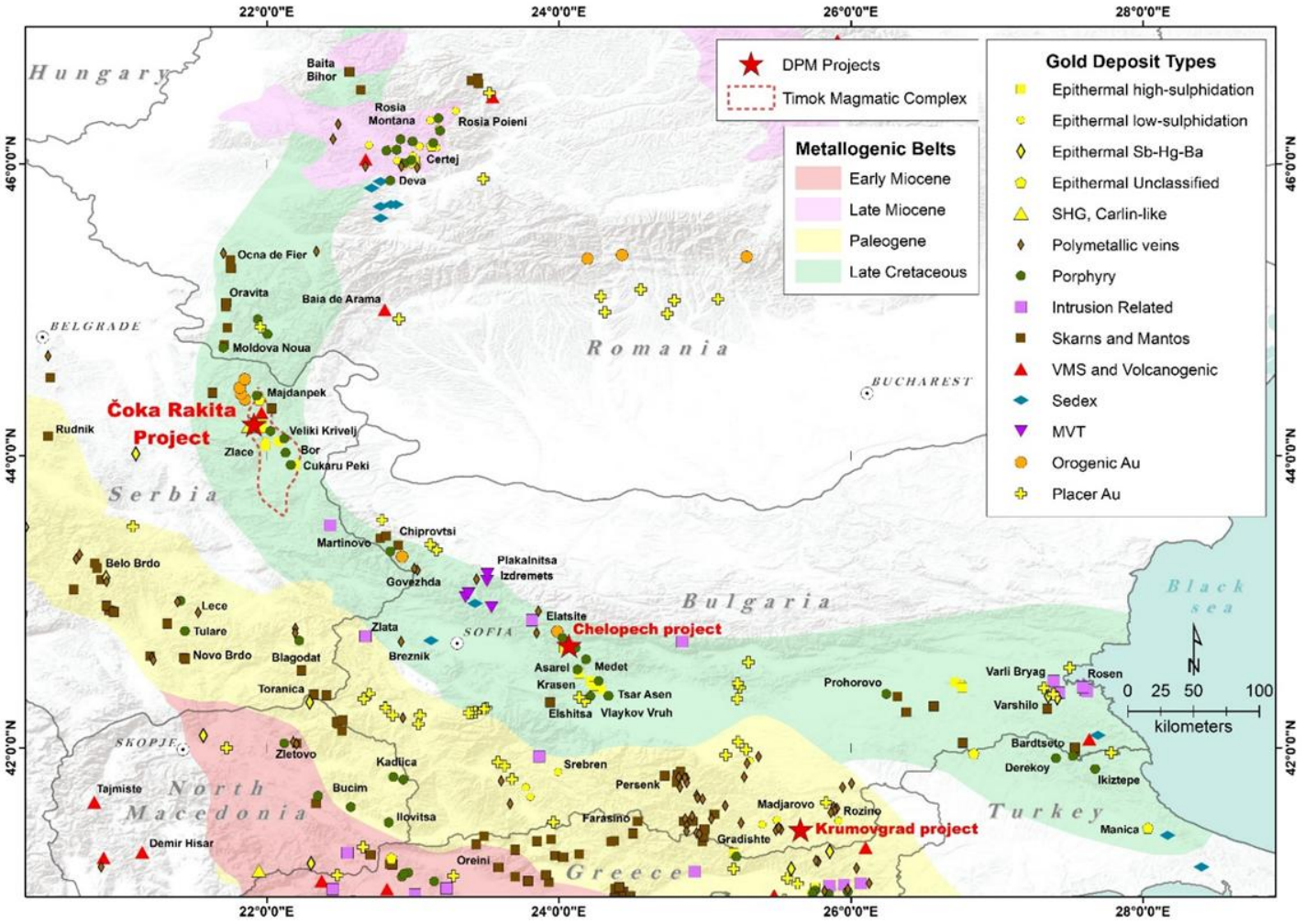
The complex arcuate geometry of the collision interface, and the presence of several micro-plates within the orogenic collage, resulted in a variety of collision products (Gallhofer et al., 2015). Some segments are characterised by extensive regional metamorphism, whereas others by calc-alkaline igneous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

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.

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



Source: DPM, 2023

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.

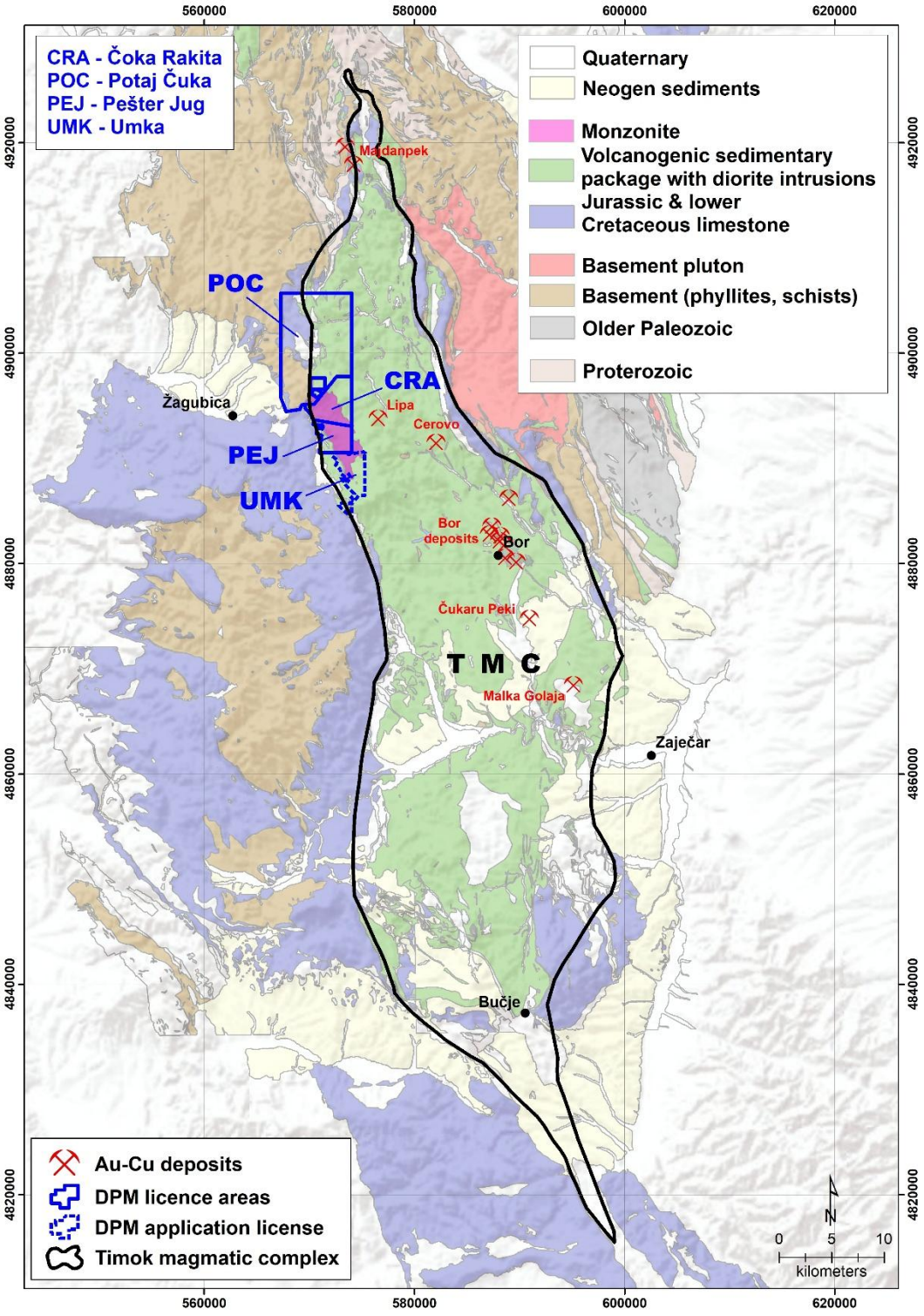
7.3 Local Geology

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, 2025

Contemporaneous sedimentation, magmatism, and hydrothermal activity were relatively continuous within the TMC throughout the entire Late Cretaceous, as illustrated in 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.

. 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.

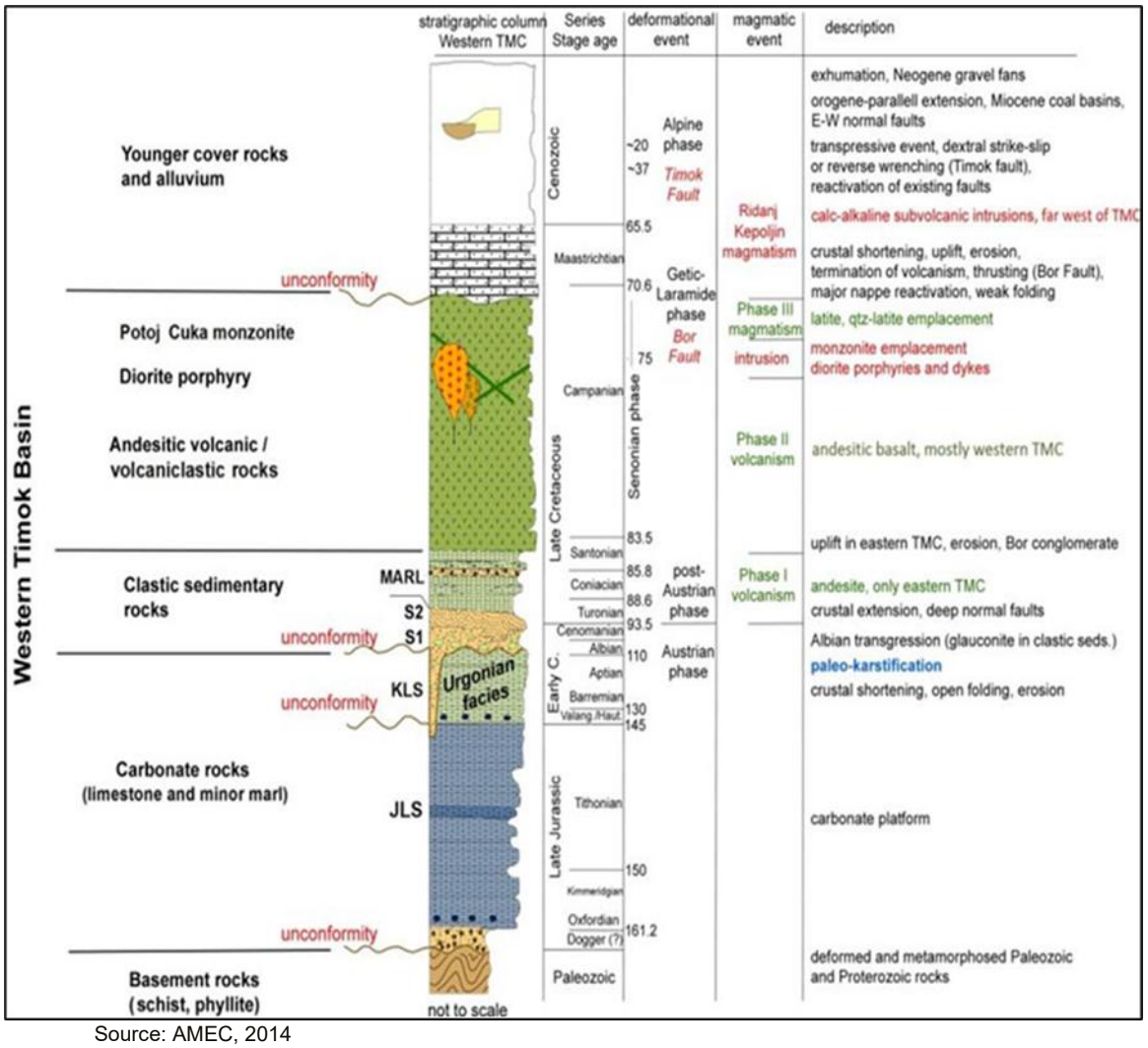
Subsequent phases of magmatism occurred from the Santonian to lower Campanian and comprised pyroxene basaltic andesite (Osnić basaltic andesite) and amphiboles andesite (Ježevica andesite). This suite is mostly found on the central and western portions of the TMC.

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.

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



Source: DPM, 2023

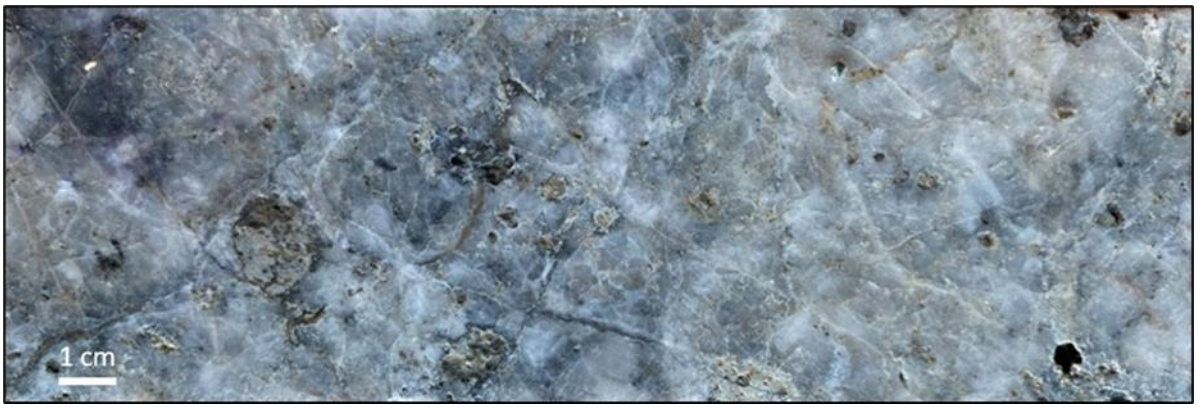
7.4.3 Calcareous Clastic Sedimentary Rocks, S1 and S2

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.

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



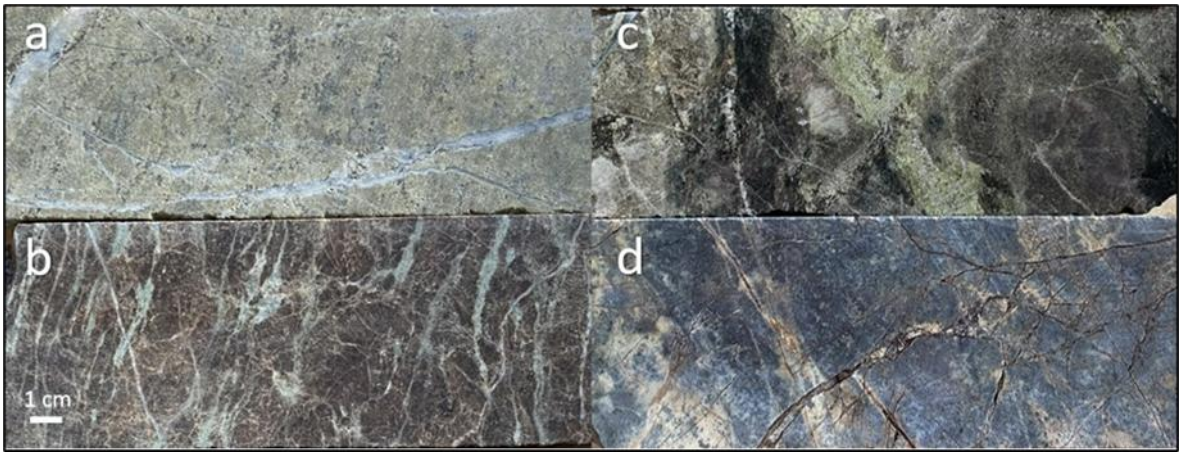
Source: DPM, 2023

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:

- Prograde, garnet-dominated skarn, with subordinate pyroxene, quartz, wollastonite; and
- 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



(a) 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

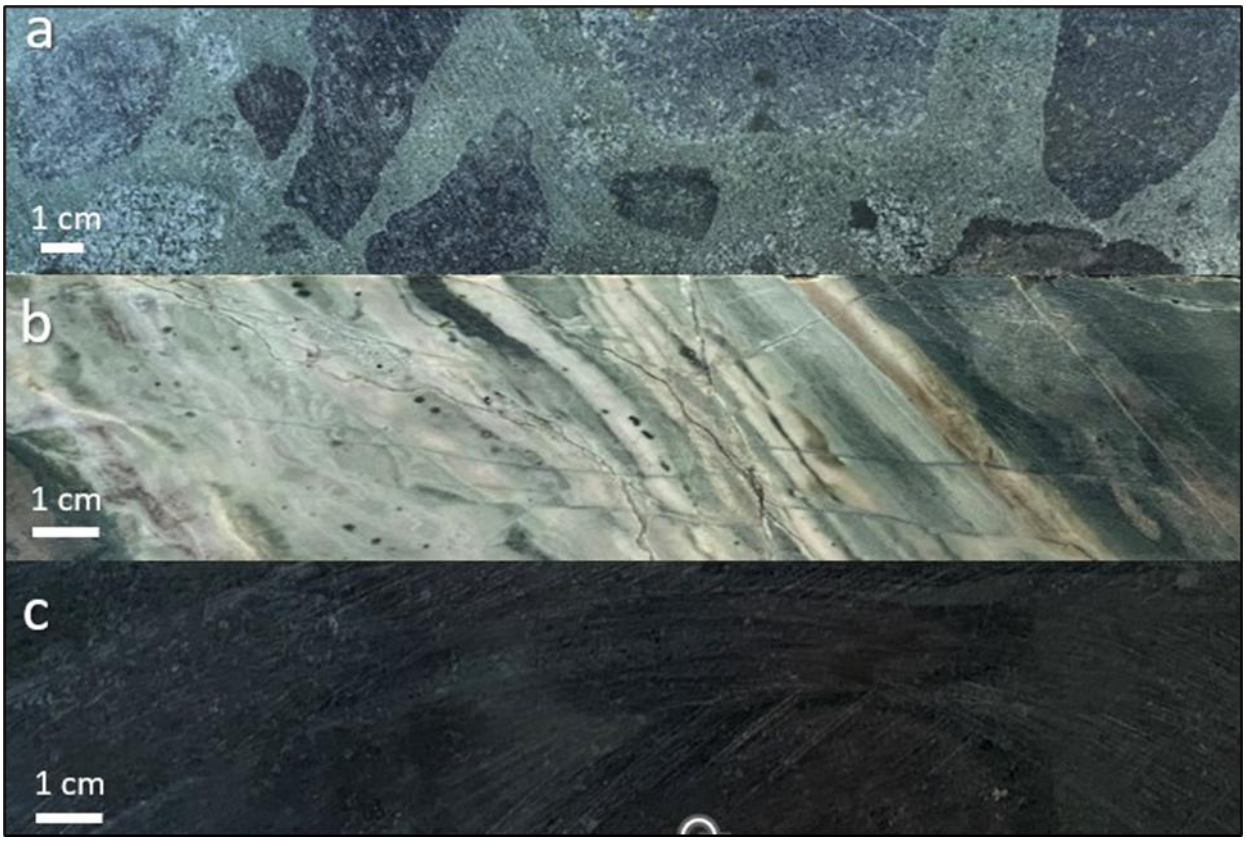
The Late Cretaceous epiclastic (SFD) unit is andesitic in composition and characterised by rapid facies changes throughout the sequence. The lower part of the epiclastic unit is characterised by polymictic basaltic andesite conglomerate and breccia, whereas the upper part is dominated by monomictic basaltic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

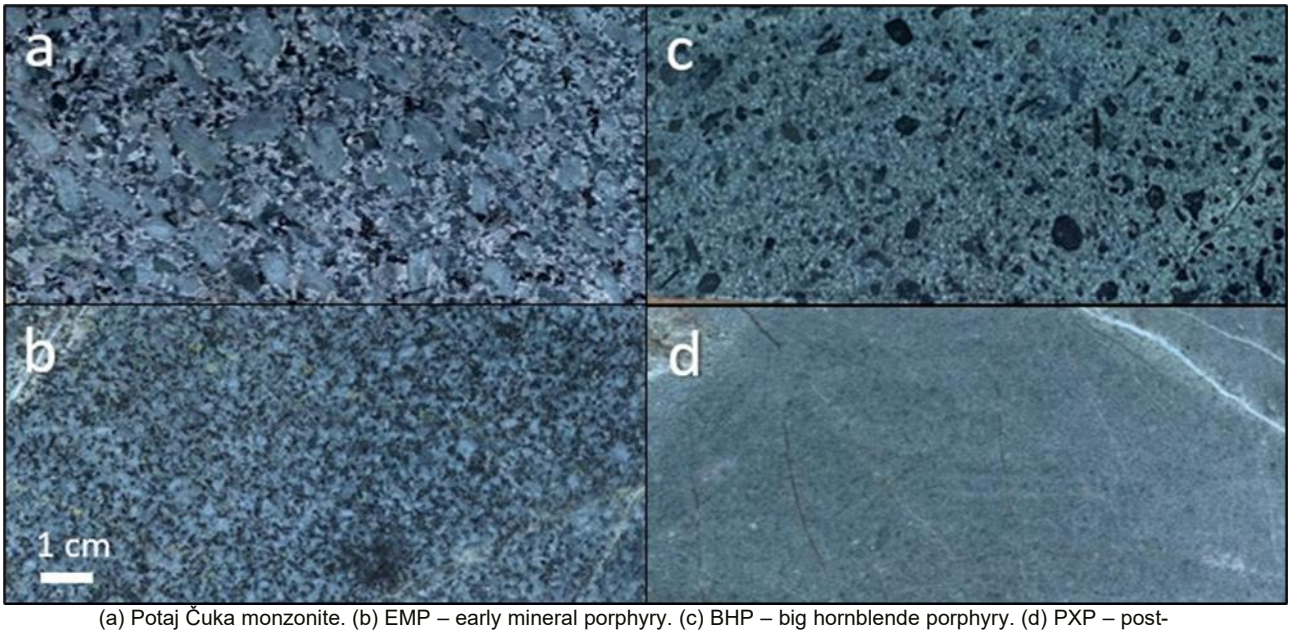
Numerous dioritic stocks, dykes, and sill-like intrusions are observed at Čoka Rakita. These bodies dominantly trend in a north-westerly orientation, which most likely represents a structural fabric in the subsurface that controlled their emplacement. The main intrusive phases identified on Čoka Rakita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

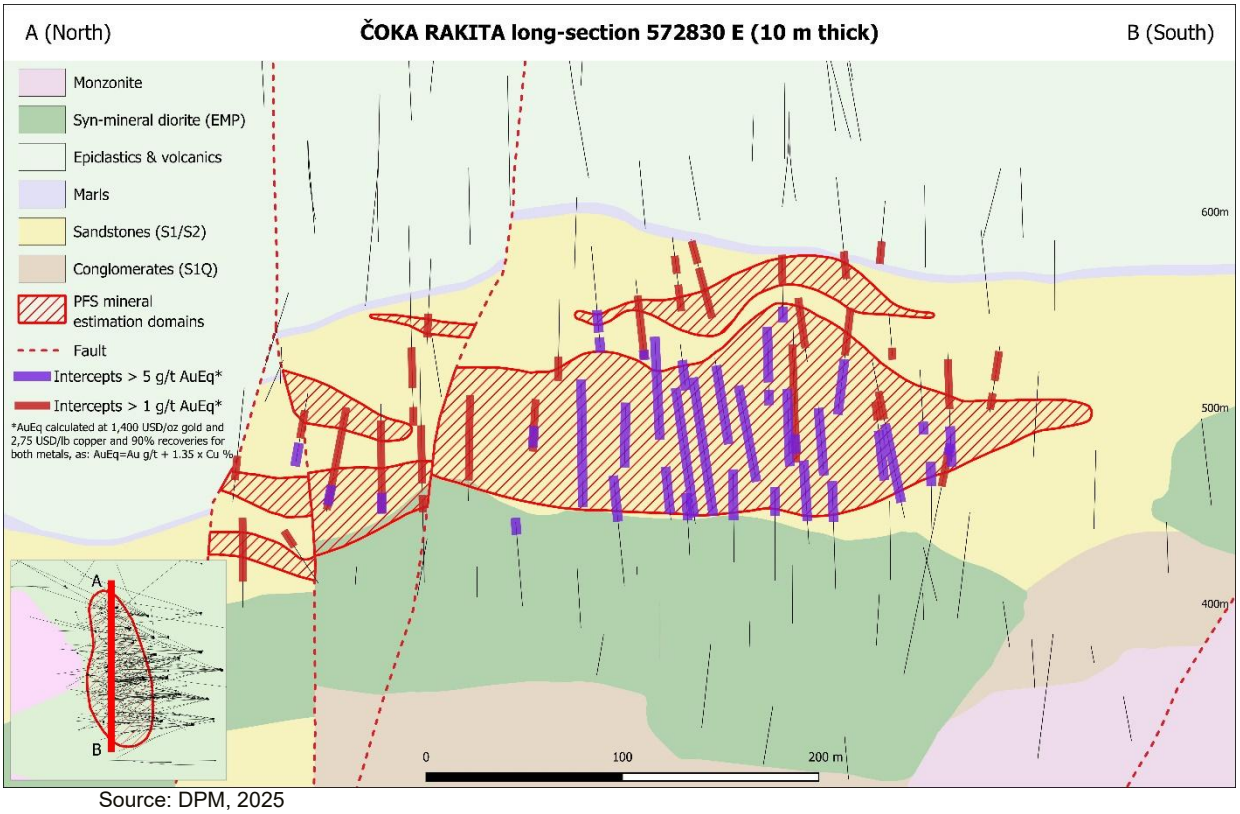
Steep north-south striking structures are observed in the central and southern parts of the deposit, dipping to the west at 70° to 80°. They appear as brittle deformation zones and often occur as hydrothermal breccia bodies. On the eastern margin of the deposit, one of those structures is

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, Drilling Intercepts and the PFS Mineral Estimation Domains within Different Litho-Tectonic Blocks, Divided by late East-West Faults



Source: DPM, 2025

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:

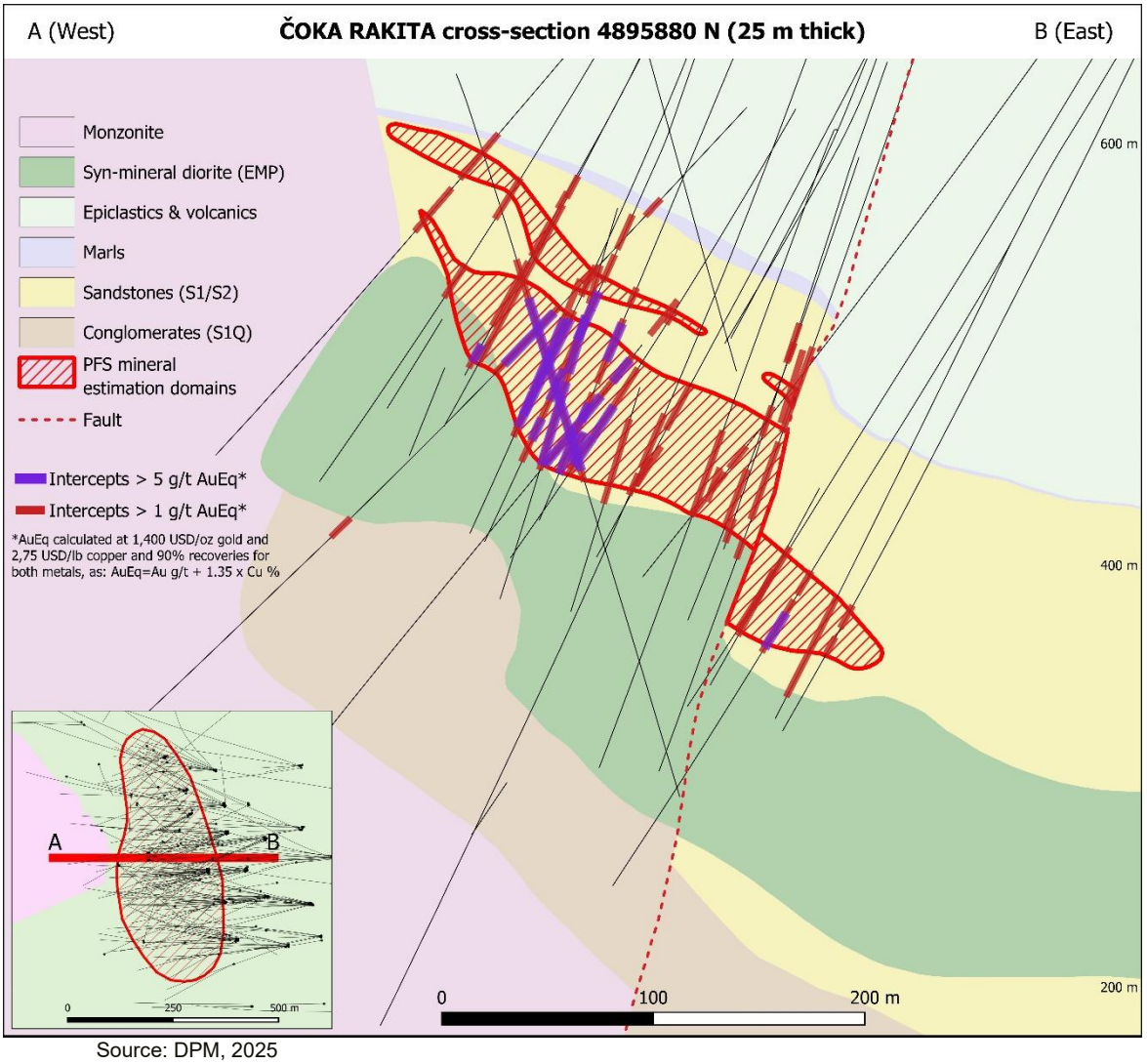
1. Stockwork quartz veinlets and disseminations related mineralisation in the potassic altered EMP, and
2. Epiclastic-hosted gold mineralisation controlled by structural and lithology contacts.

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.

Figure 7.11 – Schematic Cross Section Čoka Rakita Displaying Drilling Intercepts, Geology and the PFS Mineral Estimation Domains



Source: DPM, 2025

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

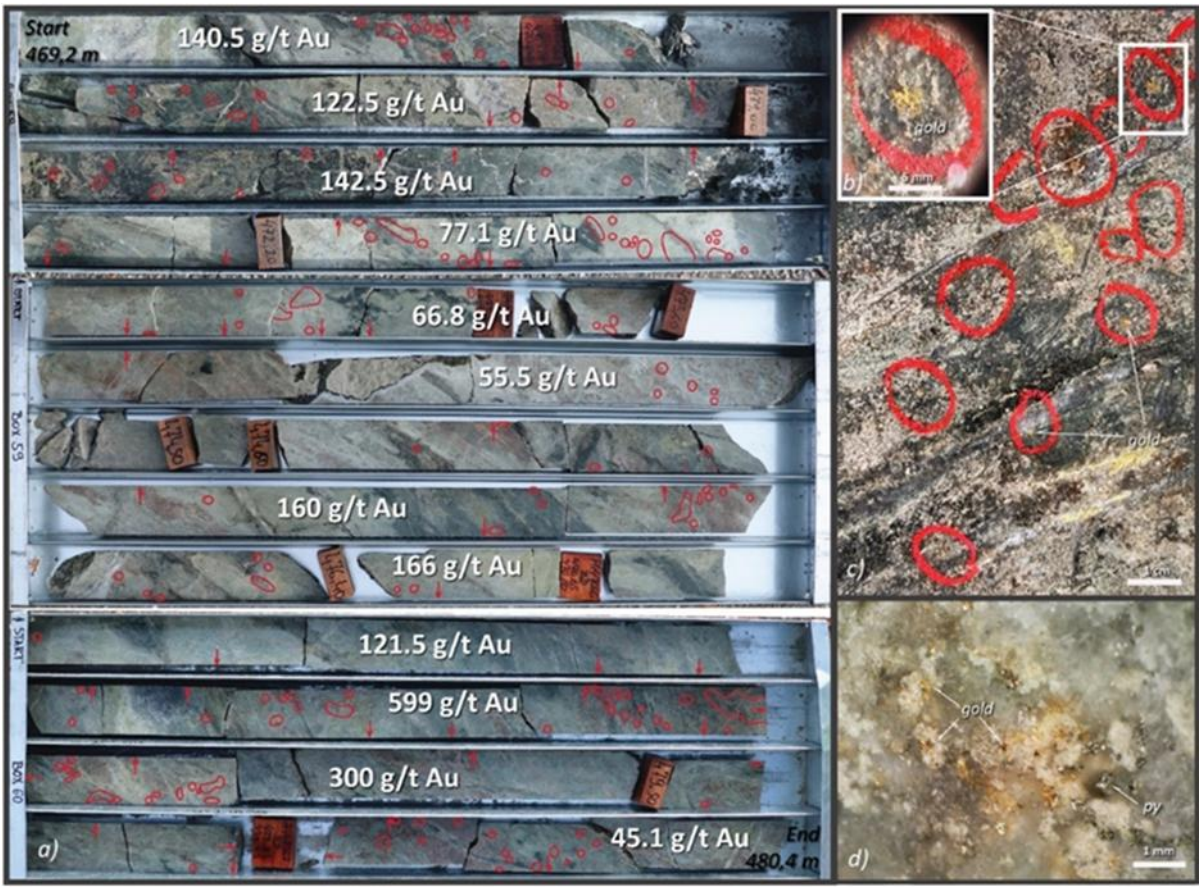
persists for a short distance within the EMP. As a second order control, a steeper north-south striking structural trend can also be determined, which is evidenced by the north-south elongation of the high-grade mineralised zones and with the occurrence of mineralised subvertical phreatic breccia zones.

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 RIDT025 is shown in Figure 7.12.

Figure 7.12 – Coarse Visible Gold from Drillhole RIDT025



(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.

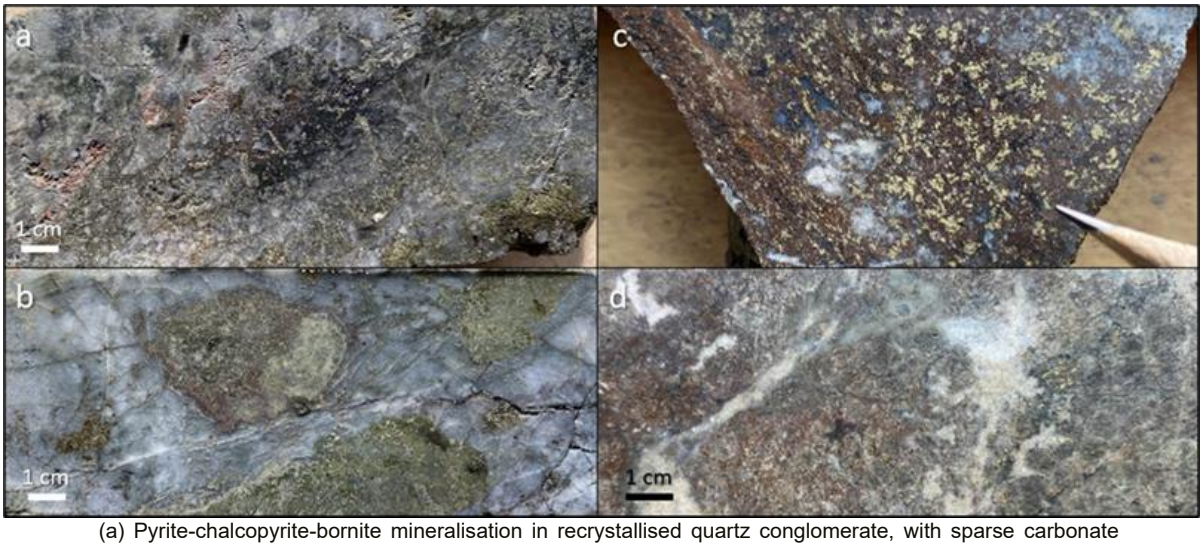
A pervasive secondary biotite±magnetite-K-feldspar assemblage formed in the EMP intrusion is associated with the low-grade disseminated and porphyry quartz-veinlet hosted copper±gold-molybdenum mineralisation. The mineralisation is represented by chalcopyrite, magnetite, pyrite and molybdenite. Although 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 marbleised limestones and on the limestone-sandstone contact. It is mainly associated with skarn alteration and intensive iron-oxide/hydroxide 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.

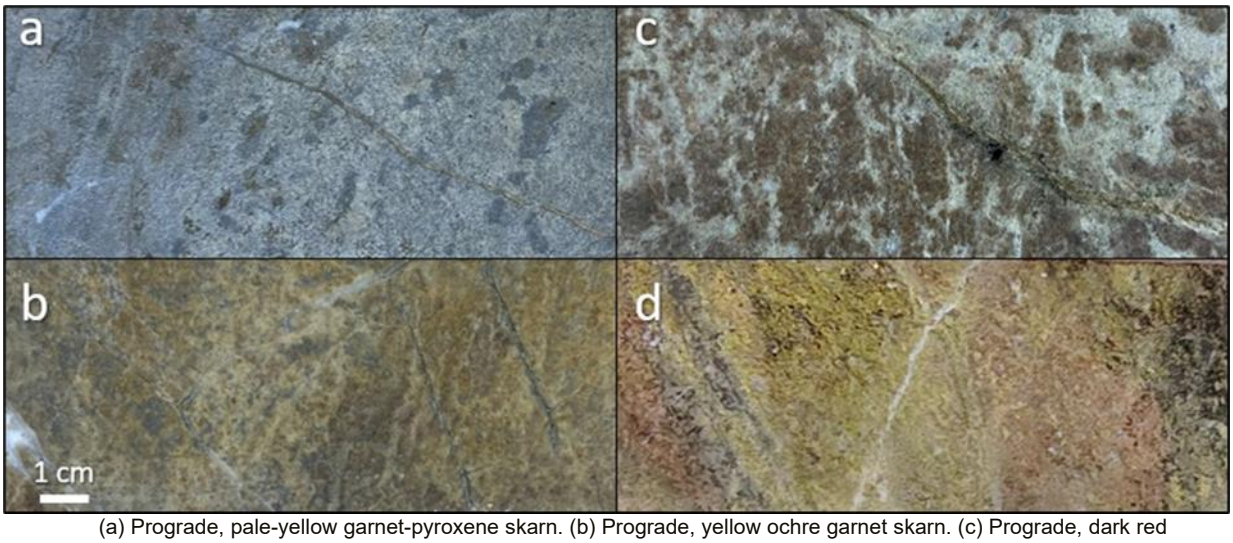
Prograde skarn is dominated by garnets of andradite-grossular series, with lesser pyroxene, wollastonite and quartz. On the western part of the project area, the skarn formation is typically dark red, with massive, banded or “leopard” texture, while to the east, its colour changes to yellow ochre. Prograde 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 mineralization 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 Alternation



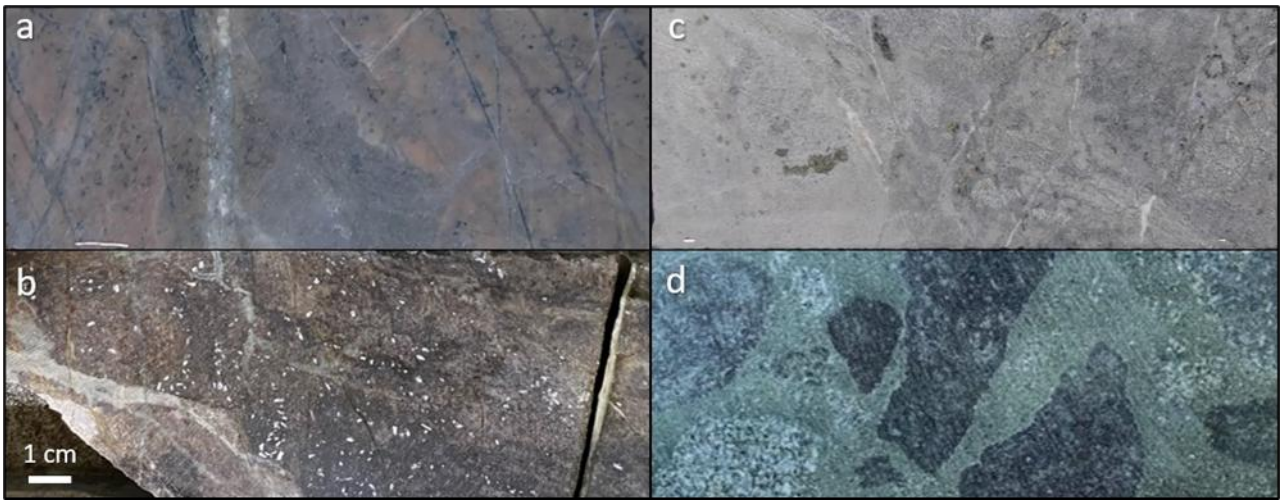
(a) Prograde, pale-yellow garnet-pyroxene skarn. (b) Prograde, yellow ochre garnet skarn. (c) Prograde, dark red garnet, wollastonite, pyroxene skarn with “leopard” texture. (d) Retrograde, epidote, actinolite, calcite skarn overprinting prograde skarn.
Source: DPM, 2023

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 Moz 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 Preliminary Economic Assessment (PEA) for the Timok Gold Project being disclosed in May 2014, based on MREs on the Bigar Hill, Korkan and Kraku Pester prospects and subsequently updated in 2019.

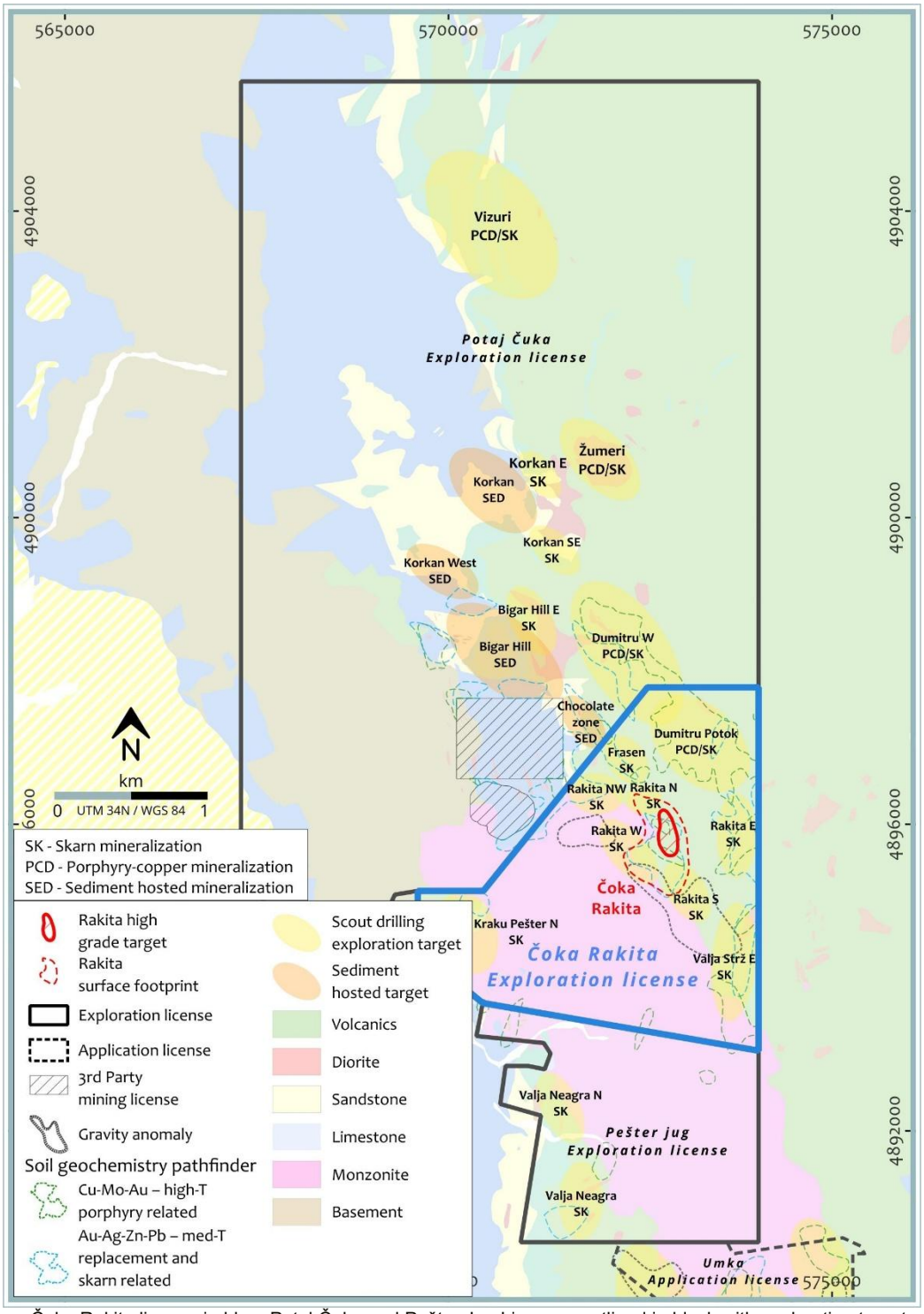
When exploration efforts were resumed, a new phase of drilling in 2016 was completed. A single deeper drillhole, that aimed to understand the potential for porphyry copper-gold mineralisation and search for potential skarn formation beneath the known extents of epiclastic-hosted gold mineralisation was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

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

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

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

Figure 9.1 – Overview Map of the Čoka Rakita Exploration License



Čoka Rakita license in blue, Potaj Čuka and Pešter Jug Licenses outlined in black with exploration targets shown on surface geology.
Source: DPM, 2025

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.

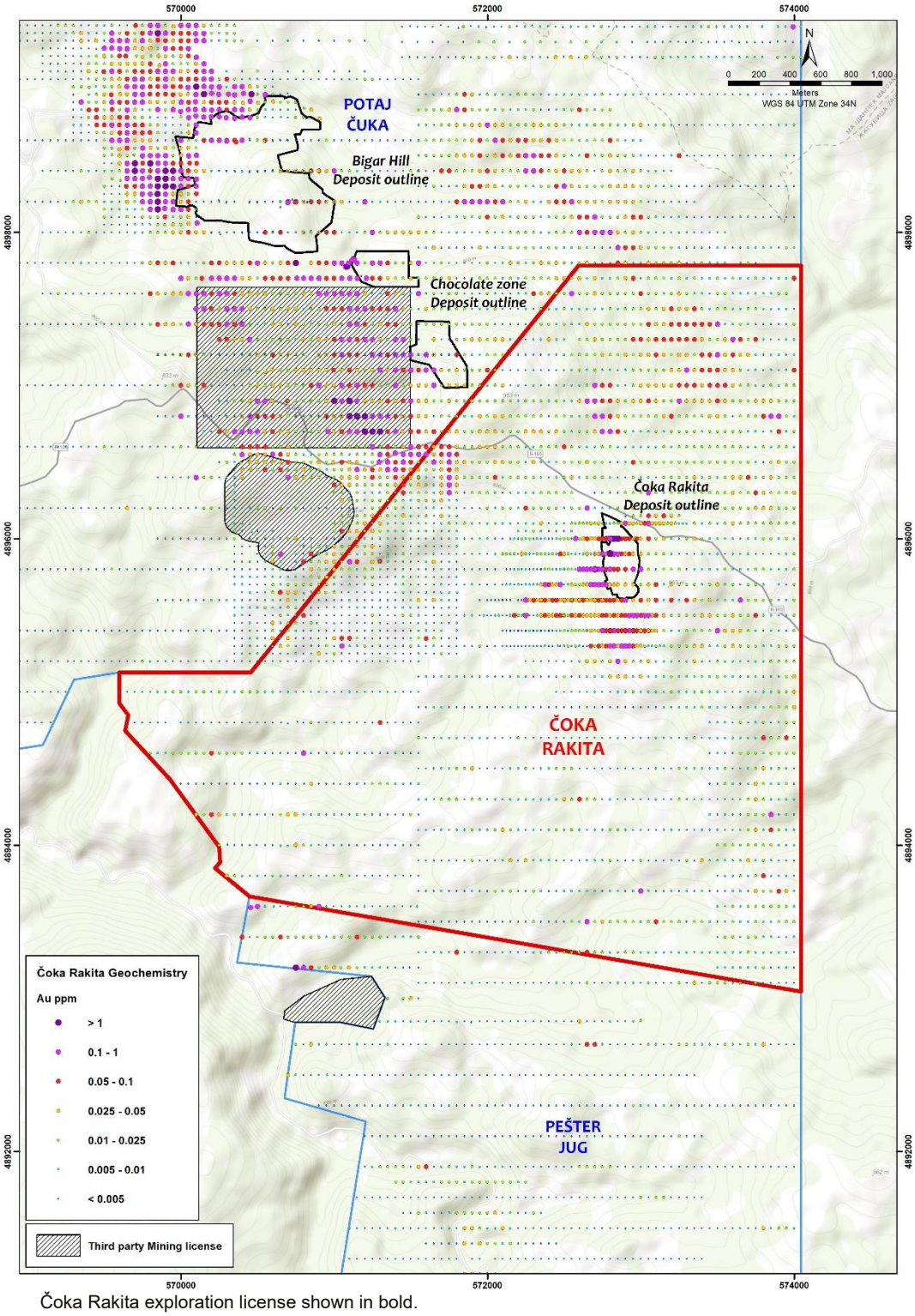
Within the Čoka Rakita exploration license, gold-in-soil anomalies to the northeast of the license are associated with porphyry mineralisation that broadly follows the trace of the Dimitru Potok porphyry system. Anomalous gold in soil results located above the Čoka Rakita deposit are related to epiclastic 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, 2025

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.

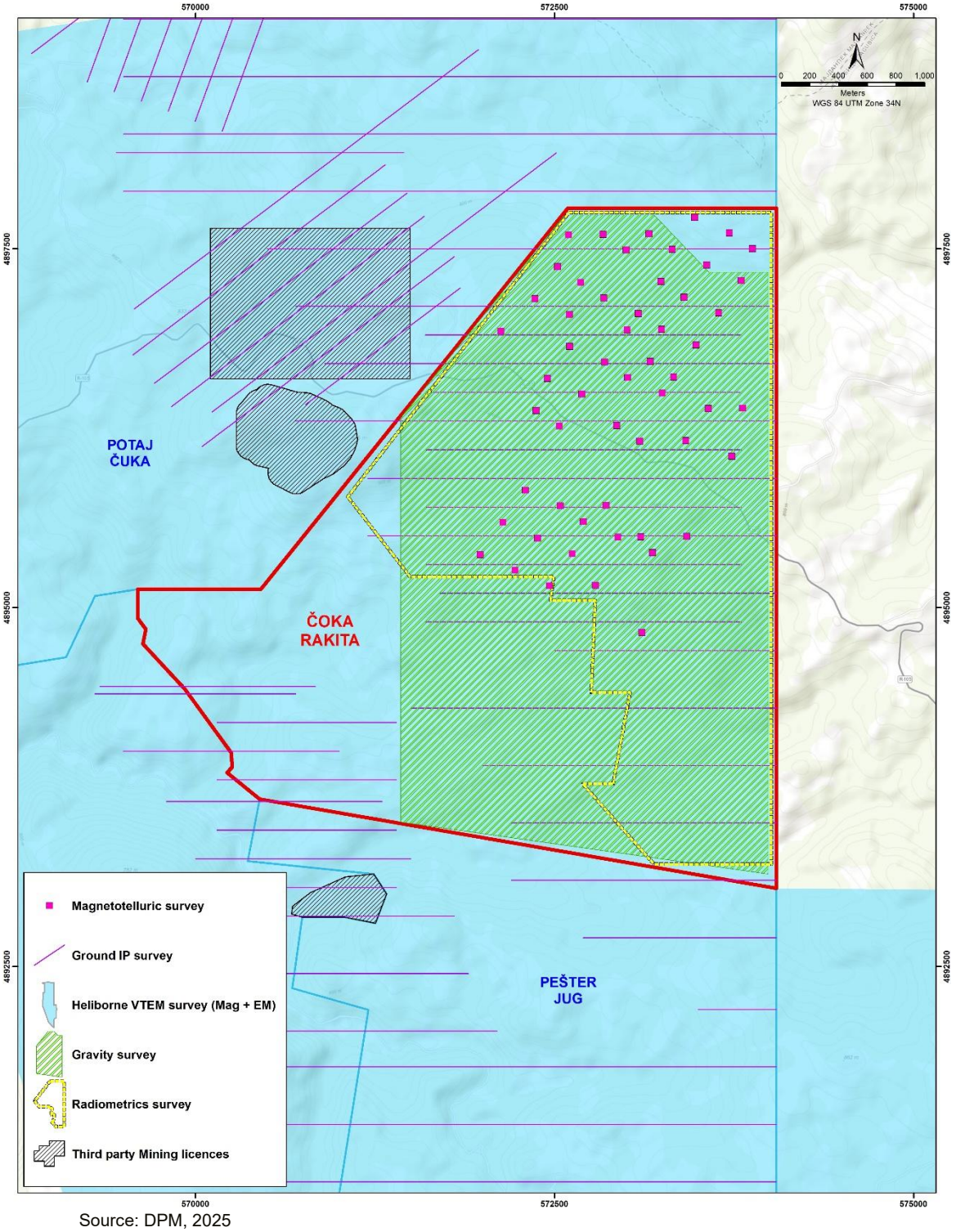
In 2022, detailed gravity surveys, using an approximately 250 m spaced random grid, identified a distinct gravity low over the Dimitru Potok porphyry system. The Čoka Rakita deposit is located in between the gravity low and on the eastern margin of positive gravity anomaly. Second rank positive gravity 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.

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



Source: DPM, 2025

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.

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 Rid (Right) at Čoka Rakita



Source: DPM, 2023

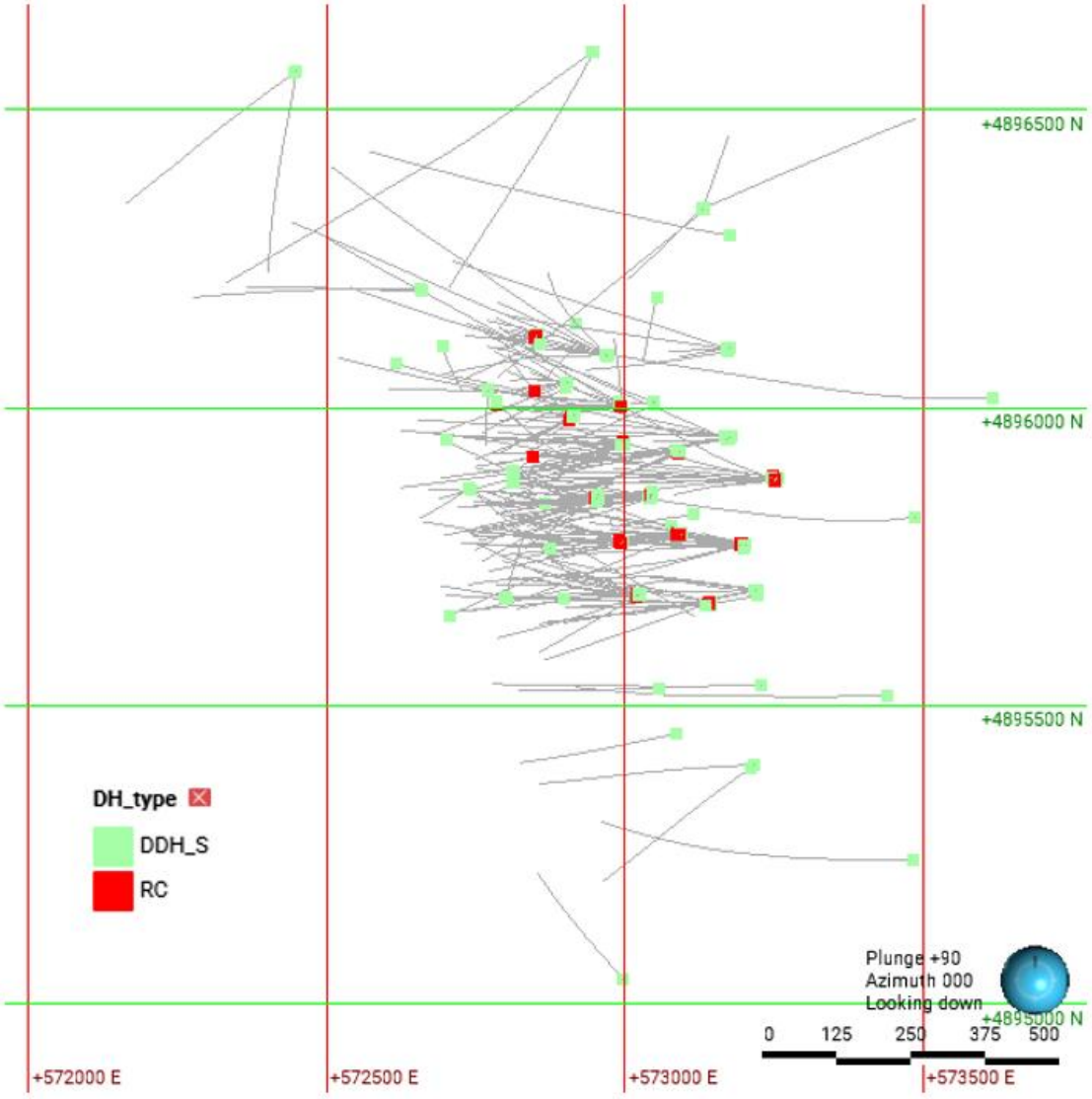
Drilling operations are summarised by area and year completed in Table 10.1. All the drilling activities on the Project have been completed during the tenure of DPM. Figures 10.2 to 10.4 present the drillholes completed at each deposit.

Table 10.1 – Summary of Drilling by Type and Year at Čoka Rakita, up to January 2025

Year	Diamond		RC		Diamond Tail		Metallurgical		Geotechnical		Hydrogeo		Total	
	#	Metres	#	Metres	#	Metres	#	Metres	#	Metres	#	Metres	#	Metres
2008	-	-	4	474	-	-	-	-	-	-	-	-	**4**	**474**
2009	9	1,319	-	-	-	-	-	-	-	-	-	-	**9**	**1,319**
2016	1	588	-	-	-	-	-	-	-	-	-	-	**1**	**588**
2017	1	225	-	-	-	-	-	-	-	-	-	-	**1**	**225**
2020	4	3,213	-	-	-	-	-	-	-	-	-	-	**4**	**3,213**
2021	24	15,116	-	-	-	-	2	1,160	-	-	-	-	**26**	**16,275**
2022	4	3,585	-	-	-	-	-	-	-	-	-	-	**4**	**3,585**
2023	63	37,914	44	7,485	33	16,520	-	-	-	-	-	-	**140**	**61,919**
2024	80	38,888	-	-	13	5,277	-	-	9	4,615	8	5,127	**110**	**53,907**
Total	**186**	**100,848**	**48**	**7,959**	**46**	**21,797**	**2**	**1,160**	**9**	**4,615**	**8**	**5,127**	**299**	**141,505**

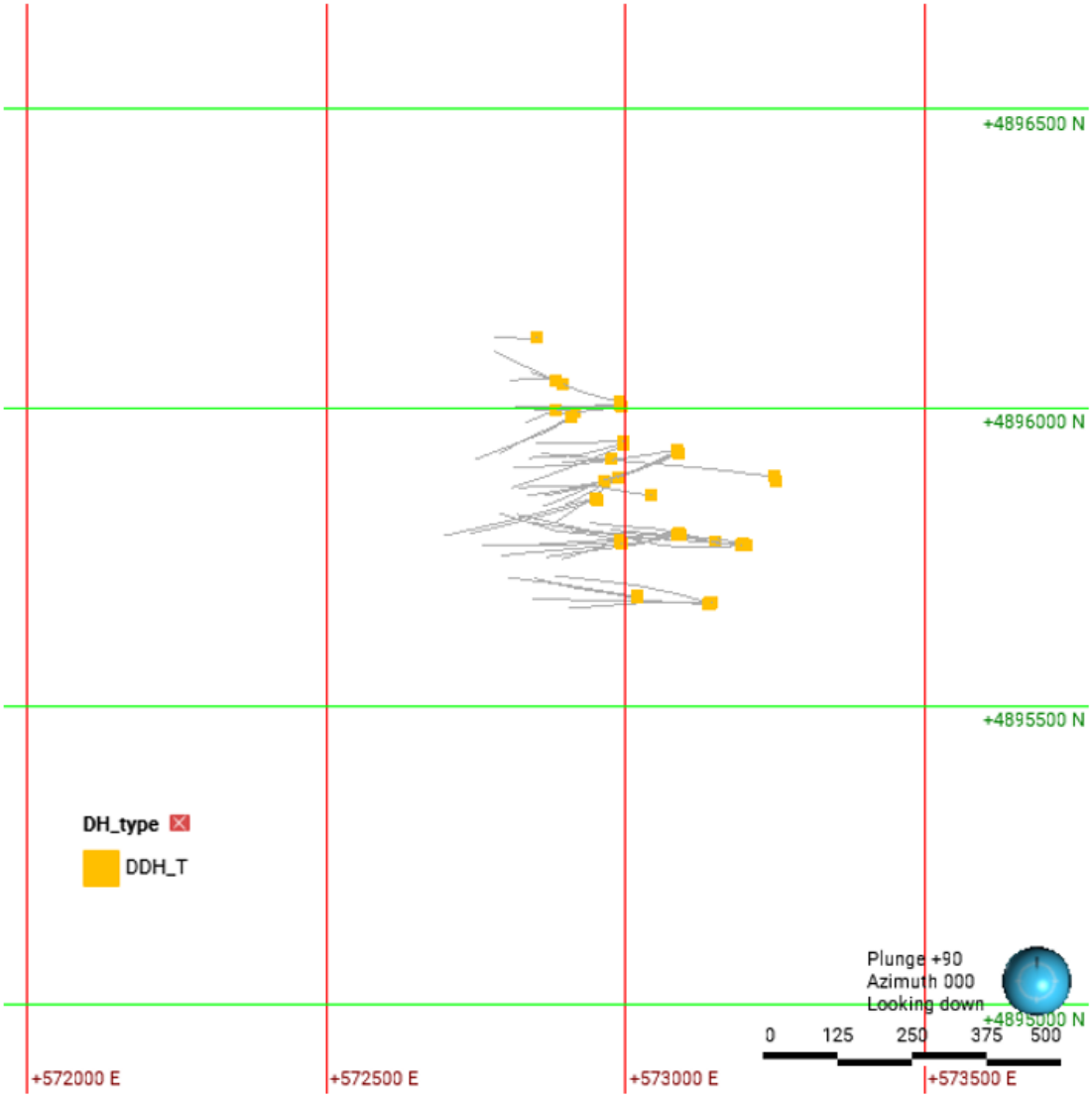
Source: ERM, 2025

Figure 10.2 – Plan Map of Diamond and RC Drillholes Completed



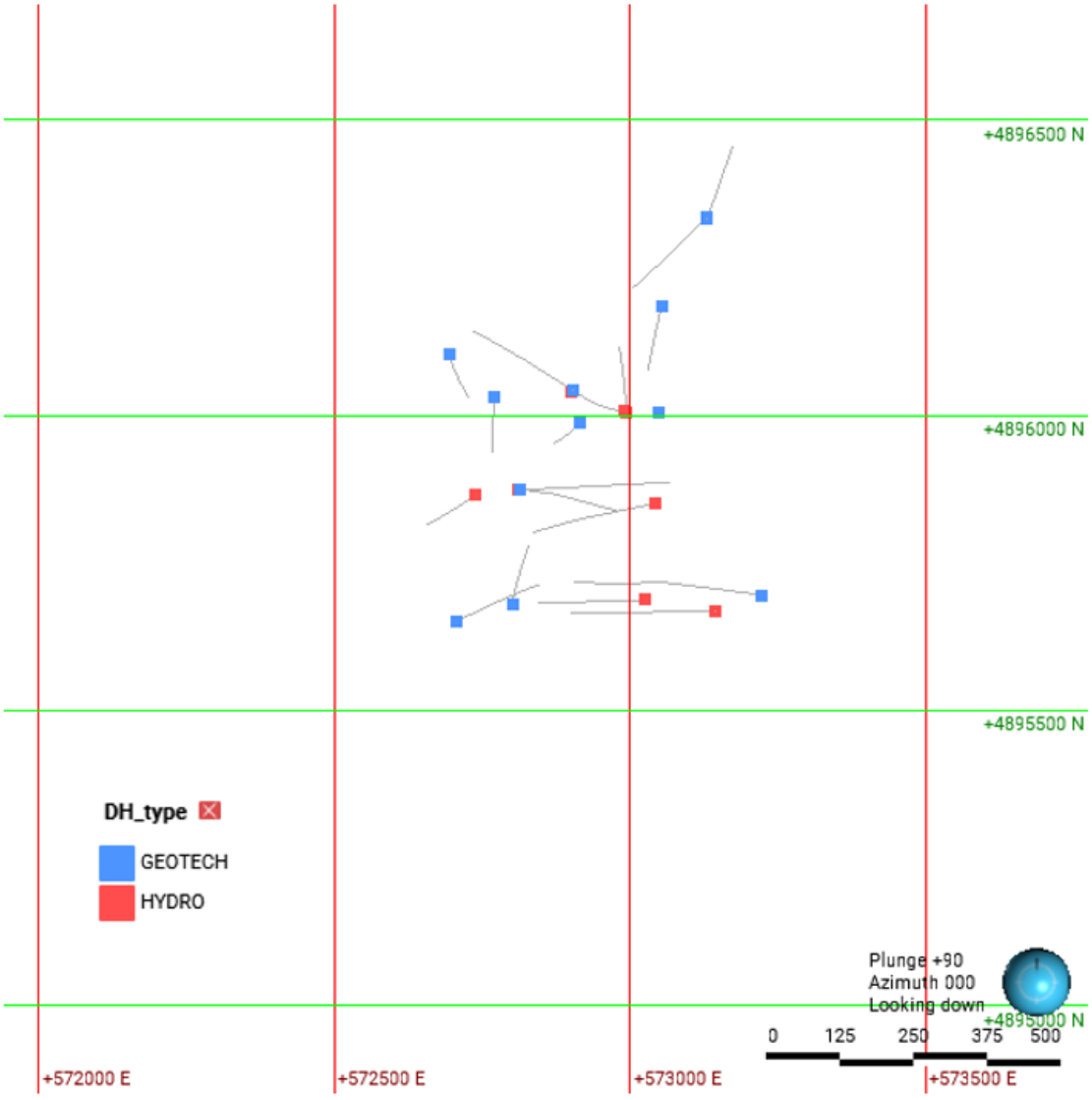
Source: DPM, 2025

Figure 10.3 – Plan Map of Diamond Tails Drillholes Completed



Source: ERM, 2024

Figure 10.4 – Plan Map of Geotechnical and Hydrogeological Diamond Holes Completed



Source: DPM, 2025

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

10.4.1 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 every drill run intervals, using a built-in electronic accelerometer, that is attached to the top of a core barrel and continuously determines the direction of the down gravity vector across the tool, which provides the orientation of the core barrel hole.

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.

10.4.2 RECOVERY

Diamond drilling core recovery averages, excluding those intervals where navigational drilling was undertaken, is 97.73% for all rock types. The majority of drill core was HQ3 size, followed by PQ3 and a small proportion of NQ2. 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.

10.5 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.

Magnetic susceptibility measurements are taken on core at regularly-spaced one metre intervals down a hole. Measurements are done on handheld pieces of core that have been removed from core trays or racks to minimize interference from adjacent magnetized material.

Point load test measurements are commonly taken on core at regularly spaced intervals down a hole.

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.

10.5.1 Downhole Televiewer Surveys

Downhole logging using an acoustic and optical Mount Sopris televiewer was performed by a DPM teams on 22 holes for a total of 5,319.0 m at the Čoka Rakita Project. The results were used for geotechnical and structural interpretation needs.

10.6 Reverse Circulation Drilling

10.6.1 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 (2) 35 m³ 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 measuring device. 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.

10.6.2 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.

10.6.3 RC Pre-Collar Drilling

No RC pre-collar drilling was completed in 2024 or 2025. 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.

10.6.4 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.

10.7 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.

10.8 Geotechnical Drillholes

Geotechnical drilling program was proposed and conducted in order to complete the geotechnical characterisation of the rock mass in the deposit area. Within the program, there were nine (9) drillholes completed with 4615 total metres of drilling Additionally, more drill holes are planned at infrastructural objects area and along the decline traces area, that will test the structural pattern, as well as the geotechnical characteristics of the ground.

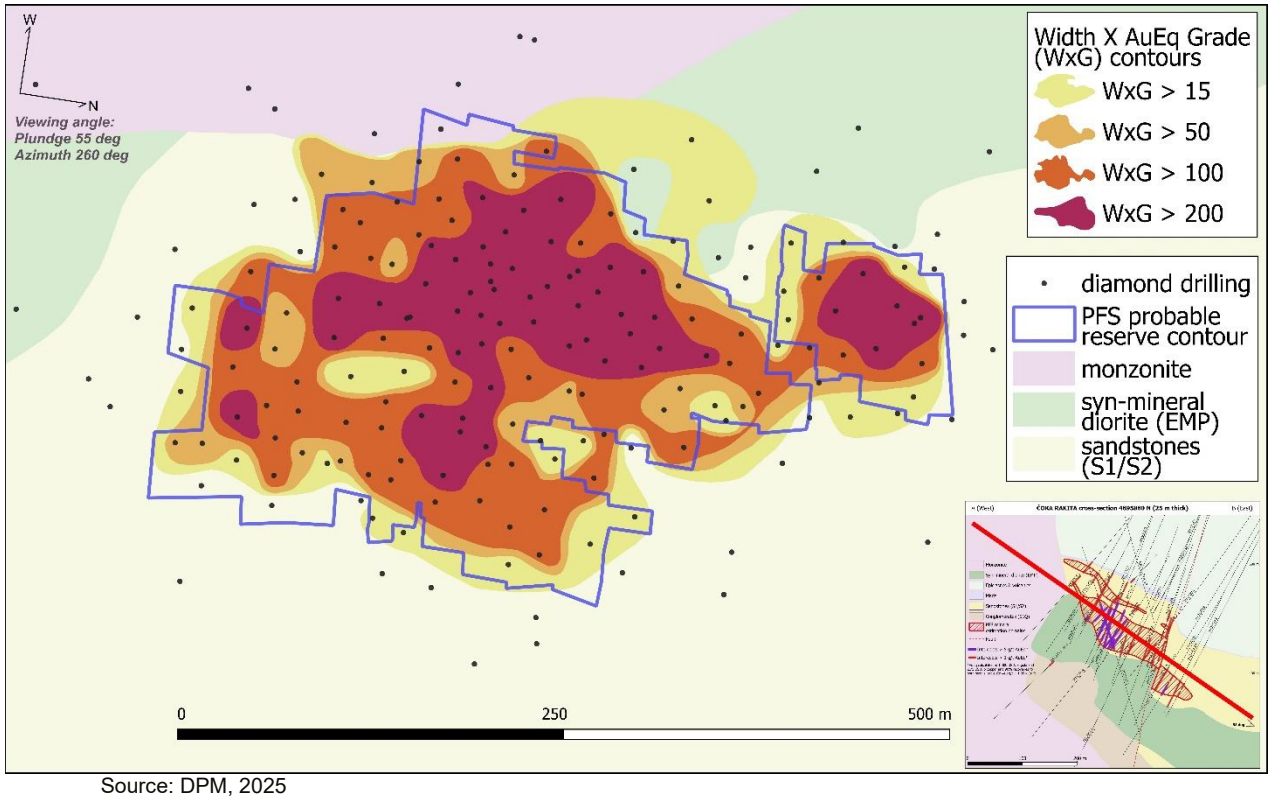
10.9 Hydrogeological Drillholes

Hydrogeological drill program was conducted in order to investigate ground water levels, quality and water regime within deposit area. Within the program there were eight (8) drillholes completed totalling 5,127 m. Another drill program will be prepared in order to test the area in and around infrastructural objects, as well as in the wider area of the deposit itself.

10.10 Drilling Results

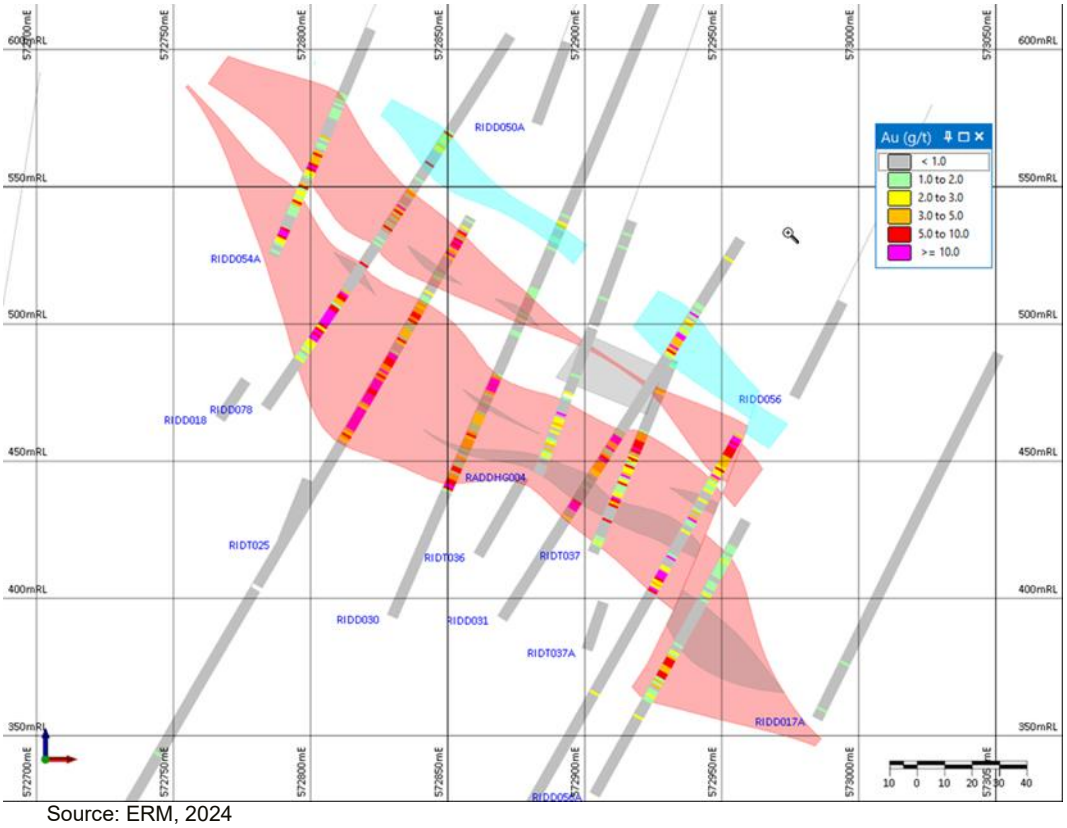
Drilling results received as of 17 January 2025, defined a wide zone of skarn-hosted gold mineralisation over a footprint of 650 m long, up to 350 m wide, and with variable thickness from less than 20 m in the margins to more than 100 m in the core of the mineralised zone. A tilted slice through the deposit is shown in Figure 10.5, 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 10.6 and 10.7.

Figure 10.5 – Tilted Slice along High-Grade Skarn Mineralisation Displaying Drilling Intercepts and Ongoing Infill Drilling at Čoka Rakita



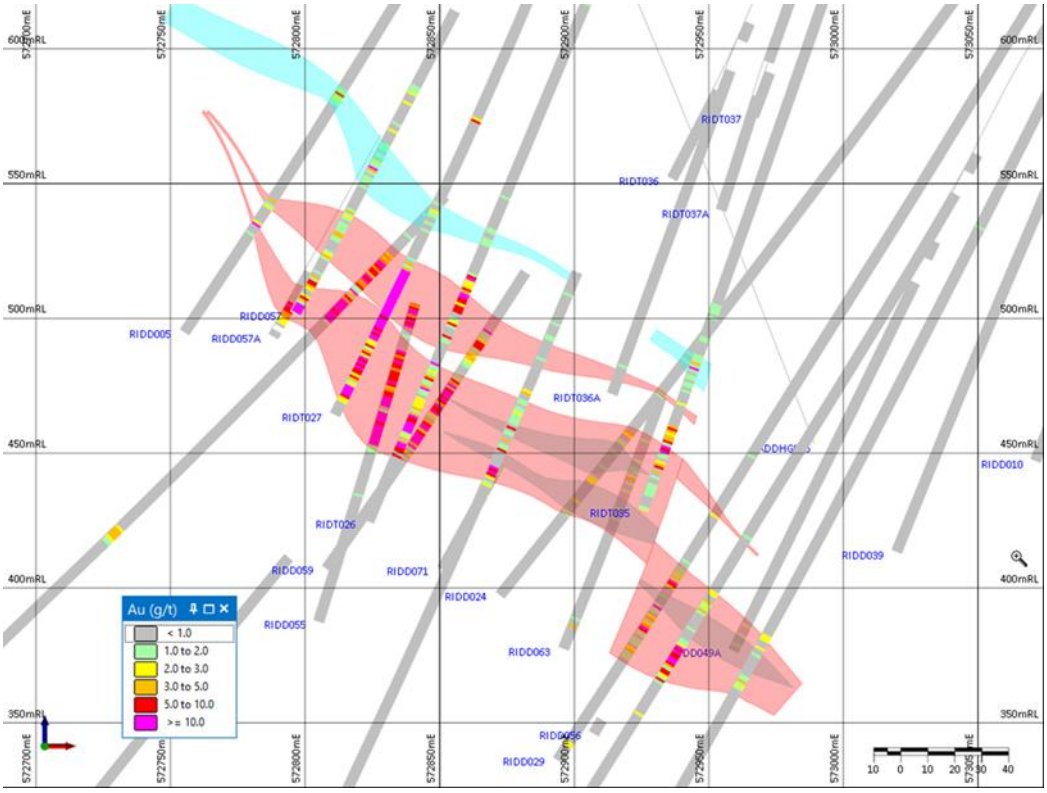
Source: DPM, 2025

Figure 10.6 – Cross Section 4895830 ME showing Drilling, Interpreted Mineralisation (Cyan, Red) and Internal Waste (Grey) at Čoka Rakita



Source: ERM, 2024

Figure 10.7 – 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, Malcolm Titley, 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 does not have any undue relationships with DPM, and that the standards used as part of the QA/QC routine adequately reflect the characteristics of the mineralisation.

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

11.2 Sampling Techniques

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

11.2.1 Dry Bulk Density Measurements

Bulk density measurements were restricted to diamond core only. Half-core samples of 20–30 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 5–10 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 are ISO-certified and fully independent of DPM.

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

Name and Location	Dates (Primary Assaying)	ISO Certification	Testwork Performed
SGS Chelopech, Bulgaria	2008 to 2010	ISO9001:2015	Gold, silver, sulphur and base metal analysis of trench, channel, RC and diamond core samples.
Genalysis/Intertek, Perth, Australia	2007 to 2008	ISO17025	Gold, silver, sulphur and base metal analysis of trench, channel, RC and diamond core samples.

Name and Location	Dates (Primary Assaying)	ISO Certification	Testwork Performed
SGS Bor, Serbia	2010 to 2025	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 2025	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 2025	ISO9001:2008 and ISO/IEC 17025:2005	Crushing and pulverising of soil, trench, channel, RC, diamond core samples and umpire QC. and screening for screen fire assay
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.
SGS Ankara, Turkey	2024	ISO9001:2015 ISO14001:2015 ISO45001:2018 and ISO/IEC 17025:2017	Gold, Silver, Sulphur and multi element analysis for some diamond core samples.

Source: ERM, 2025

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 P_{90} 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. Sulphur samples are analysed by combustion with an infrared finish.

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 (including silver) using after 4-acid digestion. In case of overrange values for Ag, Cu, Mo, Zn and Pb, re-analysis is done by atomic absorption (AAS). In 2021, the existing ICP-MS machine was upgraded with a newer version.

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 (2) per batch of 40 original samples.
- A minimum of 10% of submitted samples are subject to repeat analysis.

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

Soil samples were assayed by ALS Chemex Perth, using methods Au-TL43 (gold by aqua regia digestion with inductively coupled plasma-mass spectrometry – ICP-MS) and ME-MS41 (combined ICP-MS and inductively coupled plasma-atomic emission spectrometry (ICP-AES) dependent on concentration) for multi-elements. More recently, the same analytical methods have been undertaken at ALS Rosia Montana. Elements assayed for are silver, aluminium, arsenic, boron,

barium, beryllium, bismuth, calcium, cadmium, cerium, cobalt, chromium, caesium, copper, iron, gallium, germanium, hafnium, mercury, indium, potassium, lanthanum, lithium, magnesium, manganese, molybdenum, sodium, niobium, nickel, phosphorous, lead, rubidium, rhenium, sulphur, antimony, scandium, selenium, tin, strontium, tantalum, terbium, thorium, titanium, thallium, uranium, vanadium, tungsten, yttrium, zinc and zirconium.

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 the 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 P_{85} 4 mm, where the P_{95} exceeds 4.5 mm.

- Potential Improvements to the existing protocols:

  For SFA, using a P_{85} 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™ visible to near-infrared and 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 Prior to August 27, 2024 – QA/QC Report:

11.5.1.1 Assay QA/QC Database Checks

DPM has performed routine checks on every laboratory submission upon import to the drillhole database, using acQuire Quality Assurance/Quality Control (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.

11.5.1.2 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, OREAS) 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.

11.5.1.3 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.

11.5.1.4 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.

11.5.1.5 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 unmineralized, 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.

11.5.2 AUGUST 27, 2024 TO JANUARY 17, 2025 – QP QA/QC REPORT

DPM supplied the QP with an export of the data from the acQuire database in January 2025. 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 QA/QC reported previously (September 30, 2023 to August 30, 2024) 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 minor biases are noted both positive and negative bias but all within the acceptable limits of the individual CRMs). Precision for diamond drill samples were acceptable. Sampling procedures are appropriate and adequate security exists to minimize the risk of contamination or inappropriate mixing of samples.

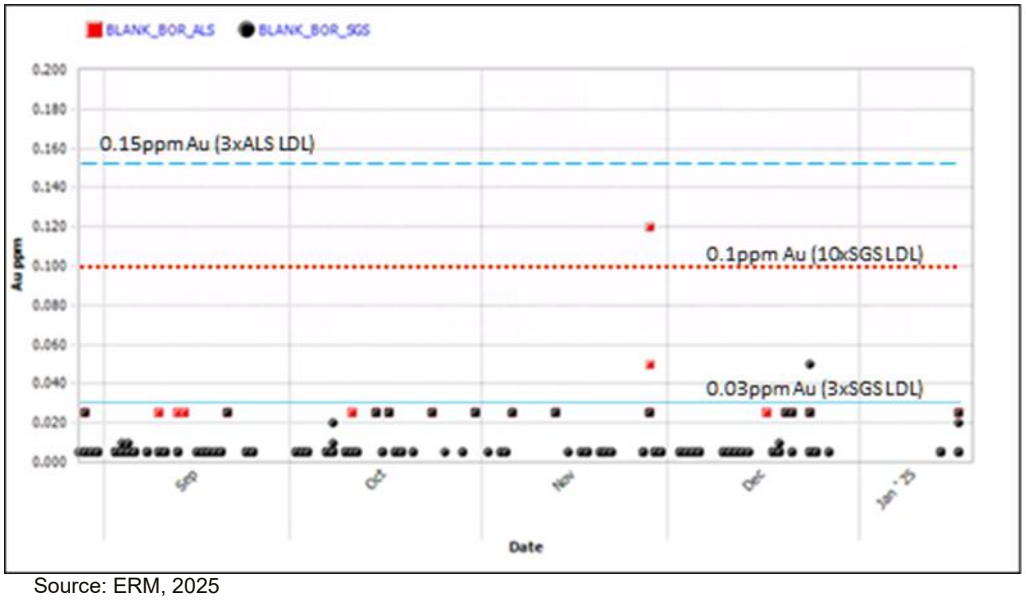
The quality control data under review for the period August 27, 2024, to January 17, 2025, for all elements also did not show any fatal flaws. There are CRMs that failed. However, these seem to be due to sample swaps or incorrect CRM names. Where CRM results display a bias, these are not material and within the acceptable limits of the respective CRMs.

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).

11.5.2.1 Contamination

In this reporting period the data for the blank, BLANK_BOR, did not show any fatal flaws. One thousand one hundred (1,100) BLANK_BOR samples were tested. One thousand (1,000) were analysed at SGS_Bor, one hundred (100) at ALS. One (1) sample from SGS Bor fell outside the expected limits, reporting 0.05 ppm Au and is not considered material.

Figure 11.1 – Blank Performance for Au for in Reporting Period (SGS BOR (N=1,000) and ALS BOR (N=100)). LDL for SGS = 0.01 ppm Au and for ALS = 0.05 ppm Au. Not Shown on Plot is 10XLDL (ALS) = 0.5 ppm Au



Source: ERM, 2025

11.5.2.2 Accuracy

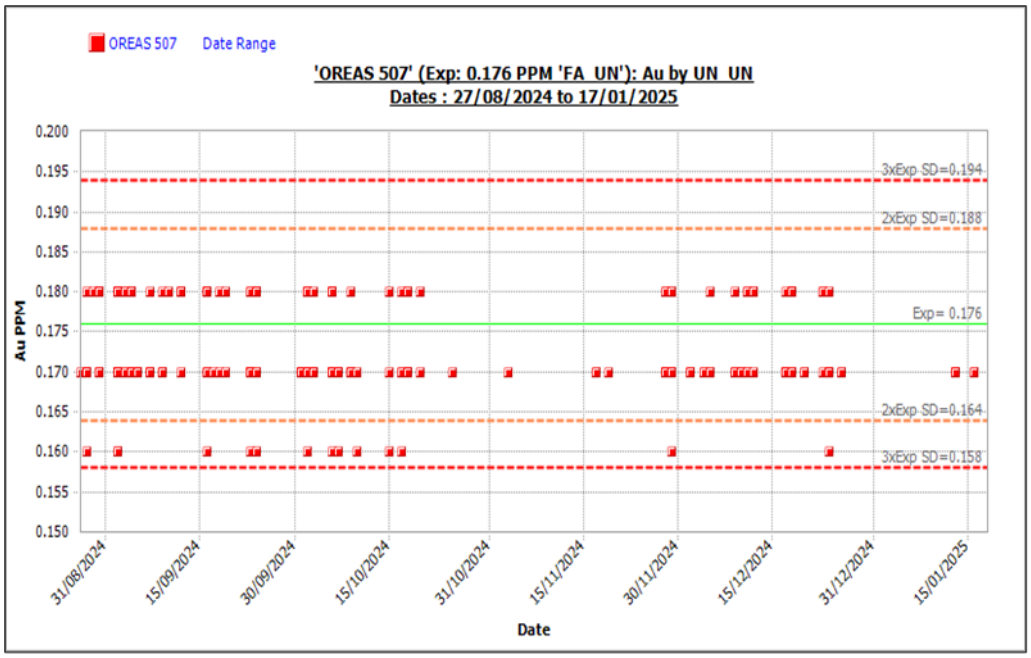
The CRM results were investigated per individual CRM for all laboratories combined (Table 11.2), as well as per CRM per laboratory. There were some biases noted but are not indicative of any systemic biases. The results do not highlight any fatal flaws, and samples that failed may be attributed to sample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

performance per CRM, laboratories combined (Figure 11.2 to Figure 11.7). A number of OREAS 507 samples reported between the -2SD and -3SD limits, but still within 10% of the expected mean (Figure 11.2), and the results of these batches, when taking into account the performance of the other CRMS and QC samples, considered sufficiently accurate for inclusion into the MRE database.

Table 11.2 – QA/QC Results per CRM

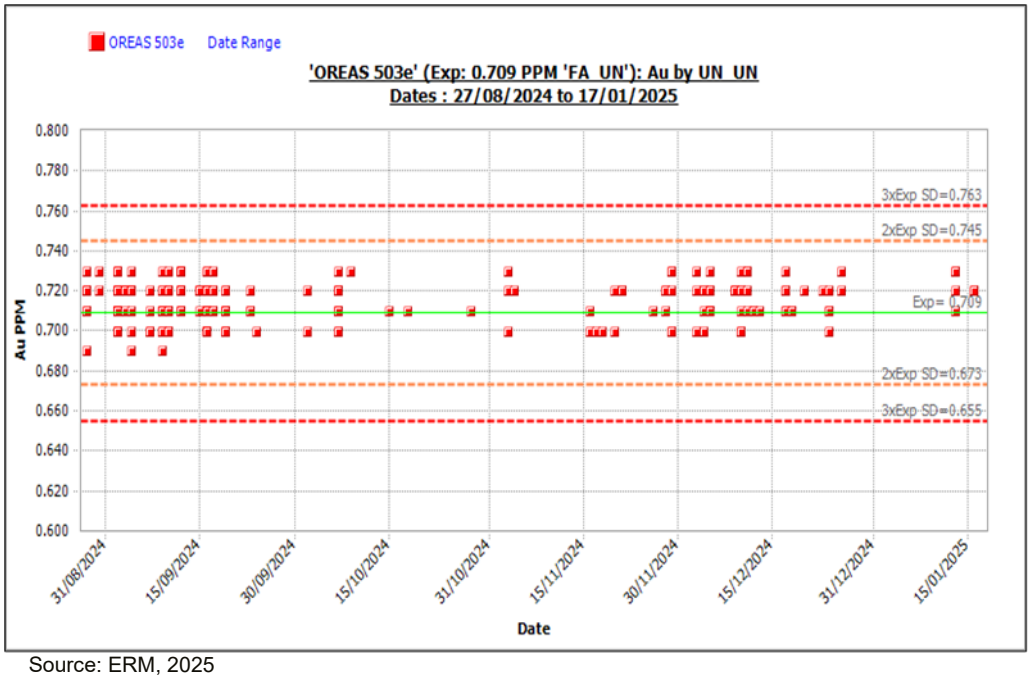
Std Code	Expected		No. of Samples	Calculated Values	SD	CV	Mean Bias
	Value	Std Dev		Mean Au			
BLANK_BOR_ALS	0.05	0.05	100	0.026	0.01	0.374	-47.60%
BLANK_BOR_SGS	0.01	0.01	1,000	0.006	0.005	0.7627	-38.85%
OREAS 240b	5.65	0.143	68	5.661	0.026	0.0046	0.19%
OREAS 501d	0.232	0.011	29	0.228	0.005	0.0216	-1.75%
OREAS 503e	0.709	0.018	210	0.714	0.01	0.0138	0.74%
OREAS 504d	1.46	0.035	167	1.477	0.009	0.006	1.14%
OREAS 507	0.176	0.006	271	0.172	0.005	0.0305	-2.15%
OREAS 508	0.47	0.016	300	0.473	0.007	0.0146	0.69%
OREAS 62h	10.54	0.291	29	10.541	0.063	0.006	0.01%
OREAS 993	56.04	0.296	3	55.767	0.321	0.0058	-0.49%

Figure 11.2 – OREAS 507 Performance for Au, All Laboratories Combined (N=271)



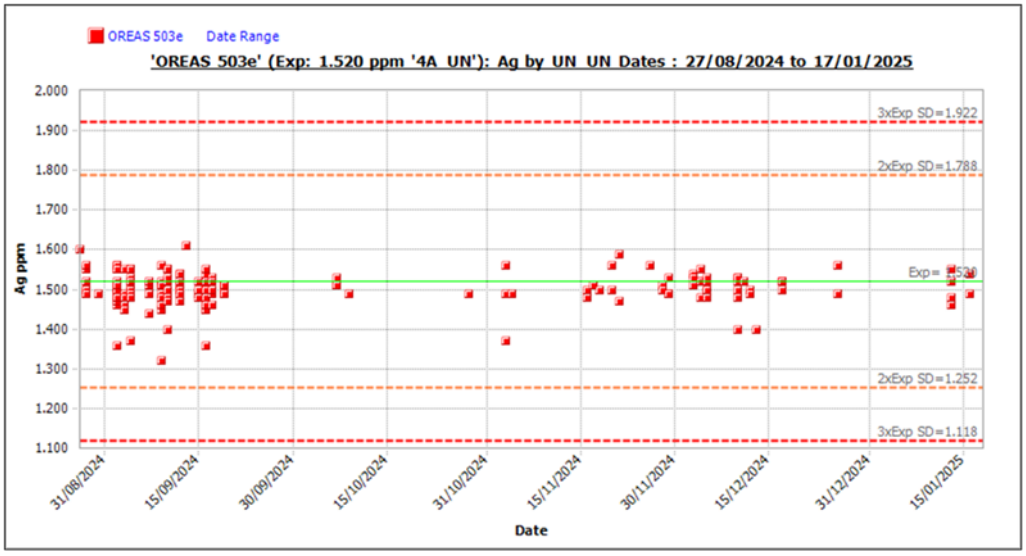
Source: ERM, 2025

Figure 11.3 – OREAS 503E Performance for Au, all Laboratories Combined (N=210)



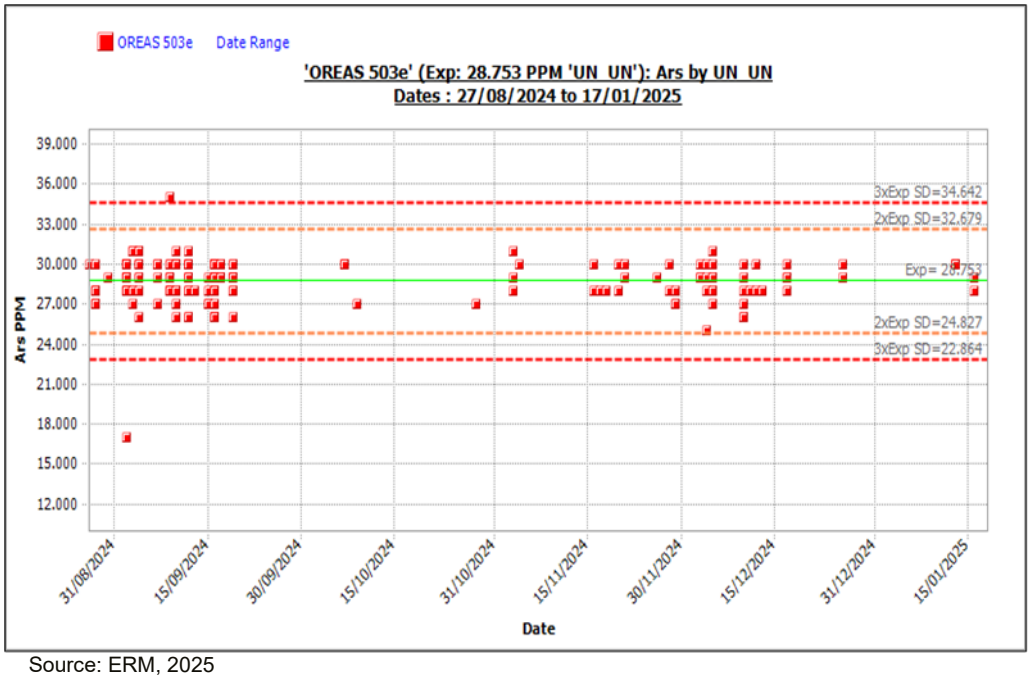
Source: ERM, 2025

Figure 11.4 – OREAS 503E Performance for Ag, all Laboratories Combined (N=178)



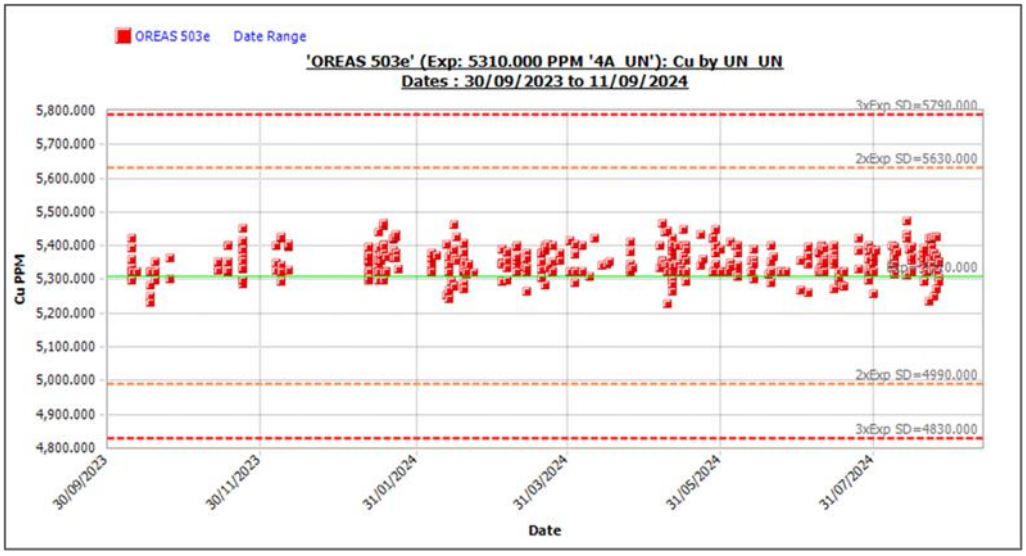
Source: ERM, 2025

Figure 11.5 – OREAS 503E Performance for As, All Laboratories Combined (N=175)



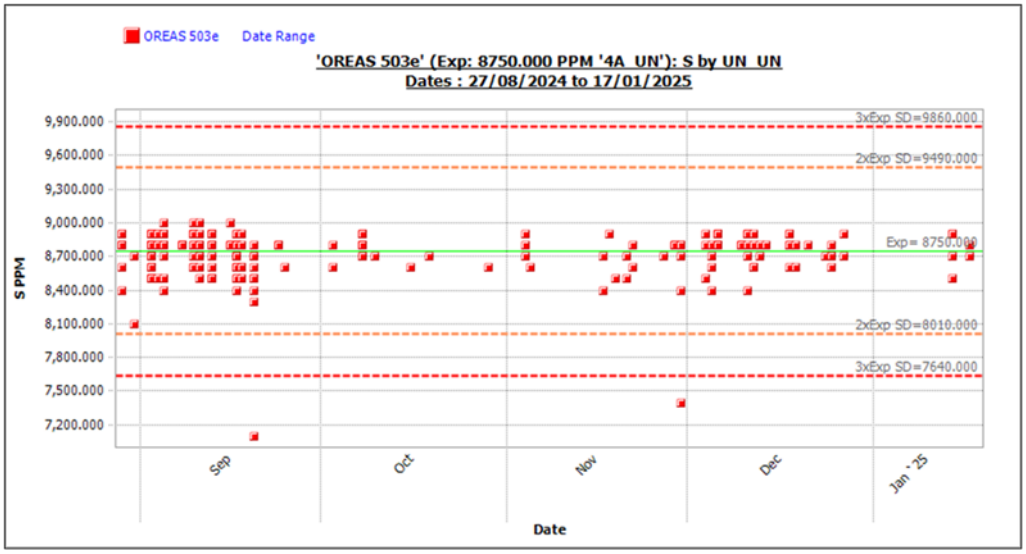
Source: ERM, 2025

Figure 11.6 – OREAS 503E Performance for Cu, All Laboratories Combined (N=178)



Source: ERM, 2025

Figure 11.7 – OREAS 503E Performance for S, all Laboratories Combined (N=210)



Source: ERM, 2025

11.5.2.3 Precision

Preparation duplicates as well as external check (SFAs as Umpire) results were compared for diamond drill samples. The duplicate results were compared for primary samples submitted to SGS Bor and ALS Bor. The precision data for gold for sample type half diamond core fall within acceptable practice limits (Table 11.3). While there is a positive bias for SFA versus FA, particularly evident in samples reporting > 5 g/t Au, this is an expected result since SFA produces more reliable results for coarse gold deposits because of:

Improved representivity (separately analysing the coarse fraction meaning larger gold particles can be accounted for).

Reduced nugget effect which reduces bias and improves accuracy especially in samples with high variability in gold particle size when compared to fire assay.

Better repeatability because results are generally more consistent when dealing with coarse gold.

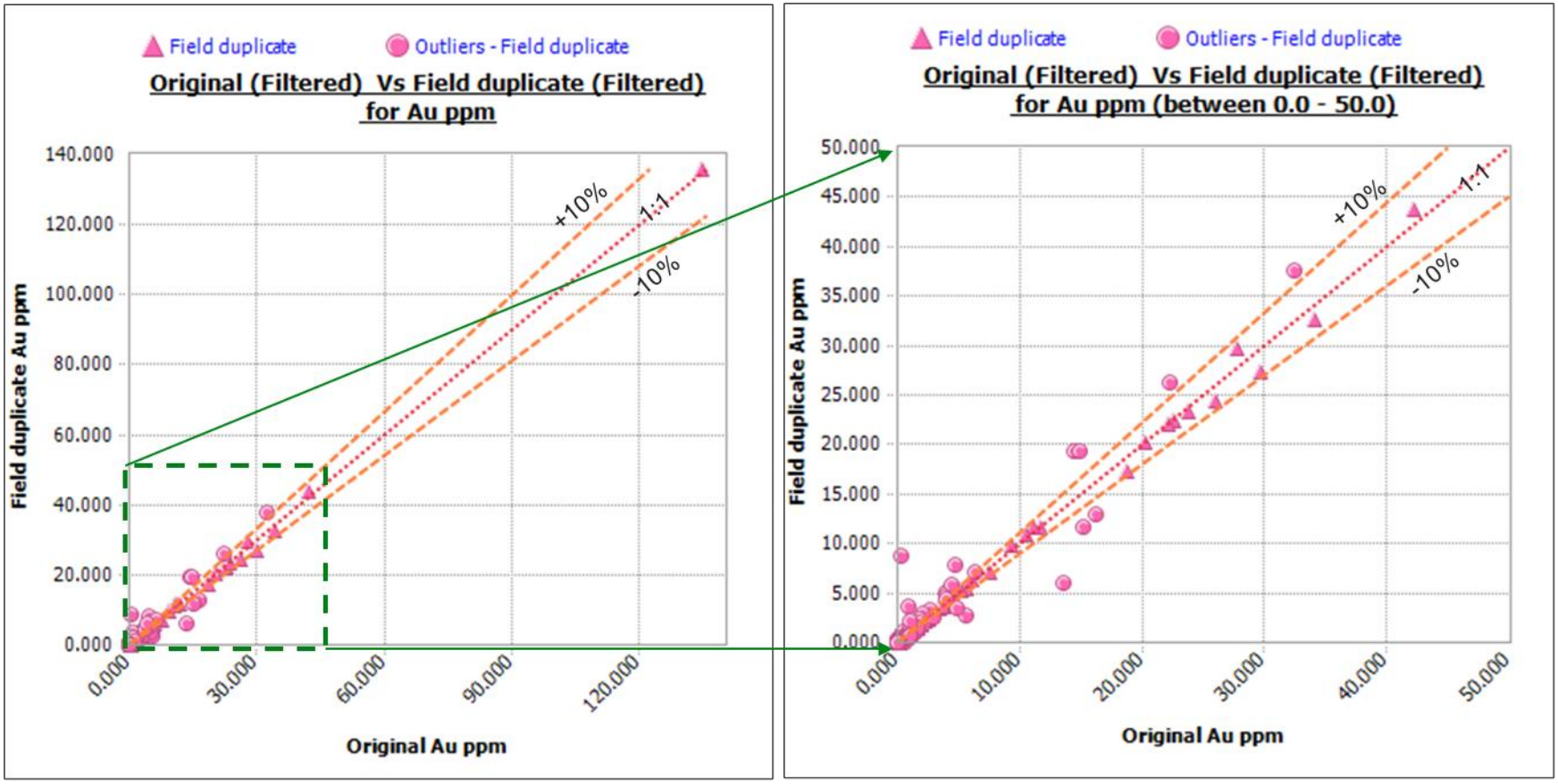
In conclusion, when coarse gold is expected, SFA is a more reliable method for obtaining more accurate and representative gold grades.

Table 11.3 – 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	1,082	395	18	2.15	2.21	3%
SFA UMPIRE	DDH ½	10	20	2,675	1,549	27	5.58	7.48	34%

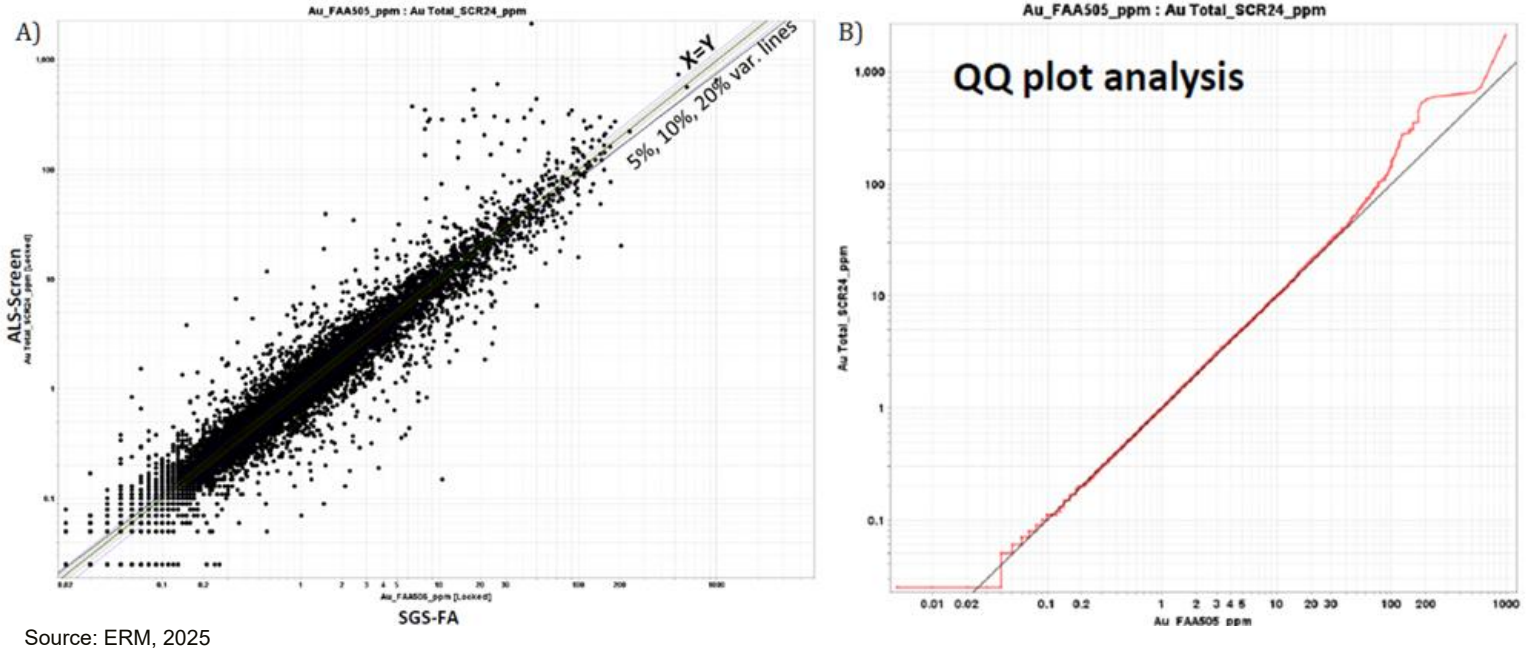
Source: ERM 2025

Figure 11.8 – Field Duplicate Paired Data (Showing all Data (Left) and for <50 ppm Au (Right))



Source: ERM, 2025

Figure 11.9 – SFA vs. FA Paired Data
Left: X-Y Scatter Plot and Right: Q-Q Plot



Source: ERM, 2025

11.6 QP's Opinion

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

- Approximately 91% of samples in mineralisation (>4,800 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. However, Serbia authorities mandate that the remaining half core is preserved within the core library for future reference and verification, so this practice has not been initiated by the company.
- 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 five (5) CRMs and five (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 mineralized 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. Additional CRM's at similar or higher Au grades would be beneficial.

- Continual vigilance is required considering the extremely high gold grade values that have been encountered while drilling.
- The QP finds that the QA/QC procedures are adequate and that there are no 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.

12.1 2023 – ERM Data Verification

The QP, Maria O'Connor, MAIG, visited the site on October 3 and 4, 2023 for the purposes of reviewing drilling activity, logging facilities, and the independent, on-site laboratory.

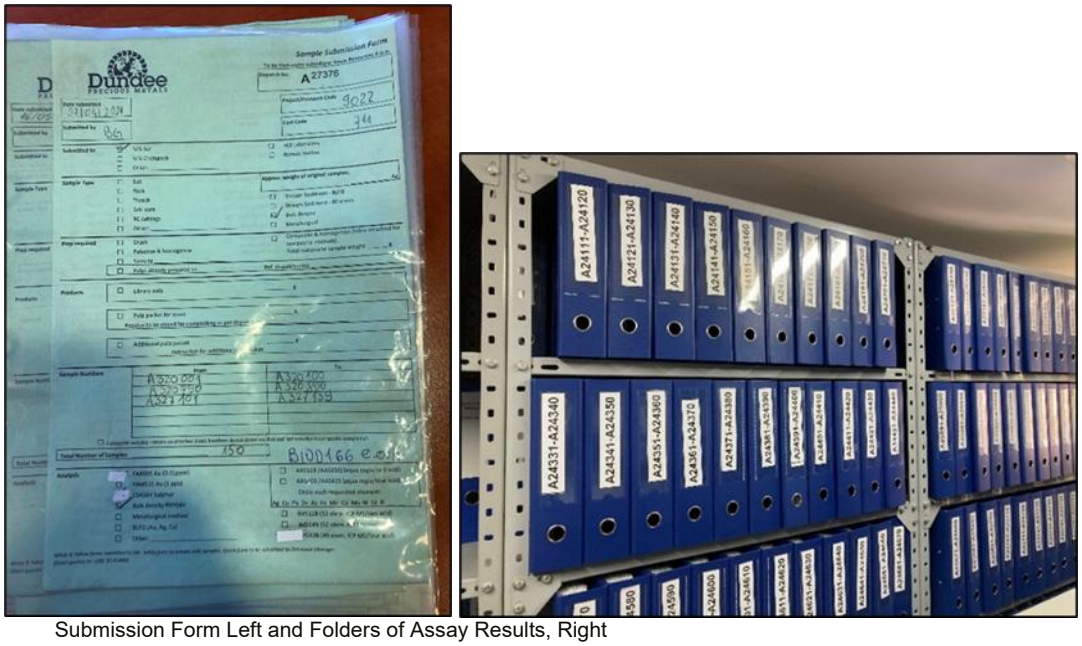
Data underwent further validation by the QP, Maria O'Connor, MAIG, through a series of spot checks for factual errors and further observations from the QP's verification program include:

- 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 previous QP and equipment was found to be clean and well managed by an experienced laboratory manager George Daher. No issues were identified.
- The previous 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).

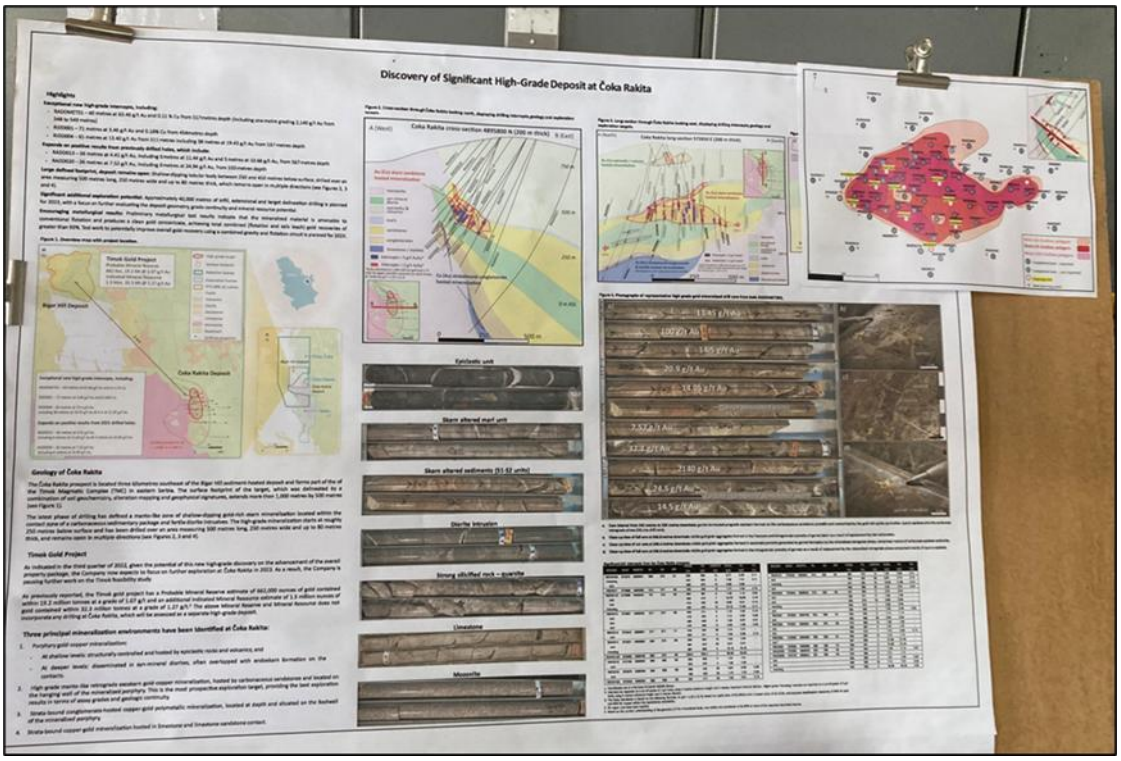
- Drilling activities were viewed on site to assess how core was recovered and treated at the drill site. Drillhole deviation is carefully monitored on an ongoing basis.
- Drilled collars were visited and collar coordinates were identified in the field (Figure 12.3) and picked up using a handheld GPS 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 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



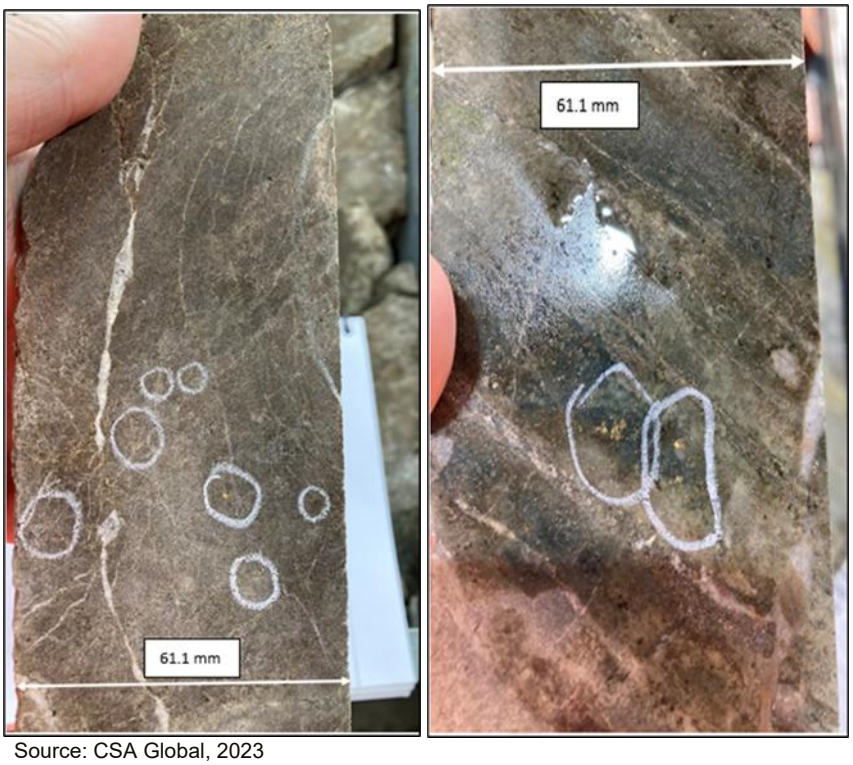
Source: CSA Global, 2023

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 previous QP is summarised below:

- Coordinates are captured at various stages using different methodologies which are ranked accordingly and those with the highest (best) ranking are captured in the “Best” field in the database. These coordinates were used in the MRE. Highest to lowest ranked methods are as follows – differential GPS->Total station->Digitised->Transformed Historic->Planned.
- Collar information was received via email from the Survey Department in pre-specified templates and imported into the acQuire database.
- There were no issues identified with the data in the collar table.
- No issues were identified with downhole surveys.
- There are unlogged intervals for geology and alteration. Industry best practice would usually have relational validation in place that would disallow this from happening, so a database audit is recommended to ensure best practice in database procedures. Since the specified intervals 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.

12.2 2025 – ERM Data Verification

A site visit was completed during the period May 13 to 15th, 2025 by the QP, Malcolm Titley. During the site visit, the following was completed:

- Review of recent drill core, geological logging and assay sampling procedures.
- Review of the SFA assay results and analytical QAQC for the drilling subsequent to the PFS MRE.
- Review of the geology interpretation and in particular updates to the interpretation subsequent to the PFS MRE.
- A general overview of the geological history and development of the current understanding of the mineralisation controls.

12.3 QP's Opinion

Based on the checks completed, the QP, Malcolm Titley, 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 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.

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

Table 13.1 – Wardell Armstrong Test Program Summary

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

Source: Wardell Armstrong, 2021

The 2023 Base Metallurgical Laboratories (BaseMet Labs) test program was performed on three (3) composite samples, representing low, medium, and high gold. 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 essentially simulates the selected flowsheet and employs optimised test conditions.

Table 13.2 summarises the gold recoveries achieved during these LCTs and compares it with a scenario where the tails from this LCT was subsequently leached using cyanide and, finally, against

a control test which simply comprises direct cyanidation of the original sample. The result shows that cyanidation of the flotation tailings consistently produced the best gold recoveries. However, DPM did not consider the use of cyanide appropriate for this Project, hence the flowsheet selected uses gravity concentration followed by flotation only.

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

Sample ID	Test Procedure	Head Au Grade (g/t)	Au Recovery (%)	Tails Au Grade (g/t)
MetCRA23-01	Gravity -> Flotation 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
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

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

Table 13.3 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 13.3 – 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

EGRG testing of the variability samples showed that the coarse gold particles are reasonably coarse with the majority within a range of P_{80} from 120 to 200 µm. Modelling of the results 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.

A set of tests which mimics the gravity-flotation flowsheet was performed on the variability samples. The test procedure comprised of grinding followed by two (2) stages of gravity concentration and two (2) stages of open-ended flotation. The results from these tests, weighted by how much each sample represents of the block model, aligned well with previous testwork outcomes. The gravity recovery, at 37%, is marginally lower than what was used in the PEA as is the overall recovery of 87.4%. For the PFS phase the design criteria was adjusted to align with these variability test results.

Table 13.4 summarises the results of the variability testing mimicking the proposed flowsheet.

Table 13.4 – BaseMet Labs Test Program (2024) Gravity-Flotation Results

Description	Units	Value
Gravity Concentrate Mass Pull	%	0.15
Gravity Concentrate Au Grade	g/t	1,188
Gravity Au Recovery	%	37.4
Flotation Concentrate Mass Pull	%	9.1
Flotation Concentrate Grade	g/t	44.4
Flotation Au Recovery	%	46.7
Overall Au Recovery	%	87.4

A hydrofloat test was performed to determine whether most of the gold can be floated at a coarser primary grind (P_{80} = 350 µm). The test was not successful as the gold distribution to the hydrofloat tails was unacceptably high at 38.5%.

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 the QP's, Niel Morrison, knowledge, there is no historical testwork for this deposit other than the testwork described in this document.

13.3 Wardell Armstrong Testwork Campaign (2021)

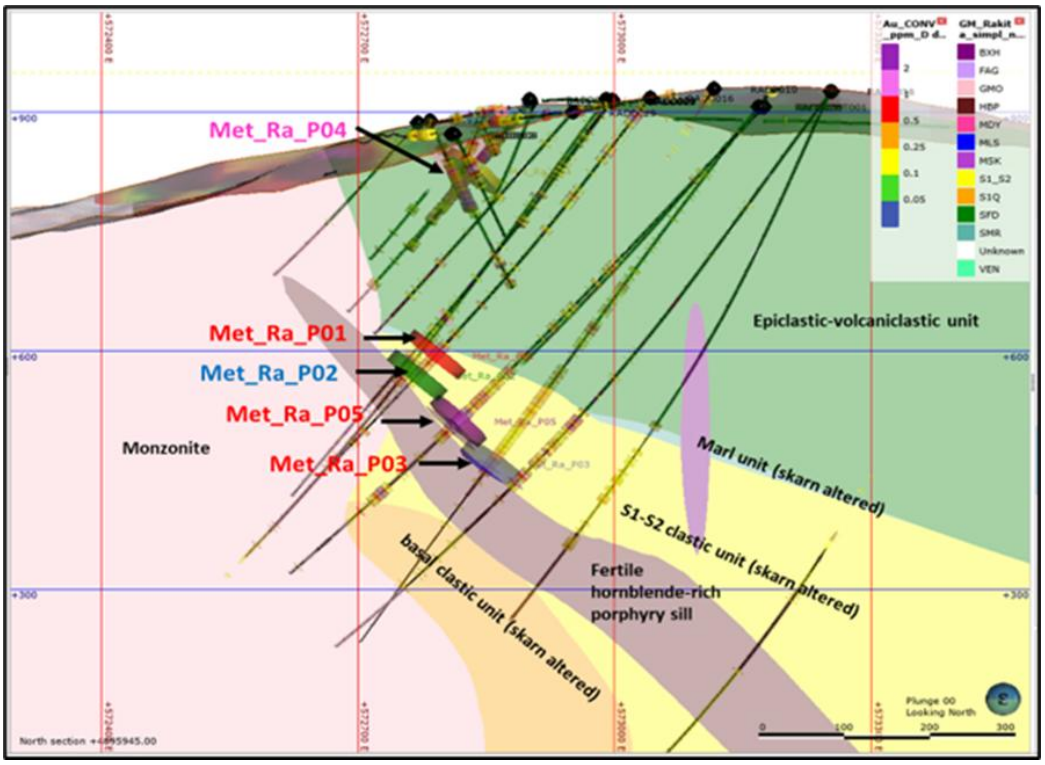
13.3.1 Sample Selection

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).
- 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 13.5 summarises the head assays determined for these samples.

Table 13.5 – 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
C(tot)	%	-	-	-	-	1.1
Cu	ppm	1,030	3,560	857	214	854
Pb	ppm	53.5	142.5	77.4	229	34
Zn	ppm	151	269	81	166	96
Fe	%	8.14	3.18	3.79	3.34	9.71

Source: Wardell Armstrong, 2021

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

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

13.3.3 MINERALOGY

Highlights from the mineralogical examination are as follows:

- Garnet and quartz are the most abundant minerals in these samples.
- 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. Table 13.6 summarises the results achieved.

Table 13.6 – Two-Stage Gravity Test Results

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

Source: Wardell Armstrong, 2021

The results show that approximately half of the gold present in these samples can be recovered using gravity concentration. Given that the grades of the samples are low compared to what the eventual plant will process it is evident that a gravity concentration step would provide several advantages and should be included in the plant flowsheet. Some of these advantages include:

- More consistent feed grades to the downstream beneficiation processes.
- Improved recovery of gold due to reduced risk of coarse gold losses in the downstream circuits.
- The gravity concentrate will be at a significantly higher grade than the flotation concentrate and will earn a higher price if sold separately.
- Including gravity concentration will reduce the total volume of concentrate to be trucked off-site.

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. Table 13.7 summarises the results obtained. 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.

Table 13.7 – Whole-Ore Cyanidation Results

Sample ID	D_{80}	Au Extraction	Reagent Consumed (kg/t)	
	(µm)	(%)	Lime	NaCN
Met_Ra_P01	105	85.5	0.30	2.10
	75	88.3	0.32	2.51
Met_Ra_P02	105	87.2	0.72	3.11
	75	88.0	0.78	3.07
Met_Ra_P03	105	88.6	0.48	2.79
	75	88.7	0.82	2.59
Met_Ra_P04	105	82.0	0.60	1.88
	75	81.7	0.72	1.54
Met_Ra_P05	105	87.2	1.04	1.61
	75	93.0	0.94	1.76

Source: Wardell Armstrong, 2021

Gold recoveries from whole-ore cyanidations ranged from 82 to 93% with the average around 87%. Cyanide consumption averaged around 2.3 kg/t which is higher than a typical gold plant and indicates that a cyanide consumption strategy would be required if cyanidation is included in the

flowsheet. The strategy could attempt to either curb consumption or to recover cyanide after leaching.

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 13.8 summarises the results of the scouting rougher flotation tests performed in 2021. Rougher flotation tests were performed after grinding the samples to 80% passing 105 µm, 75 µm, 45 µm and 30 µm respectively. The slurry samples were conditioned with $CuSO_4$ prior to adding PAX and MIBC at the start of the flotation stage. Flotation was performed over a total duration of 15 minutes (with intermediate sampling after each 3-minute interval). It is noted that the results for sample P02 are omitted for the sake of brevity because it is far below the cut-off grade.

Table 13.8 – 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 variations in recovery at all grinds. The recovery versus grade curves (see Wardell Armstrong, 2021) are very steep in some instances which may indicate that flotation was not completed within the allotted duration.

A further set of tests were performed at grinds of 75 µm and 45 µm, where a cleaner step was added after rougher flotation to assess how much the grades could be improved through cleaning. Table 13.9 summarises the results for the 75 µm set (45 µm results omitted for the sake of brevity as trends are similar).

Table 13.9 – Summary of Rougher-Cleaner Flotation Tests at a P_{80} of 75 µm

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

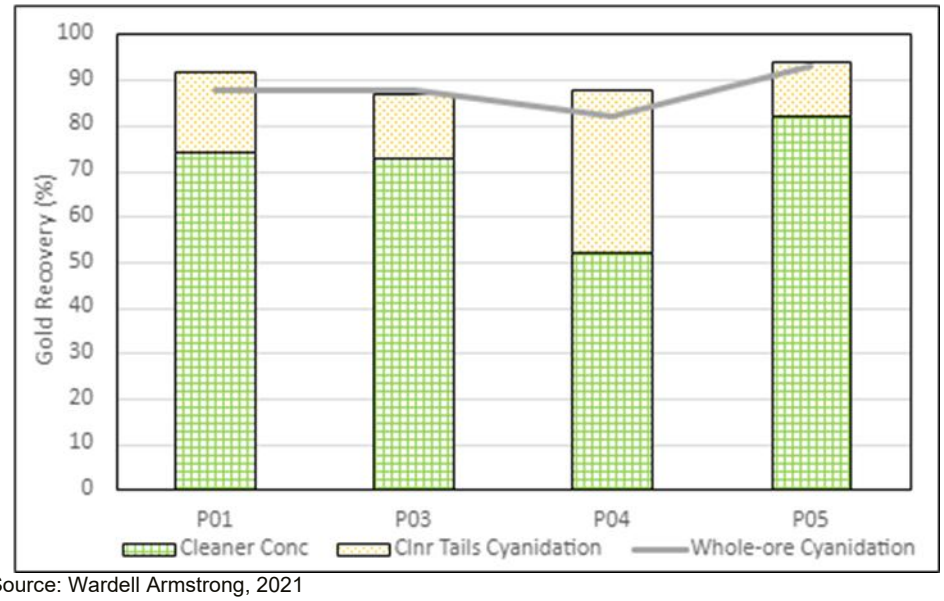
Source: Wardell Armstrong, 2021

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

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

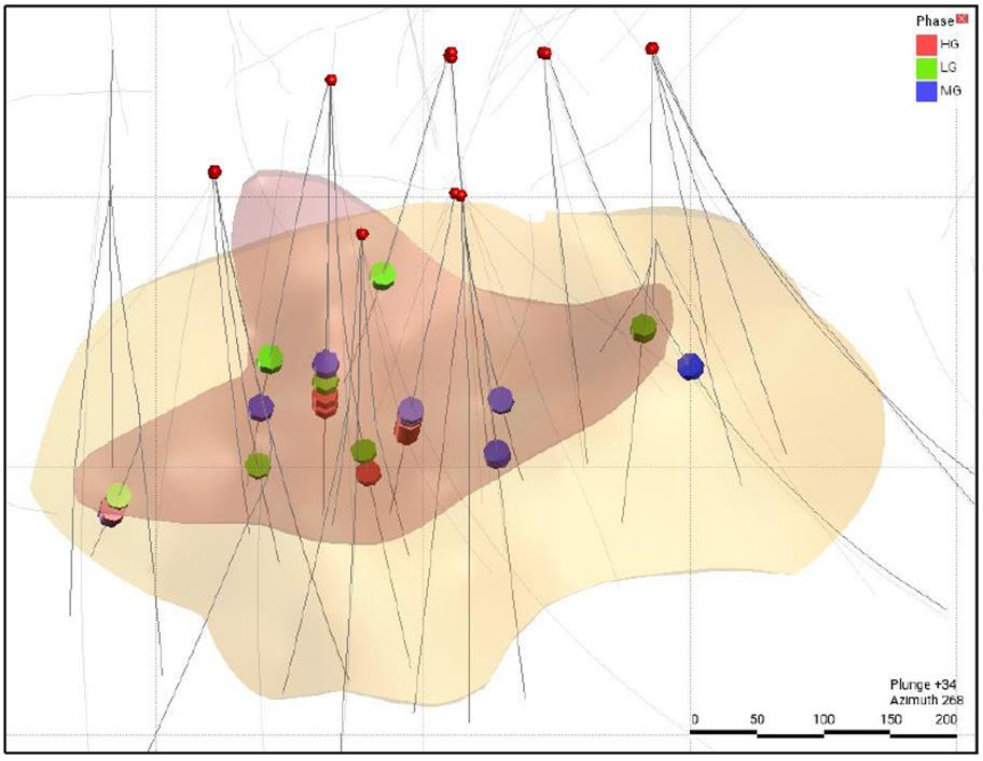
Cancha software was used to select the intervals for the three (3) composites. Quarter cores were used. Older holes were not considered as these may be oxidised and were hence deemed unsuitable. A minimum cut-off grade of 3.0 ppm Au was specified at the time of sample selection. The aim with the sample 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.

The low-grade composite sample (MetCRA23-01) consists of ten (10) intervals selected from ten (10) drillholes for a total mass of 131 kg. The depth below surface for the selected intervals range from 183 m to 587 m. The weighted average gold grade of the cores selected is 3.16 g/t.

The medium-grade composite sample (MetCRA23-02) contains nine (9) intervals selected from eight (8) drillholes for a total mass of 128 kg. The depth below surface for the selected intervals range from 216 m to 608 m. The weighted average gold grade of the cores selected is 6.36 g/t.

The high-grade composite sample (MetCRA23-03) contains nine (9) intervals selected from seven (7) drillholes for a total mass of 131 kg. The depth below surface for the selected intervals range from 428 m to 598 m. The weighted average gold grade of the cores selected is 12.52 g/t.

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 13.10 summarises the head analysis performed on the three (3) composites tested in 2023. This table shows acceptable agreement between the gold analysis through fire assay (FA) and through the screened metallics (SM) procedure. Sample MetCRA23-02 has a similar head gold grade to the LOM grade of around 5.5 g/t.

Table 13.10 – Head Assay for BaseMet Labs Test Program (2023)

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

Source: BaseMet Labs, 2023

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

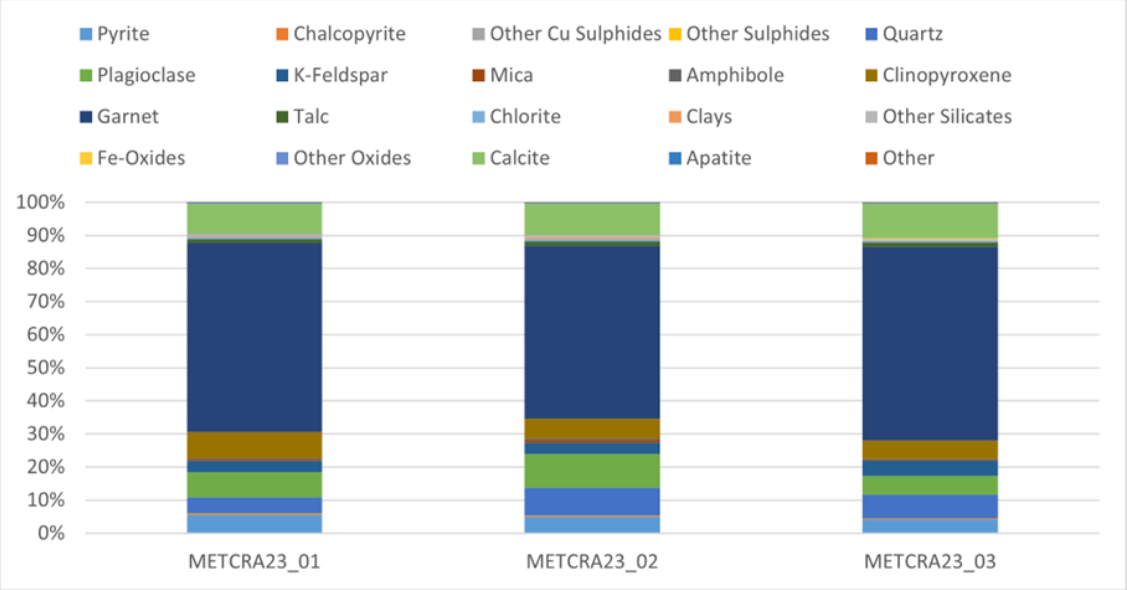
13.4.3 MINERALOGY

AGAT Geology Dept (directed by BaseMet Labs) performed XRD analysis on powdered samples of the three (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%.

Figure 13.4 shows the mineral abundance results of these three (3) samples.

Figure 13.4 – Mineral Abundance Results



Source: BaseMet Labs, 2023

13.4.4 LASER ABLATION ANALYSIS

Table 13.11 presents the results of the laser ablation analysis of the pyrite showing the distribution of metals.

Table 13.11 – 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 13.12 summarises the comminution results produced for the PEA phase of the Project. Three (3) metallurgical composite samples were tested individually. There was no testing of comminution parameters during the 2021 program performed by Wardell Armstrong.

Table 13.12 – 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. Table 13.13 summarises the results for the combined concentrate collected at all grinds.

During sample preparation, a 20-kg portion of each master composite, Met-01 to Met-03 was split and stage crushed to 100% passing 1.7 mm to produce a target K_{80} of ~1,200 µm. This material was utilised for EGRG testing. The EGRG test is conducted by passing the entire sample of crushed material through a Knelson MD-3 concentrator at a force of 60 Gs. The concentrate is retained and sized for assay; the tailings are sub-sampled for sizing.

The tailings are ground in a laboratory rod mill and repassed (Pass 2) at a grind target K_{80} of 250 µm, the concentrate and tailings are sampled as per the initial pass before regrinding the tailings and repassing (Pass 3). The Pass 3 tailings were reground prior to repass (Pass 4). The Grind targets for Passes 3 and 4 were 105 and 54 µm, respectively. The final tailings are sampled, sized and assayed by size. All assaying included Au by fire assay with concentrate fractions assayed to extinction.

Table 13.13 – EGRG Results Summary

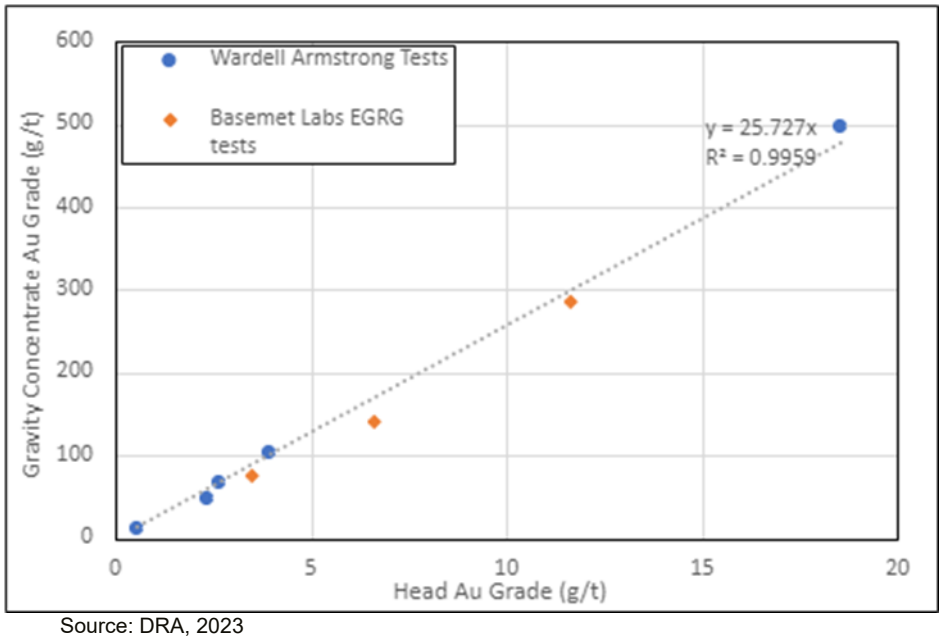
Composite (Test)	Product	Feed Size K_{80} (µm) per Stage	Mass (%)	Assay Au (g/t)	% Au Distribution
Met_CRA23_01 GRG (T-05)	Stage 1 Conc.	1281	0.56	90.0	14.4
	Stage 2 Conc.	228	0.56	127	20.3
	Stage 3 Conc.	96	0.74	70.1	14.9
	Stage 4 Conc.	54	0.65	32.2	6.0
	Tailing		94.3	1.65	44.4
	Comb Conc. 1-4		2.52	77.3	55.6
	Feed (calc.)			3.50	
Met_CRA23_02 GRG (T-05)	Stage 1 Conc.	1040	0.51	236	18.3
	Stage 2 Conc.	71	0.54	179	14.9
	Stage 3 Conc.	110	0.61	112	10.4
	Stage 4 Conc.	58	0.58	61.7	5.4
	Tailing		94.6	3.55	51.0
	Comb Conc. 1-4		2.25	143	49.0
	Feed (calc.)			6.58	

Composite (Test)	Product	Feed Size K_{80} (µm) per Stage	Mass (%)	Assay Au (g/t)	% Au Distribution
Met_CRA23_03 GRG (T-05)	Stage 1 Conc.	1241	0.53	441	19.9
	Stage 2 Conc.	258	0.56	532	25.7
	Stage 3 Conc.	106	0.65	208	11.6
	Stage 4 Conc.	57	0.76	69.2	4.5
	Tailing		94.5	4.71	38.2
	Comb Conc. 1-4		2.50	288	61.8
	Feed (calc.)			11.6	

Source: BaseMet Labs, 2023

Figure 13.5 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.

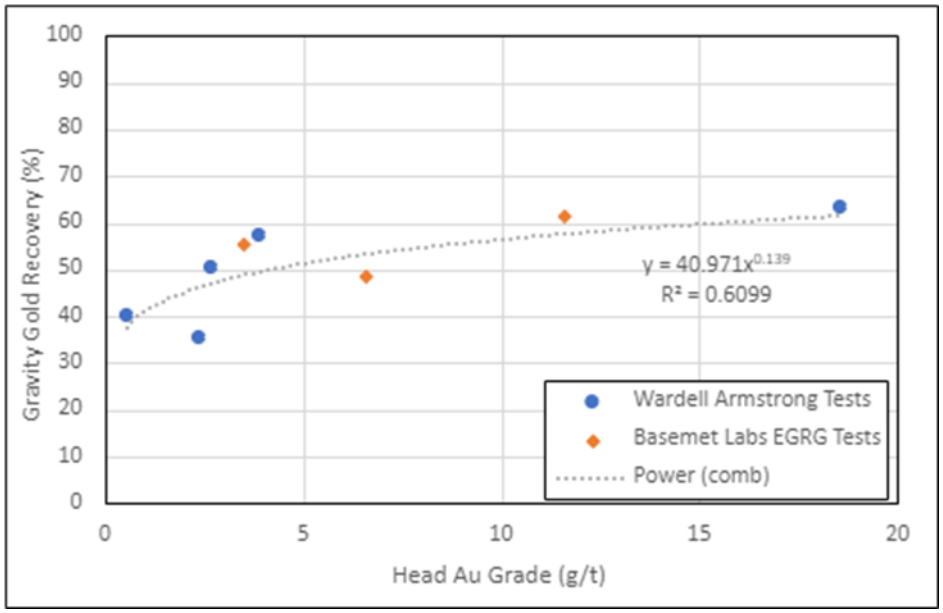
Figure 13.5 – Gravity Concentrate Gold Grade vs. Head Grade



Source: DRA, 2023

Figure 13.6 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 13.6 – Gravity Gold Recovery versus Head Grade



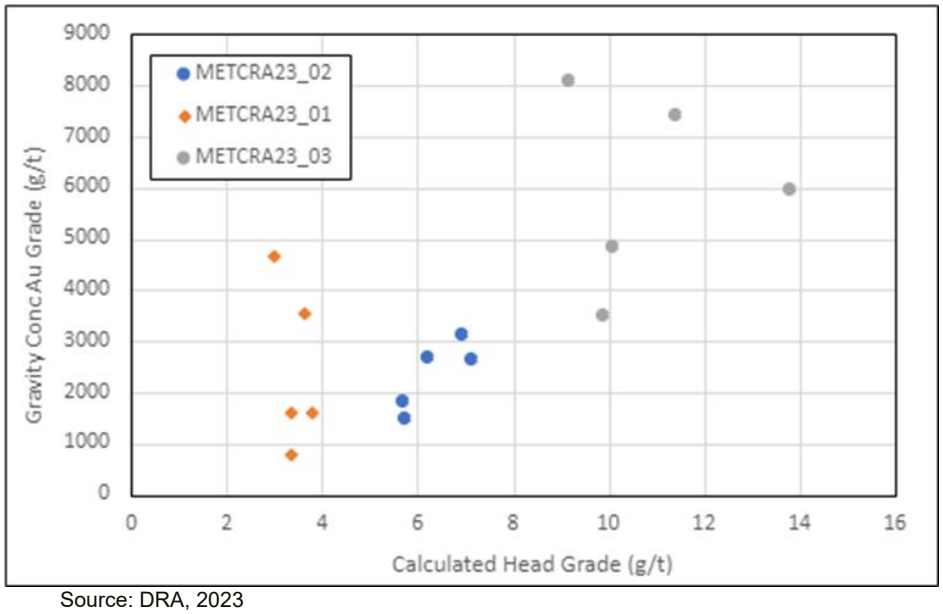
Source: DRA, 2023

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 (around 0.1 % as opposed to the approximately 2% recorded by Wardell Armstrong and BaseMet Labs' own EGRG test). It appears the settings on the shaking table and/or Knelson must have been modified to produce a higher-grade gravity concentrate. Because of this change in test procedure, it is not possible to compare these different sets of results directly.

Figure 13.7 shows the graph of gravity concentrate grade versus head grade for this set of tests (at all five (5) grinds per sample). The fit is not as smooth as it was for the data shown in Figure 13.3 (Wardell Armstrong and EGRG tests). Despite the increase in variability, it is still evident that there is a proportional correlation between the two (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. Concentrating a similar mass of high-density material into a much smaller concentrate mass amplifies the nugget effect which then manifests in amplified variances around the average trendline.

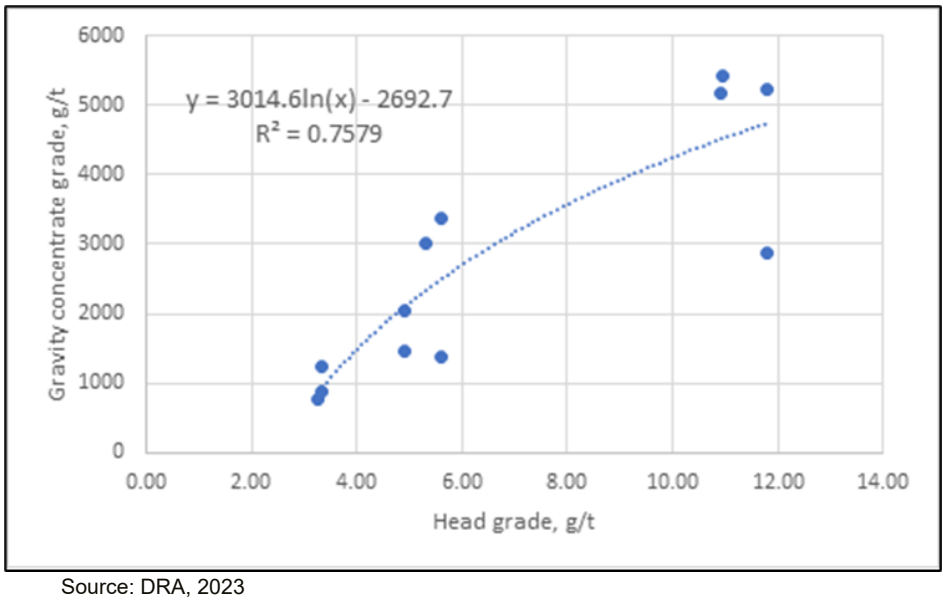
Figure 13.7 – Gravity Concentrate Gold Grade versus Sample Head Grade



Source: DRA, 2023

Figure 13.8 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.

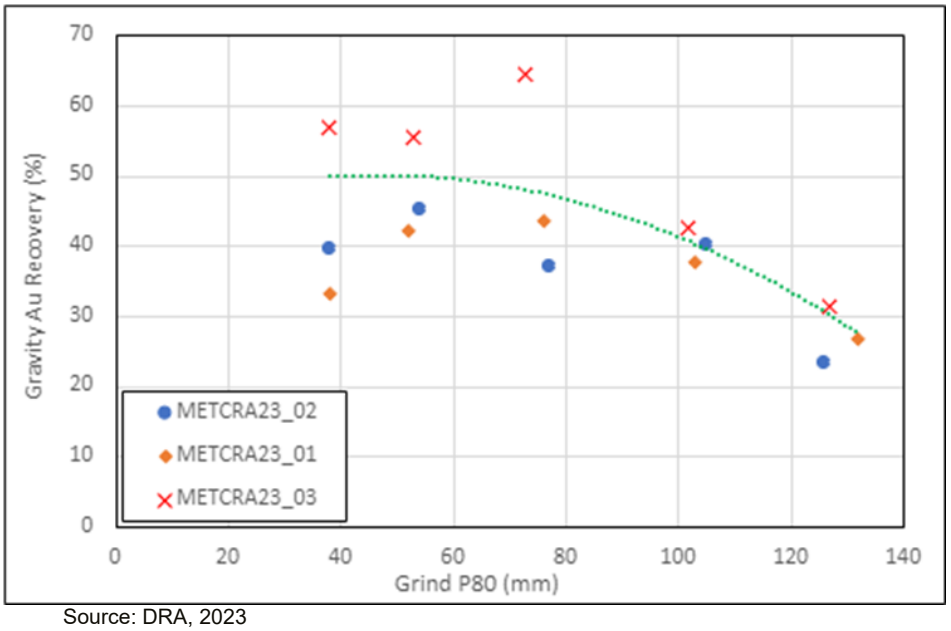
Figure 13.8 – Gravity Concentrate Grade versus Head Grade (53 µm grind)



Source: DRA, 2023

Figure 13.9 presents the gravity gold recoveries recorded at different grind sizes for this series of gravity-rougher flotation tests conducted at BaseMet Labs.

Figure 13.9 – Gravity Gold Recovery versus Grind Size



Source: DRA, 2023

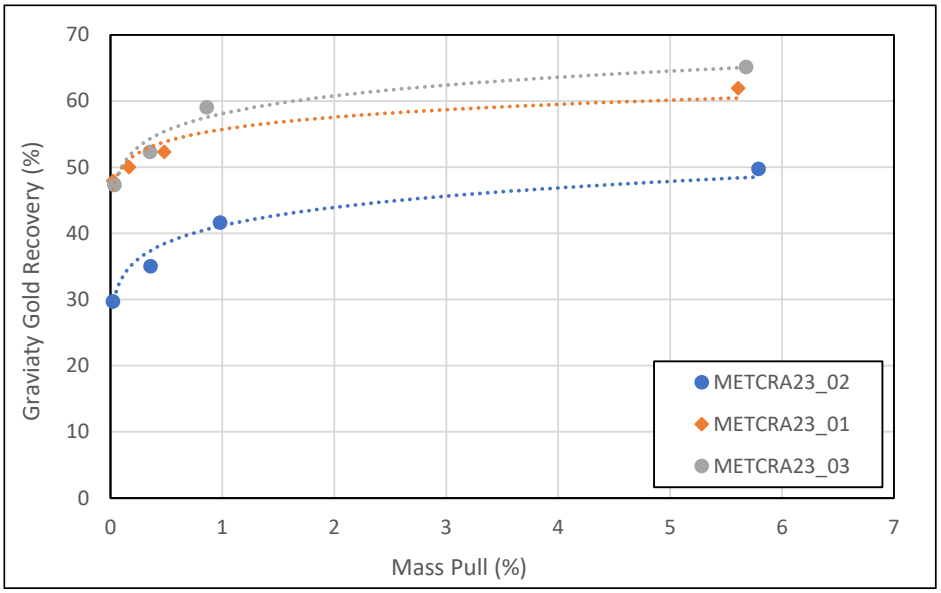
From Figure 13.9, it can be concluded that gravity gold recovery decreases as the grind coarsens above 53 µm. Below 53 µm the recovery of coarse gold appears to be unaffected by the grind size. It appears all gravity recoverable gold is fully liberated at a grind of 80% passing 53 µm.

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 13.10 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 13.10 – Gravity Gold Recovery vs. 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. Table 13.14 summarises the results. 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.

Table 13.14 – Whole-Ore Cyanidation Results

Sample ID	Au Extraction	Reagent Consumed (kg/t)	
	(%)	Lime	NaCN
Met_Ra_P01	88.4	1.2	2.0
Met_Ra_P02	92.1	1.9	1.4
Met_Ra_P03	95.4	1.7	1.2

Source: BaseMet Labs, 2023

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

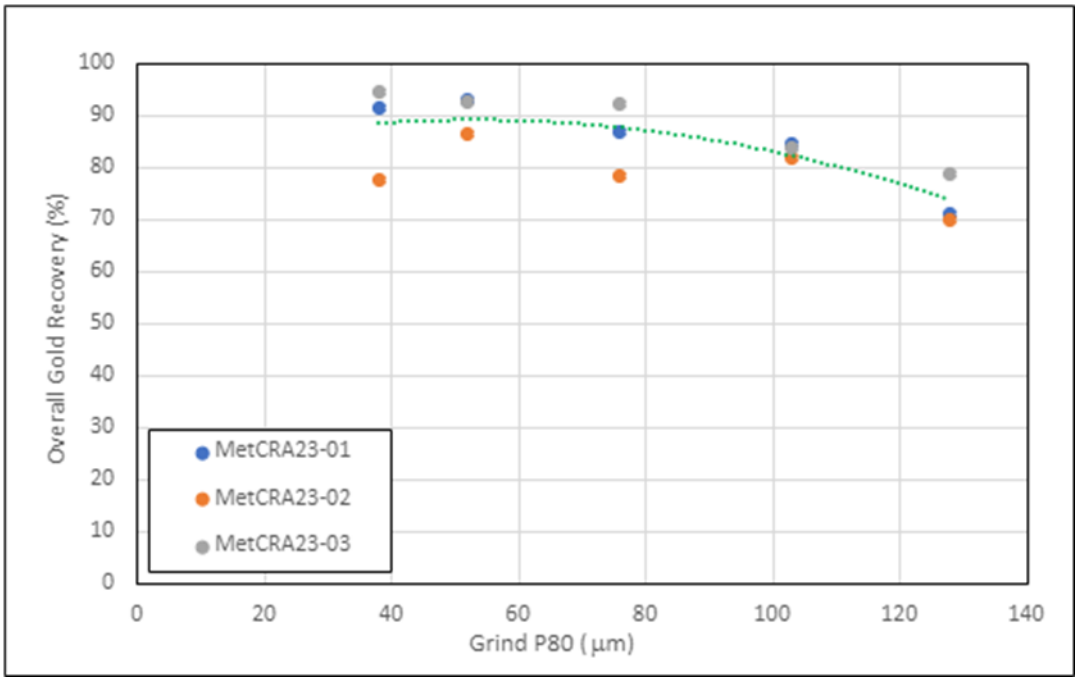
BaseMet Labs subjected their three (3) samples to a series of tests comprising gravity separation followed by rougher flotation. The program investigated, primarily, the role of grinding but also explored extended flotation times and adding more reagent.

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. MIBC was added throughout the tests as required and was hence not exactly the same for each test. Five rougher concentrate samples were collected after incremental durations of 1, 2, 3, 4 and 5 minutes respectively for a total duration of 15 minutes. Each concentrate was analysed separately.

Figure 13.11 summarises the results by showing the overall gold recovery from both gravity concentration and rougher flotation versus the grind P_{80} size.

Figure 13.11 – Overall (Gravity-Rougher Float) Gold Recovery vs. 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).

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

BaseMet Labs also explored the effects of adding additional PAX, extending flotation duration and adding collector A3477 on flotation performance. Table 13.15 summarises the overall (gravity +rougher float) gold recoveries recorded with these changes from the base case test procedure.

Table 13.15 – Summary of Gravity Plus Rougher Flotation Recovery at a P_{80} of 53 µm

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

Source: BaseMet Labs, 2023

There is no clear benefit with any of the changes made from the base case procedure. However, this may be entirely due to how the nugget effect masks other effects. Case in point is the ~8% decrease in overall gold recovery for sample Met_CRA23_01 when increasing PAX and extending the duration. There is no reason why the recoveries should be worse for these two (2) changes, so it is likely that the nuggets are again responsible for the unexpected results.

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 13.16 summarises the results of this set of tests.

Table 13.16 – Gravity-Rougher-Cleaner Tests (2023)

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

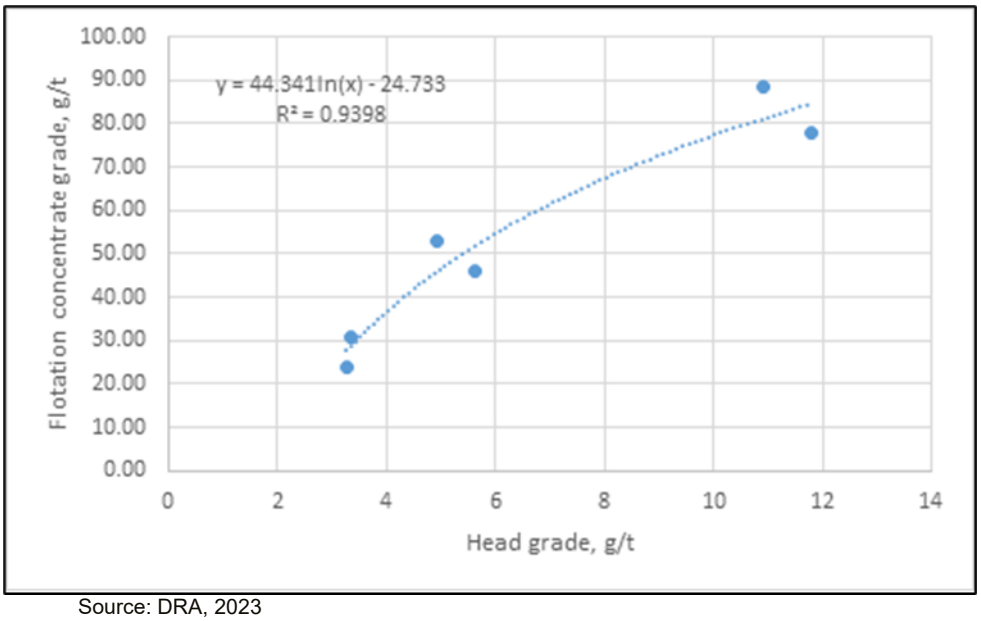
Source: BaseMet Labs, 2023

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

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

Figure 13.12 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 13.12 – Flotation Cleaner Concentrate Grade vs. Head Grade

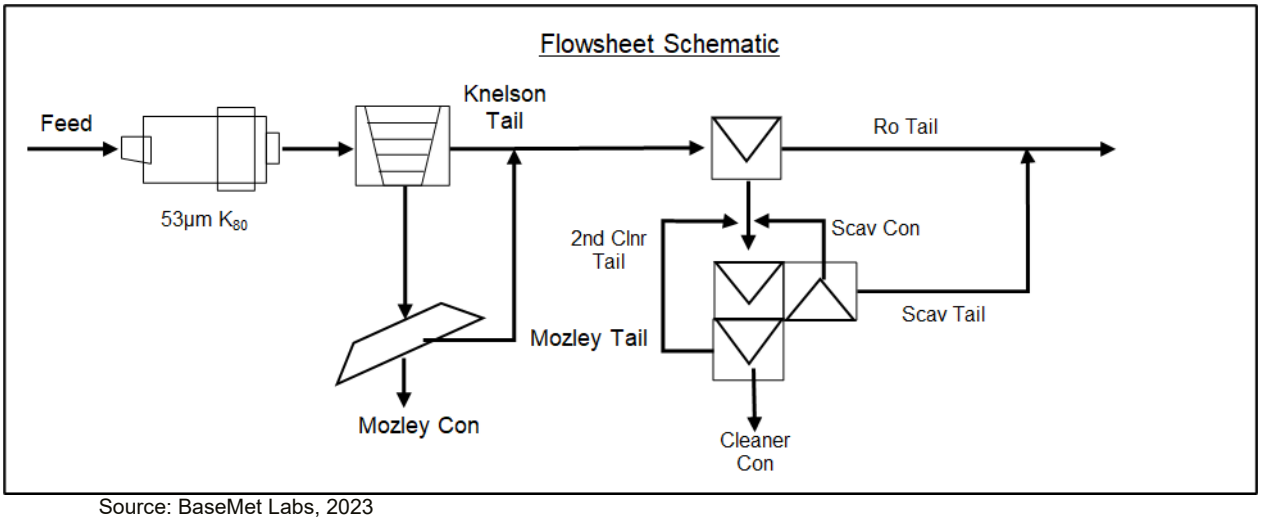


Source: DRA, 2023

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 13.13 shows a schematic of the procedure employed (courtesy of BaseMet Labs). The locked-cycle procedure involves using the products that are recycled from the preceding cycle of batch tests in the next cycle. After a few cycles, the products would effectively begin to resemble what would be produced by a continuous operation.

Figure 13.13 – 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 minutes conditioning of rougher feed with 250 g/t $CuSO_4$.
- 125 g/t PAX and 125 g/t A3477 collector addition to roughers.
- Total rougher float time = 30 minutes.
- Cleaner 1 duration = 15 minutes and cleaner 2 duration = 10 minutes.
- Cleaner scavenger = 6 minutes.
- Add 20 g/t PAX and 3477 to cleaner 1.
- Add 10 g/t PAX and 3477 to cleaner 2 and cleaner scavengers.
- MIBC addition varied depending on visual assessment of the froth.

Table 13.17 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 13.17 – 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

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 %
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%. It is recommended that future testwork should explore why this gold was not recovered by rougher flotation. This could be done by subjecting the tails sample to a diagnostic leach procedure which can be used to identify whether the gold lost to tails was locked within non-sulphide minerals and/or whether this could still be recovered through cyanidation.

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

13.4.10.1 Analysis of Concentrate Samples

The final concentrates from the LCTs as well as the gravity concentrate were analysed for elemental constituents. Table 13.18 summarises the most significant elements:

Table 13.18 – 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

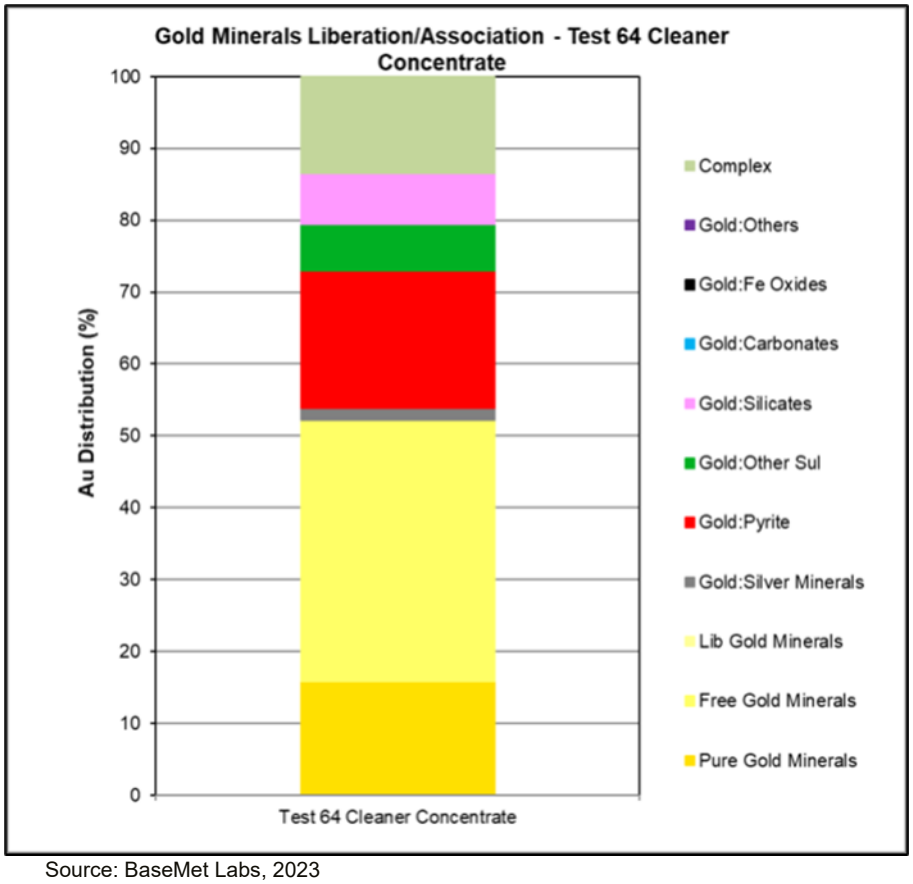
The silver contents of the three (3) flotation concentrates are too low for a revenue credit as it is only payable above 30 g/t. Arsenic is present, but below the penalty threshold of 2,000 ppm. The high copper and iron contents are as expected and due to their association with sulphur.

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. Figures 13.14 and 13.15 are reproduced here from the BaseMet Labs report.

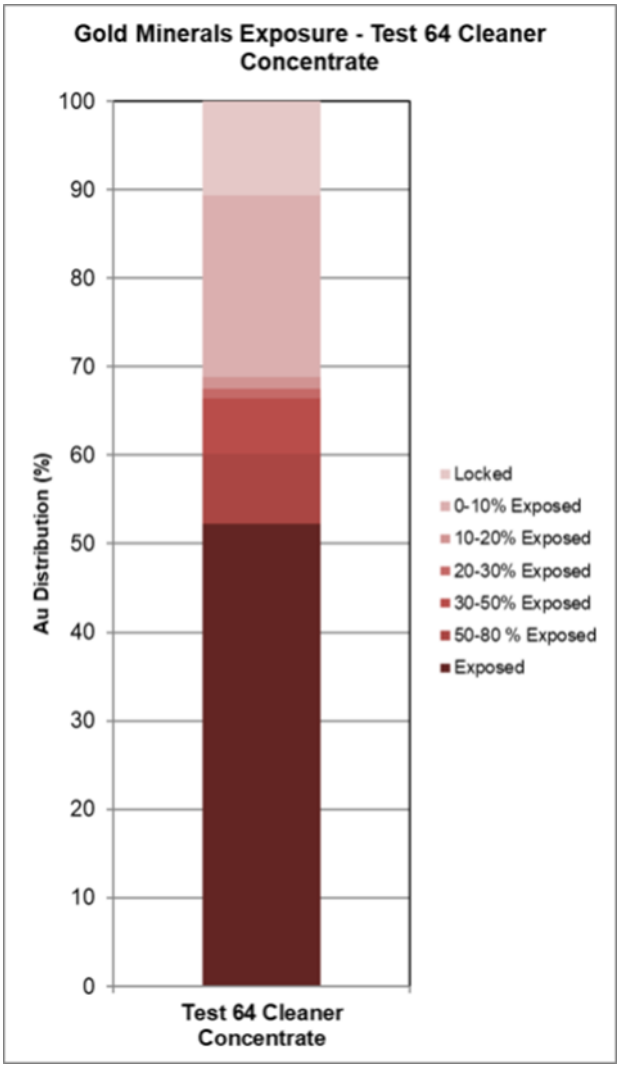
From Figure 13.14, only 10% of the gold is locked which explains why the sample was amenable to cyanidation. More than half of the gold in the sample was fully exposed.

Figure 13.14 – Gold Minerals/Association for a Sample of Cleaner Concentrate



Source: BaseMet Labs, 2023

Figure 13.15 – Gold Exposure in a Sample of Cleaner Concentrate

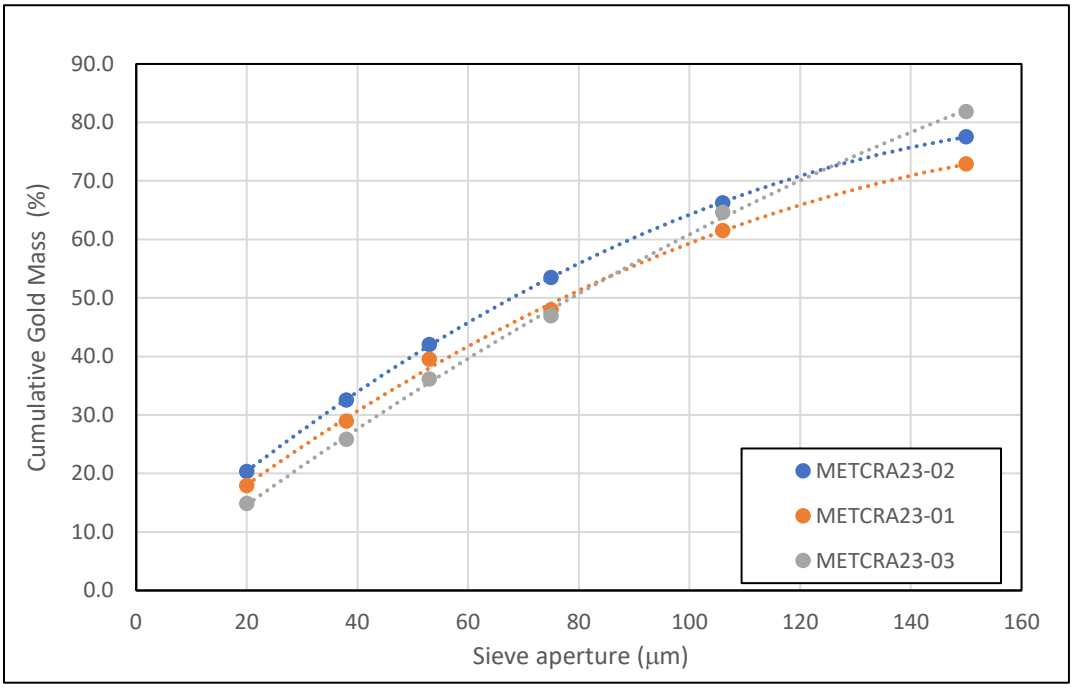


Source: BaseMet Labs, 2023

13.4.10.2 Gold Distribution by Size in Feed and Tails Samples

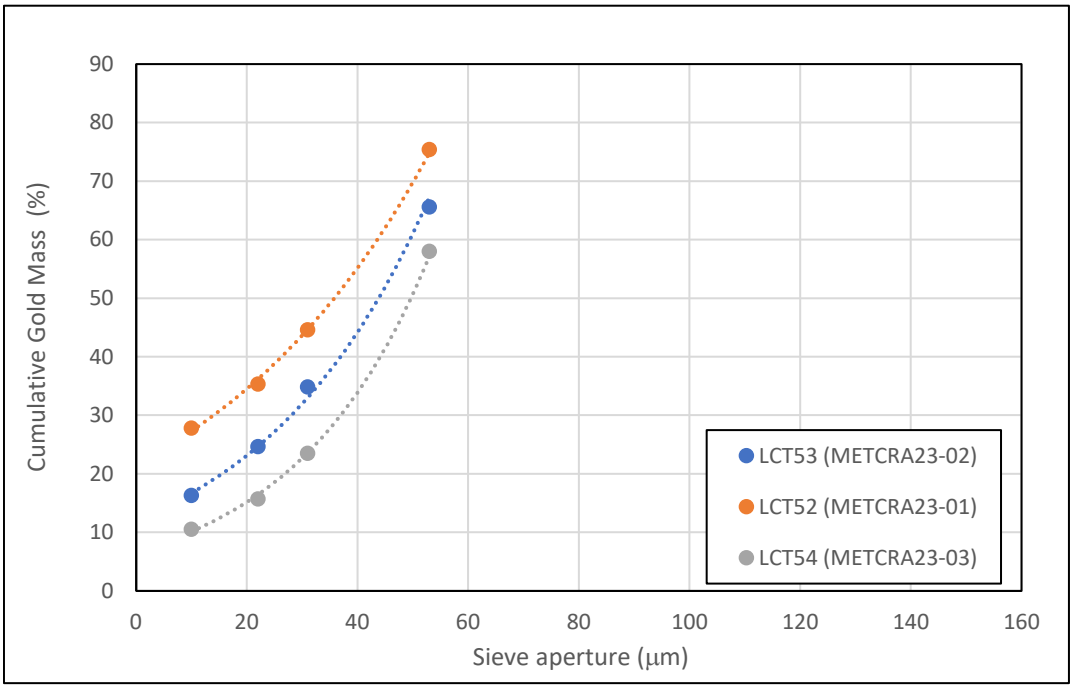
The three (3) feed samples and three (3) locked-cycle test tails samples were screened into size fractions and then each size was analysed for gold grade. Figures 13.16 and 13.17 illustrate the cumulative gold mass versus size curves for these samples.

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



Source: DRA, 2023

Figure 13.17 – Cumulative Gold Distribution by Size Class for the LCT Tails Samples



Source: DRA, 2023

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

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

13.4.11 Cyanidation of Locked Cycle Tails

The two (2) rejects products from each of the three (3) LCTs were combined and subjected to a 48-hour cyanidation. Table 13.19 summarises the results.

Table 13.19 – 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 recovery 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.

The average cyanide and lime consumptions recorded during these tests were 0.4 kg/t NaCN and 1.9 kg/t lime. Based on the kinetic curves and the reported solution analyses a total leach duration of 24 hours is sufficient. The >96% recovery is somewhat optimistic in that the pregnant leach solution and concentrates produced still requires further processing which may incur a small amount of recovery loss.

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-hours leach.

If cyanide is ever introduced to site together with a carbon circuit, electrowinning, and gold smelter to produce doré it stands to reason that the logical next step would be to attempt to also process the flotation concentrate further so that its gold can also be added into the doré product. To do this, the sulphides in the flotation concentrate would need to be destroyed to liberate the locked gold. Possible ways of achieving this would be ultrafine grinding followed by oxidation (Albion process), POX, BIOX, etc. Of these, the first option is by far the easiest and cheapest and it is recommended that this should be explored in future test programs when considering the introduction of cyanide to the site. A trade-off would be required to evaluate whether the incremental gold recovery justifies the significant increase in costs associated with these oxidation processes.

13.4.12 Cyanidation of Gravity Concentrates

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

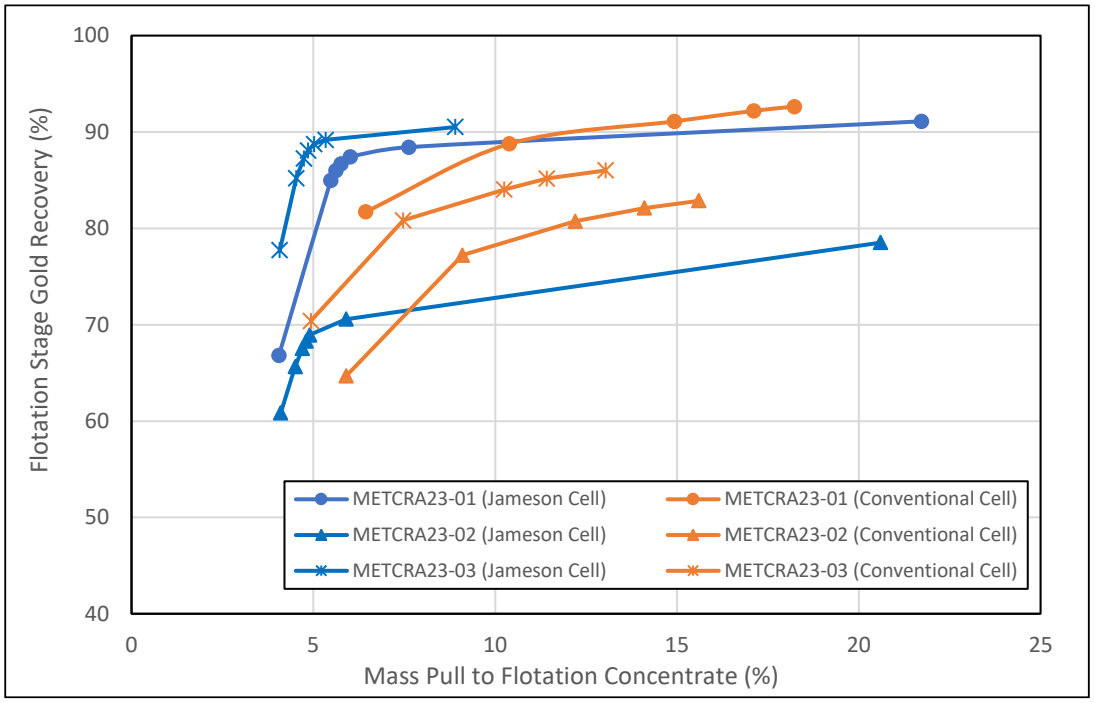
The results showed that near complete leaching is achievable (99.8% to 99.9%) after 72 hours of cyanidation. Leach kinetics was slow with about 7% of the leaching occurring in the final interval from 48 hours to 72 hours. Cyanide consumptions ranged from 30 kg/t to 80 kg/t, but this is possibly due to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

13.4.13 Jameson Flotation Cell Tests

BaseMet Labs performed three (3) tests using a procedure designed to mimic the operation of Jameson flotation cells. The tests were preceded by the normal gravity step to remove coarse gold ahead of the flotation procedure. Figure 13.18 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 13.18 – 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.
- 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 13.20. 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 13.20 – 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 13.21. 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 13.21 – 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 it does 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 ore body. 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].

- 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.
  - 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%.
  - 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 ore body. The remaining 18 samples were composited from 41 intervals. Table 13.22 summarises pertinent details pertaining to the variability samples. The proportion of the block model was calculated based on the block model developed during the PEA.

Table 13.22 – PFS (Phase 3) Samples Summary

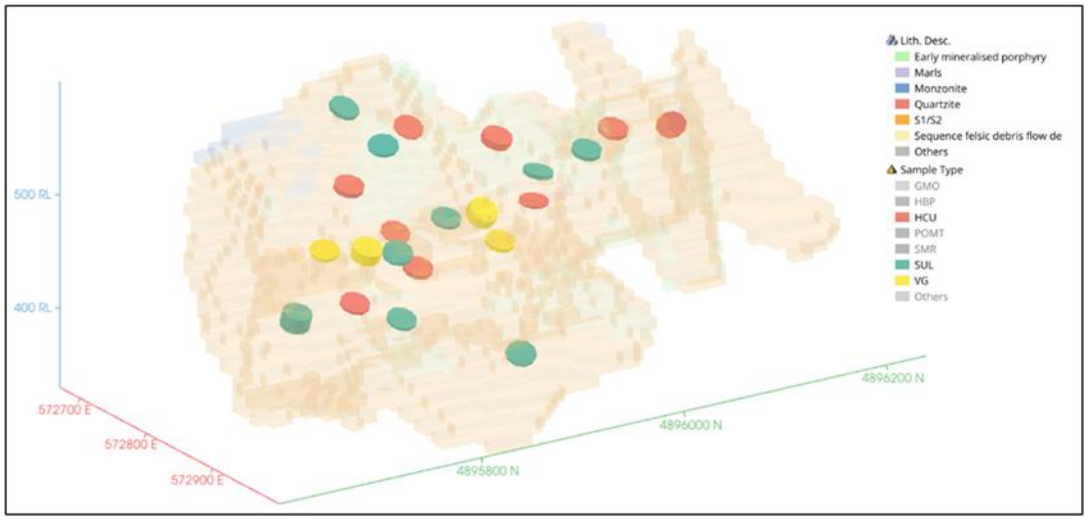
Lithology	Sample ID	Selection Criteria	Category	Block Model Ore Proportion
Diorite (Early mineralised porphyry)	HBP_1	Diorite Lithology	Diorite	4.60%
	HBP_2			
	HBP_3			
S1/S2	HCU_1	S1/S2 Lithology, Cu Mineralisation	S1/S2	86.76%
	HCU_2			
	HCU_3			
	SUL_1	S1/S2 Lithology, Sulphur Mineralisation		
	SUL_2			
	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%
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

It is worth noting that 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 represents 99.7% of the block model.

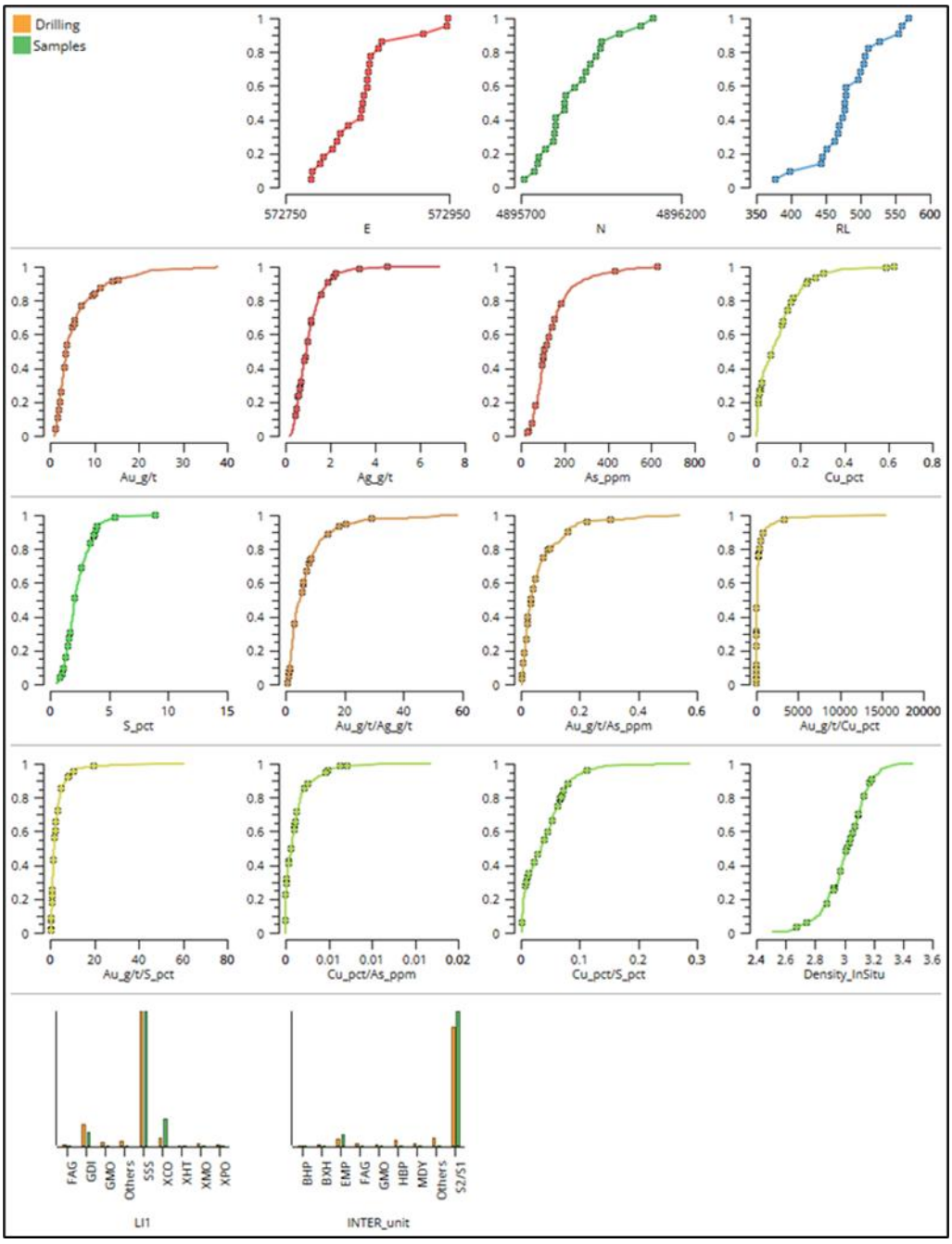
The Cancha software package was then used to confirm sample representativity. Figure 13.19 presents the spatial distribution of the intervals selected for the variability samples as an overlay on the block model. Figure 13.20 plots the Cancha distribution graphs for various elements and spatial distribution. In DRA'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 13.19 – Spatial Distribution of Intervals Selected for Variability Samples



Source: DPM, 2024

Figure 13.20 – 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.

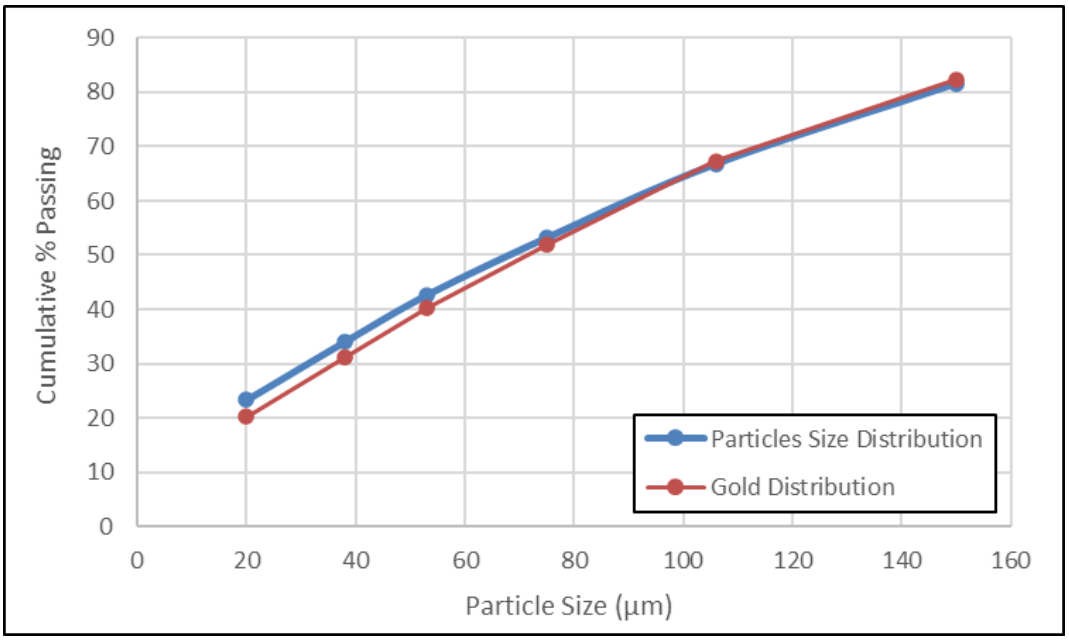
Table 13.23 provides the cumulative particle size distribution as well as the cumulative distribution of gold per particle size class for the 18 variability samples. Figure 13.21 shows the averaged profiles for both the particle sizes and gold deportment by size. The gold content is evenly distributed 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.

Table 13.23 – Cumulative Particle Size Distribution and Size Distribution of Gold

Cumulative % Passing		Particle Size Distribution						Size Distribution of Gold					
Sieve Size (µm)		20	38	53	75	106	150	20	38	53	75	106	150
Sample ID	HBP-1	23.3	36.0	45.9	58.7	74.7	90.7	24.5	38.3	48.0	60.9	75.6	92.6
	HBP-2	24.3	36.2	44.1	53.1	64.2	77.3	28.1	41.6	48.6	60.6	69.5	81.3
	HBP-3	22.8	33.8	42.1	52.7	65.9	81.9	28.2	41.3	49.1	59.3	72.0	85.9
	GMO-1	22.7	33.3	40.4	49.5	62.5	77.6	6.5	14.1	21.5	32.3	52.0	84.5
	SMR-1	21.9	32.2	40.5	51.4	65.6	81.0	25.4	37.7	49.9	64.1	76.0	88.1
	SMR-2	29.1	36.3	44.3	54.7	69.3	85.0	27.8	37.7	48.7	64.5	81.3	93.5
	POMT-1	27.1	37.6	45.7	55.3	67.4	80.0	20.1	30.5	39.6	51.7	70.7	83.2
	POMT-2	27.1	35.0	43.5	53.4	65.5	77.9	18.0	23.7	31.7	42.3	63.7	81.2
	POMT-3	23.9	36.9	46.4	57.5	70.7	84.8	25.7	44.7	53.3	65.0	77.4	89.8
	POMT-4	20.7	31.0	40.0	50.7	64.1	78.0	21.9	33.2	40.7	54.1	69.5	81.3
	HCU-1	21.8	32.0	40.8	51.3	64.5	77.7	13.8	21.6	31.3	45.1	62.5	86.6
	HCU-2	22.7	34.4	42.5	53.0	66.2	81.2	16.1	26.7	35.1	46.9	68.5	83.6
	HCU-3	24.7	36.9	46.5	58.2	74.0	88.9	15.0	27.1	39.8	50.7	64.2	78.4
	SUL-1	20.1	31.7	40.9	52.1	65.8	80.8	22.9	33.6	43.8	56.7	70.9	83.9
	SUL-2	26.8	37.4	46.0	55.4	66.9	79.1	21.7	30.7	41.1	54.1	75.7	85.5
	SUL-3	20.0	31.2	39.8	50.9	65.8	83.5	11.0	17.8	22.1	27.9	36.8	47.8
	VG-1	15.9	25.3	34.7	46.7	63.6	81.6	12.1	22.8	31.7	42.2	54.3	71.5
	VG-2	24.7	35.2	43.3	52.9	65.9	81.0	25.4	37.4	46.7	55.9	70.3	83.1

Source: BaseMet Labs, 2024

Figure 13.21 – Size by Size Gold Deportment Profiles



Source: DRA, 2024

13.5.3 HEAD ANALYSIS

Table 13.24 shows the head analyses performed on all 18 samples. Three (3) different methods were used, namely flame atomic absorption spectrometry (FAAS) for gold and iron contents, inductively couple 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 block plan, it stands to reason that there could be prolonged periods of weeks or even months where a high proportion of POMT will be mined. It would be prudent for the mine plan to be cognizant of the need to limit the POMT in the ROM blend to a maximum of 50% of plant feed. This risk is explored in greater detail in the flotation section.

Table 13.24 – Head Analysis for PFS Testwork (2024)

Element		Au	S	As	Fe	Ag
Method		FAAS	LECO	ICP	FAAS	ICP
Conc 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

Table 13.25 gives the weighted average grades calculated from the block model for each sample.

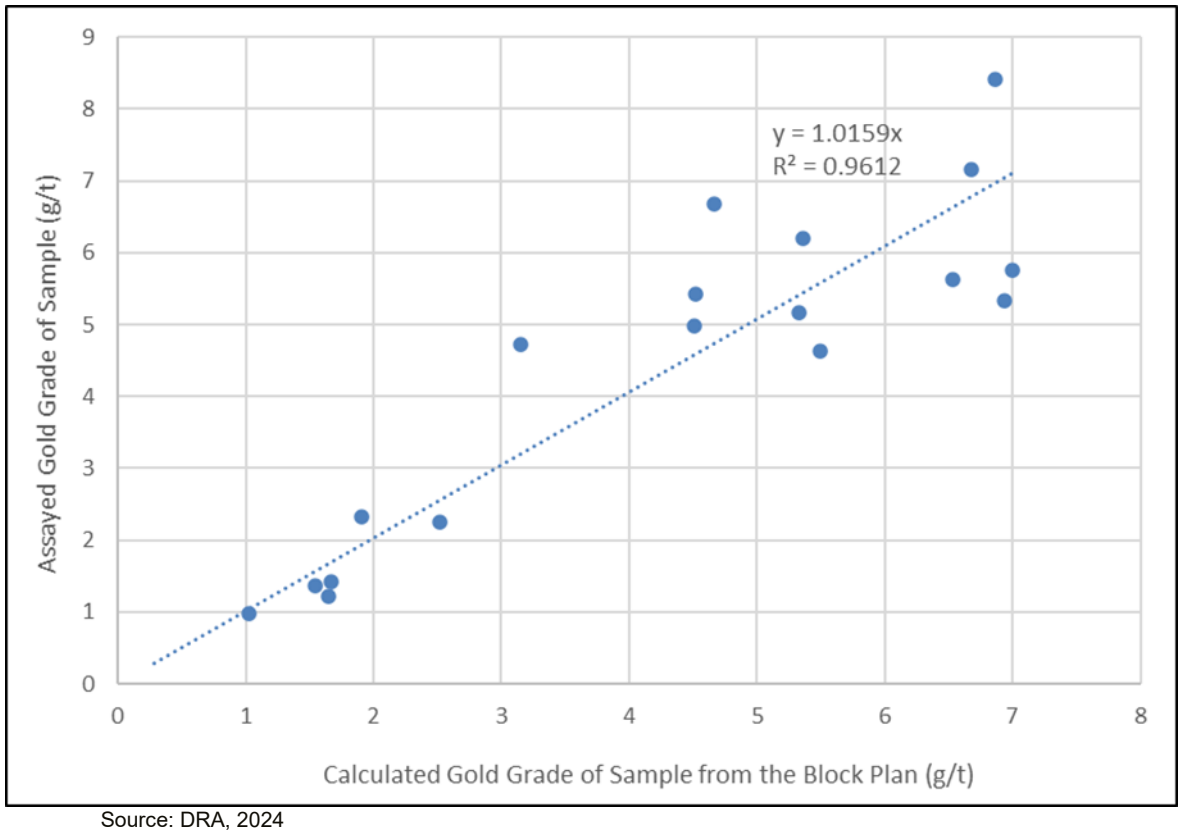
Table 13.25 – Block Model Grades

Element		Au	Ag	Cu	Pb	Zn	As	S	Mo	Sb
Conc Unit		ppm	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm
Sample ID	HBP_1	5.36	0.79	2,001.15	278.00	771.83	58.50	4.73	5.24	87.26
	HBP_2	2.52	0.45	1,733.04	23.10	48.37	38.90	1.90	19.99	10.19
	HBP_3	4.51	6.60	3,914.18	1,109.00	3,215.30	183.27	7.94	1.28	137.87
	GMO_1	6.86		1,286.36				2.50		
	SMR_1	1.02	0.69	647.34	58.60	100.40	85.50	2.63	1.68	25.42
	SMR_2	1.65	0.74	430.97	78.90	98.38	275.20	2.39	3.25	18.94
	POMT_1	4.66	3.35	76.58	74.60	117.35	893.40	7.03	1.39	20.65
	POMT_2	5.49	2.04	149.26	222.64	774.13	530.00	5.18	1.22	33.62
	POMT_3	1.54	1.65	861.96	82.20	30,608.82	499.00	8.28	1.20	27.43
	POMT_4	1.67	0.99	94.70	153.00	173.16	502.80	2.40	1.50	59.26
	HCU_1	6.53	3.26	7,395.00	57.50	245.97	98.90	8.43	3.98	28.42
	HCU_2	4.52	1.60	2,964.25	70.17	150.20	81.17	3.85	7.94	81.73
	HCU_3	7.00	2.11	1,065.88	390.70	7,456.96	154.90	4.14	12.74	38.67
	SUL_1	1.91	3.89	3,542.90	304.75	1,507.50	135.75	11.46	1.53	157.85
	SUL_2	3.15	1.28	1,495.75	49.67	80.77	748.33	5.20	1.04	61.28
	SUL_3	5.33		151.88				2.08		
	VG_1	6.94	0.66	57.19	75.00	120.08	125.40	0.91	1.70	7.69
	VG_2	6.68	0.75	667.99	66.00	711.71	74.71	0.97	2.41	17.06

Source: DRA, 2024

Figure 13.22 presents a plot of the assayed gold grades for each sample versus the calculated grades using the grades for the intervals that were selected for each sample in the block plan. The graph indicates that there was a good correlation between these values and no bias as the trendline through the (0,0) intercept had a high R^2 of 0.96 and the slope of the line is very close to 1. The good correlation provides confidence that the samples are representative of the block model and vice versa that the grades in the model are accurate.

Figure 13.22 – Comparison of Assayed vs. Calculated Grades for the 18 Variability Samples



Source: DRA, 2024

13.5.4 COMMINUTION

Table 13.26 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. The wt% in column 3 represents the weighting of this sample as a percentage of the block model. These weightings were used in calculating the averages and percentile values for the dataset.

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 implication is that the grinding section should be relatively easy to operate and control as variations in feed characteristics will not introduce significant instability. It also implies that blending of the ROM feed is not essential to smoothen hardness parameters. There might be an imperative to blend to limit the arsenic variation, but blending is not required from a throughput perspective.

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, and it was 59.8 for the three PEA samples.

Table 13.26 – Summary of Comminution Results (PFS Testwork)

Parameter		wt%	SG	SMC	DWi	SPI	BBWi	Ai
				A×b	kWh/m³	min	kWh/t	g
Sample ID	**HBP_1**	1.29	2.95	36.2	8.20	103	15.4	0.213
	HBP_2	1.29	2.74	28.5	9.53	117	17.7	0.201
	HBP_3	1.29	2.91	46.3	6.28	88	16.3	0.148
	GMO_1	0.03	2.70	35.1	7.66	170	19.7	0.229
	SMR_1	0.05	3.31	39.2	8.47	112	15.4	0.207
	SMR_2	0.05	3.41	52.2	6.51	80	16.0	0.170
	POMT_1	2.28	3.10	65.8	4.72	82	12.0	0.067
	POMT_2	2.28	3.23	40.3	8.01	139	15.4	0.098
	POMT_3	2.28	3.06	112.5	2.72	29	9.1	0.033
	POMT_4	2.28	3.09	56.3	5.47	103	14.7	0.102
	HCU_1	10.86	2.92	61.0	4.80	75	13.3	0.114
	HCU_2	10.86	2.77	65.5	4.22	69	15.0	0.103
	HCU_3	10.86	3.08	59.8	5.17	60	12.4	0.079
	SUL_1	10.86	3.17	84.7	3.73	47	10.8	0.064
	SUL_2	10.86	3.12	59.1	5.30	72	13.9	0.056
	SUL_3	10.86	3.33	48.1	6.92	103	14.9	0.167
	VG_1	10.86	3.30	72.6	4.54	73	13.4	0.157
	VG_2	10.86	3.10	44.6	6.96	112	15.2	0.170
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. Table 13.27 summarises the results for the concentrates collected at all stages of grinding.

During sample preparation, an approximate 10-kg portion of each sample was split and stage crushed to 100% passing 1,700 µm to produce a target K_{80} of around 1,200 µm. The EGRG test was conducted by passing the entire sample of crushed material through a Knelson MD-3 concentrator at a force of 60 G's. The concentrate was retained and sized for assaying. The tailings were ground in a laboratory rod mill for a second pass to a target K_{80} of around 250 µm. This material was then processed through the Knelson. The resulting concentrate was again screened and analysed as during the first pass. The procedure was repeated for a third pass aiming at a target K_{80} of 53 µm. After Pass 3, the final tailings were directly sampled, sized, and assayed by size.

Table 13.27 – EGRG Results Summary

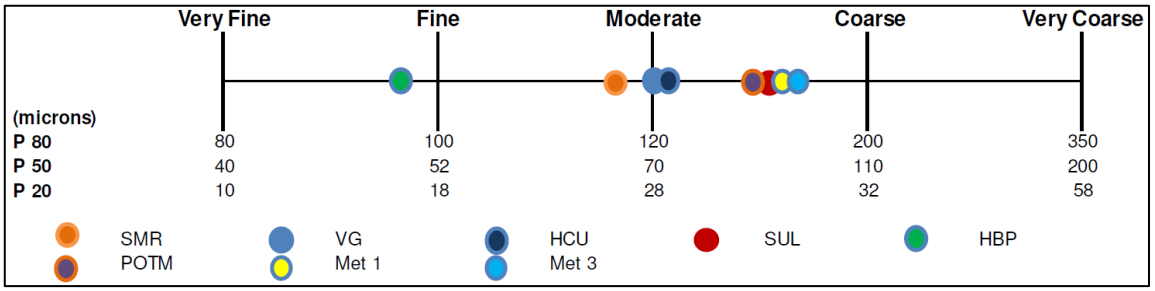
Composite (Test)	**Product**	**Feed Size K_{80} (µm) per Stage**	**Mass** (%)	**Assay Au** (g/t)	**% Au Distribution**
SMR GRG (T-40)	Stage 1 Conc	1,295	1.07	15.7	13.1
	Stage 2 Conc	241	1.05	16.4	13.4
	Stage 3 Conc	79	0.89	21.0	14.6
	Tailings		92.5	0.82	59.0
	Comb Conc 1-3		3.00	17.5	41.0
	Feed (calc.)			1.28	
VG GRG (T-41)	Stage 1 Conc	1,253	1.04	131	20.9
	Stage 2 Conc	253	1.00	148	22.6
	Stage 3 Conc	96	0.92	89.0	12.5
	Tailings		92.0	3.13	44.0
	Comb Conc 1-3		2.96	124	56.0
	Feed (calc.)			6.53	
HCU GRG (T-42)	Stage 1 Conc	1,225	0.79	139	20.8
	Stage 2 Conc	398	0.93	95.8	16.8
	Stage 3 Conc	83	0.89	131	22.1
	Tailings		95.1	2.23	40.2
	Comb Conc 1-3		2.61	121	59.8
	Feed (calc.)			5.28	

Composite (Test)	Product	Feed Size K_{80} (µm) per Stage	Mass (%)	Assay Au (g/t)	% Au Distribution
SUL GRG (T-43)	Stage 1 Conc	1,162	0.89	73.2	14.4
	Stage 2 Conc	282	0.86	85.9	16.2
	Stage 3 Conc	64	0.76	116	19.3
	Tailings		95.0	2.40	50.1
	Comb Conc 1-3		2.51	90.3	49.9
	Feed (calc.)			4.55	
HBP GRG (T-44)	Stage 1 Conc	1,242	0.78	63.0	9.7
	Stage 2 Conc	246	0.95	54.6	10.2
	Stage 3 Conc	68	0.78	96.5	14.9
	Tailings		94.8	3.49	65.2
	Comb Conc 1-3		2.51	70.2	34.8
	Feed (calc.)			5.08	
POMT GRG (T-45)	Stage 1 Conc	1,160	0.52	79.8	12.6
	Stage 2 Conc	254	0.48	114	16.8
	Stage 3 Conc	59	0.46	114	15.9
	Tailings		95.7	1.88	54.7
	Comb Conc 1-3		1.46	102	45.3
	Feed (calc.)			3.29	

Source: BaseMet Labs, 2024

The full set of laboratory results was provided to FLSmidth for modelling expected products from operating various Knelson concentrator models. Figure 13.23 shows the classification, by FLSmidth, of the variability samples and the two composites tested on the Amira scale. For most of the samples tested the GRG was classified as “moderate” to “coarse” with only the HBP sample classified as “fine”.

Figure 13.23 – GRG Fineness on the Amira Scale



Source: FLS, 2024

Table 13.28 summarises the modelling results performed with 40% of circulating load (125 tph solids or 120% of fresh feed) diverted from the cyclone underflow to the gravity circuit. Two (2) flowsheet scenarios were modelled as follows:

Two stages of gravity concentration with a primary centrifugal concentrator followed by a shaking table operating at average efficiency according to the FLSmidth benchmark. This is shown in Table 13.28 by the three (3) columns of numbers under the “To Table Concentrate” heading.

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.

For each of these flowsheets, FLSmidth modelled the use of a single QS30, two (2) parallel QS30s and a single QS48 unit; for the two (2) parallel QS30s each would receive 62.5 tph solids only.

Table 13.28 – Modelled Gravity Recovery Results for Cyclone Underflow

Sample ID	Total Gold Recovery (%)					
	To Table Concentrate			To ICU Pregnant Solution		
	1 QS30	2 QS30	1 QS48	1 QS30	2 QS30	1 QS48
SMR	20	21	22	23	24	25
VG	30	32	33	35	37	38
HCU	34	35	37	38	40	42
SUL	25	27	28	30	31	33
HBP	12	13	14	17	18	19
POTM	25	26	27	28	30	31
Weighted Average	**28.5**	**30.0**	**31.4**	**33.0**	**34.6**	**36.3**

Source: FLSmidth, 2024

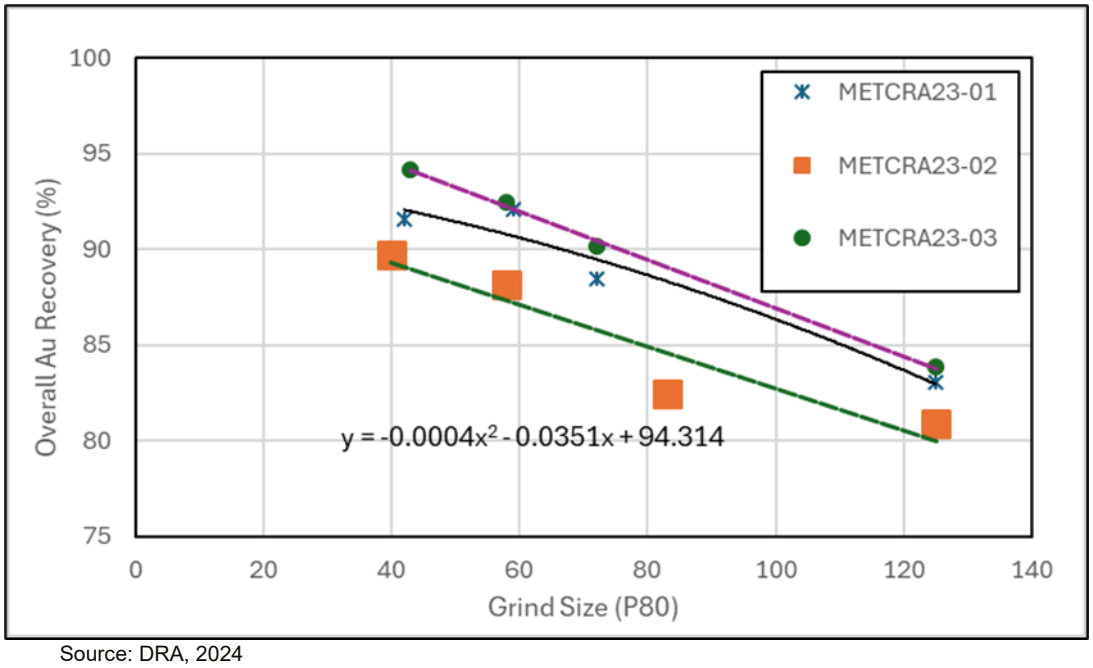
FLSmidth recommended installing two (2) parallel units so that there is still some removal of GRG from the circuit, albeit at a slightly reduced efficiency, should one of the units become unavailable. It should be noted that this efficiency only applied to how much gold reports to the gravity concentrate, it did not apply to overall gold recovery. Based on discussions with FLSmidth, the overall gold recovery will only be affected negatively if there is no gravity concentration at all (i.e., if a single machine is used and it is bypassed for a prolonged period). In such a scenario, the GRG would recirculate through the grinding circuit until it became very fine and then reports to the cyclone overflow stream. Because of its fineness, this gold content would then be lost to the final flotation tails stream.

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 (2023) 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 (3) 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 (3) were then ground to three (3) finer grinds. Rougher flotation tests were then conducted using these sub-samples at the four (4) 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 13.24 illustrates a plot of the overall gold recovery versus the grind P_{80} size, whereas Figure 13.25 depicts a plot of the rougher tailings gold grade versus the grind P_{80} size.

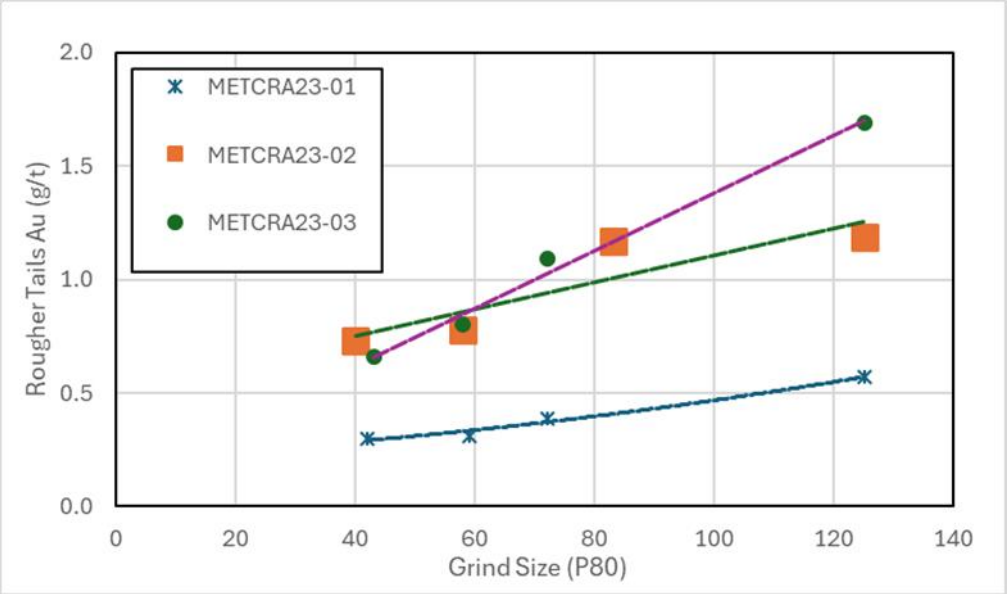
Figure 13.24 – Overall Gold Recovery vs. Grind P_{80}



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 to 130 µm).

Figure 13.25 – Rougher Tailings Gold Grade vs. Grind P_{80}



Source: DRA, 2024

DRA performed a high-level profitability analysis to determine the optimum grind size. As inputs to this evaluation the following criteria was assumed:

- Throughput = 105.5 tph
- Grinding Circuit = SSAG (assumed for simplicity)
- Gold Price = $74/gram or $2,302 per Oz
- Head Grade = 5.6 g/t Au
- Power Cost = $0.10/kWh
- BBWI = 13.9 kWh/t (average of variability samples weighted by lithology)
- Ai = 0.125 (average of PEA and PFS test results)
- Liner Price = $4/kg
- Media Price = $1.9/kg
- Media Consumption = 0.026 kg/kWh (based on DRA in-house correlation with Ai)
- Overall Recovery (%) = 92 - 0.00044*P_{80}^2 - 0.0351*P_{80} (based on grind optimisation tests, but with intercept of y-axis decreased from 94 to 92 to scale it to other test results)

With these criteria assumptions the revenue at each grind was calculated as shown in Table 13.29.

Table 13.29 – Revenue Estimate at Each Grind P_{80}

Grind Size P_{80} (µm)	105	74	53	38
Recovery (%)	83.9	87.2	89.0	90.1
Revenue ($/h)	36,682	38,128	38,917	39,386

Source: DRA, 2024

Table 13.30 shows the calculations and additional assumptions made with regards to the Opex estimate at each grind. The factor for fine grinding is an empirical exponential equation, defined by DRA for this exercise, to account for the cost increase at finer grinds. This factor was calibrated to yield a number close to 1 at P_{80} =75 µm. This factor was used in the calculation of grinding energy in to compensate for the underestimation of grinding energy by Bond's 4th law for grinds below 75 µm. This factor was then also used to estimate ancillary power, reagent consumptions and even maintenance costs at the finer grinds – the assumption being that these costs will all also increase exponentially as the grind becomes finer. The reagent cost is from the PEA and includes flocculant and all flotation reagents.

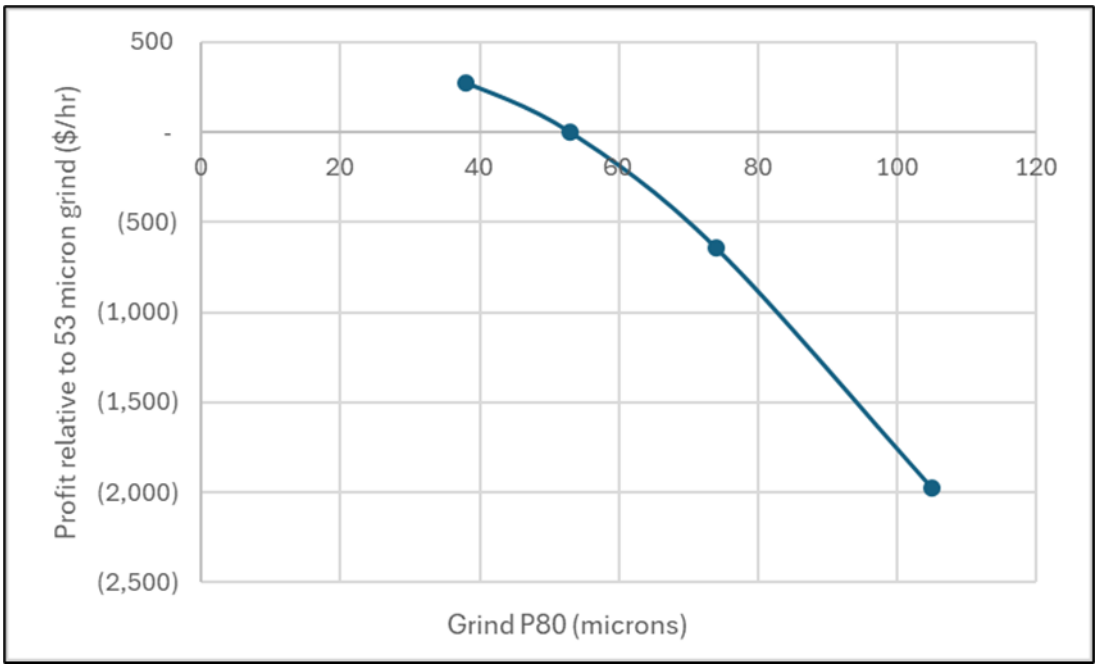
Table 13.30 – Opex Estimate at Each Grind P_{80}

Grind Size P_{80} (µm)	105	74	53	38
Factor for Fine Grinding	0.852	0.993	1.150	1.332
Factor for SSAG Circuit	1.2	1.2	1.2	1.2
Grinding Pinion Power (kW)	1,424	1,987	2,729	3,741
Ancillaries Power (kW)	236	250	268	294
Power Costs ($/h)	166	224	300	403
Media Costs ($/h)	71	100	137	188
Liners Costs ($/h)	18	25	34	47
Reagents Costs ($/h)	78	91	105	122
Maintenance Allowance ($/h)	45	52	61	70
Total Costs ($/h)	**378**	**492**	**637**	**830**

Source: DRA, 2024

Figure 13.26 demonstrates the profitability of the plant at each grind P_{80} relative to the baseline of a 53 µm grind. It was estimated that a decrease of grind P_{80} size from 53 to 38 µm would yield an additional profit of $275/h or $2.2 million per year (assuming 8,000 hours per year).

Figure 13.26 – Estimated Profit Relative to 53 µm Grind



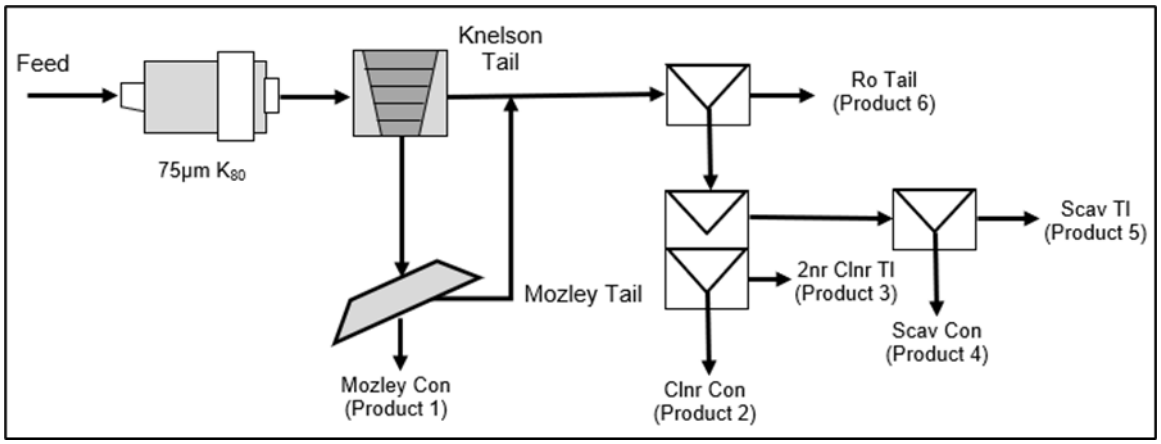
Source: DRA, 2024

However, this analysis disregards the additional Capex required to achieve said finer grind. Decreasing from 53 to 38 µm would require more grinding mills, larger thickeners, larger filters for both concentrate and tailings, etc. For the Čoka Rakita project, this additional profit would likely be insufficient to pay back the capital required to go from 53 to 38 µm. There would also be additional new equipment Capex for Čoka Rakita as the Ada Tepe equipment would not be sufficient (e.g. tailings thickener, reagents systems). Considering these Capex effects, DRA recommends retaining the 53 µm grind from the PEA.

13.5.7 FLOTATION

All variability testwork samples were subjected to a series of tests comprising of gravity separation followed by both rougher and cleaner flotation (as shown in Figure 13.27 courtesy of BaseMet Labs). The purpose of these tests was to observe both the mass pull and gold recovery of the variability samples.

Figure 13.27 – Schematic of Flotation Variability Testwork



Source: BaseMet Labs, 2024

The gravity step comprised of 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 13.31 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 were added to the overall recovery. Based on DRA's experience it was assumed that around 80% of the gold contained in these intermediate streams would eventually report to the final product stream if it were recycled.

Table 13.31 – Summary of Gravity and Flotation Tests at k_{80} of both 53 and 75 µm

Sample ID	Grind k_{80} (µm)	Gravity Conc			Rougher Conc			Cleaner Conc			Overall Rec (%)
		wt %	Au (g/t)	Rec (%)	wt %	Au (g/t)	Rec (%)	wt %	Au (g/t)	Rec (%)	
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 Conc			Rougher Conc			Cleaner Conc			Overall Rec (%)
		wt %	Au (g/t)	Rec (%)	wt %	Au (g/t)	Rec (%)	wt %	Au (g/t)	Rec (%)	
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

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

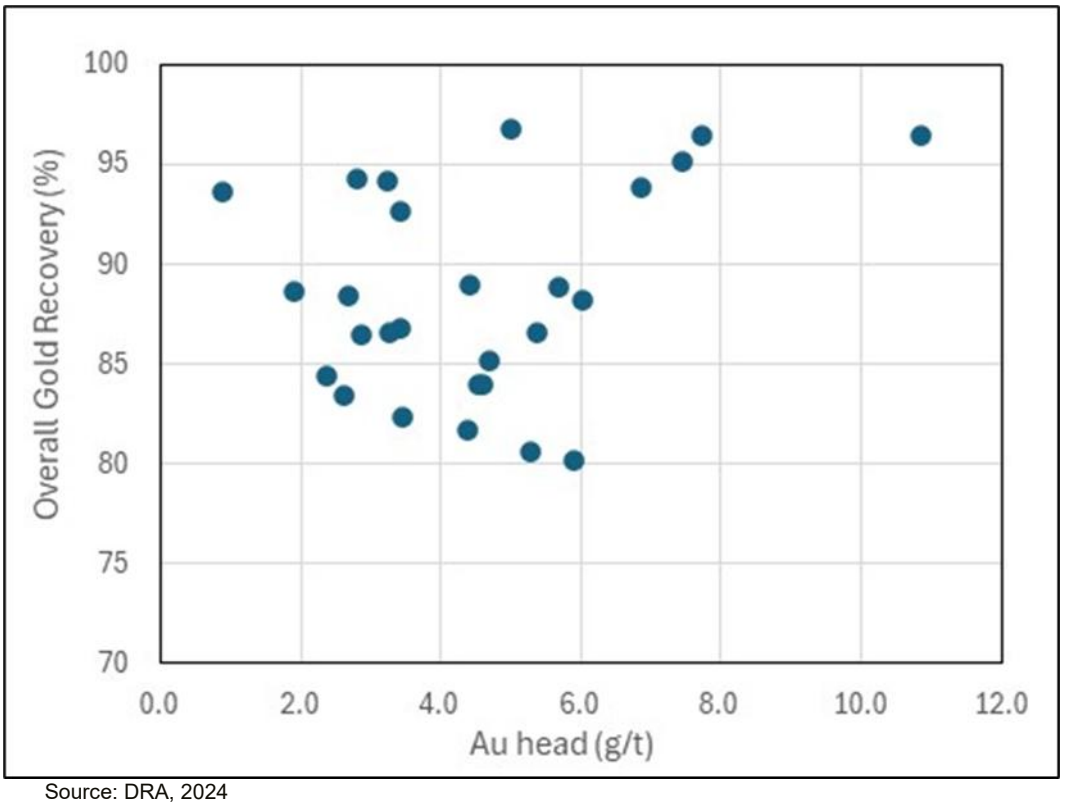
- Mass pull to gravity concentrate = 0.15%.
- Gold recovery to gravity concentrate = 37.4%.
- Mass pull to the final flotation concentrate = 9.1%.
- Gold distribution to flotation concentrate = 50.0%.
- Overall recovery of gold = 87.4%.

The gravity gold recovery number 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 (refer to Section 13.5.1). Based on this a lower gravity recovery value will be adopted for the PDC of this 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 13.28 shows the population of recoveries versus head grade for all the tests conducted on all 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 ore body based on the limited dataset of results from this testwork program and that the PDC will use the weighted average recovery for design purposes.

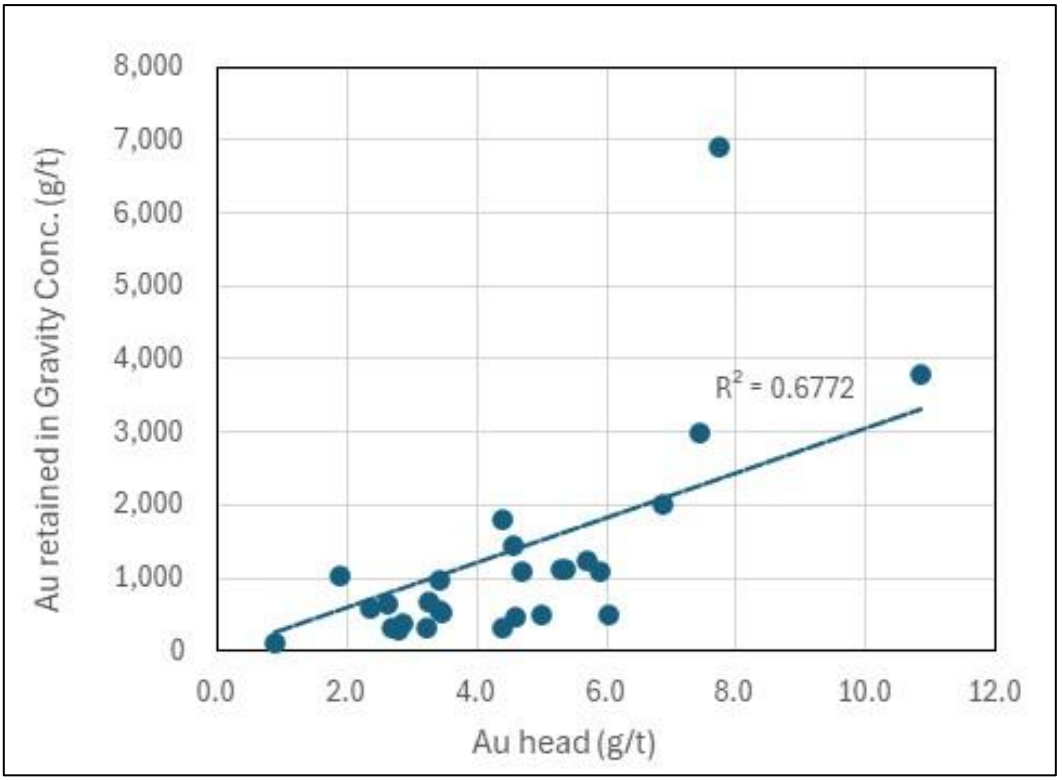
Figure 13.28 – Overall Gold Recovery vs. Head Grade



Source: DRA, 2024

Figure 13.29 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 13.29 – Overall Gold Recovery vs. Head Grade

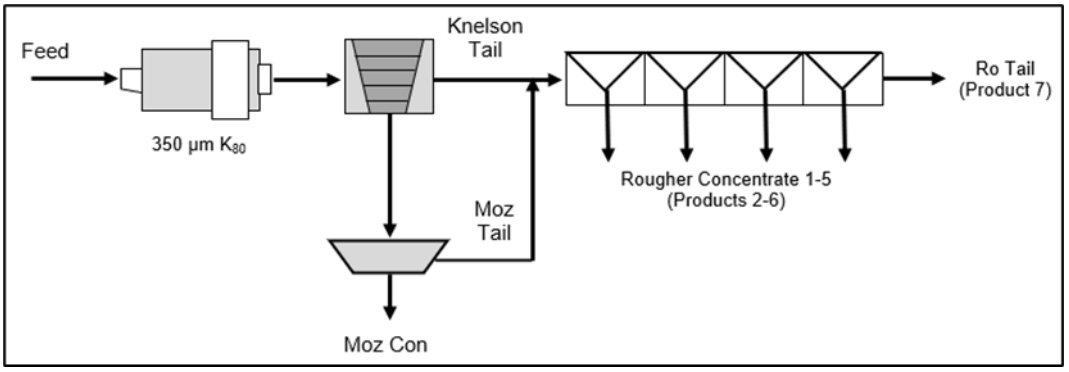


Source: DRA, 2024

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 (CPF) tests to investigate the viability of including CPF 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 13.30 – Schematic of Rougher Test

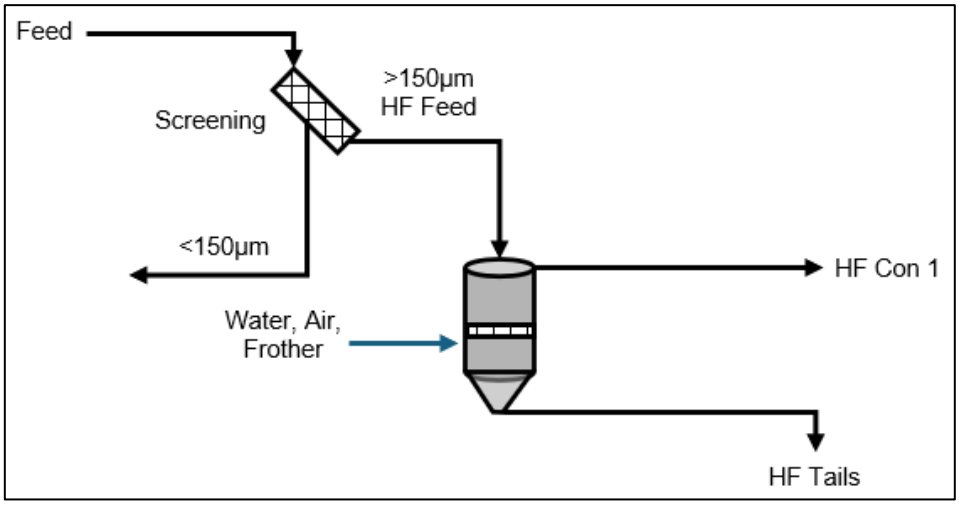


Source: DRA, 2024

Figure 13.30 provides the schematic of the rougher test procedure that served as a comparative test and resembled a traditional flotation circuit with five (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.

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 13.31 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 13.31 – Schematic of Exploratory Hydrofloat Test



Source: BaseMet Lab, 2024

Table 13.32 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).

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.

The stage recovery when floating the desliming screen undersize sample was 80.5%. This is also lower than the recovery of around 87% which was typically achieved when floating ROM samples with a P_{80} of 53 µm.

It is concluded that coarse particle flotation is not a viable flowsheet option for Čoka Rakita.

Table 13.32 – Rougher and Preliminary Hydrofloat Test Results

Product	wt %	Au (g/t)	Au Distrib (%)
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

13.6 Responsible Mining Solutions 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 testwork 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. Additionally, UCS tests evaluated backfill strength using varying binder contents over different curing times.

The RMS report includes details pertaining to the sample receipt and preparations.

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 Semi-Quantitative X-Ray Diffraction (XRD) Analysis

Semi-quantitative X-Ray Diffraction (XRD) analysis was conducted on the FPT to identify the crystalline mineral components present in the samples. The results are summarised in Table 13.33.

Table 13.33 – Mineralogy Results Summary

Mineral Name	Chemical Compound	SG	Mohs Hardness	FPT (wt%)
Grossular	$Ca_3Al_2(SiO_4)_3$	3.6	6.5 - 7	47.8
Calcite	$CaCO_3$	2.7	3	11.2
Quartz	SiO_2	2.7	7	10.2
Orthoclase	$K(AlSi_3O_8)$	2.6	6	9.2
Albite	$Na\ (AlSi_3O_8)$	2.6	6 - 6.5	8.8
Clinochlore	$Mg_5Al\ (AlSi_3O_{10})(OH)_8$	2.8	2 - 2.5	8.4
Laumontite	$CaAl_2Si_4O_{12}\ 4H_2O$	2.3	3.5 - 4	3.0
Annite	$KFe^{2+}{}_3(AlSi_3O_{10})\ (OH)_2$	3.3	2.5 - 3	0.9
Barrerite	$(Na,K,Ca)2[Al_2Si_7O_{18}]\ 7H_2O$	2.3	3 - 4	0.3
Actinolite	$Ca_2(Mg,Fe^{+2})5Si_8O_{22}(OH)_2$	3.1	5 - 6	0.2
Total				**100**

Source: RMS, 2024

13.6.3 Rheology

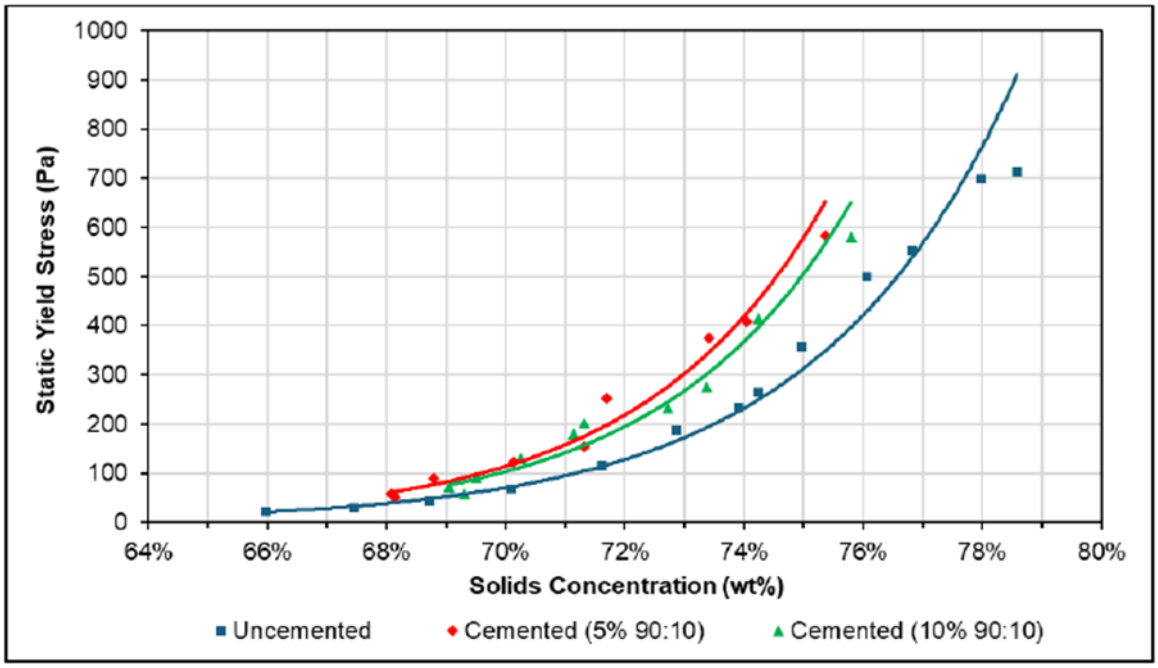
The rheological tests evaluated the flow properties of the FPT, which are critical for pumping and handling during backfill operations. The results showed that the uncemented FPT has a yield stress of 405 Pa at a 175 mm slump and 234 Pa at a 250 mm slump, indicating that it requires significant force 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 13.34, Figure 13.32 and Figure 13.33 show the summary of the static yield stress testing.

Table 13.34 – 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

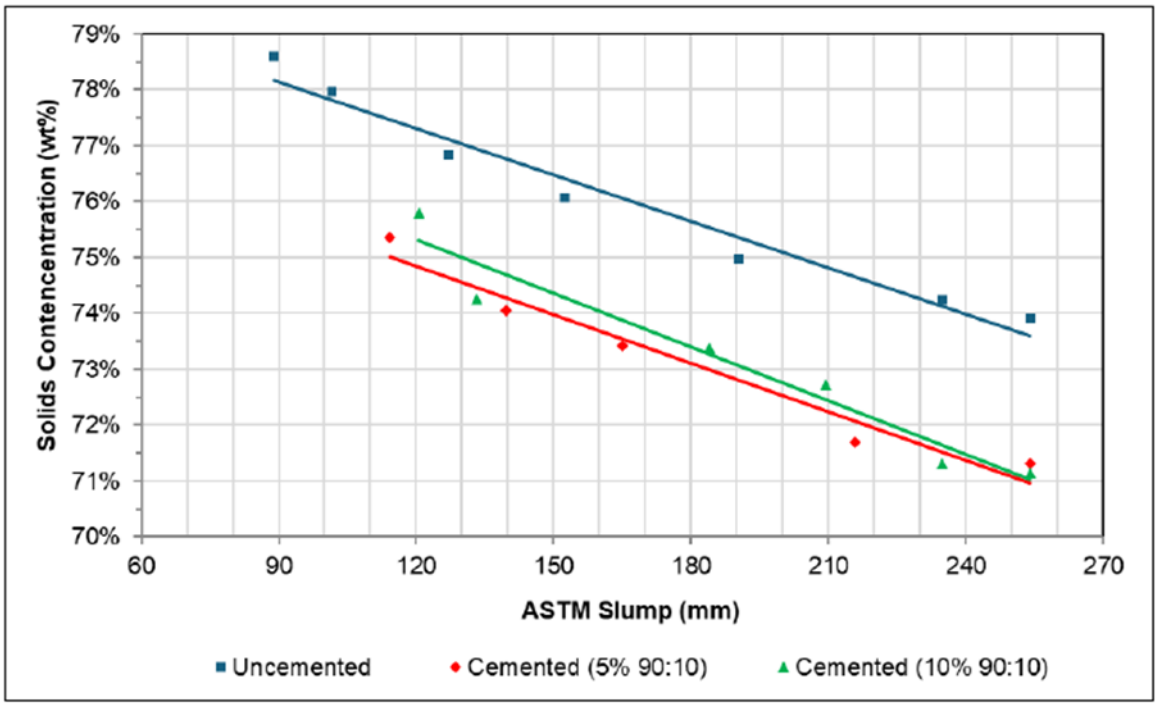
Source: RMS, 2024

Figure 13.32 – Static Yield Stress Curves vs. Solids Concentration



Source: RMS, 2024

Figure 13.33 – ASTM Slump vs. Solids Concentration

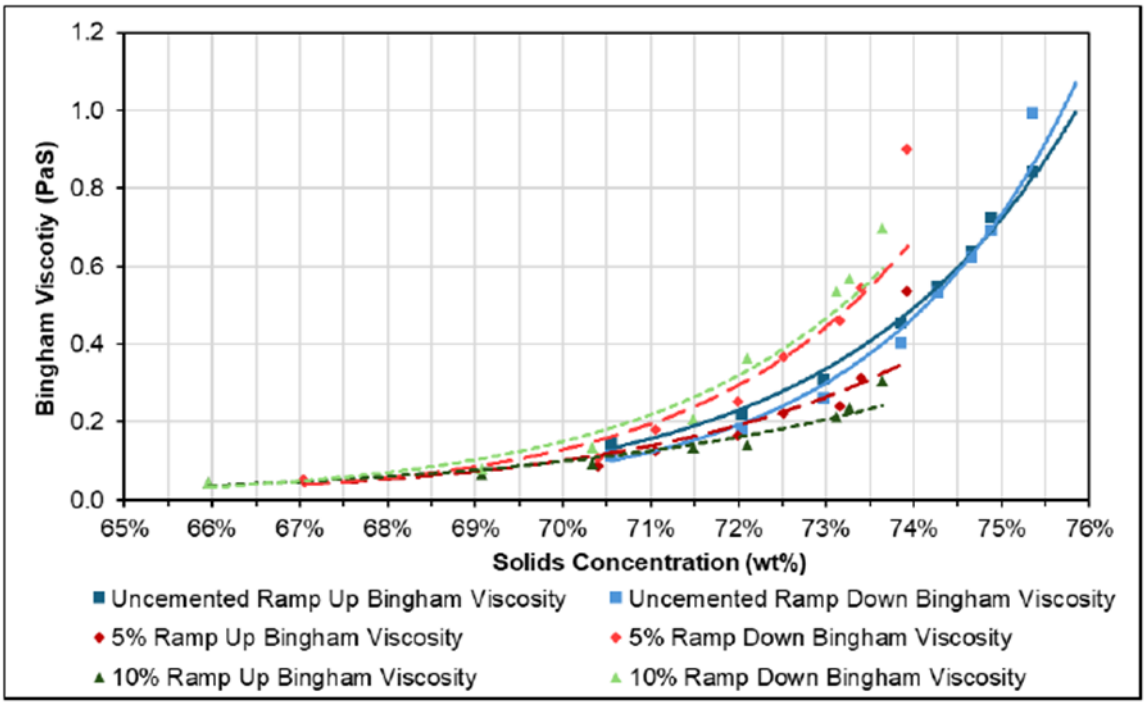


Source: RMS, 2024

13.6.4 VISCOSITY AND DYNAMIC YIELD STRESS

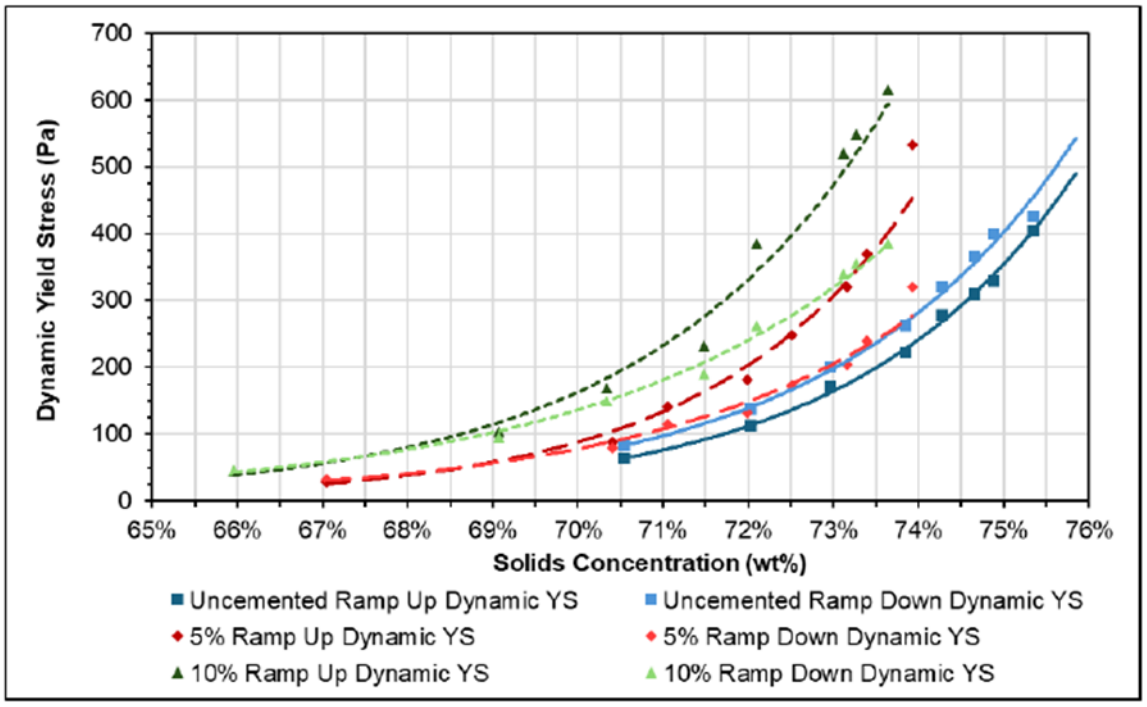
Viscosity measures the internal friction within a fluid, providing insight into its resistance to flow, and is useful for estimating friction losses and material transport requirements. Paste is generally considered a non-Newtonian fluid with plastic properties, so a Bingham regression is applied to the data to determine both the viscosity and dynamic yield stress. The results for viscosity and dynamic yield stress are shown graphically in Figure 13.34 and Figure 13.35 respectively.

Figure 13.34 – Bingham Viscosity vs. Solids Concentration



Source: RMS, 2024

Figure 13.35 – Dynamic Yield Stress vs. Solids Concentration



Source: RMS, 2024

13.6.5 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.

Table 13.35 and Figure 13.36 present the results summary for all flocculants tested.

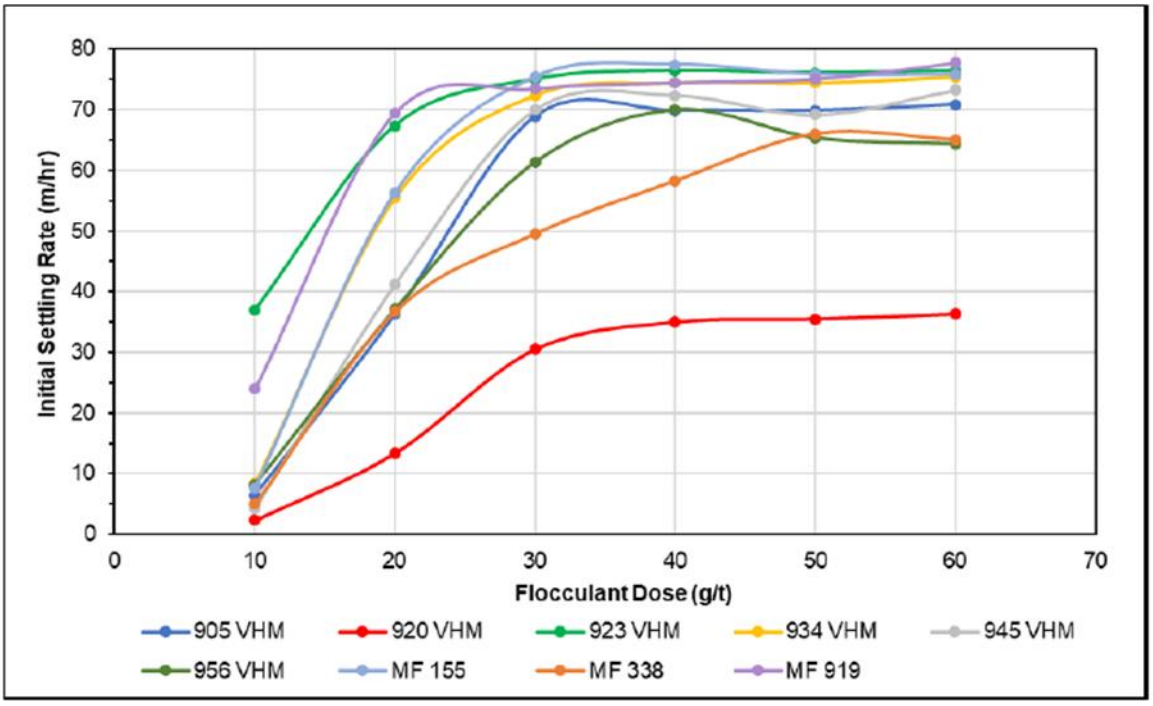
Table 13.35 – Flocculant Screening Summary

Flocculant	Dosage (g/t)	Initial Settling Velocity (m/h)	Overflow Clarity
AN 905 VHM	10	6	Cloudy
	20	36	Very Clear
	30	69	Very Clear
	40	70	Very Clear
	50	70	Very Clear
	60	71	Very Clear
AN 923 VHM	10	37	Cloudy
	20	67	Clear
	30	75	Clear
	40	76	Very Clear
	50	76	Very Clear
	60	76	Very Clear
AN 934 VHM	10	8	Cloudy
	20	56	Clear
	30	72	Clear
	40	74	Clear
	50	74	Clear
	60	75	Clear

Flocculant	Dosage (g/t)	Initial Settling Velocity (m/h)	Overflow Clarity
AN 945 VHM	10	4	Cloudy
	20	41	Slightly Cloudy
	30	70	Slightly Cloudy
	40	72	Clear
	50	69	Clear
	60	73	Very Clear
MF 155	10	8	Cloudy
	20	56	Clear
	30	75	Clear
	40	77	Clear
	50	76	Clear
	60	76	Clear
MF 919	10	24	Cloudy
	20	69	Clear
	30	73	Clear
	40	74	Clear
	50	75	Very Clear
	60	78	Very Clear

Source: RMS, 2024

Figure 13.36 – Initial Settling Velocity vs. Flocculant Dose



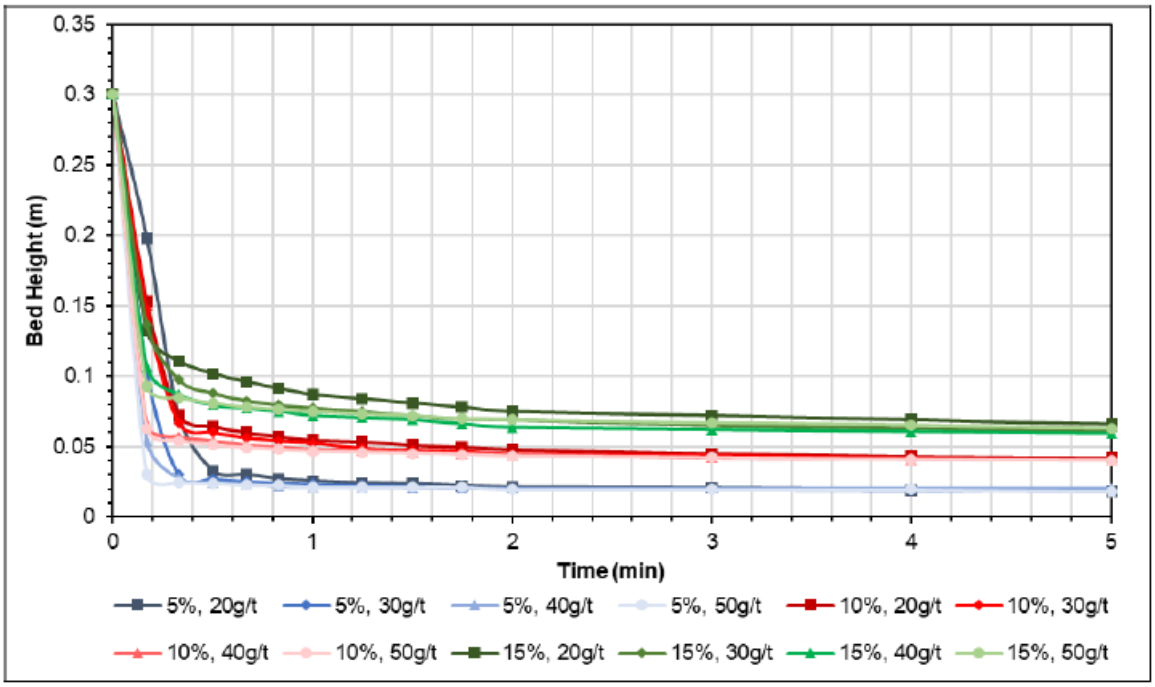
Source: RMS, 2024

13.6.6 FEED DENSITY SCREENING

Feed screening involved testing a range of feed densities with the selected flocculant and dosage range determined during the flocculant screening phase. Tests were carried out in 1000 mL graduated cylinders, tracking the bed height (volume) over a 30-minute period. The initial settling velocity, underflow (UF) density, and overflow (OF) clarity were evaluated to identify the optimal feed density.

Feed screening was performed at 5, 10, and 15 wt% solids using AN 923 VHM. Figure 13.37 displays the data graphically.

Figure 13.37 – Bed Height vs. Time (first 5 minutes)



Source: RMS, 2024

13.6.7 4 L Static Thickening

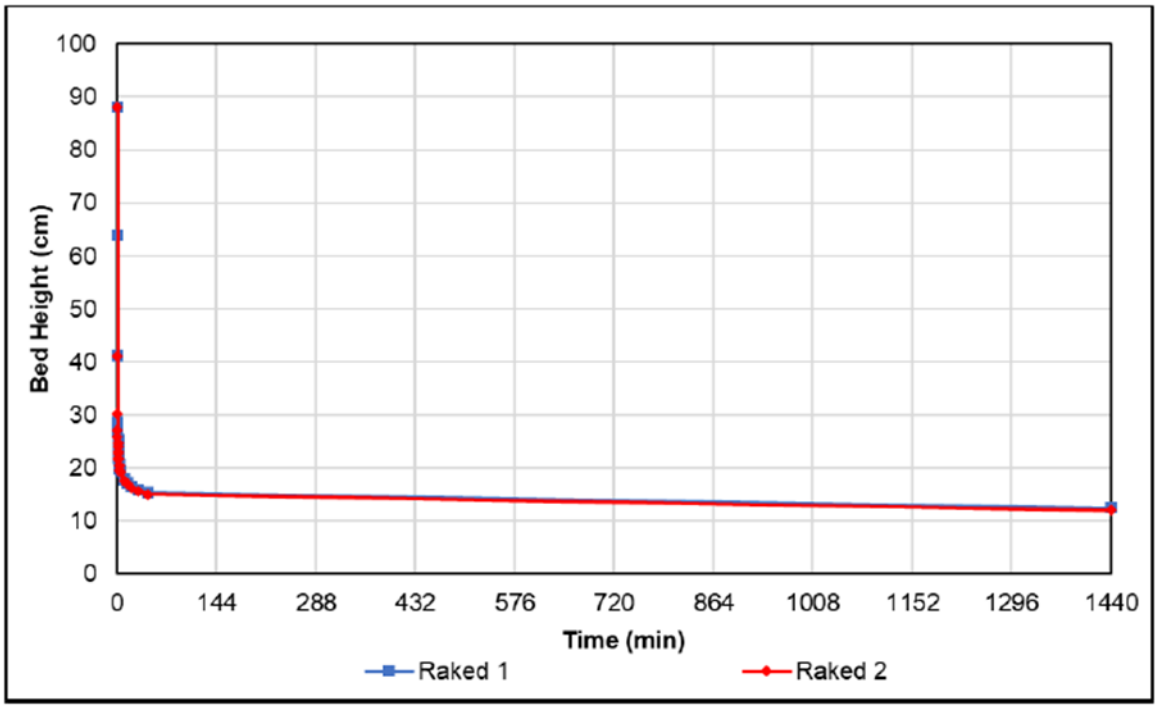
Based on the parameters established during previous testing, two (2) 4 L static thickening cylinders were utilised to scale up the process, validate earlier results, and assess the impact of raking on underflow compaction. The contents of the cylinders were homogenised and allowed to settle while monitoring the bed volume over time. After 45 minutes of settling, the cylinders were raked and then left to settle and compact for 24 hours. After the 24-hour period, the resulting plugs were removed, sheared, and tested for static yield stress and solids content. Table 13.36 summarises the static thickening results and Figure 13.38 and Figure 13.39 show bed height and underflow solids vs. time.

Table 13.36 – Static Thickening Results Summary

Sample	**Feed Solids** (wt%)	**Flocculant Dose** (g/t)	**Bed Density** (wt%)	**Overflow TSS @ 45 min** (mg/L)	**Initial Settling Velocity** (m/h)	**Sheared Yield Stress** (Pa)
Raked 1	15	40	70	24	105	103
Raked 1	15	40		17	85	

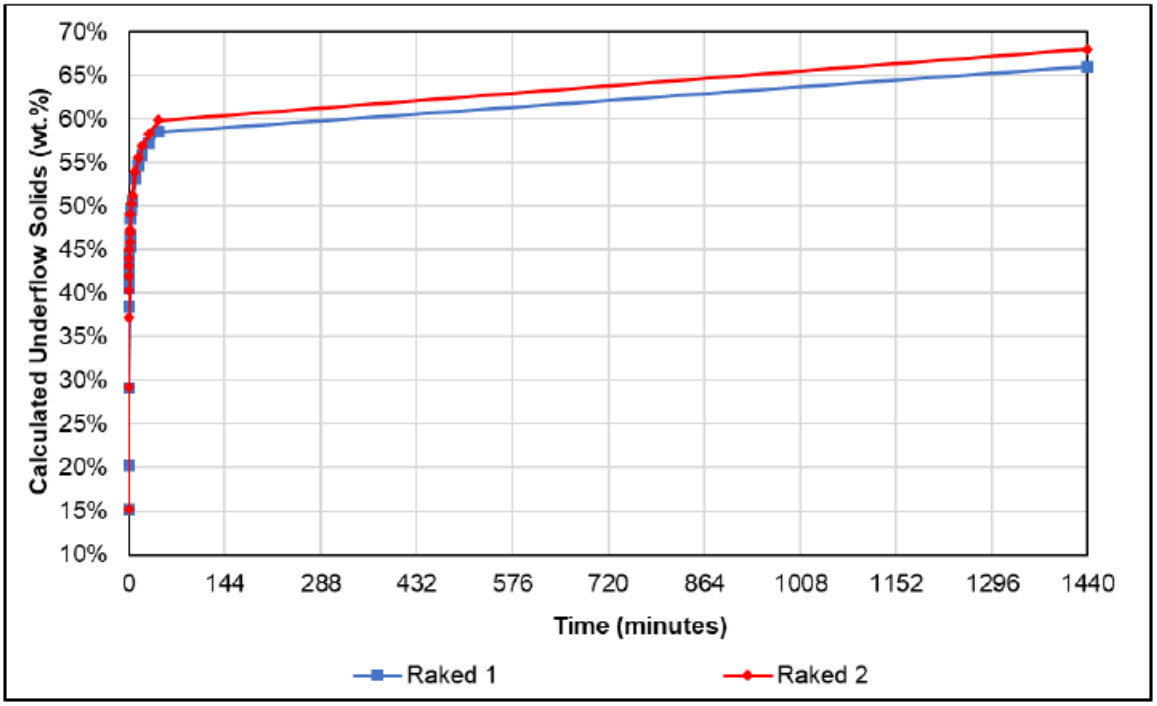
Source: RMS, 2024

Figure 13.38 – Bed Height vs. Time



Source: RMS, 2024

Figure 13.39 – Calculated Underflow Solids vs. Time



Source: RMS, 2024

13.6.8 DYNAMIC THICKENING

Dynamic thickening was carried out using a 102 mm High-Rate Thickener test rig, applying parameters identified in the static screening tests. The feed stream consisted of a low-density slurry, prepared to the desired feed solids content, which was continuously pumped at a controlled rate into the thickener feed well. Inside the thickener, the settled material was continuously raked to simulate the raking process used in dewatering. The test concluded once the bed height reached 240 mm, after which a subsample of the underflow was collected to measure the unsheared yield stress and solids content.

Testing was conducted at four distinct flux rates ranging from 0.5 to 1.3 t/m²/h with a flocculant dosage of 40 g/t. An additional test was performed at 50 g/t to assess the effect of an increased flocculant dosage. Results showed that increasing the flocculant dosage had little effect on settling properties. Another test, using lower-density feed at 10 wt% solids yielded similar results to that of the tests at 15 wt% solids.

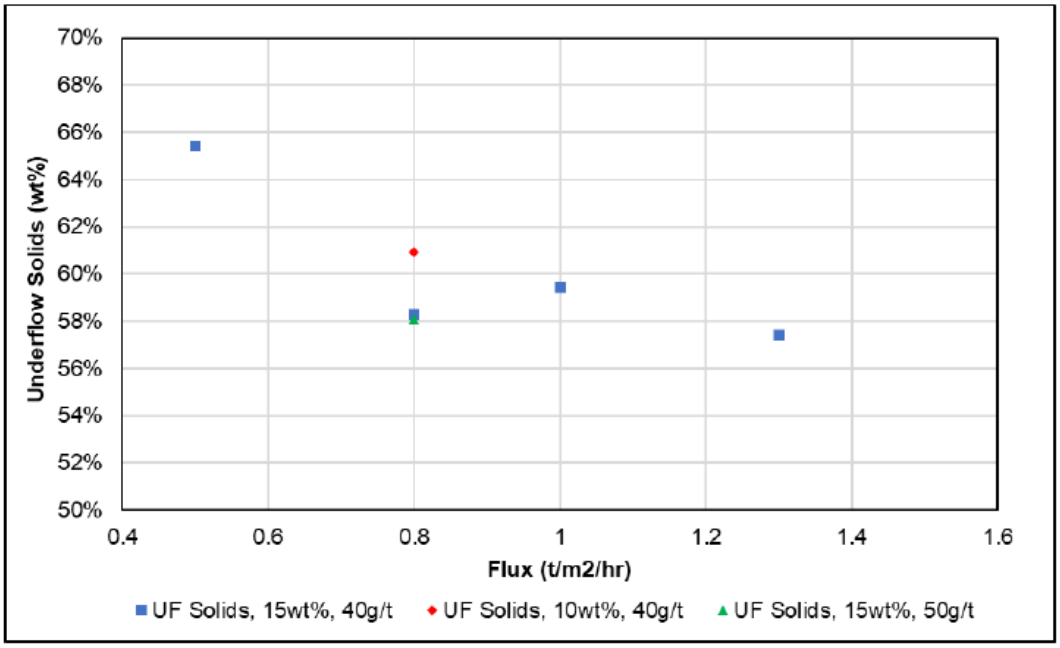
All tests were conducted at a pH of 7.5 using AN 923 VHM. The results of the testing are summarised in Table 13.37 and Figures 13.4 and 13.41.

Table 13.37 – Dynamic Thickening Results Summary

Run	Flux Rate(t/m²/h)	Feed Solids (wt%)	Flocc Dosage (g/t)	UF Solids (wt%)	Final TSS (ppm)	Unsheared Yield Stress (Pa)
1	1.0	15	40	59	136	28
2	1.3	15	40	57	84	27
3	0.8	15	40	58	53	25
4	0.5	15	40	65	69	71
5	0.8	15	50	58	54	31
6	0.8	10	40	61	46	35

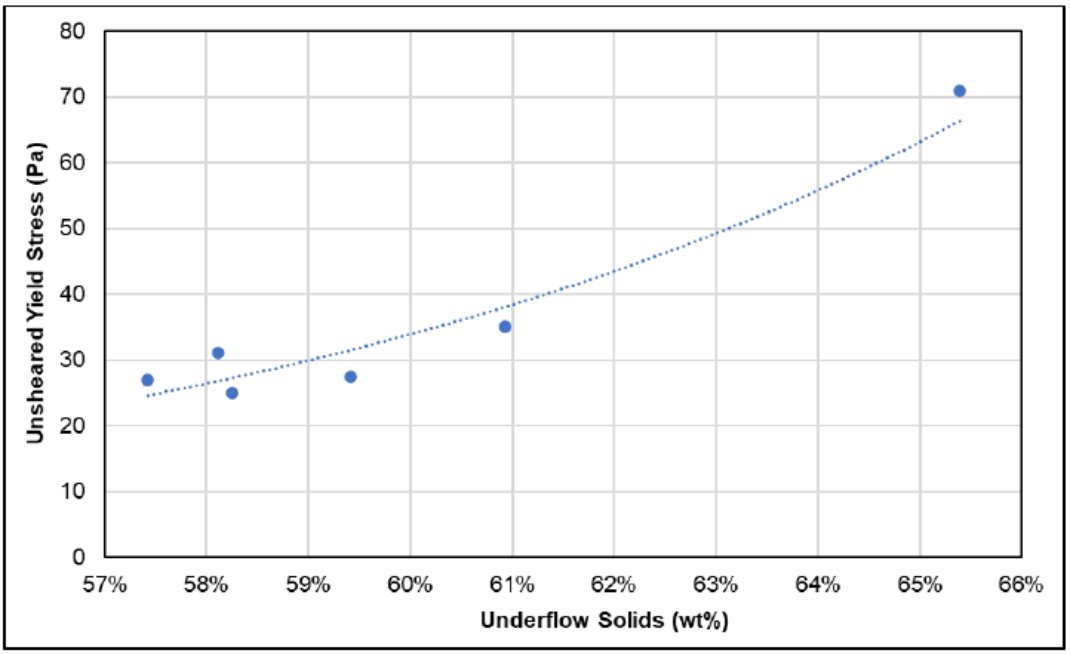
Source: RMS, 2024

Figure 13.40 – Underflow Densities and Overflow Solids Contents vs. Flux Rate



Source: RMS, 2024

Figure 13.41 – Unsheared Yield Stress vs. Underflow



Source: RMS, 2024

13.6.9 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 13.38 shows a summary of this test.

Table 13.38 – Membrane Screening Results Summary

Cloth ID	Cloth Type	Feed Solids (wt%)	Vacuum (bar)	Cycle Time (sec)	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

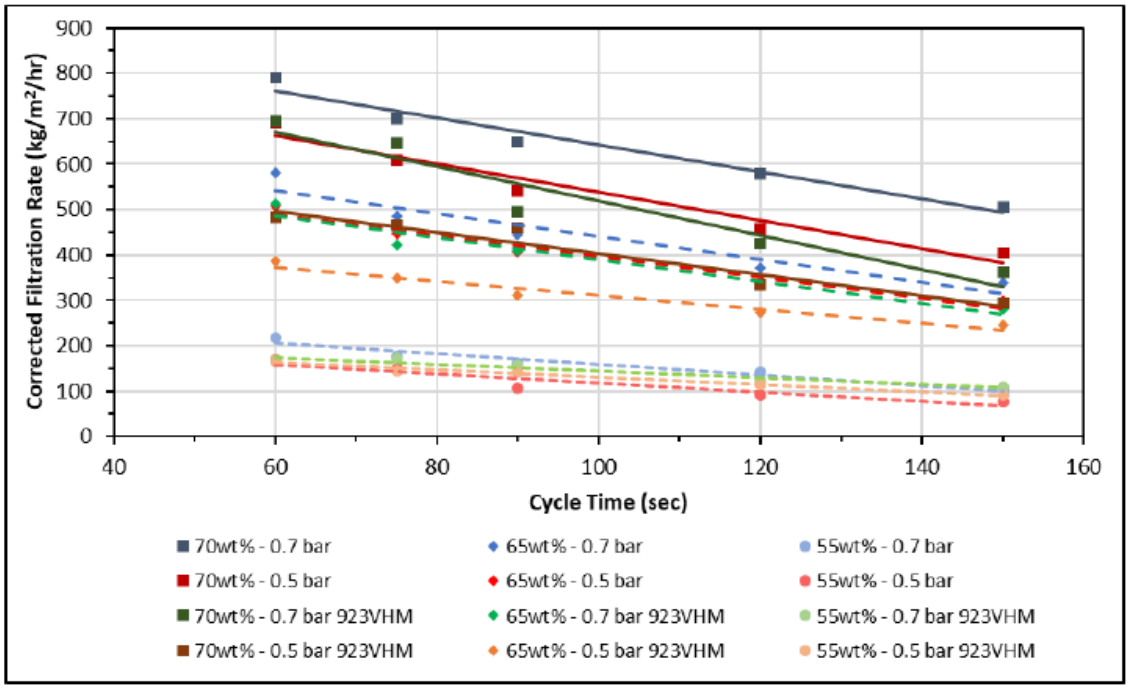
Source: RMS, 2024

13.6.9.1 Vacuum Disk Filtration

Vacuum disc filtration testing was conducted at three (3) 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.

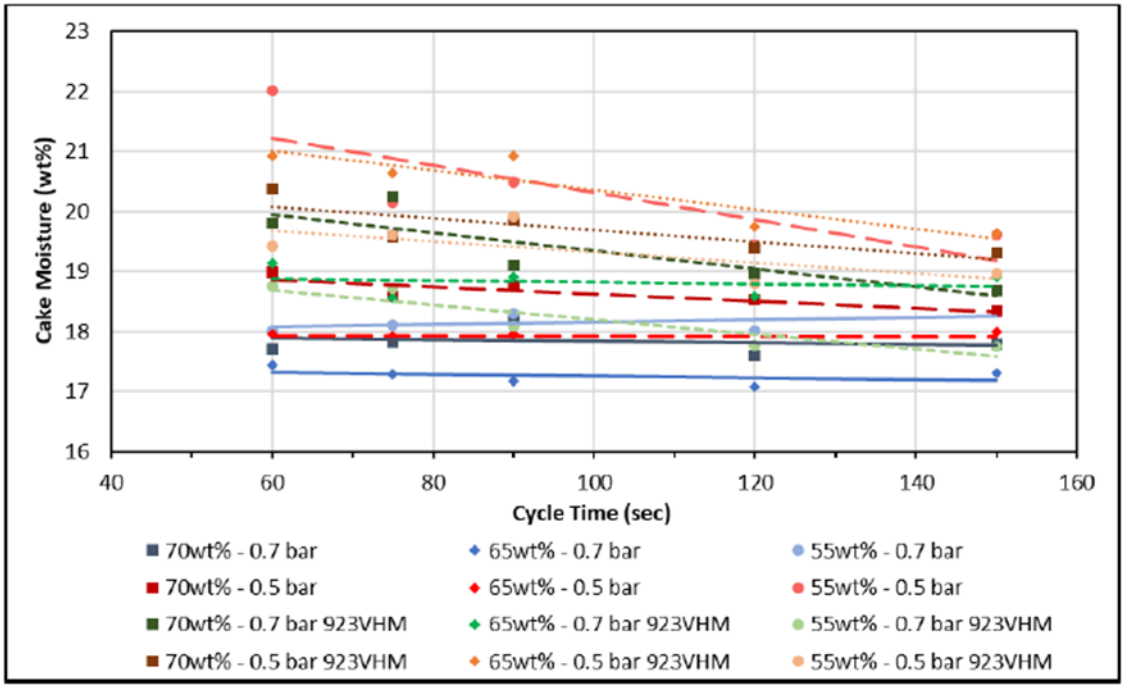
These results can be used for equipment sizing, including thickeners, feed tanks, filter sizes, and determining vacuum pump and auxiliary equipment needs. The data provided a range of values for efficiency, power consumption, and desired cake production rates. A vacuum pressure of 0.7 bar, corrected 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, 65, and 55 wt% solids, a laboratory temperature of 20°C ± 3, and a pH of 7.5. Results are summarised in Figure 13.42, and Figure 13.43, with a 5 mm cake thickness identified as ideal for disc filters.

Figure 13.42 – Corrected Filtration Rate vs. Cycle Time



Source: RMS, 2024

Figure 13.43 – Cake Moisture vs. Cycle Time



Source: RMS, 2024

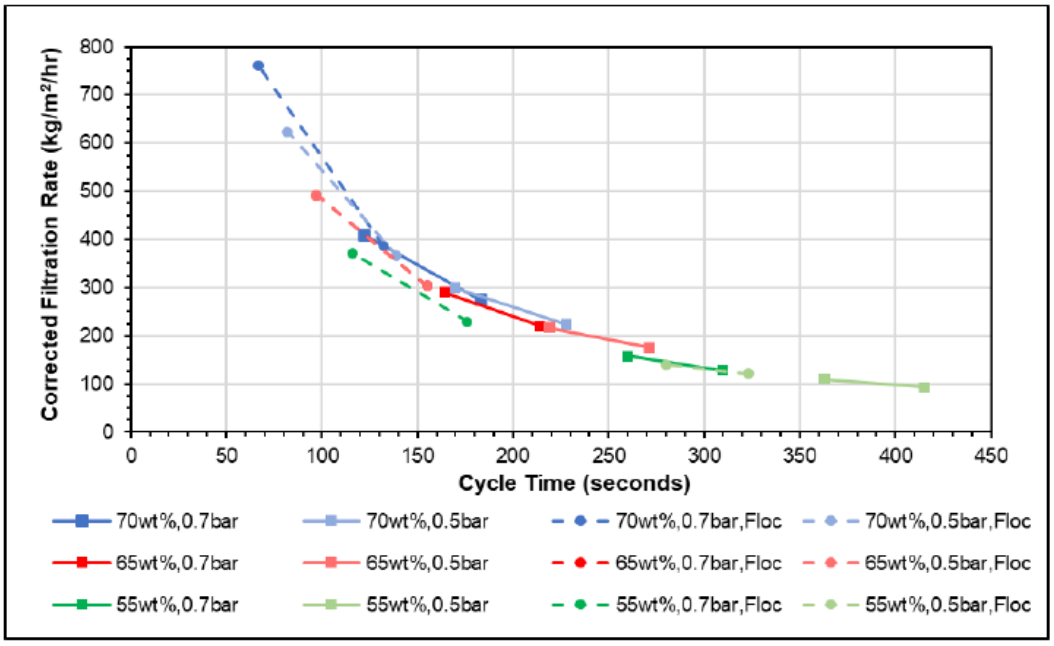
The vacuum disc filtration tests show that performance is adequate to good when the filter feed density is at least 65 wt%. However, at 55 wt%, the filtration results were poor, with the least dry filter cakes, suggesting challenges at lower densities. The use of a filtration aid (flocculant) did not 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.

13.6.9.2 Vacuum Horizontal Belt Filtration

Horizontal belt filtration testing was performed at three (3) 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, 200, and 100 g were selected to analyse varying cake thicknesses. Two (2) drying times (90 and 30 seconds) were tested, with vacuum pressures of 0.7 and 0.5 bar, adjusted for site elevation and an 80% efficiency factor.

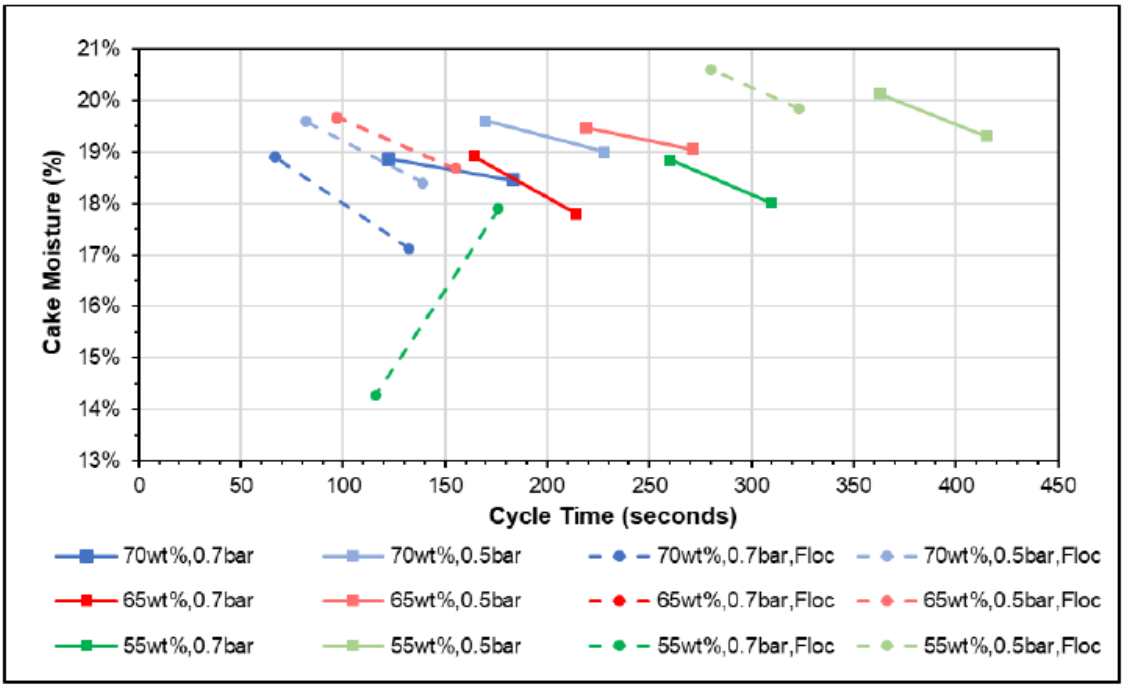
The filter feed densities increments were 70, 65, and 55 wt. The filtration cloth used was a Sefar 05-678-122-NFG, and a flocculant (AN 923 VHM) was added at 40 g/t. Filtration capacities were derated by 70% to account for realistic continuous production conditions on a belt filter. Results for corrected filtration rates and cake moisture for the different subsample sizes (300 g, 200 g, and 100 g) are shown in Figure 13.44 to Figure 13.49, with longer cycle times associated with the 90-second dry time and shorter cycle times with the 30-second dry time.

Figure 13.44 – Corrected Filtration Rate vs. Cycle Time for 300 g Samples



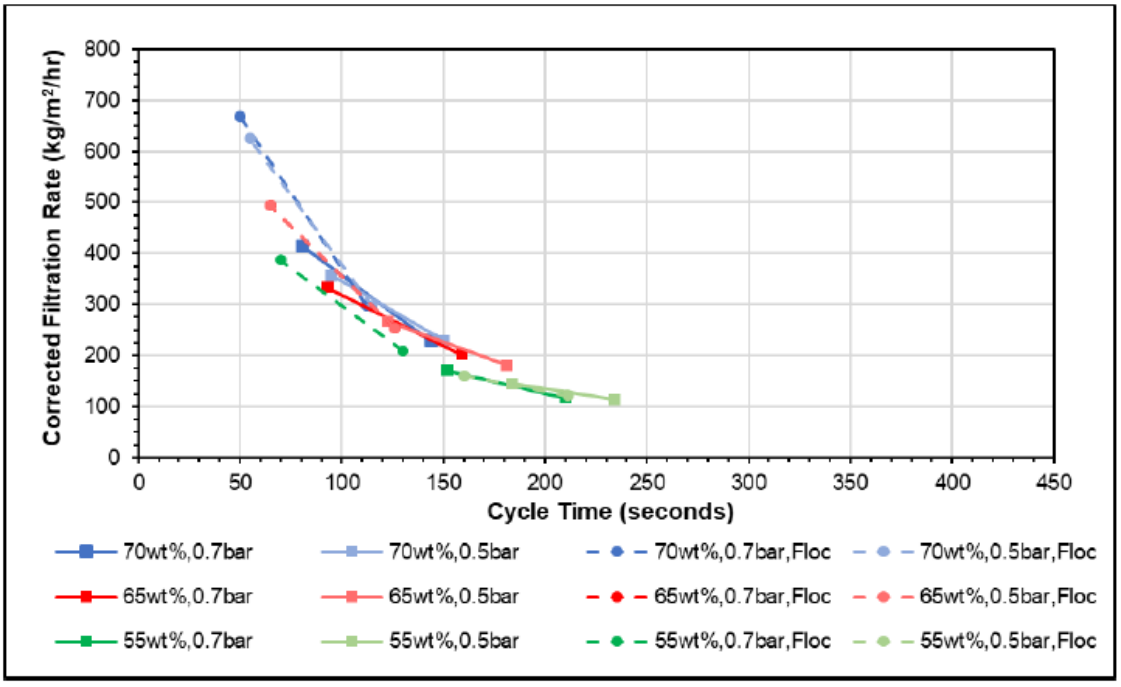
Source: RMS, 2024

Figure 13.45 – Cake Moisture vs. Cycle Time for 300 g Samples



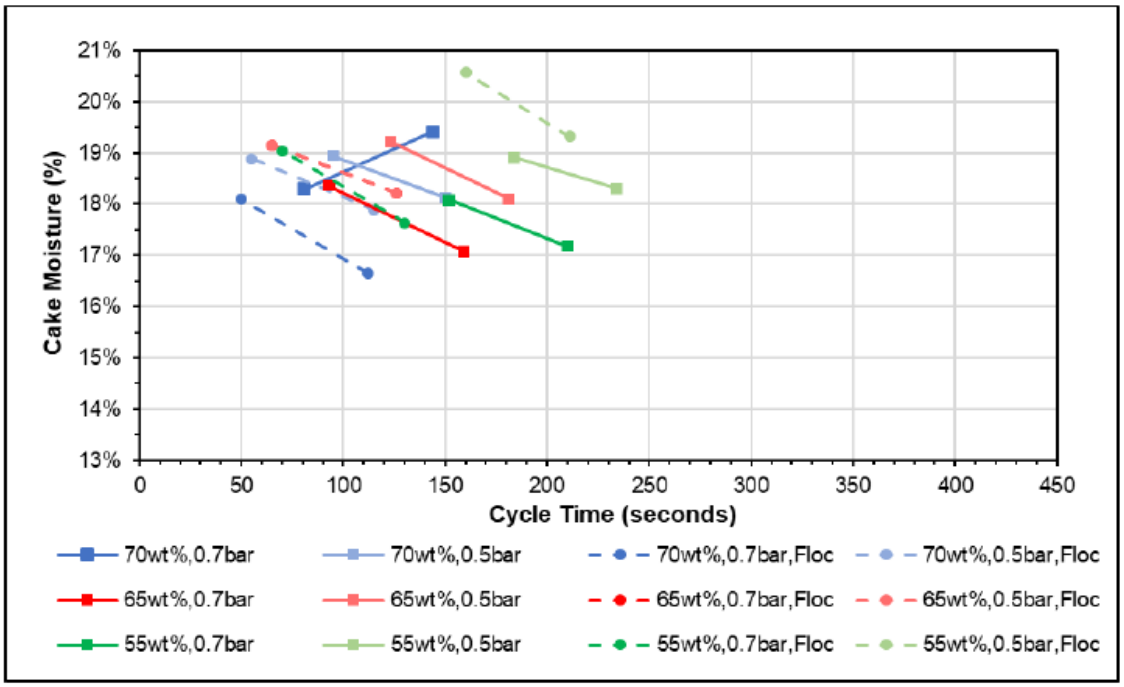
Source: RMS, 2024

Figure 13.46 – Corrected Filtration Rate vs. Cycle Time for 200 g Samples



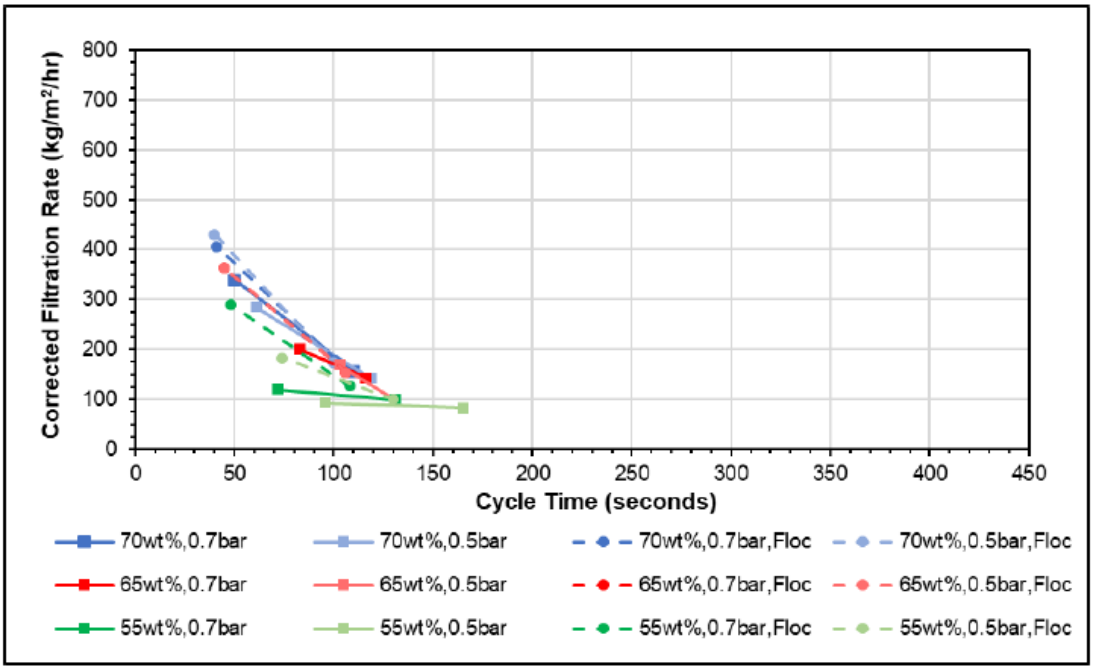
Source: RMS, 2024

Figure 13.47 – Cake Moisture vs. Cycle Time for 200 g Samples



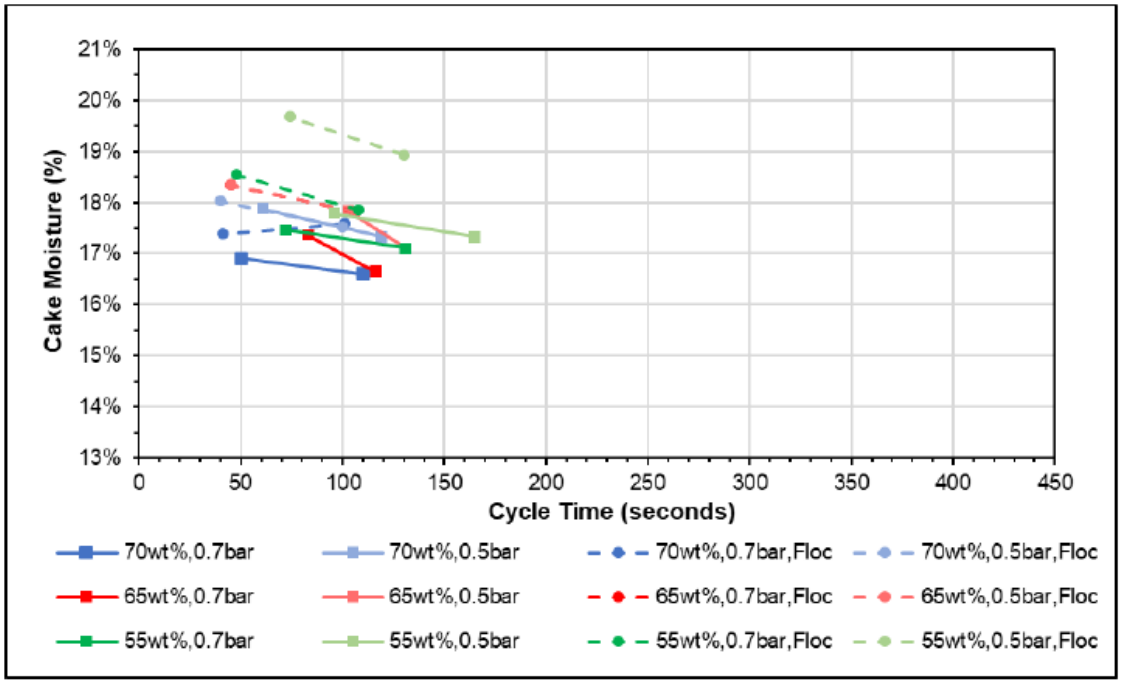
Source: RMS, 2024

Figure 13.48 – Corrected Filtration Rate vs. Cycle Time for 100 g Samples



Source: RMS, 2024

Figure 13.49 – Cake Moisture vs. Cycle Time for 100 g Samples



Source: RMS, 2024

The graph shows an inverse relationship between filtration rate and cycle time with all of the points on a similar curve which indicates that the operating point shifts on this curve depending on the conditions applied. Increasing the density was most impactful, shifting the operating point to the left (i.e., lower cycle time and higher filtration rate). Adding flocculant and increasing the vacuum pressure was also beneficial, as expected.

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 filtration aid (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.

13.6.10 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 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 13.39 summarises the desiccation environments and parameters.

Table 13.39 – 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.

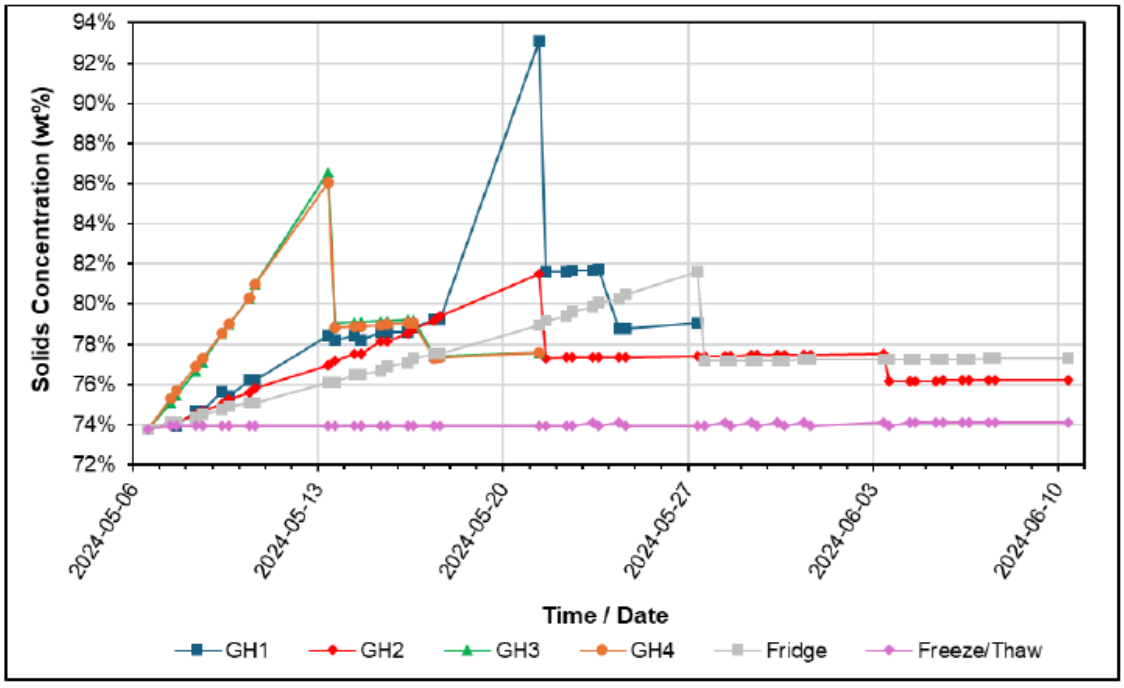
Cold jointing was observed between the layers when analysing the desiccated material after testing was completed. It is important to note that the layers were not altered or compressed after deposition. No material segregation was detected either during the deposition of the layers or upon visual inspection over time. Table 13.40, Figure 13.50 and Figure 13.51 show the lifts summary of samples and its relation to time.

Table 13.40 – Lifts Summary

Lift #	GH1	GH2	GH3	GH4	FR	FT
1 (Start Date)	06 May 24	06 May 24	06 May 24	06 May 24	06 May 24	06 May 24
2	21 May 24	21 May 24	13 May 24	13 May 24	27 May 24	-
3	23 May 24	03 Jun 24	16 May 24	16 May 24	-	-
End Date	27 May 24	10 Jun 24	21 May 24	21 May 24	-	-

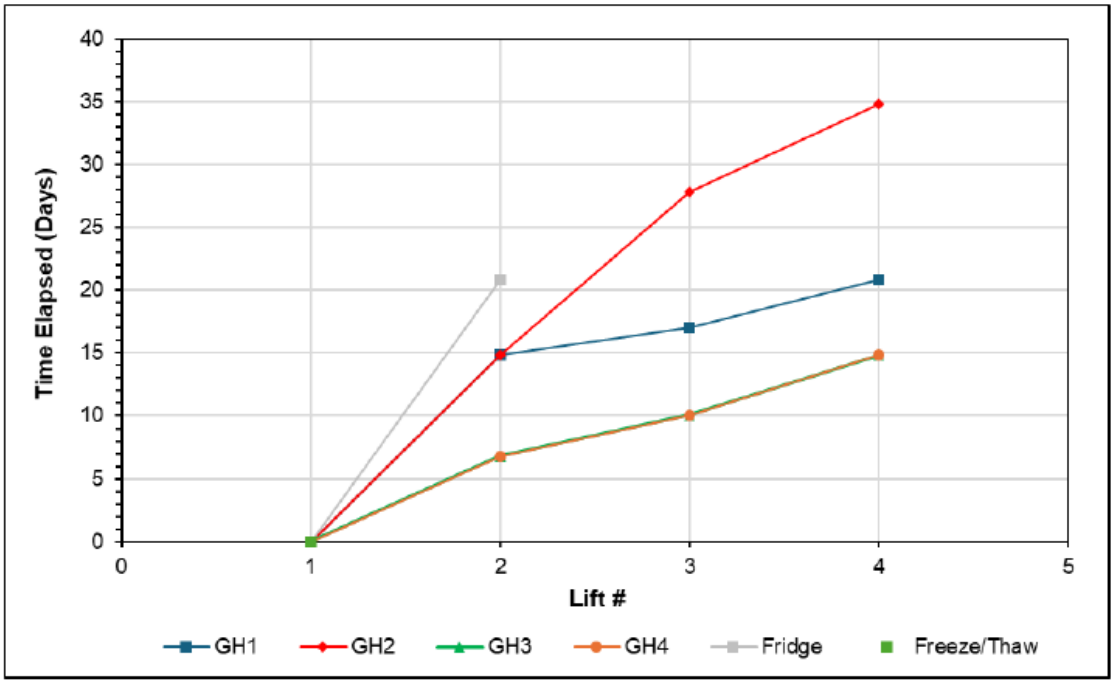
Source: RMS, 2024

Figure 13.50 – Solids Concentration vs. Time



Source: RMS, 2024

Figure 13.51 – Lift Number vs. Time Elapsed



Source: RMS, 2024

13.6.11 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 13.41 and Figure 13.52 suggest that the GGIBFS II blend is an optimal solution for producing high-strength paste backfill for underground mine operations.

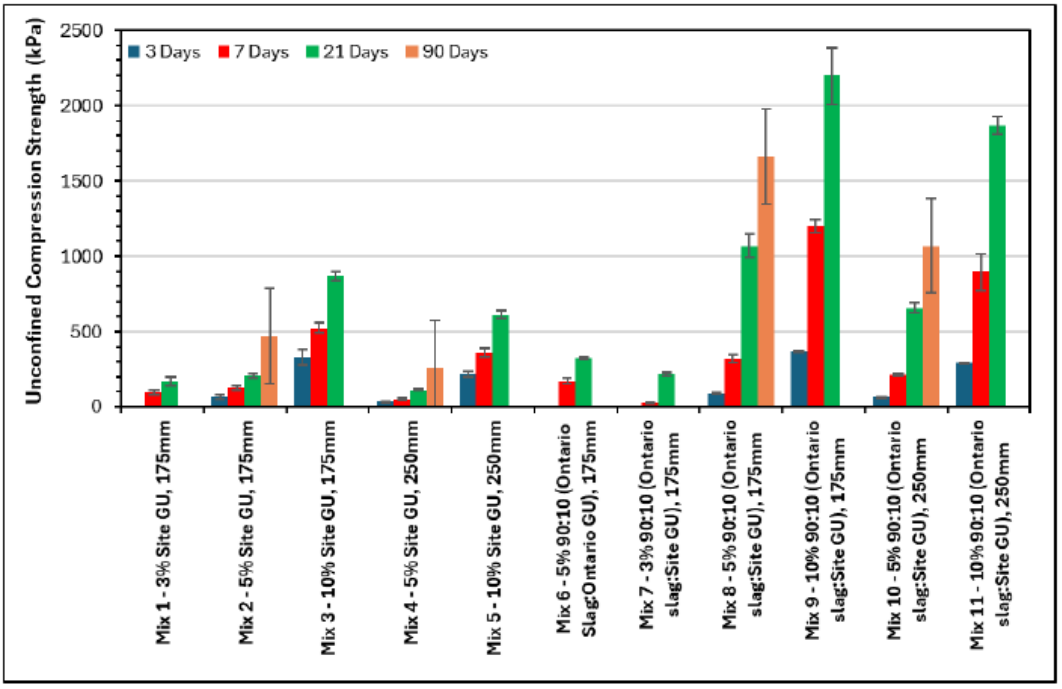
Table 13.41 – UCS Test Results Summary

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

Mix #	Binder	Slump (mm)	Binder Conc (wt%)	Curing Period (Days)			
				3	7	21	90
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 13.52 – UCS Results Summary



Source: RMS, 2024

13.7 BaseMet Labs FS Testwork Program 2025

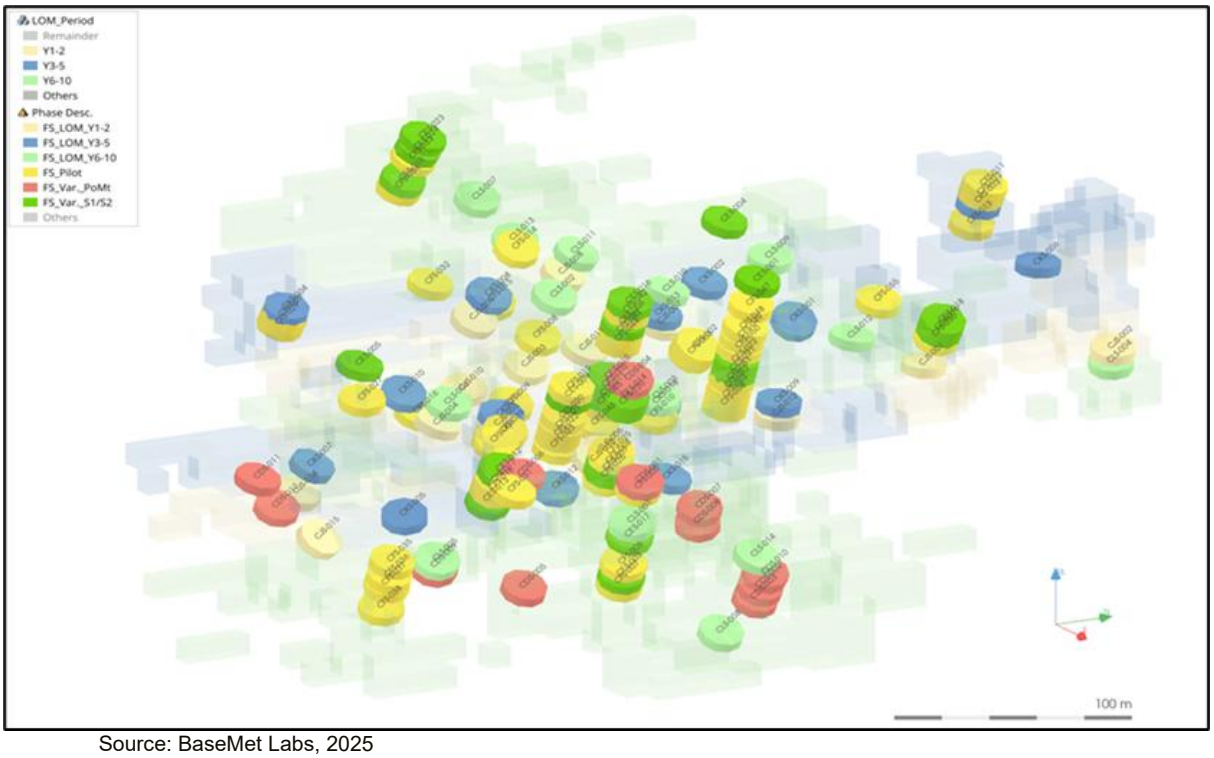
13.7.1 Sample Selection

The purpose of the FS testwork program was, primarily, to understand how the metallurgical performance of the ores will vary over the course of the mine's life. For this purpose, three (3) LOM composite samples were selected to represent the initial two (2) years of operation, the middle years (Years 3 to 5) and the final years (from Year 6 to 10). Another Master Composite (MC) sample,

representing the entire LOM, was produced for comparison and for Jameson cell pilot testing. These new LOM and MC samples were selected from the samples that were chosen for this testing program to represent the entire block model. In addition, 36 variability samples, which represent a wide range of the mine geological characteristics from both the Pyrrhotite-Magnetite (PoMt) and non-PoMt domains, were used in the variability test program.

The Cancha software package was used to confirm samples representativity. Figure 13.53 depicts the spatial distribution of the intervals selected for the pilot test, three (3) LOM periods and variability samples as an overlay on the block model.

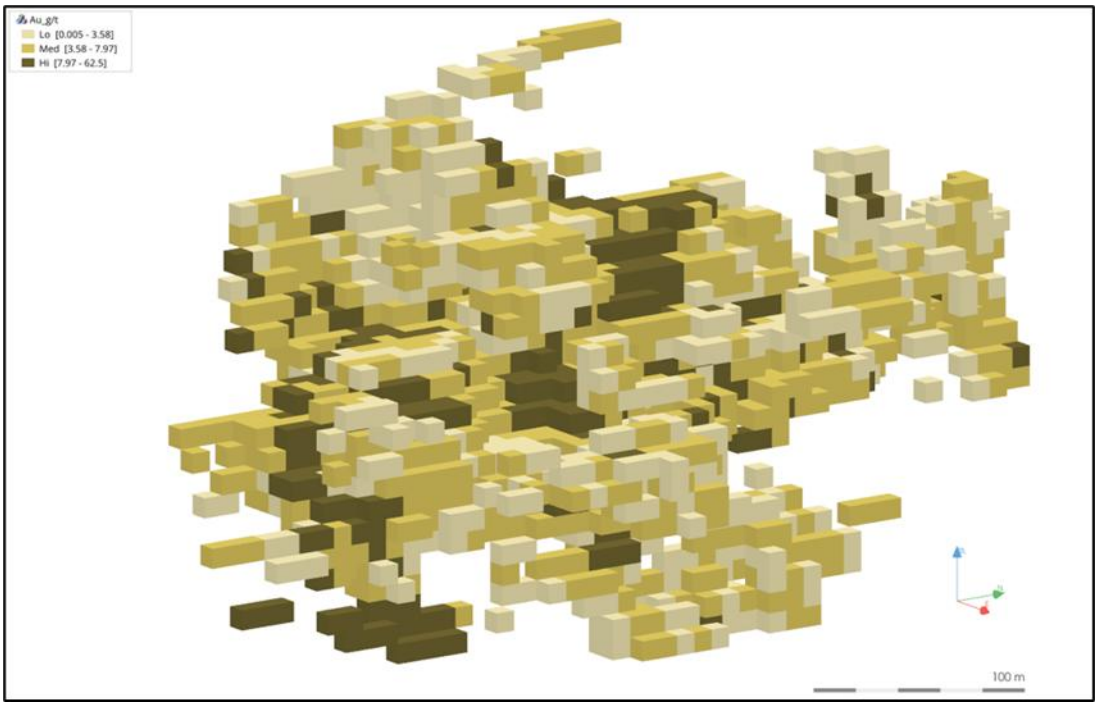
Figure 13.53 – Spatial Distribution of Intervals Selected Samples



Source: BaseMet Labs, 2025

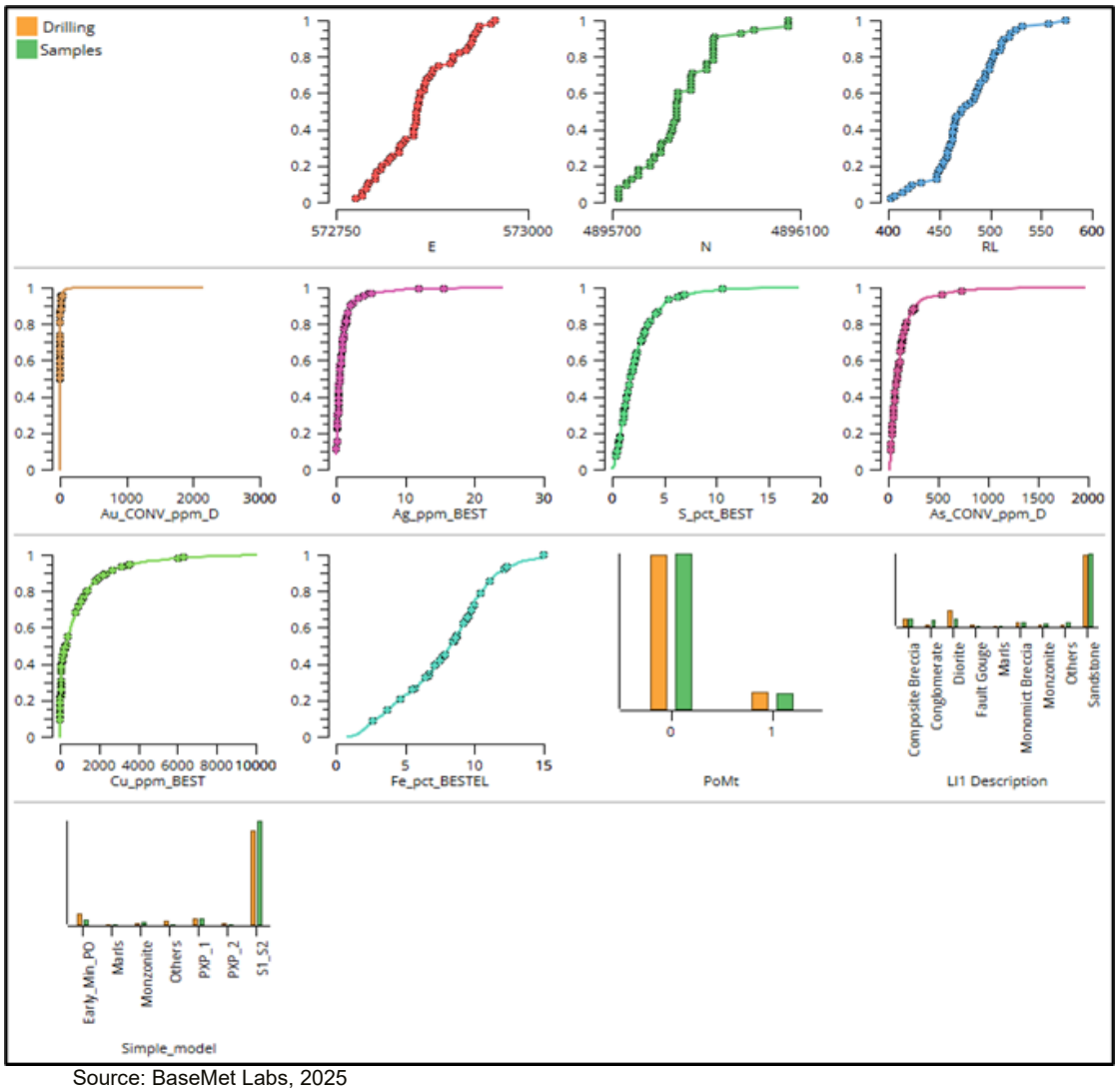
Figure 13.54 illustrates the block model of MC sample selection and Figure 13.55 plots the Cancha distribution graphs for various elements and spatial distribution of MC sample.

Figure 13.54 – Cancha Block Model for Master Composite Sample



Source: BaseMet Labs, 2025

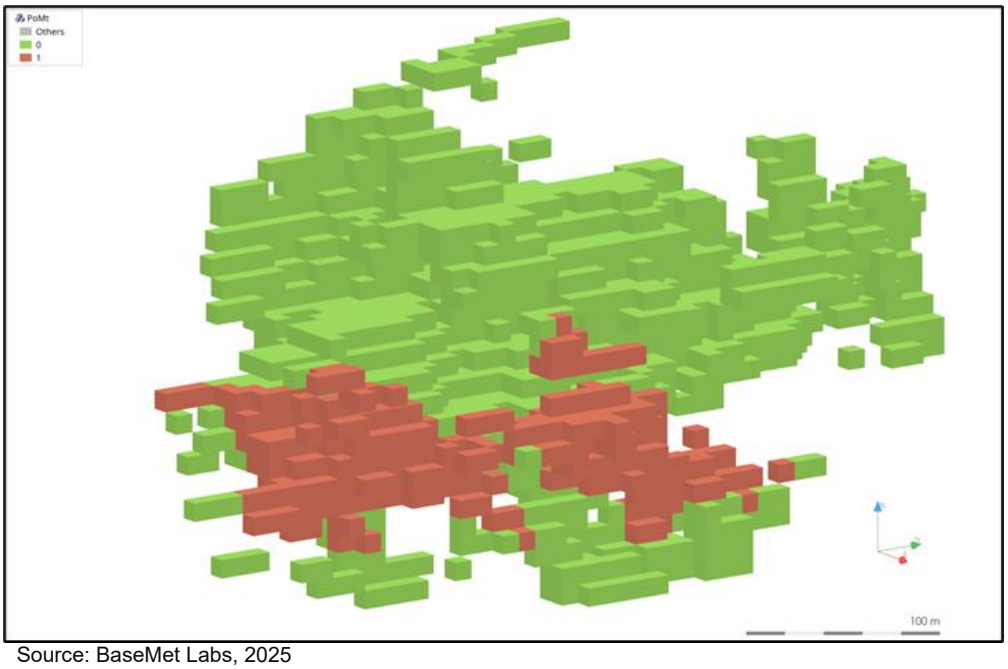
Figure 13.55 – Cancha Distribution Graphs for Master Composite Sample



Source: BaseMet Labs, 2025

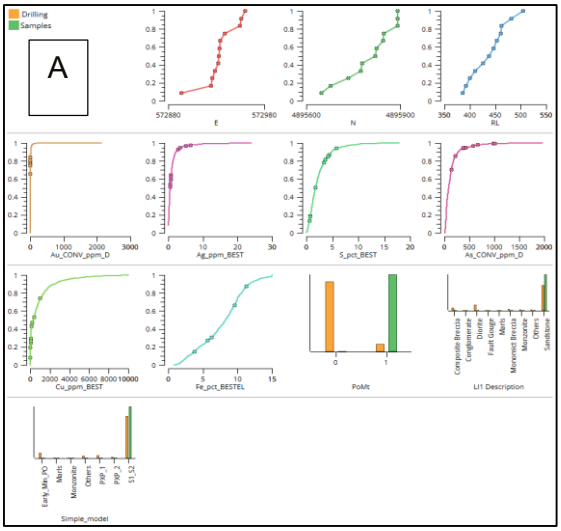
Figure 13.56 depicts the block model of variety samples (0 represents green for non PoMt; 1 represents brown for PoMt domain) and Figure 13.57 plots the Cancha distribution graphs for various elements and spatial distribution of variety samples (A for Pyrrhotite-Magnetite Mineralisation and B for Non Pyrrhotite-Magnetite Mineralisation).

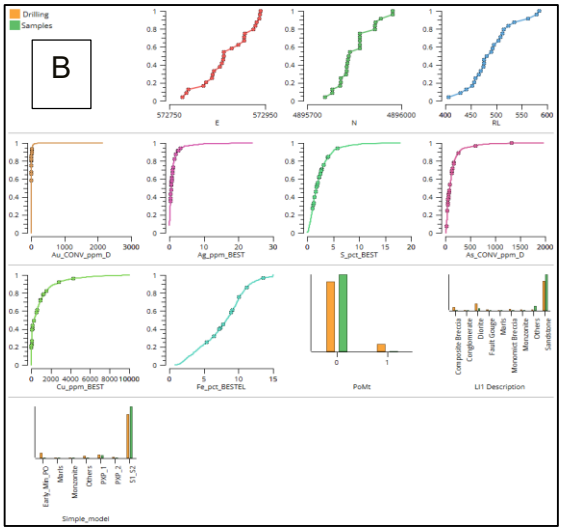
Figure 13.56 – Cancha Block Model for Variety Samples



Source: BaseMet Labs, 2025

Figure 13.57 – Cancha Distribution Graphs for Master Composite Sample

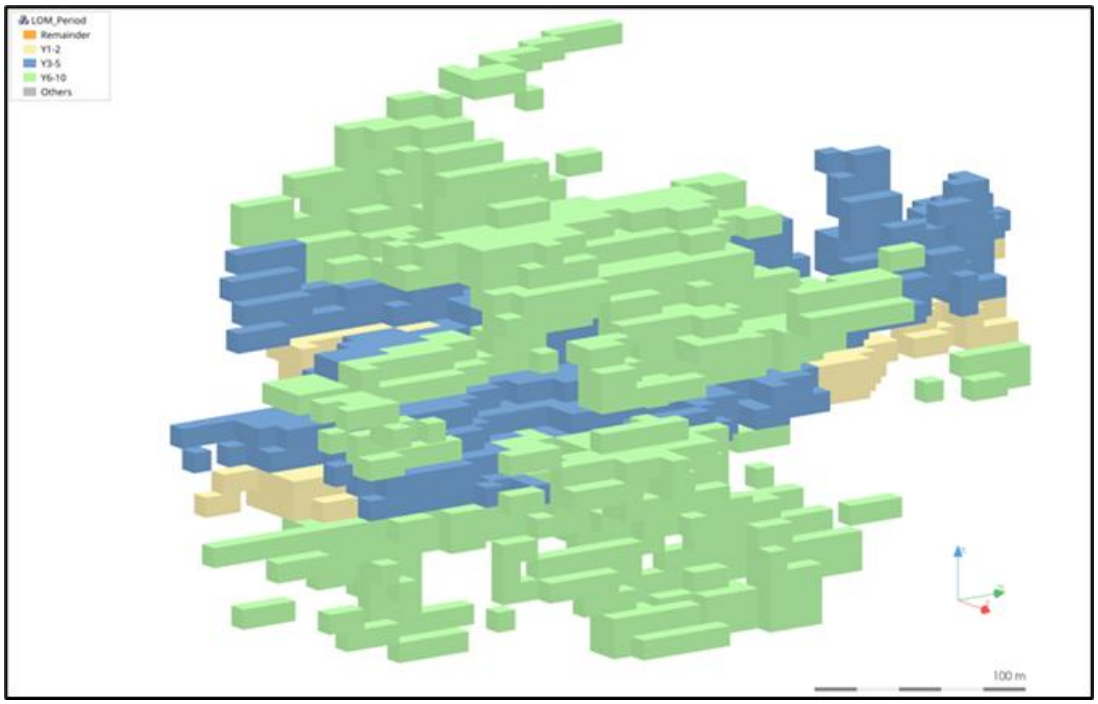




Source: BaseMet Labs, 2025

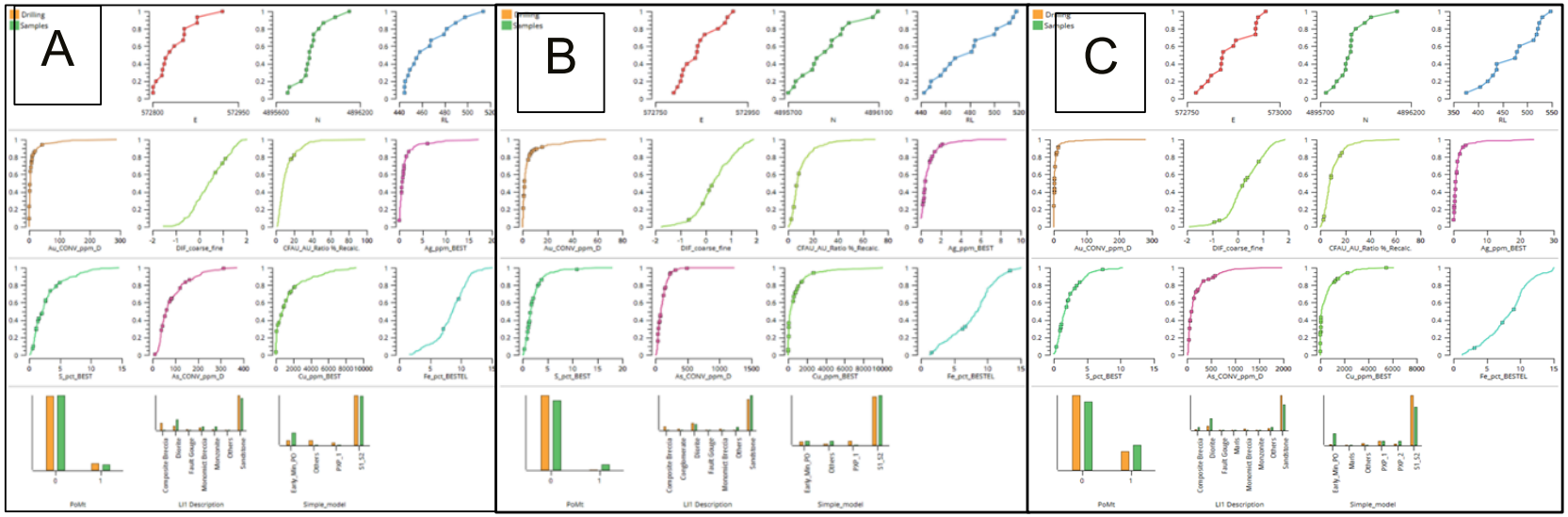
Figure 13.58 illustrates the block model of LOM periods (Yellow for Years 1-2; Blue for Years 3-5; and Green for Years 6-10) and Figure 13.59 plots the Cancha distribution graphs for various elements and spatial distribution of LOM periods samples (A for Years 1-2 and B for Years 3-5 and C for Years 6-10).

Figure 13.58 – Cancha Block for LOM Periods



Source: BaseMet Labs, 2025

Figure 13.59 – Cancha Distribution Graphs for LOM Periods Samples



Source: BaseMet Labs, 2025

13.7.2 HEAD ANALYSIS

Table 13.42 presents the head analyses performed on all 36 head samples prior to compositing. Three (3) different methods were used, namely flame atomic absorption spectrometry (FAAS) for gold and iron contents, inductively couple plasma (ICP) for arsenic and silver contents, and LECO for sulphur content.

Table 13.42 – Head Analysis for FS Testwork (2025)

	Element	Au	S	As	Fe	Ag
	Method	**FAAS**	**LECO**	**ICP**	**FAAS**	**ICP**
	Conc Unit	g/t	%	g/t	%	g/t
Sample ID	Var-01 Hd 1	4.61	5.25	564	9.48	2.5
	Var-02 Hd 1	5.69	0.72	254	3.24	0.5
	Var-03 Hd 1	4.68	3.54	341	4.68	0.9
	Var-04 Hd 1	3.71	4.71	392	8.12	5.7
	Var-05 Hd 1	6.01	2.17	277	5.24	1.2
	Var-06 Hd 1	4.25	8.40	884	11.1	1.3
	Var-07 Hd 1	5.45	3.82	817	7.20	3.1
	Var-08 Hd 1	8.68	4.14	928	8.00	4.7
	Var-09 Hd 1	2.50	0.53	102	5.92	0.5
	Var-10 Hd 1	2.60	0.89	255	5.64	0.6
	Var-11 Hd 1	4.09	5.05	299	14.3	6.0
	Var-12 Hd 1	3.02	0.84	78	7.56	0.5
	Var-13 Hd 1	2.58	4.13	150	10.7	2.8
	Var-14 Hd 1	1.87	1.71	36	7.20	0.8
	Var-15 Hd 1	6.81	2.69	152	6.12	2.3
	Var-16 Hd 1	20.5	2.55	54	4.80	0.9
	Var-17 Hd 1	12.0	2.46	65	6.60	0.9
	Var-18 Hd 1	1.77	1.66	63	10.3	0.3
	Var-19 Hd 1	3.59	1.77	86	7.36	0.4
	Var-20 Hd 1	5.78	1.88	85	9.60	0.5
	Var-21 Hd 1	10.4	1.74	50	5.72	0.5
	Var-22 Hd 1	5.87	1.17	31	7.40	0.7
	Var-23 Hd 1	6.25	1.21	38	7.84	0.7

	Element	Au	S	As	Fe	Ag
	Method	FAAS	LECO	ICP	FAAS	ICP
	Conc Unit	g/t	%	g/t	%	g/t
Sample ID	Var-24 Hd 1	4.84	2.99	99	6.76	0.6
	Var-25 Hd 1	7.16	3.46	85	4.80	2.2
	Var-26 Hd 1	4.71	5.87	342	8.84	6.5
	Var-27 Hd 1	3.10	4.60	51	7.84	3.7
	Var-28 Hd 1	14.2	2.67	48	11.6	1.2
	Var-29 Hd 1	8.01	1.42	204	7.56	0.7
	Var-30 Hd 1	5.93	5.90	930	11.6	1.8
	Var-31 Hd 1	10.1	3.32	440	26.3	1.5
	Var-32 Hd 1	5.19	4.24	309	7.56	4.4
	Var-33 Hd 1	3.24	1.45	39	6.88	0.7
	Var-34 Hd 1	11.0	1.06	113	1.96	1.3
	Var-35 Hd 1	7.01	4.12	54	9.20	1.5
	Var-36 Hd 1	5.37	2.47	35	8.36	2.5

Source: BaseMet Labs, 2025

The head gold grade of the master composite sample is 5.48 g/t. The head assay results for all four (4) samples are shown in Table 13.43.

Table 13.43 – Head Assay for Master Composite and Variety Samples (2025)

	Element	Au	Cu	S	As	Fe	Ag
	Method	FAAS	FAAS	LECO	ICP	FAAS	ICP
	Conc Unit	g/t	%	%	g/t	%	g/t
Sample ID	Master Comp	5.48	0.11	2.74	96	7.70	1.2
	Years 1-2 Composite	8.58	0.09	2.24	77	6.56	1.0
	Years 3-5 Composite	4.08	0.07	2.39	104	6.28	0.8
	Years 6-10 Composite	3.62	0.11	2.49	188	6.16	1.0

Source: BaseMet Labs, 2025

13.7.3 MINERALOGY

XRD analysis was performed on powdered sub-samples of the master composite, three (3) LOM composites and the final concentrate from the pilot test. The results of quantitative phase analysis by Rietveld refinements are given in Table 13.44.

Table 13.44 – Results of Quantitative Phase Analysis (wt.%)

Mineral	Master Comp (wt.%)	Years 1-2 Head Feed (wt.%)	Years 3-5 Head Feed (wt.%)	Years 6-10 Head Feed (wt.%)	Final Concentrate (wt.%)
Andradite – Grossular	54.3	49.2	54.0	44.6	20.7
Augite	9.1				5.6
Calcite	7.7	9.5	9.1	10.3	4.1
Chalcopyrite					2.3
K-feldspar (orthoclase)	4.7	7.8	6.1	3.9	1.4
Montmorillonite	1.7*	3.8	4.2	4.4	1.8*
Plagioclase (andesine)	3.7	5.4	4.9	7.8	2.3
Pyrite	3.6	3.1	3.6	3.6	53.8
Quartz	7.3	9.7	4.3	6.7	3.0
Stilbite	1.8	1.8	1.4	1.9	0.9

* Abundance calculated by PONKCS models

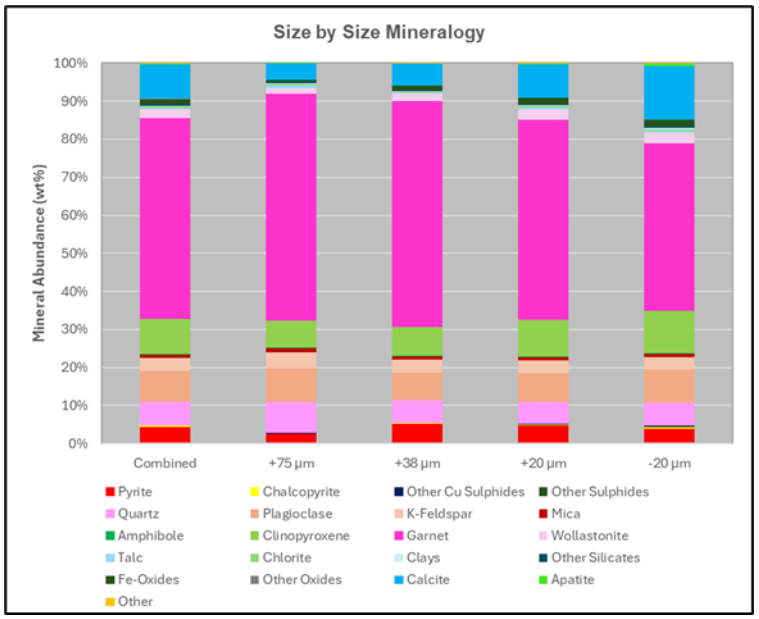
A QEMSCAN study was performed on the master composite sample to assess mineral abundance, liberation, and deportment. This analysis examined the distribution and associations of key minerals, focusing on size-by-size mineralogy, copper and sulphur deportment, and the liberation of pyrite and copper sulphides.

Highlights from the size-by-size mineralogy are as follows:

- The dominant mineral present in all size fractions was Garnet at around 53% of the total mass.
- Calcite (9%), Clinopyroxene (9%), Plagioclase (8%) and Quartz (6%) are also significant.
- Pyrite was the only significant sulphide mineral at around 4.4%.
- Talc, Clays, and Chlorite were observed but were insignificant.

Figure 13.60 illustrates the mineral abundance results of Master Composite sample.

Figure 13.60 – Mineral Abundance Results



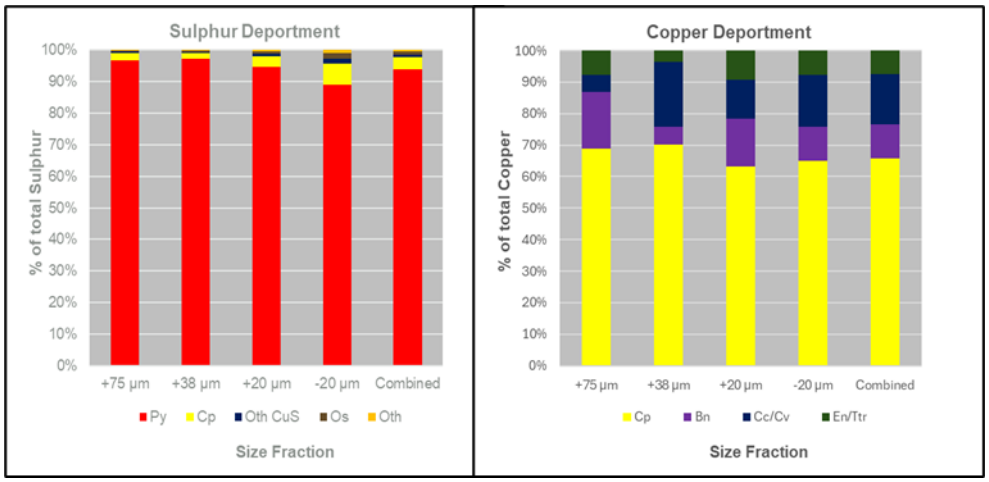
Source: BaseMet Labs, 2025

Highlights from the deportment study are as follows:

- The majority of sulphur is associated with pyrite (Py, red color) across all size fractions.
- A small percentage of sulphur is found in chalcopyrite.
- The sulphur distribution remains fairly consistent across different size fractions, indicating that pyrite is the dominant sulphur-bearing phase in this ore.
- Chalcopyrite is the major copper-bearing mineral in all size fractions, followed by bornite and then chalcocite/covellite, with minor contributions from enargite/Tetrahedrite.

Figure 13.61 presents the sulphur and copper deportment study results of Master Composite sample.

Figure 13.61 – Sulphur and Copper Deportment Results

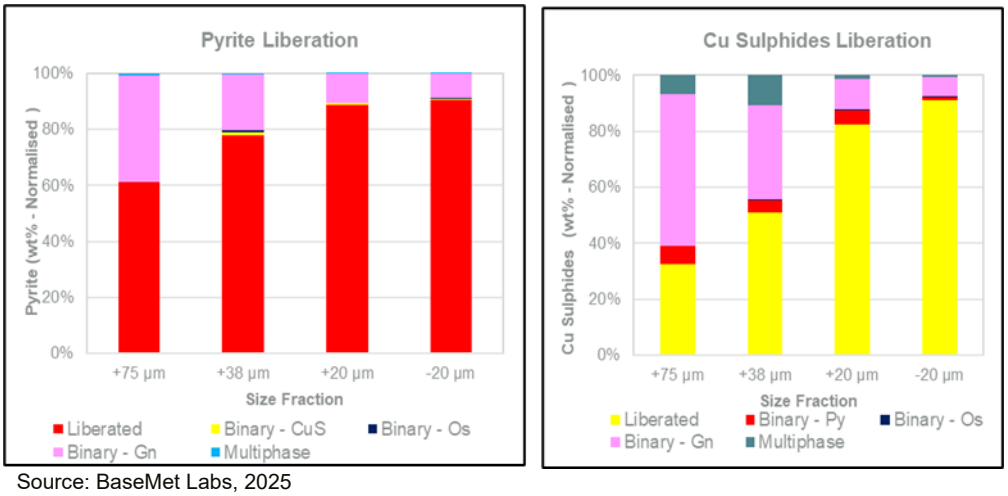


Source: BaseMet Labs, 2025

The pyrite and copper sulphide liberation study showed that the coarser fractions (+75 µm, +38 µm) have lower liberation compared to finer fractions (+20 µm, -20 µm). This suggests better liberation of pyrite at finer grind sizes.

Figure 13.62 illustrates the pyrite and copper sulphide liberation results for the Master Composite sample.

Figure 13.62 – Pyrite and Copper Sulphides Liberation Results



Source: BaseMet Labs, 2025

It is again evident how liberation improves with an increase in fineness. This confirms the observations made before during previous test programs on the need to grind to 80% passing 53 µm.

13.7.4 LASER ABLATION ANALYSIS

Laser ablation analysis of the pyrite showed the following distribution of metals:

Table 13.45 – Average Minor to Trace Element Concentrations Within Pyrite

Sample ID	Average Element Concentration within Pyrite (ppm)											
	Mn	Co	Ni	Cu	Zn	As	Ag	Cd	Sn	Sb	Au	Pb
Master Comp	4.09	413	337	5.56	1.39	421	1.47	0.02	0.11	6.60	0.13	12.98

The low gold concentrations indicates that there is very little refractory gold locked in pyrite. This is in agreement with similar ablation analysis performed during Phase 2 testwork.

13.7.5 COMMINUTION

13.7.5.1 Eliason Test

The Eliason test procedure was performed on a sub-sample of the master composite to determine the grinding energy required for fine grinding in a vertical stirred mill. The procedure consists of consecutive periods of grinding in a jar with fine media beads with intermittent analysis of the grind size at timed intervals. The sub-sample was pre-ground to a P_{80} of 207 µm which is approximately what is expected as the transfer size (T_{80}) from the primary AG milling circuit. Table 13.46 summarises the outcome of this test.

Table 13.46 – Eliason Secondary Grinding Test Results

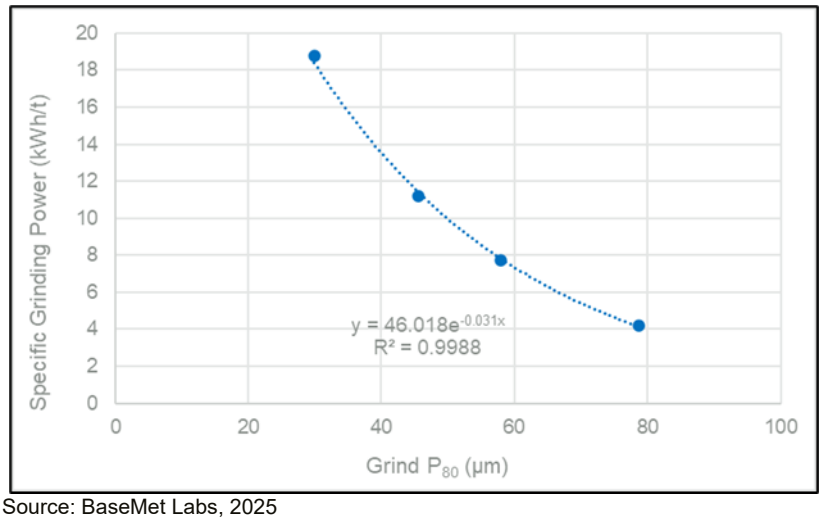
Grinding Energy (kWh/t)	P_{80} (µm)
4.2	78.7
7.7	58.0
11.2	45.5
18.8	29.9

Figure 13.63 illustrates the result graphically with an exponential empirical fit to the data. At the target grind of 53 µm the predicted specific energy consumption is 8.9 kWh/t. The modelling performed by DRA during the PFS yielded a power draw of 766 kW for the fresh feed throughput of 105 t/h, which results in a specific energy consumption of 7.3 kWh/t. This is approximately 23% lower than what the test results predict.

However, in scaling the test results to full scale an efficiency discount of 25% need to be applied to compensate for the fact that the full-scale circuit contains a classification step which removes correctly sized particles from the circuit which prevents overgrinding. The cyclone also scalps fines

from the fresh feed stream because the fresh feed reports to the cyclone pumpbox as it enters the secondary grinding circuit. Once this correction is applied the test result aligns very well with the design criteria established during the PFS. The implication is that this test supports and confirms the accuracy of the grinding circuit modelling performed during the PFS.

Figure 13.63 – Secondary Grinding Mill Specific Energy versus Grind P_{80}



Source: BaseMet Labs, 2025

13.7.5.2 Levin Test

The Levin test procedure was performed on a sub-sample of the Master Composite to estimate the specific grinding energy required for fine grinding in a regrind application. The procedure involves successive grinding stages in a standard ball mill using fine media, with periodic sampling to measure particle size distribution after defined energy inputs. The sub-sample was pre-ground to a P_{80} of 200 µm which is approximately what is expected as the transfer size (T_{80}) from the primary AG milling circuit. A series of grinding steps at energy levels of 5, 10, 15, and 20 kWh/t was completed, and particle size distributions were determined to assess the grindability response of the sample. Table 13.47 summarises the outcome of this test.

Table 13.47 – Levin Secondary Grinding Test Results

Grinding Energy (kWh/t)	**P_{80}** (µm)
5	112
10	78
15	60
20	51

13.7.6 GRAVITY CONCENTRATION

All four (4) composite samples underwent an EGRG procedure comprising staged recovery of gravity recoverable gold at increasingly finer grinds. Table 13.48 summarises the results for the concentrates collected at all stages of grinding.

For sample preparation 20-kg portion of each master composite and three (3) LOM samples were split and stage crushed to 100% passing 1,700 µm to produce a target K_{80} of around 1,200 µm. The EGRG test was conducted by passing the entire sample of crushed material through a Knelson MD-3 concentrator at a force of 60 G's. The concentrate was retained and sized for assaying. The tailings were ground in a laboratory rod mill for a second pass to a target K_{80} of around 200 to 250 µm. This material was then processed through the Knelson. The resulting concentrate was again screened and analysed as during the first pass; the tailings were ground to a target K_{80} of around 100 µm for a third pass. The procedure was repeated for a third pass. After Pass 3, the final tailings were directly sampled, sized, and assayed by size.

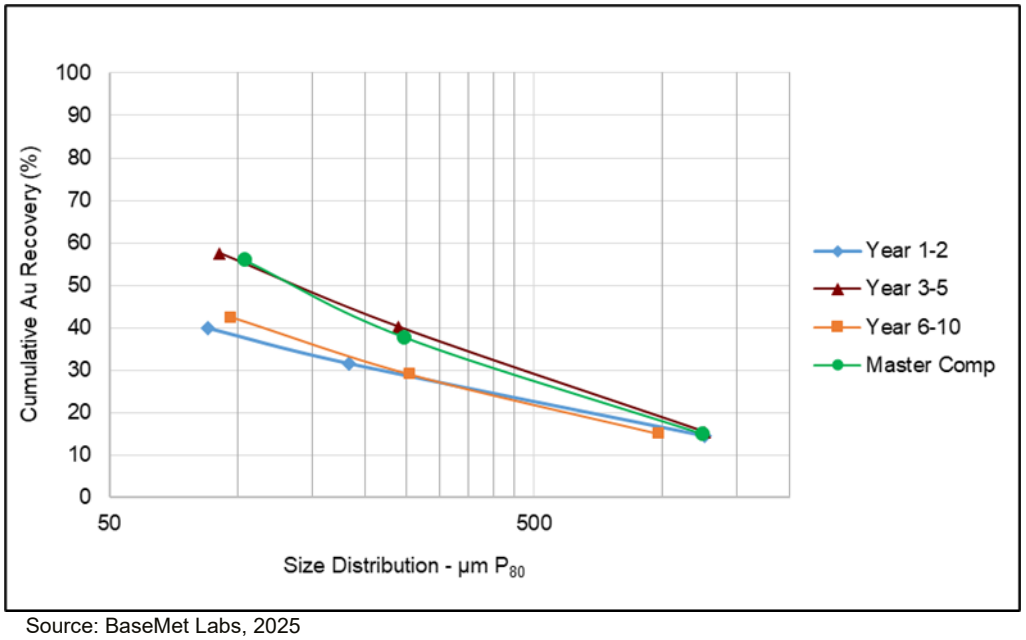
Table 13.48 – EGRG Results Summary (BaseMet Labs 2025)

Composite (Test)	**Product**	**Feed Size K_{80} (µm) per Stage**	**Mass** (%)	**Assay Au** (g/t)	**% Au Distribution**
Years 1-2 EGRG (T69)	Stage 1 Conc	1,257	0.52	226	14.6
	Stage 2 Conc	183	0.55	248	17.0
	Stage 3 Conc	85	0.49	135	8.3
	Tailings		98.4	4.89	60.1
	Comb Conc 1-3		1.56	205	39.9
	Feed (calc.)			8.01	
Years 3-5 EGRG (T70)	Stage 1 Conc	1,261	0.53	190	15.4
	Stage 2 Conc	239	0.57	285	24.9
	Stage 3 Conc	91	0.57	197	17.3
	Tailings		98.3	2.80	42.5
	Comb Conc 1-3		1.66	225	57.5
	Feed (calc.)			6.49	
Years 6-10 EGRG (T71)	Stage 1 Conc	986	0.50	135	15.0
	Stage 2 Conc	255	0.54	117	14.0
	Stage 3 Conc	97	0.55	112	13.5
	Tailings		98.4	2.65	57.6
	Comb Conc 1-3		1.59	121	42.4
	Feed (calc.)			4.53	

Composite (Test)	Product	Feed Size K_{80} (µm) per Stage	Mass (%)	Assay Au (g/t)	% Au Distribution
Master Comp EGRG (T21)	Stage 1 Conc	1,246	0.61	161	15.0
	Stage 2 Conc	247	0.55	274	22.8
	Stage 3 Conc	104	0.56	213	18.2
	Tailings		98.3	2.94	44.0
	Comb Conc 1-3		1.72	214	56.0
	Feed (calc.)			6.57	

Source: BaseMet Labs, 2025

Figure 13.64 – Gravity Separation Gold Recovery in Different Size Distribution



Source: BaseMet Labs, 2025

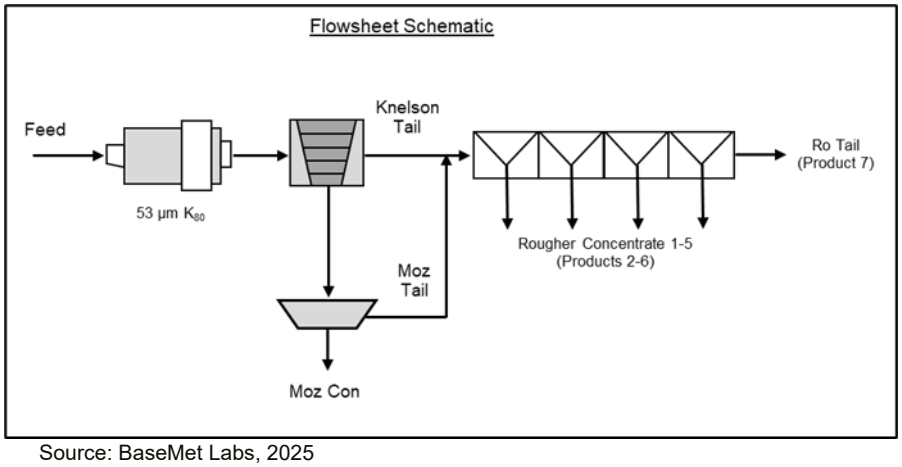
It is somewhat surprising that the Years 1-2 composite yielded the lowest gravity recovery at 39.9% given that the head grade for this sample is the highest at 8 g/t. Given the variability in gravity recoveries observed throughout the various testwork campaigns this is likely again due to the nugget effect. It is furthermore noteworthy that the gravity recoveries recorded for these four (4) composite samples are more aligned to the recoveries recorded in Phase 1 (around 40%) than the somewhat lower recoveries recorded for the variability samples of Phase 2 (approximately 36%).

13.7.7 Rougher Flotation Tests Program

BaseMet Labs subjected the master composite to rougher flotation tests (Figure 13.65) to investigate the effects of reagent dosage and retention time. In addition, one (1) sample was tested with water

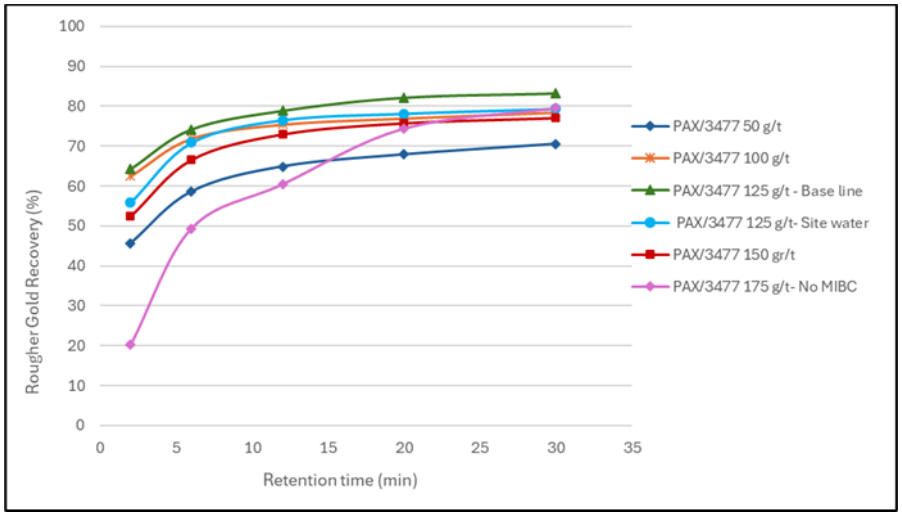
from the Čoka Rakita site to evaluate its effect on rougher efficiency. The results are shown in Figure 13.66.

Figure 13.65 – Schematic of Rougher Test



Source: BaseMet Labs, 2025

Figure 13.66 – Effect of Reagents Dosage on Rougher Efficiency



Source: BaseMet Labs, 2025

The results show that the highest recovery in the rougher stage was achieved by adding 125 g/t of each PAX and Aero3477 collectors. Using site water slightly decreased rougher recovery. The absence of MIBC resulted in slower flotation kinetics.

13.7.8 Flotation Tests Program – Optimisation

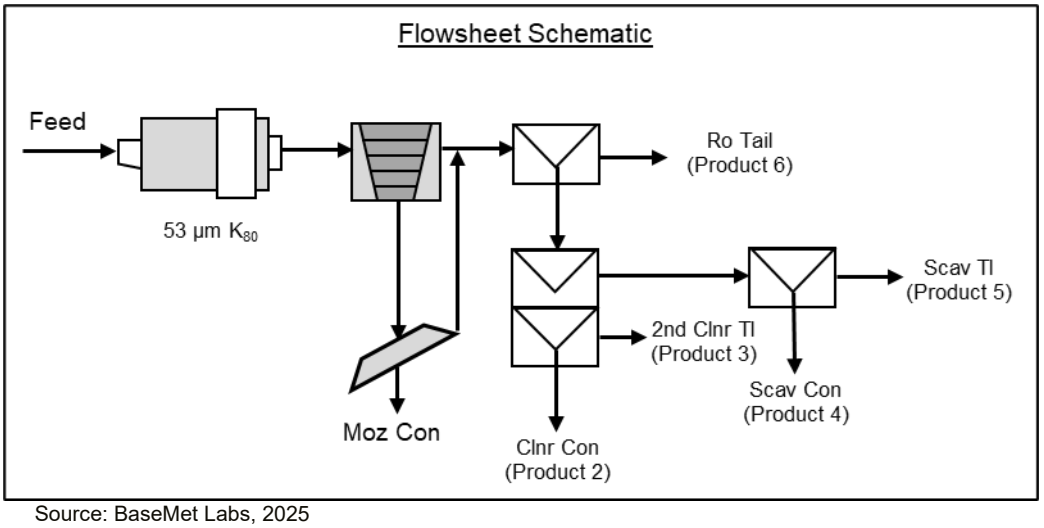
BaseMet Labs conducted flotation tests on sub-samples of the master composite, including gravity separation followed by rougher and cleaner flotation (as illustrated in Figure 13.67). The tests

optimise reagents in both stages prior to the variability tests. Based on the results, 35 g/t of PAX and Aero3477 in the cleaner stage and 325 g/t $CuSO_4$ in the rougher stage were deemed optimal.

13.7.9 VARIABILITY FLOTATION TESTS

All 36 variability testwork samples underwent a series of tests comprising gravity separation followed by rougher and cleaner flotation (as depicted in Figure 13.67). The objective was to observe mass pull and gold recovery for the individual variability samples using a procedure that reproduces the proposed Čoka Rakita flowsheet. All samples were ground to 80% passing 53 µm.

Figure 13.67 – Schematic of Flotation Variability Testwork



Source: BaseMet Labs, 2025

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

Table 13.49 summarises the mass pulls and gold recoveries recorded after gravity concentration, rougher flotation and cleaner flotation. Note that, as these are open-circuit tests (i.e., with no recycling of the intermediate products as would occur in an operating plant), it was assumed that, if recycled, the gold contained in both the second cleaner tails (Product 3) and the scavenger concentrate (Product 4) would find its way into the final flotation concentrate. Consequently, the gold in these two (2) streams were added when calculating the overall recovery for each test.

Table 13.49 – Summary of Gravity and Flotation Tests

Sample ID	Gravity Concentrate (GC)			Rougher Concentrate			Cleaner Concentrate (CC)			Overall GC + CC
	wt (%)	Au (g/t)	Au Rec (%)	wt (%)	Au (g/t)	Au Rec (%)	wt (%)	Au (g/t)	Au Rec (%)	Au Rec (%)
Met Var-1	0.054	1,494	26.87	26.65	6.69	58.85	11.48	12.99	49.24	76.1
Met Var-2	0.046	1,062	12.70	6.17	39.59	63.62	0.75	255.04	50.05	62.7
Met Var-3	0.053	429	4.29	16.90	24.70	78.03	6.56	46.21	56.66	60.9
Met Var-4	0.070	1,440	22.48	12.28	23.21	63.47	7.17	32.8	52.34	74.8
Met Var-5	0.076	2,003	32.75	13.82	16.23	48.19	2.65	63.05	35.94	68.7
Met Var-6	0.135	2,283	53.33	20.28	11.47	40.33	14.05	14.59	35.55	88.9
Met Var-7	0.080	1,672	32.79	21.93	10.55	56.86	5.78	29.64	42.12	74.9
Met Var-8	0.053	3,371	23.62	15.01	27.61	54.33	8.86	39.61	46.00	69.6
Met Var-9	0.079	581	12.91	5.35	46.37	70.00	0.55	391.98	60.63	73.5
Met Var-10	0.151	55	3.38	7.77	17.40	55.24	1.54	73.83	46.51	49.9
Met Var-11	0.185	544	29.32	15.45	14.79	66.74	9.36	22.64	61.89	91.2
Met Var-12	0.115	855	37.41	10.37	12.42	48.97	0.98	102.39	38.31	75.7
Met Var-13	0.128	502	21.24	12.47	16.93	69.54	7.04	25.99	60.32	81.6
Met Var-14	0.146	297	20.20	11.59	12.32	66.62	2.27	50.4	53.28	73.5
Met Var-15	0.134	624	18.95	6.74	37.61	57.57	2.93	59.61	39.63	58.6
Met Var-16	0.273	1,477	31.86	16.32	45.51	58.71	3.69	171.73	50.05	81.9
Met Var-17	0.137	787	10.15	15.26	54.93	78.86	4.24	163.79	65.38	75.5
Met Var-18	0.196	238	15.55	9.25	25.26	77.82	2.94	75.19	73.53	89.1
Met Var-19	0.145	754	22.87	9.34	35.36	69.00	2.18	120.97	55.00	77.9
Met Var-20	0.109	1,882	42.18	7.44	32.33	49.29	2.95	66.6	40.24	82.4
Met Var-21	0.206	2,099	49.09	9.82	39.89	44.47	2.63	122.29	36.55	85.6
Met Var-22	0.219	516	17.25	5.76	87.80	77.02	1.56	282.45	67.13	84.4
Met Var-23	0.144	959	24.73	8.38	45.58	68.54	1.94	167.9	58.43	83.2
Met Var-24	0.217	517	26.03	11.20	24.53	63.88	3.98	56.29	52.10	78.1
Met Var-25	0.364	148	9.63	11.77	27.03	56.90	6.40	40.61	46.47	56.1
Met Var-26	0.221	454	29.19	39.42	5.29	60.64	10.94	15.98	50.78	80.0
Met Var-27	0.315	348	32.08	13.50	15.55	61.37	7.64	25.36	56.62	88.7
Met Var-28	0.228	527	30.56	12.04	19.65	60.09	3.11	57.58	45.41	76.0
Met Var-29	0.222	2,163	65.35	16.54	12.81	28.78	3.58	54.34	26.43	91.8
Met Var-30	0.127	452	9.32	19.70	21.81	69.84	6.97	50.47	57.15	66.5

Sample ID	Gravity Concentrate (GC)			Rougher Concentrate			Cleaner Concentrate (CC)			Overall GC + CC
	wt (%)	Au (g/t)	Au Rec (%)	wt (%)	Au (g/t)	Au Rec (%)	wt (%)	Au (g/t)	Au Rec (%)	Au Rec (%)
Met Var-31	0.148	1,539	27.50	49.97	10.81	65.22	3.59	85.95	37.23	64.7
Met Var-32	0.154	411	10.92	14.11	33.88	82.71	6.90	58.69	70.11	81.0
Met Var-33	0.267	1,826	68.52	10.34	15.78	22.94	2.23	52.78	16.55	85.1
Met Var-34	0.210	1,197	42.79	6.53	37.39	41.62	1.92	101.04	33.08	75.9
Met Var-35	0.234	441	18.14	13.88	26.24	64.16	7.39	42.07	54.72	72.9
Met Var-36	0.178	485	22.75	13.59	18.59	66.38	4.79	49.78	62.69	85.4

Source: BaseMet Labs, 2025

The following statistics were calculated based on the variability test results (note its raw testwork results without any scale-up to expected full-scale operation):

Overall gold recovery:

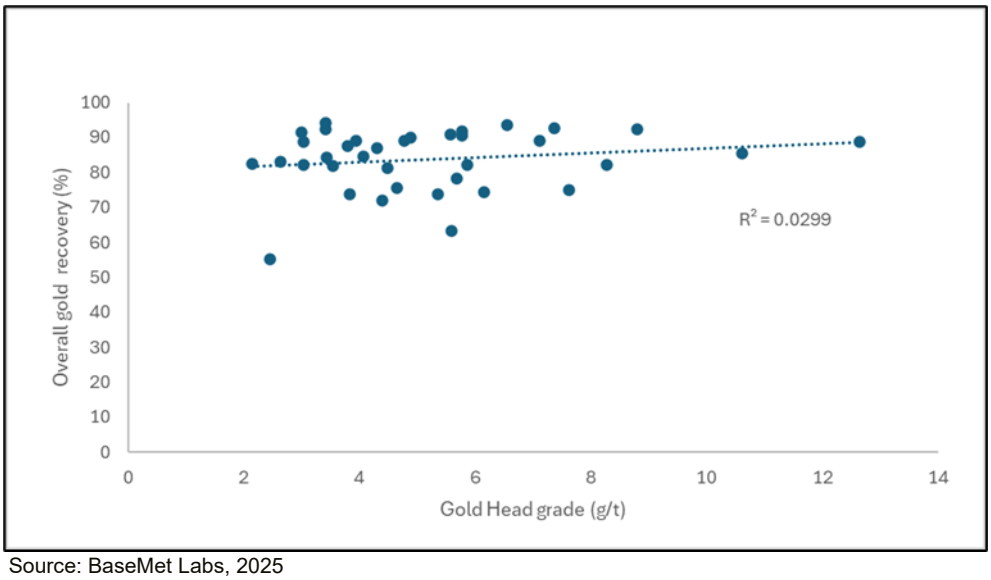
- Minimum: 49.9%.
- Median: 76%.
- Average: 76.2%.
- 90th Percentile: 88.8%.
- Maximum: 91.8%.

The gravity gold recovery varies significantly from 3.4% to 68.5%, which is again indicative of the presence of coarse gold and a justification for including the gravity step in the flowsheet.

The average overall gold recovery of 76.2% is lower than what was recorded in previous testwork campaigns where the gold recovery was around 87%. The lower value for this set of tests is due to a few very low outliers e.g., sample Met Var-10 which yielded only 49.9% overall recovery. On the other hand, the 90th percentile overall gold recovery of 88.8% is slightly higher than previous testwork campaign 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 13.68 illustrates the population of recoveries versus head grade for all the tests conducted on all the variability samples. There is a positive correlation between recovery and grade despite it being very weakly correlated. The implication is again that it is not feasible to define a recovery model, as a function of head grade, for this ore body and that the PDC will use the average recovery for design purposes.

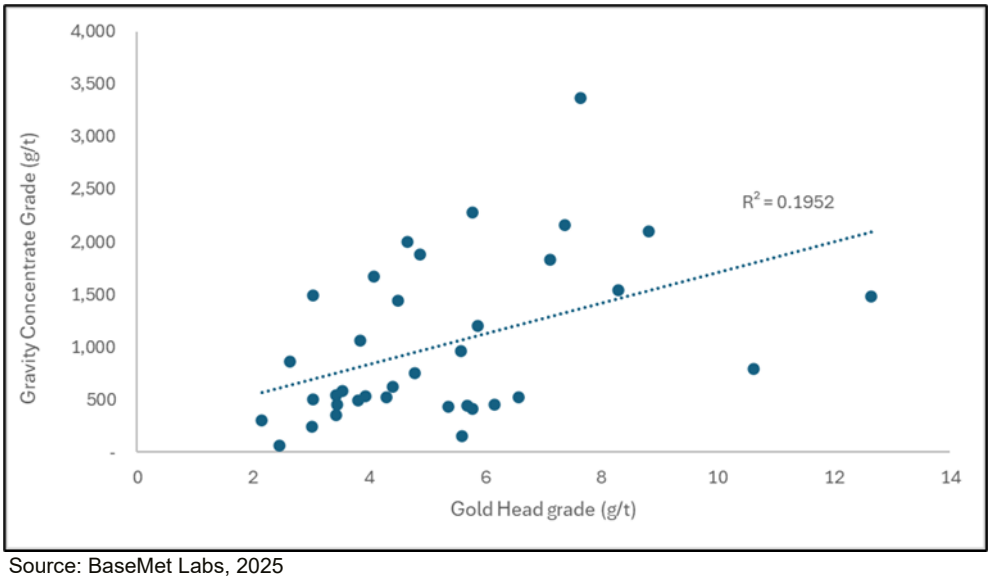
Figure 13.68 – Overall Gold Recovery vs. Head Grade



Source: BaseMet Labs, 2025

Figure 13.69 illustrates the gravity concentrate grade for each variability sample against its head grade. There is a positive correlation between gravity concentrate and head grade despite it being weakly correlated, which supports the assumption of a constant upgrade ratio made during the PFS.

Figure 13.69 – Gravity Concentrate Grade vs Head Grade

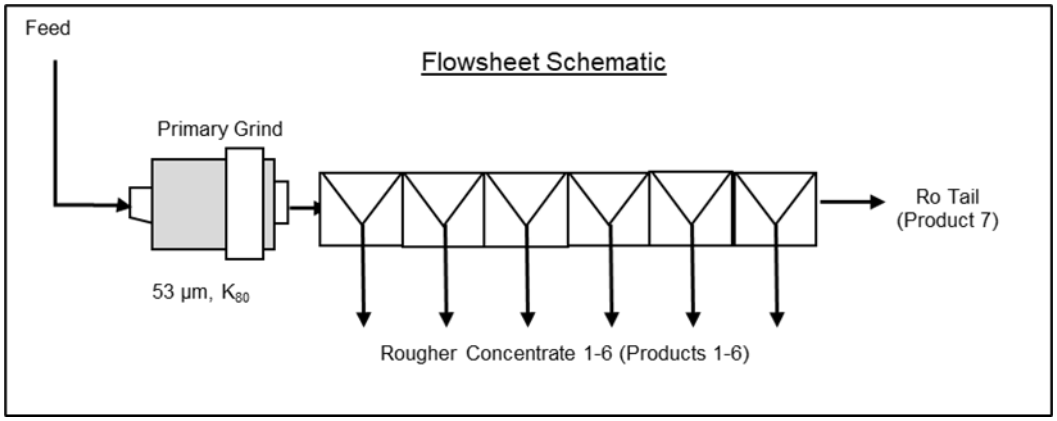


Source: BaseMet Labs, 2025

13.7.10 ROUGHER :150 PILOT TEST

BaseMet Labs conducted two (2) sub-samples of the master composite, which were 16 kg and 60 kg, to L150 rougher pilot testwork. A Jameson Cell L150 pilot scale test unit was used to confirm the rougher flotation performance. The tests comprising of gravity separation followed by rougher flotation. The primary grind size for gravity test was 212 µm and gravity tails were reground to K_{80} 53 µm and fed to rougher circuit. The test schematics is illustrated in Figure 13.70.

Figure 13.70 – Rougher L150 Pilot Test



Source: BaseMet Labs, 2025

Test conditions were as follows:

- Test No. BL1724-15 with 16 kg of Master Composite.
- Test No. BL1724-18B with 60 kg of Master Composite.
- Water type is Čoka Rakita Site Water (well).
- Feed percent Solid: 30%.
- 325 g/t $CuSO_4$ addition to rougher.
- 125 g/t PAX and 125 g/t A3477 collector addition to rougher.
- 50 g/t MIBC addition to rougher.

Table 13.50 summarises the mass and pertinent gold data obtained from the Jameson pilot test. Note that the data for some of the streams were calculated.

Table 13.50 – L150 Rougher Pilot Tests Result

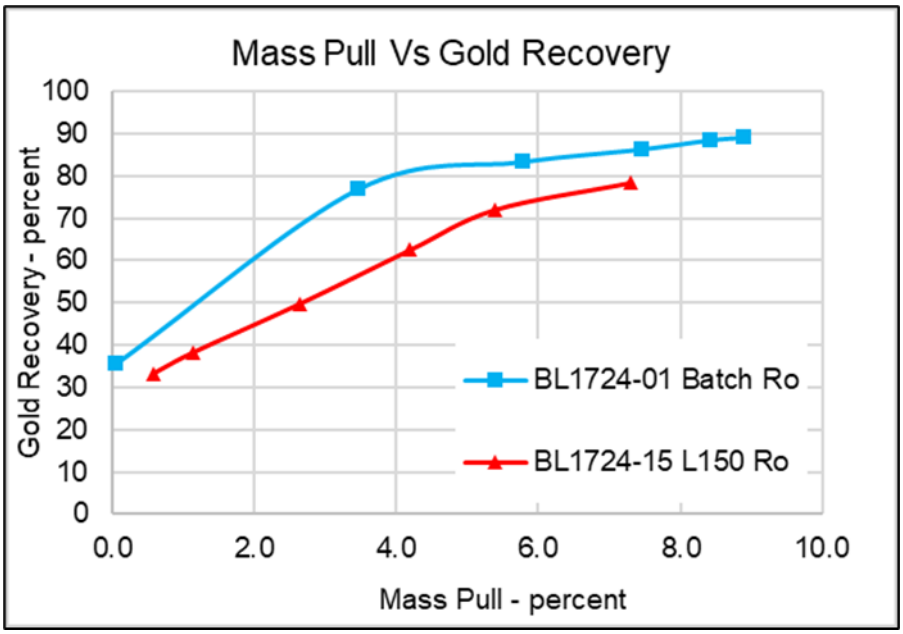
Sample ID	BL1724-15 Master Composite			BL1724-18B Master Composite		
	Wt (%)	Au (g/t)	Au Distribution (%)	Wt (%)	Au (g/t)	Au Distribution (%)
Feed		4.21			4.97	
Gravity Concentrate	0.015	7,457	26.1	0.003	51,764	31.4
Rougher Concentrate	7.31	30.2	52.3	17.53	13.8	48.7
Rougher Tails	92.7	0.98	21.6	82.5	1.20	19.9

Source: BaseMet Labs, 2025

The combined overall gold recovery was 78.4% and 80.1% for the master composite samples. Note that for test 18B a small amount of gravity concentrate was collected, resulting in a very high grade of 52 kg/t or 5.2% gold by mass.

Figure 13.71 depicts the results of the rougher bench-scale and rougher L150 pilot-scale tests of master composite. As illustrated, both cases exhibit a direct correlation between mass pull and gold recovery. The batch test demonstrates higher gold recovery at a given mass pull compared with the pilot-scale test, indicating potentially better performance or different operational conditions in the batch process. The recovery increases rapidly at lower mass pull percentages before reaching a plateau, particularly in the batch test, where recovery stabilises above 85%. In contrast, the pilot-scale test, a similar trend is observed, but with lower overall recovery. If the tests were extended and more mass were recovered, the overall recoveries recorded for these two (2) tests may reach the expected recovery of around 87%.

Figure 13.71 – Rougher L150 Pilot Test vs Rougher Bench Scale Test (Master Composite)

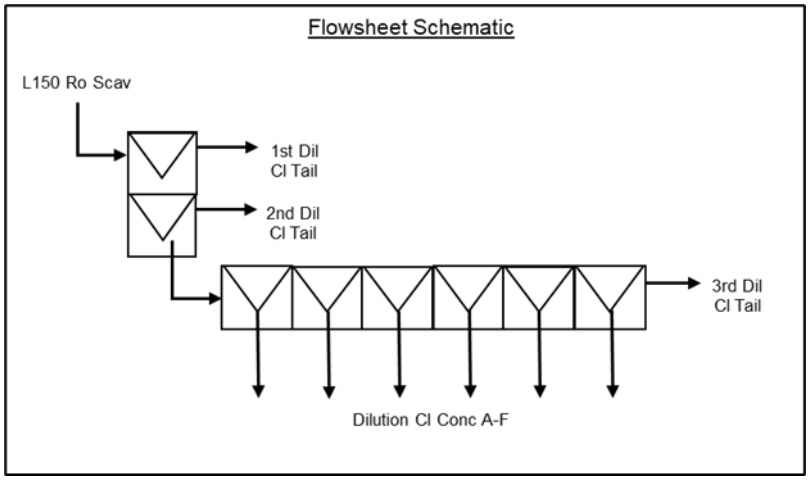


Source: BaseMet Labs, 2025

13.7.11 CLEANER BENCH TEST – MASTER COMPOSITE SAMPLE

Basemet performed three (3) bench cleaner tests to evaluate cleaner conditions for further testwork. The L150 rougher scavenger concentrate was passed through three (3) stages of dilution cleaning. The schematic is illustrated in Figure 13.72 and the results presented in Figure 13.73.

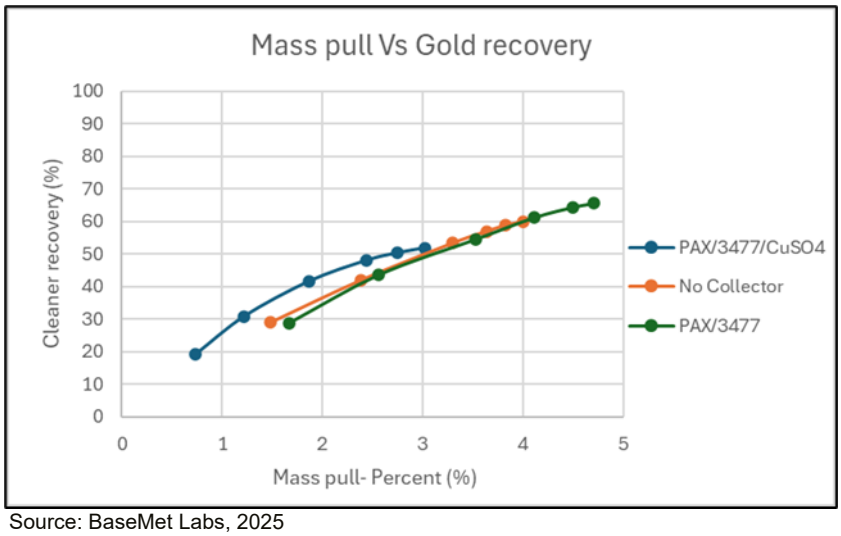
Figure 13.72 – Schematic of Bench Cleaner Dilution Test



Source: BaseMet Labs, 2025

The results indicated that all conditions follow a similar trend where gold recovery increases with mass pull. The addition of PAX, A3477 and $CuSO_4$ in the cleaner tests yielded a higher recovery at lower mass pull values, which demonstrates the enhanced performance with $CuSO_4$ as an activator. Comparing the test with PAX and 3477 against the test without any collector exhibit similar recovery trends but with a higher recovery achieved when using the collectors.

Figure 13.73 – Bench Cleaner Dilution Test with Different Reagents Dosage

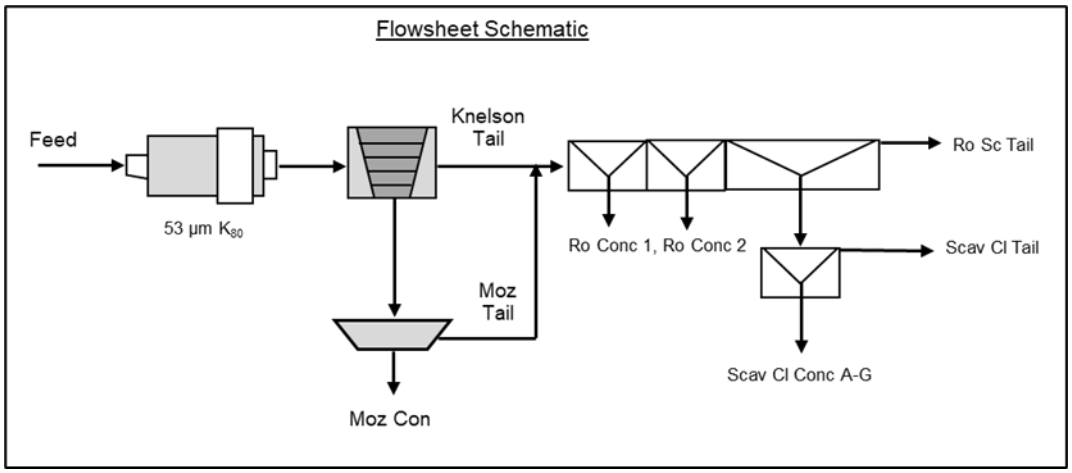


Source: BaseMet Labs, 2025

13.7.12 JAMESON PILOT TEST – MASTER COMPOSITE SAMPLE

BaseMet Labs performed a Jameson pilot test using 200 kg master composite sample. A Jameson Cell, L150 pilot scale test unit was used to confirm the flotation performance. The sample was first ground to 80%passing 53 µm. The ground sample was then passed through two (2) stages of gravity concentration with the Knelson followed by a Mozley table. Mozley table concentrate were collected as the final gravity concentrate product. Both gravity steps' tails were combined and used as feed for the Rougher L150 test. Figure 13.74 shows a schematic of the procedure employed.

Figure 13.74 – Schematic of Jameson Pilot Test (L-150 Rougher, Scavenger and Cleaner)



Source: BaseMet Labs, 2025

Test conditions were as follows:

- Čoka Rakita Site Water (well).
- Feed percent Solid: 30%.
- 300 g/t $CuSO_4$ to rougher feed.
- 100 g/t PAX and 100 g/t A3477 collector, and 20 g/t MIBC addition to rougher.
- 50 g/t PAX,50 g/t A3477, 30 g/t MIBC and 100 g/t $CuSO_4$ addition to rougher Scavenger.
- Add 50 g/t PAX and 50 g/t PAX 3477 to scavenger cleaner.

Table 13.51 summarises the mass and pertinent gold data obtained from the Jameson pilot test. Note that the data for some of the streams were calculated.

Table 13.51 – Jameson Pilot Test Results

Sample ID	BL1724-20B		
	Wt (%)	**Au** (g/t)	**Au Distribution** (%)
Feed	100	6.14	100
Mozley Con	0.002	74,694	30.4
Rougher Con1	2.35	64.2	24.5
Rougher Con 2	2.78	47.7	21.6
Rougher Con (1+2)	5.13	55.2	46.2
Scavenger Cleaner Con	1.96	28	8.9
Rougher Con (1+2) + Scav Cl Con	7.09	47.7	55.1

Sample ID	BL1724-20B		
	Wt (%)	Au (g/t)	Au Distribution (%)
Scavenger Cleaner Tail	3.55	4.09	2.4
Rougher Scavenger Tails	89.36	0.84	12.2

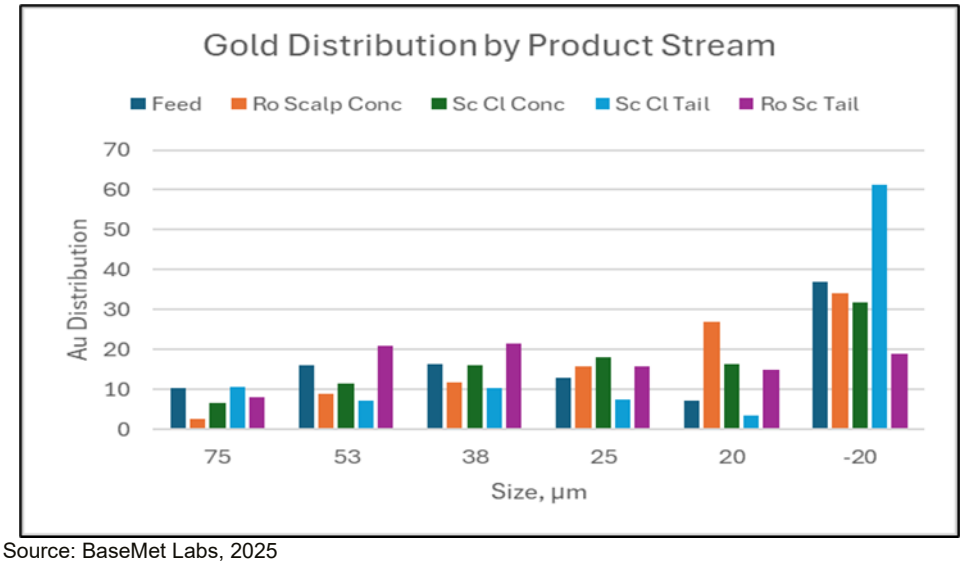
Source: BaseMet Labs, 2025

The combined overall gold recovery was 85.5% for this master composite sample, which excludes the recoverable gold from the scavenger cleaner tail. In a full-scale operation this product will be recycled so some of the gold in this stream would be recovered. Note that this stream contained 2.4% of the feed gold. If some or most of this gold is recovered when this stream is recycled the overall gold recovery will be in the vicinity of the 87% overall gold recovery that was adopted for the PFS. A locked cycle test will be required to determine how much gold will be recovered from recycled scavenger cleaner tails.

The final flotation concentrate mass pull was 7.09% with the gold grade of 47.7 g/t. The final rougher tails mass weight was 89.36% with the gold grade of 0.84 g/t.

Gold distribution in each size fraction for each product stream is illustrated in Figure 13.75. It shows around 60 percent of the gold that presented in the cleaner scavenger tails and around 20% of the gold presented in rougher scavenger tails are less than 20 µm.

Figure 13.75 – Gold Distribution by Product Stream in Different Size Fraction

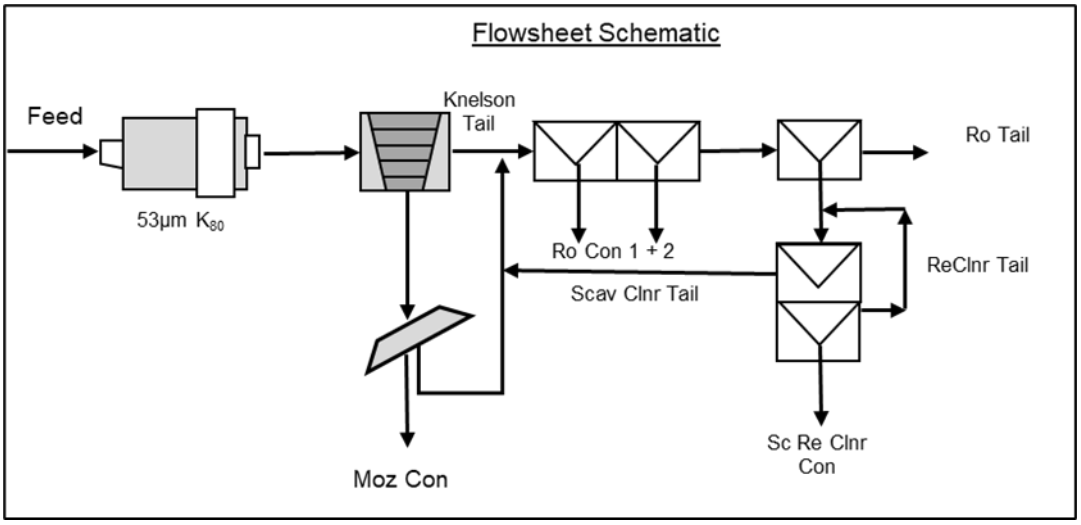


Source: BaseMet Labs, 2025

13.7.13 LOCKED CYCLE FLOTATION TESTS

BaseMet Labs subjected the MC sample to a standard locked-cycle flotation test. One of the objectives of this test was to estimate how much of the gold could be recovered from the scavenger cleaner tails when the circuit is closed. Figure 13.76 illustrates a schematic of the procedure used. The locked-cycle procedure involves using the products that are recycled from the preceding cycle of batch tests in the next cycle. After a few cycles the products would effectively begin to resemble what would be produced by a continuous operation.

Figure 13.76 – Schematic of Locked-Cycle Test



Source: BaseMet Labs, 2025

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. Both gravity steps' tails and scavenger cleaner tails were combined and used as feed for the rougher flotation step.

Test conditions were as follows:

- 5 mins conditioning of rougher feed with 250 g/t $CuSO_4$.
- 100 g/t PAX and 100 g/t A3477 collector addition to rougher.
- 25 g/t PAX,25 g/t A3477, 100 g/t $CuSO_4$ and 56 g/t MIBC addition to rougher Scavenger.
- Total rougher float time = 30 mins.
- Scavenger cleaner duration = 18 mins.
- Scavenger re-cleaner = 14 mins.
- Add 50 g/t PAX and 50 g/t PAX 3477 to scavenger cleaner.

Table 13.52 summaries the mass and pertinent gold data obtained from the LCT. Note that the data for some of the streams were calculated.

Table 13.52 – Locked Cycle Test Results

Sample ID	BL1724-68 Cycle D+E		
	Wt (%)	**Au** (g/t)	**Au Distribution** (%)
Feed	100	6.19	100
Mozley Con	0.10	2,706	45.9`
Rougher Con1	2.21	64.1	22.9
Rougher Con 2	2.14	26.6	9.2
Rougher Con (1+2)	4.35	24.9	17.5
Scav ReCl Con	2.19	23.3	8.2
Rougher Con (1+2) + Scan ReCl Con	6.54	38.2	40.3
Rougher Tails	93.4	0.92	13.8

The combined overall gold recovery was 86.2 for the master composite sample, which is consistent with previous test results. Note that 45.9% of the gold in this sample reported to the gravity concentrate which is close to what is expected on average for this deposit.

The final rougher flotation concentrate mass pull was 4.35% with the gold grade of 24.9 g/t and the final cleaner flotation concentrate mass pull was 2.2% with the gold grade of 23.3 g/t. This mass pull is again marginally higher than the average reported for previous test campaigns (which were just below 5%) – it is prudent to increase the design factor for mass pull to ensure the downstream dewatering section can handle peaks in concentrate production.

13.7.14 JAMESON 3-STAGE DILUTION CLEANER TEST – LOM VARIABILITY SAMPLES

BaseMet Labs performed Jameson 3-stage dilution cleaner tests on the three (3) variability samples which represent distinct periods over the mine's life. The flotation steps were preceded by the normal gravity step to remove coarse gold ahead of the Jameson test. The gravity concentration test results are shown in Table 13.53.

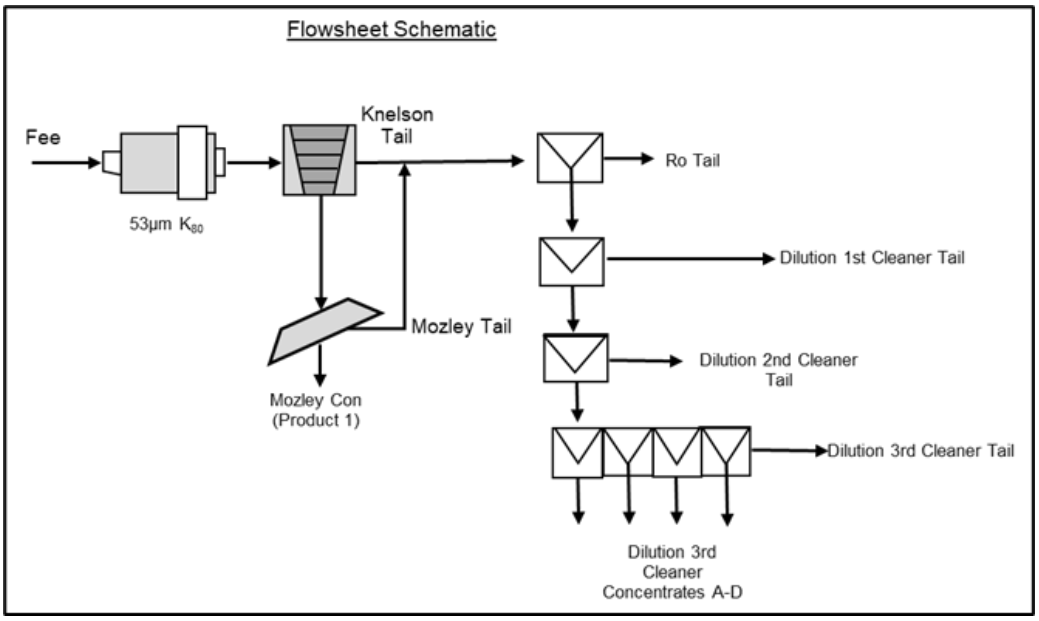
Table 13.53 – Gravity Test Results (Preparing Feed for Jameson Dilution Cleaner Test)

Composite (Test)	Product	Feed Size K_{80} (µm)	Mass (%)	Assay Au (g/t)	% Au Distribution
Years 1-2 G58	Moz Con A		0.002	35,881.6	15.1
	Moz Con B		0.019	462	1.5
	Moz Con C		0.029	95.8	0.5
	Comb Conc A-C		0.051	1,935	17.1
	Knelson and Mozley Tail		99.95	4.81	82.9
	Feed (calc.)	53		5.8	100
Years 3-5 G59	Moz Con A		0.003	84,269.7	47.1
	Moz Con B		0.014	422.9	1.2
	Moz Con C		0.029	124.3	0.7
	Comb Conc A-C		0.046	5,342	49
	Knelson and Mozley Tail		99.95	2.57	51
	Feed (calc.)	53		5.04	100
Years 6-10 G60	Moz Con A		0.003	6,314.6	7.4
	Moz Con B		0.010	328.1	1.1
	Moz Con C		0.019	60.7	0.4
	Comb Conc A-C		0.032	792	9.0
	Knelson and Mozley Tail		99.97	2.6	91.0
	Feed (calc.)	53		2.8	100

Gravity recovery was highly variable, as usual, ranging from as little as 9% only for the Years 6-10 sample to 49% for the middle years sample.

Knelson and Mozley tails were used for the Jameson dilution cleaner tests based on the Glencore protocol. The goal of the Jameson dilution cleaner testing was to assess how flotation efficiency would vary over the mine's life. The flowsheet for the 3-stage cleaner dilution test is illustrated in Figure 13.77.

Figure 13.77 – Flowsheet Schematic of Jameson 3-Stage Dilution Cleaner Test



Tests conditions were as follows:

- Čoka Rakita Site Water (well).
- Feed size is 80% passing 53 µm.
- 5 mins conditioning of rougher feed with 250 g/t $CuSO_4$.
- Addition of 150 g/t PAX,150 g/t A3477, and 100 g/t $CuSO_4$ to rougher.
- Total rougher float time = 30 mins.
- Total cleaner float duration = 45 mins.

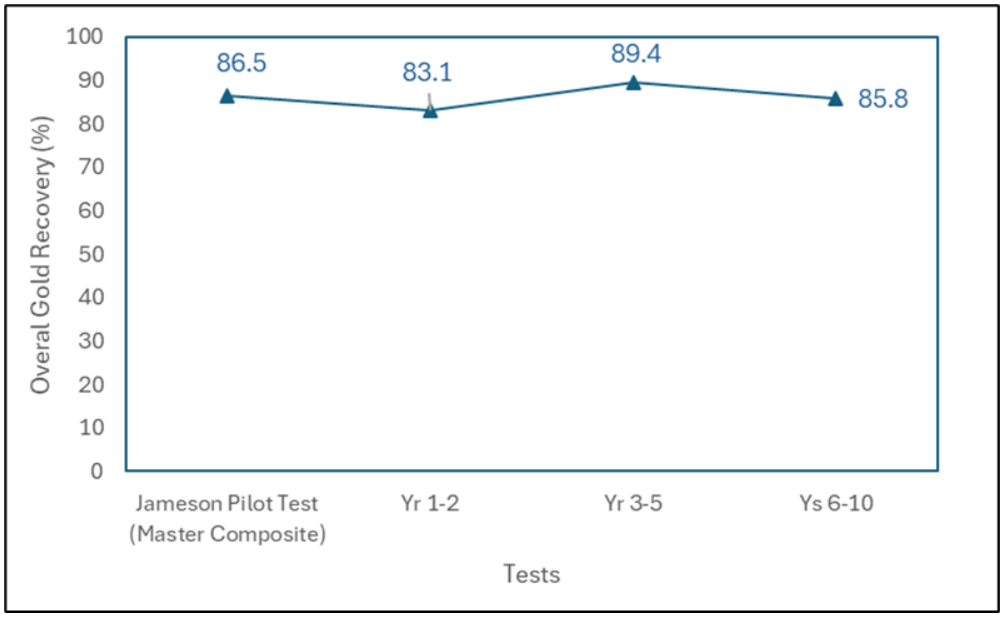
Table 13.54 summarises the mass and pertinent gold data obtained from the Jameson variability composites tests. Note that the data for some of the streams were calculated.

Table 13.54 – Jameson Variety Dilution Cleaner Test Results

Sample ID	(Years 1-2) BL1724-58C			(Years 3-5) BL1724-59C			(Years 6-10) BL1724-60C		
	Wt (%)	Au (g/t)	Au Distribution (%)	Wt (%)	Au (g/t)	Au Distribution (%)	Wt (%)	Au (g/t)	Au Distribution (%)
Recalc. Feed		5.79			5.03			2.80	
Mozley Con	0.05	1,935	17.1	0.046	5,342	48.9	0.030	792	8.5
Ro Con & Scav con	14.04	27.2	66	26.1	7.8	40.5	43.31	5.0	77.4
2nd Di Cl Con	1.9	60.4	20.1	2.4	33.4	16.2	1.6	25.3	14.5
Dil 1st Cl Con	3.7	33.5	21.3	3.0	29.6	17.5	8.0	7.65	21.7
Dil Ro Con(Scav con)	9.4	15.0	24.3	21.2	4.98	21.0	34.7	2.29	28.4
Ro Scav Tail	85.9	1.14	16.9	73.8	0.72	10.6	56.7	0.70	14.2
R+S stage recovery			83.1			89.4			85.8

Figure 13.78 illustrates the comparison of Jameson pilot test overall gold recovery on master composite (L150-20 pilot test) and the three (3) mine life variability samples. Both the Years 1-2 and Years 6-10 samples yielded unexpectedly low gravity recoveries of 17.1% and 8.5% respectively, which might be the nugget effect or the way the tests were performed. Nugget presence in the sample usually has a direct effect on assay and gravity tests.

Figure 13.78 – Gold Recovery Comparison in Master Composite Sample and Variety Samples



13.7.15 VARIABILITY CLEANER TESTS WITH JAMESON FLOWSHEET ARRANGEMENT

BaseMet Labs performed four (4) cleaner tests (master composite, and three (3) mine life periods composites) with Jameson flowsheet arrangement illustrated in Figure 13.79. Summary of the tests result are shown in Table 13.55.

Figure 13.79 – Cleaner Tests with Jameson Flowsheet Arrangement

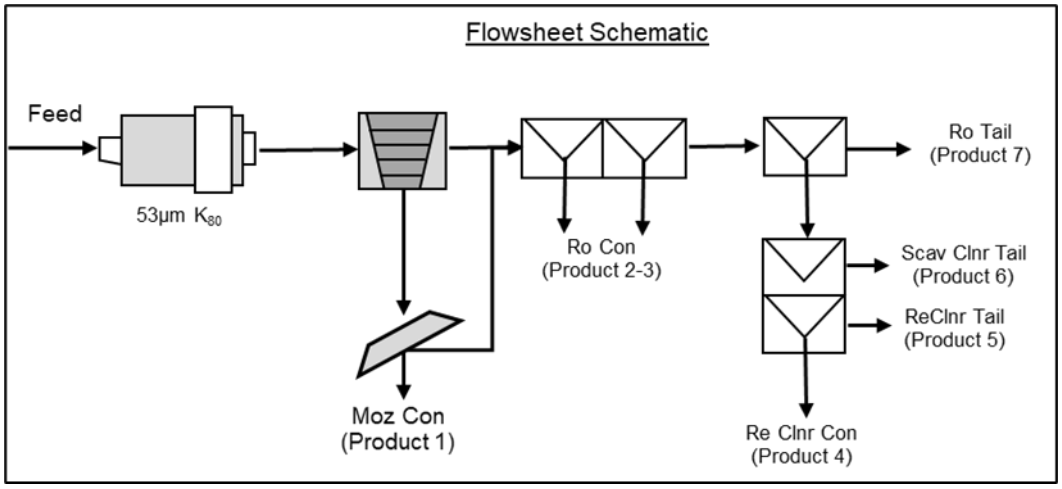
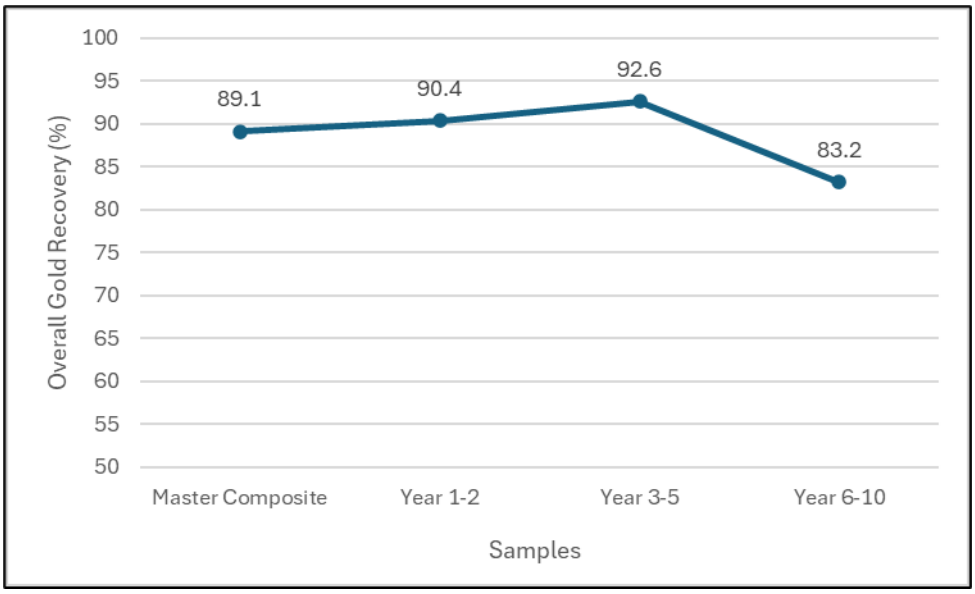


Table 13.55 – Results Variety Cleaner Tests with Jameson Arrangement

Sample ID	BL1724-67 Master			BL1724-62 (Yr 1-2)			BL1724-64 (Yr 3-5)			BL1724-66 (Yr 6-10)		
	Wt%	Au (g/t)	Au Rec (%)	Wt%	Au (g/t)	Au Rec (%)	Wt%	Au (g/t)	Au Rec (%)	Wt%	Au (g/t)	Au Recy (%)
Recalc. Feed		5.83			7.71			6.31			3.75	
Grav + Final Cons	6.8	76.6	89.1	12.1	57.5	90.4	11.7	49.9	92.6	7.4	42.1	83.2
Ro Tail	91.1	0.65	10.2	82.9	0.80	8.6	83.8	0.50	6.6	88.7	0.64	15.2

The overall gold recovery for each sample is illustrated in Figure 13.80. It shows that gold recovery was lowest for the Years 6 to 10 head sample which is as expected and in line with its reduced head grade.

Figure 13.80 – Overall Gold Recovery for Cleaner Tests with Jameson Arrangement



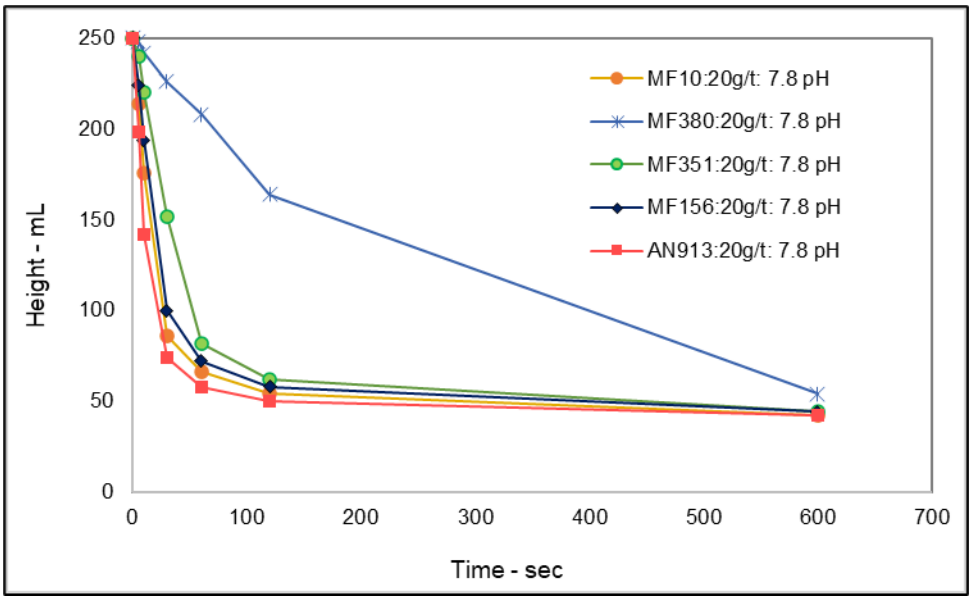
13.7.16 SOLID-LIQUID SEPARATION – CONCENTRATE

The flotation concentrates dewatering testwork program included flocculant scoping, static and dynamic settling, underflow rheology, and underflow pressure filtration. All tests were performed with the L150 final concentrate combined sample.

- Flocculant scoping test

  Flocculant scoping tests consisted of evaluating the performance of five (5) types of flocculants at pH =7.8. AN913 yielded the best settling rate and an acceptably clear supernatant solution. The flocculant scoping tests results are presented in Figure 13.81.

Figure 13.81 – Flocculant Scoping Test



- Static Thickening test

  Static settling tests were done to evaluate the concentrate settling rate at flocculant dosage rates of 20 and 40 g/t and after elevating the pH. The static thickening results are summarised in Table 13.56.

  With respect to the final concentrate sample the following outcomes were observed:

  - Increasing flocculant dosage from 20 to 40 g/t had no impact on u/flow density but improved clarity and free settling velocity.
  - Increasing pH from 7.8 to 9 had minimal impact on u/flow density, which decreased from 59 to 58.1%, but it did improve the free settling rates by approximately 30%. However, adding another reagent preparation system would be costly and the resultant plant effluent may require neutralization prior to discharge. Given these considerations a marginally larger thickener size is preferable.

Table 13.56 – Static Thickening Summary (BML 2025)

Test	Sample	Size (µm)	Flocculant type	Flocculant (g/t)	pH	Initial Density (%)	Final Density (%)	Free Settling Velo (m/h)
S1	T20 Final concentrate	52	AN913SH	20	7.8	13.4	59.0	14.5
S2				40	7.8	13.4	59.0	18.2
S3				40	9	13.5	58.1	25.0

- Dynamic Thickening Test

Dynamic thickening tests results for the concentrate sample are presented in Table 13.57. The tests were all performed at pH 9, a grind of 80% passing 52 µm, feed density of 15%w/w solids and with coagulant added.

Test D1-A, which was the only test performed without adding coagulant, yielded significantly higher turbidity readings compared to the other four (4) tests. Over the range of loading rates tested underflow densities exceeded 58%w/w solids.

Based on the test results, a flocculant and coagulant dosage of 20 g/t each, with a solid loading rate of 1 tph/m², achieved an underflow density of 67% w/w solids while also producing acceptably clear overflows. The loading rate of 1 tph/m² can be used for the sizing of the concentrate thickener. These results are similar to what was achieved during the previous phase testing with the most significant difference being that the underflow density in the previous phase was even higher at 76%w/w solids at a lower flocculant dosage of 15 g/t and with no coagulant or pH modification. For the PDC an underflow density of 65%w/w solids was adopted, combined with a flocculant dosage of 15 g/t flocculant and a design rate of 1.0 tph/m². The latest test results confirms that the criteria are applicable but with the requirement for coagulant to ensure overflow clarity.

Table 13.57 – Dynamic Settling Tests- Final Concentrate

Test	Density (%)	Flocculant		Coagulant		pH	Turbidity	Loading Rate	Rise Rate	Yield stress
	U/F	Type	g/t	Type	g/t		ppm	tph/m²	m/h	Pa
D1-A	60.6	AN913SH	40	DB45SH	0	9.0	326	0.5	3.1	64
D1-B	65.2		30		30	9.0	19	0.7	4.2	136
D1-C	65.6		30		30	9.0	17	1.0	6.1	104
D1-D	58.0		30		30	9.0	16	1.3	7.9	119
D1-E	66.8		20		20	9.0	17	1.0	6.0	104

- Pressure Filtration Testing

  Pressure filtration tests were carried out on the combined final concentrate sample generated from L150 Test 20, which was performed on the Master Composite and thickened to around 58% solids through dynamic thickening.

  Pressure plate filtration tests were performed using a Micronics plate-and-frame pressure filtration unit to evaluate the dewatering characteristics of the sample. The filtration process was divided into three (3) distinct phases:

- Form: Slurry was pumped into the filtration unit using a diaphragm pump operating at pressures ranging from 60 to 90 psi.
- Press: The filter chamber was compressed for 30 seconds at a pressure of 120 psi to initiate dewatering.
- Blow: Compressed air was applied to the chamber, with the filtrate recorded after blow durations of 1 and 3 minutes to calculate the residual water content in the filter cake at that juncture.
- A summary of the cake moisture and filtration rate for each test is presented in Tables 13.58 and 13.59.

Table 13.58 – Pressure Filtration Test Results – Final Concentrate

Test	Chamber Thickness	Feed		Cake Moisture (%)				Total time	Discharge
	mm	Density (% Sol)	Pressure (psi)	Formed	Press	Blow 1 min	Blow 3 min	Sec	Dry, Sticky
PF-1-A	25	52.2	60	24.7	17.5	14.4	11.7	290	Dry
PF-1-B	25	55.8	75	23.2	17.3	14.0	11.9	290	Dry
PF-1-C	25	52.6	90	22.7	17.0	13.8	11.0	290	Slightly Wet
PF-1-D	35	58.2	75	22.7	17.5	14.8	12.7	310	Dry
PF-1-E	35	56.2	75	21.1	17.2	13.8	10.7	280	Dry

Table 13.59 – Pressure Filtration Final Concentrate Filtration Rate Results

Test	Filtration Rate (kg/m²/h)				Turb.
	Formed	Press	Blow 1 min	Blow 3 min	FAU
PF-1-A	1167	849	406	322	406
PF-1-B	1209	879	421	334	247
PF-1-C	1247	907	434	344	125
PF-1-D	1274	980	509	411	183
PF-1-E	1065	745	339	266	190

The results show that the cake moisture content decreases by around 2 to 3%w/w when extending the blow time to three (3) minutes. Given that the cost of transportation is by mass

this reduction in mass is worth the additional capital required for a larger filter press. The moisture content adopted into the design criteria was increased from 9 to 10% in recognition of these results. Additionally, the design filtration rate was downgraded from 400 to 300 kg/m²/h to align with the latest test results.

- Vacuum Filtration testing

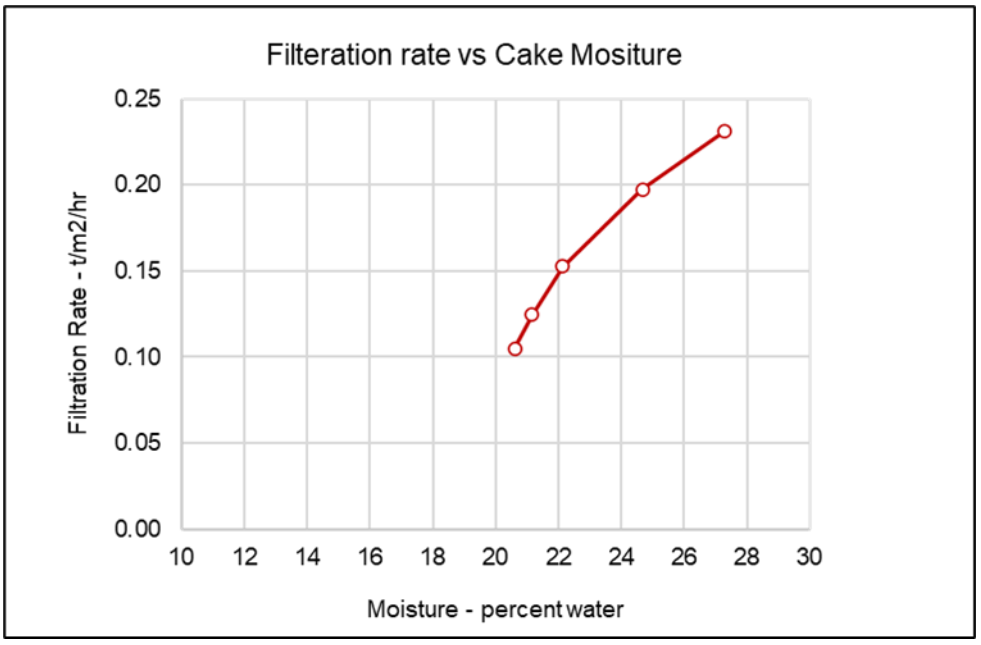
  Vacuum filtration tests were performed using a sub-sample of the concentrate from the Jameson pilot test. For each drying cycle the amount of filtrate recovered was measured allowing the calculating of cake moisture content and filtration rate data as summarised in the Table 13.60. The test was performed with a slurry density of approximately 48%w/w solids, solid SG of 4.15, and vacuum pressure of 240 mm Hg over cycle time from 350 to 770 second.

  The vacuum filtration result is somewhat disappointing as a moisture content below 20%w/w appears to be impractical. Given that this concentrate will be shipped off-site a lower moisture content is desirable. Pressure filtration with cake blow (as per the FS design) remains the preferred dewatering solution for this application.

Table 13.60 – Vacuum Filtration Tests Summary

Sample	Cycle Time (Second)	Filter Cake			
		Total Weight (g)	Water (g)	Moisture (%)	Rate (t/m²/h)
Final Concentrate	0	227.2	67.8	29.8	
	350	219.2	59.8	27.3	0.23
	410	211.6	52.2	24.7	0.20
	530	204.7	45.3	22.1	0.15
	650	202.2	42.8	21.2	0.12
	770	200.8	41.4	20.6	0.11

Figure 13.82 – Filtration Rate vs. Cake Moisture



13.8 Process Design Criteria Derived from Test Results

- A×b = 59.8 (average) and 44.6 (design) used the Phase 2 variability numbers as they are more conservative than the PEA numbers. The PEA design Axb was 52.5.
- Ai = 0. 12 (average for Opex) – Used the average of the PEA results and the PFS results.
- Bond Ball Mill Work Index = 13.9 kWh/t (average) and 15.2 kWh/t (design) – Used the PFS variability results as the design value increased significantly from 13.4 to 15.2 kWh/t. This increase is prudent to accommodate variability in hardness. It also explains why a second grinding stage is recommended for the PFS comminution circuit.
- Optimum primary grind for gravity plus rougher flotation route is 53 µm as confirmed by the PFS grind optimisation results.
- Used the LCT, Jameson pilot and optimisation tests condition from the FS as design criteria for flotation circuits:
  - 5 minutes conditioning with 325 g/t $CuSO_4$.
  - 125 g/t of both PAX and A3477 collector addition to roughers or rougher conditioning tank.
  - Add 35 g/t of both PAX and 3477 to the cleaner or cleaner conditioning tank.
- Thickener solids loading rate of 1 tph/m² can be used with underflow densities of 60% solids.

- Flocculant addition rates of 50 g/t for the tailings thickener and 15 g/t for the concentrate thickener.
- Coagulant addition of 20 g/t to the concentrate thickener.
- Filtration rate of 300 kg/m²/h and a cake moisture content of 10%w/w solids for the flotation concentrate pressure filter design.

13.9 Recovery Model

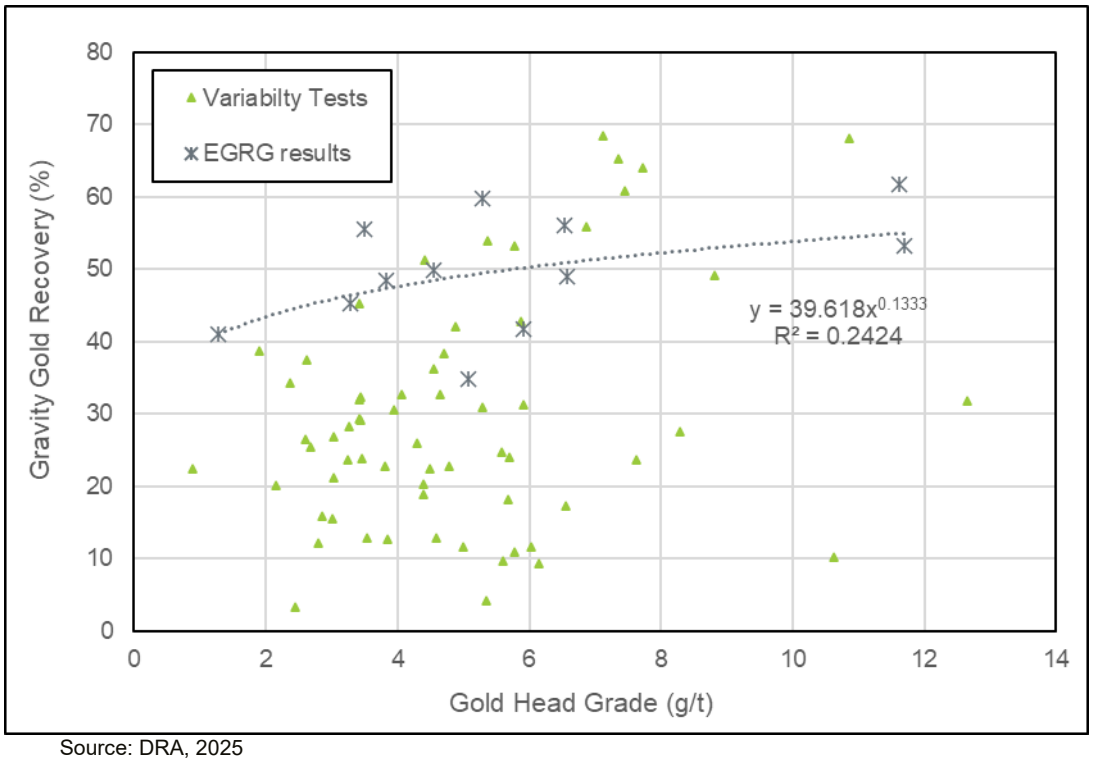
With respect to gravity concentration, two (2) observations were made from the test results that have been adopted into the design criteria to predict the gravity gold recovery and concentrate grade:

- The different sets of tests all yielded good correlations for a constant upgrade ratio from head gold grade to gravity concentrate gold grade. The actual value of this upgrade ratio would depend on how the gravity concentration step was performed and varied considerably from set of tests. For the process design criteria an upgrade ratio was selected that closely mimics the locked-cycle test results as these tests were also used as design criteria for the downstream flotation section. This trend was confirmed by the variability testing results in the PFS and FS.

- All of the gravity gold recoveries recorded during the various testwork campaigns are plotted on Figure 13.83. The various variability results are shown as triangles while the EGRG results from three (3) different sets were plotted as a separate series. It is evident that the variability results yielded, on average, lower gravity gold recoveries than the EGRG results. This was especially true at the lower end of the head grade range. An empirical correlation was fitted to the EGRG results as these are perceived to be more detailed and accurate. The correlation is Gravity Recovery = $39.6*[\text{Head grade}]^{0.133}$.

It should be noted that this correlation is similar to what was used before in the PEA and PFS studies, partly due to the fact that about half of the data is from those two (2) phases.

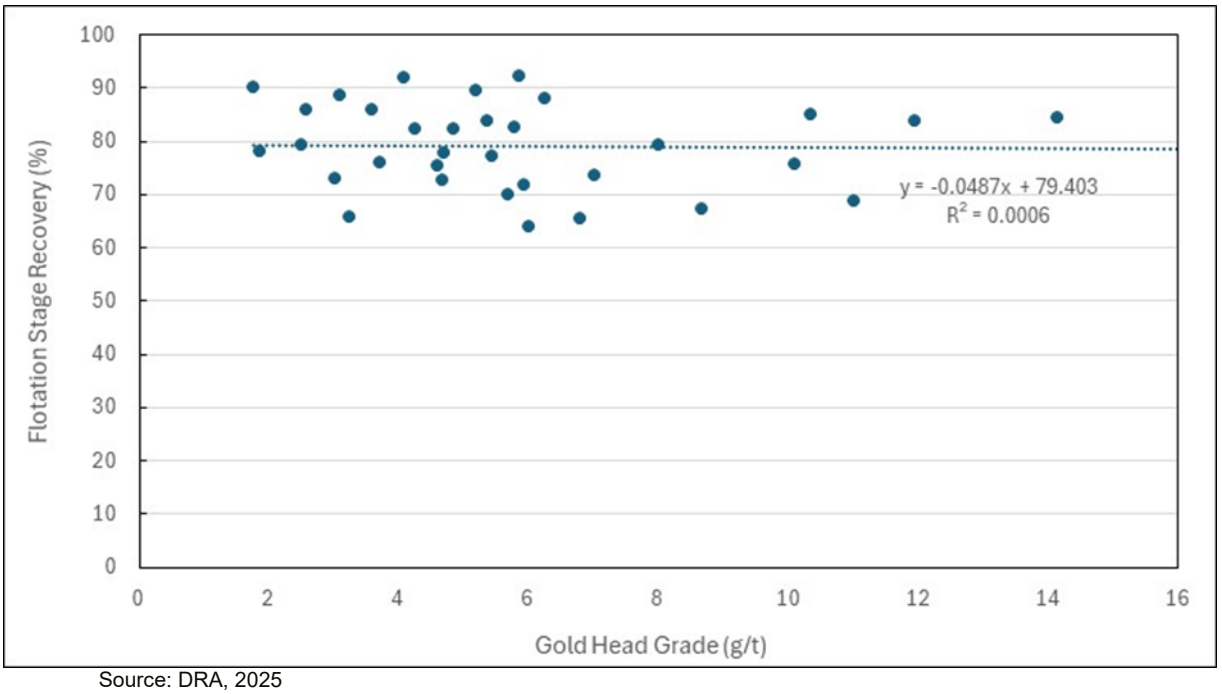
Figure 13.83 – Gravity Gold Recovery versus Head Grade for all Test Programs Combined



Source: DRA, 2025

With respect to flotation stage recovery, there is no clear correlation that could be established. Figure 13.84 shows the results for all 26 variability samples (Phase 3 FS) tests performed by BaseMet Labs for the set of rougher cleaner batch tests. It is evident that the correlation is weak and that the most appropriate model for flotation stage recovery is to adopt a simple constant recovery of 79%.

Figure 13.84 – Flotation Stage Recovery vs. Gold Head Grade



Source: DRA, 2025

The combination of the gravity model with a constant 79% flotation stage recovery yields a predicted overall gold recovery of 87.4% for the design head grade of 6.38 g/t, which aligns well with the weighted average overall recovery recorded for the variability samples during the PEA phases as well as all subsequent phases.

13.10 Deleterious Elements

Arsenic was present in the final product (concentrates) samples. For the three (3) samples tested by BaseMet Labs 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 to 3,466 g/t. These represent less than 20% of the block model. To avoid penalties the concentrate from these regions of the block model must be blended with low arsenic concentrate to ensure the shipped concentrate remains below the threshold. Provision has been made in the 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 for Čoka Rakita was estimated in February and March 2025 using drilling data collected since 2009 but dominated by data collected in 2021, 2022, 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 PFS MRE, reported on August 30, 2024, 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, 2025.

The MRE was estimated by ERM under the guidance and supervision of QP, Malcolm Titley. 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 as reported in the MRE.

14.2 Database Cut-Off

The database cut-off was January 17th, 2025, 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_Alt_170125.csv
- CSA_Assay_BM_170125.csv – for gold, SFA used in preference to FA where available
- CSA_Assay_ME_170125.csv – for gold, SFA used in preference to FA where available
- CSA_BulkDens_170125.csv
- CSA_Collar_170125.csv
- CSA_Geotech_170125.csv
- CSA_Lith_170125.csv
- CSA_MagSusc.csv
- CSA_ScreenFireAssay_170125.csv
- CSA_Structure.csv
- CSA_Sulph.csv
- CSA_Survey_170125.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	4	3,213
	2021	26	16,275
	2022	4	3,585
	2023	63	37,914
	2024	97	48,630
Diamond – Subtotal		**205**	**111,749**
Diamond Tail	2023	33	16,520
	2024	13	5,277
Diamond Tail – Subtotal		**46**	**21,797**
RC	2008	4	474
	2023	44	7,485
RC – Subtotal		**48**	**7,959**
Total		**299**	**141,505**

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 – RADD049, RADD050, RADD051 and RIDD103 and not used in the MRE.
- For “missing” assays, there were three (3) categories:
  - Twelve (12) holes had no samples/assays recorded for any part of the drill hole – RADDGTH008A, 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 - 3,559 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 diamond drillholes had logged geology and alteration except for 12. These were drilled in 2024, meaning they are likely to be awaiting data.
- There are 132 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 surrounding data.

Overall, the drillhole database is clean and as complete as possible given drilling continues on the Project and the database was closed on the January 17th,2025, the effective date of the MRE. It is, therefore, reasonable that some drillholes contain missing logging data as they wait to get processed.

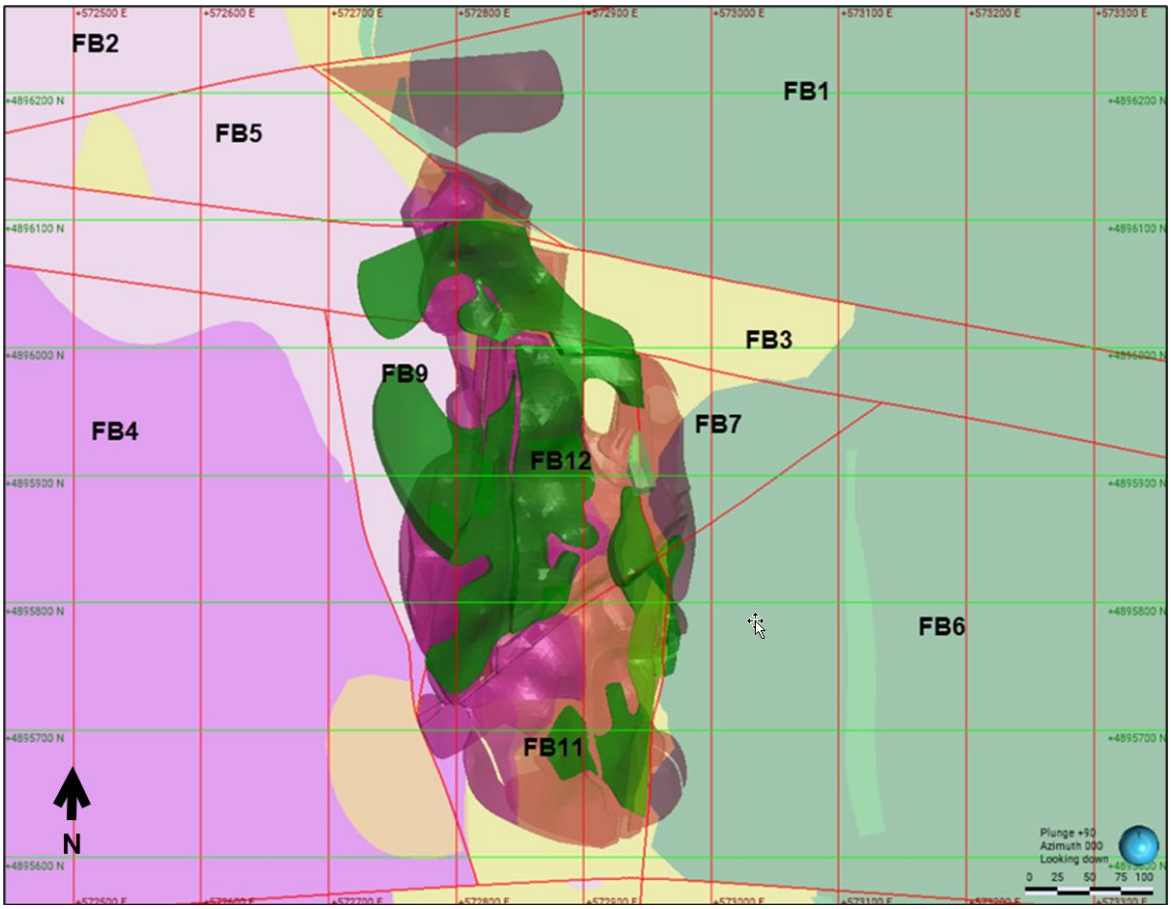
14.2.1 Data Excluded

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 did not intercept the area of interest, 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 interpretation. Major offsetting faults, that crosscut the area were used to further refine these blocks.

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



Source: ERM, 2025

The Leapfrog Geo project containing the lithology, structure and oxidation model for the area was provided by DPM. The QP 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:

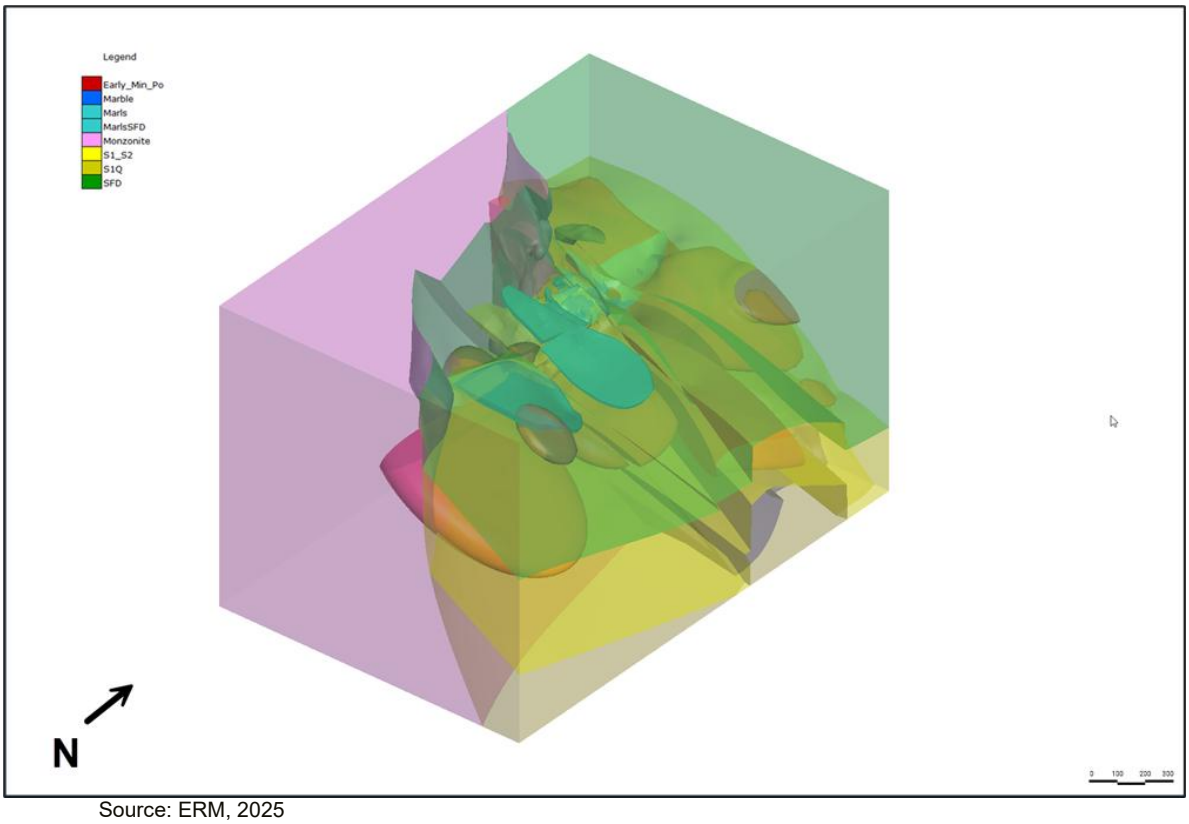
- The lithology model was exported individually so that the mineralisation wireframes could be reviewed by lithology, which was not possible to do when mineralisation was integrated into the lithology model.
- Removal of the topography surface used to constrain the upper limit of the LeapFrog model to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

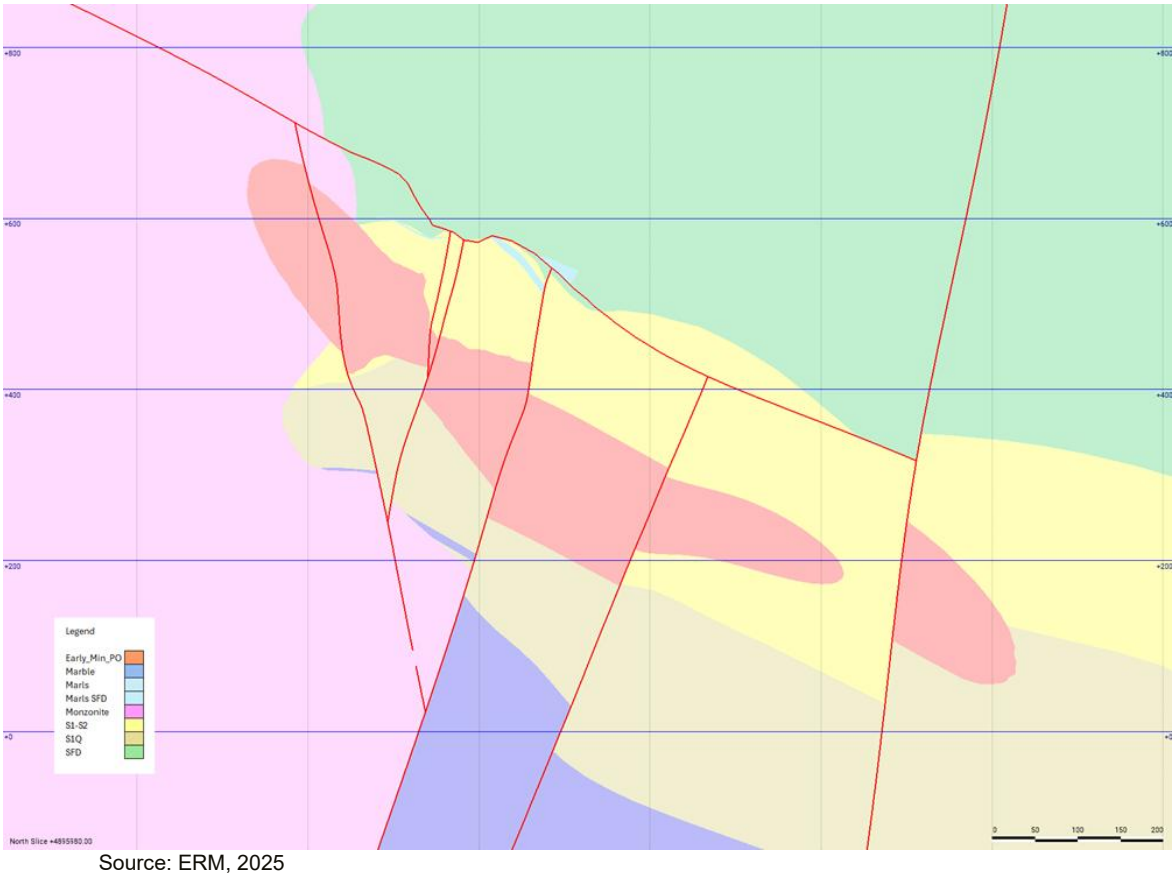
- Early mineralised porphyry (Early_Min_PO) – modelled as an intrusive body with a moderate dipping trend towards the east-northeast, trends varied slightly between faulted blocks, using an elongate spheroidal ellipse.
- Monzonite – modelled as an intrusive body using a spherical model.
- Marls – modelled 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, 2025

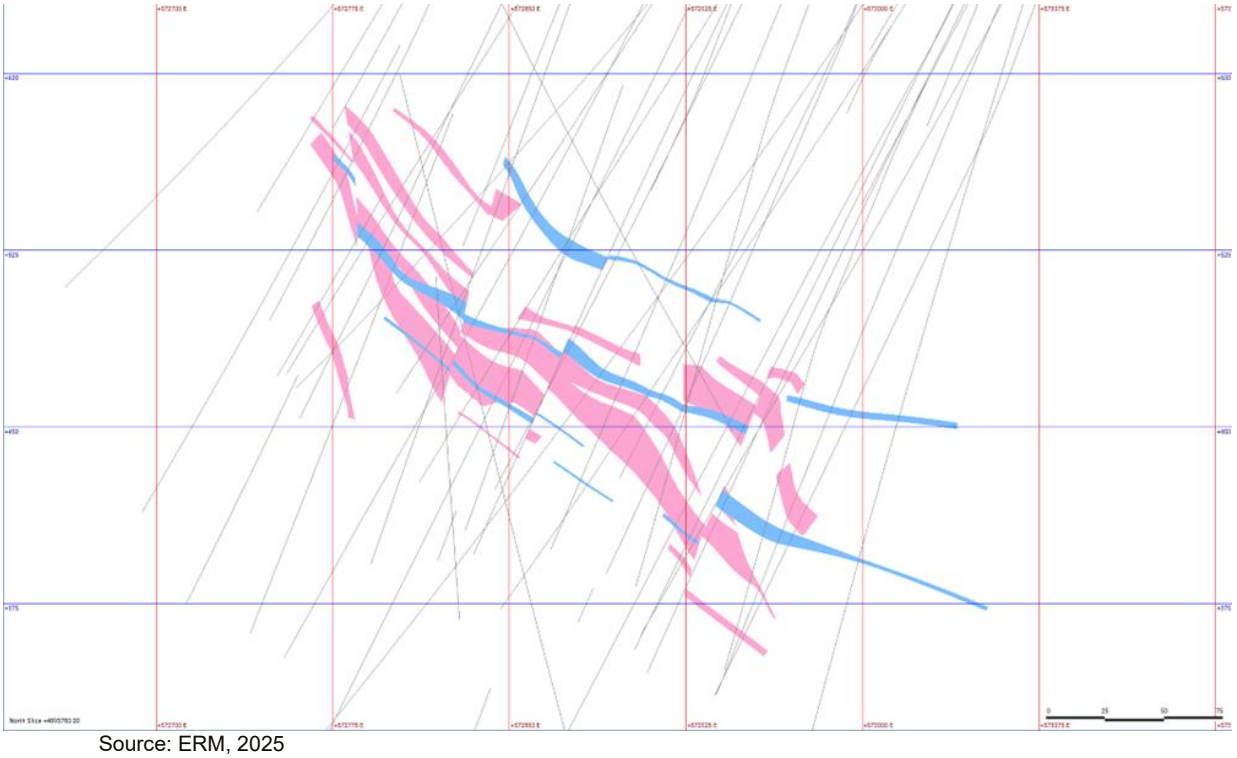
Figure 14.3 – Cross Section Showing Lithology Units and Drillholes, View Looking North, 4895780MN



Source: ERM, 2025

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) and Late-Stage Intrusives (Blue) and Drillhole Traces (Black) – View towards North, Slice View ±30 M



Source: ERM, 2025

14.3.2 MINERALISATION

The Leapfrog Geo model supplied by DPM and verified by the QP, Malcolm Titley, contained 3D models of mineralised material as part of the lithology model which were constructed based upon geological observations – mainly metasomatic intensity of skarn formation 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 including internal waste material where it was considered to be part of the larger mineralisation volume. 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/metasomatic intensity. 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 (3) 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.

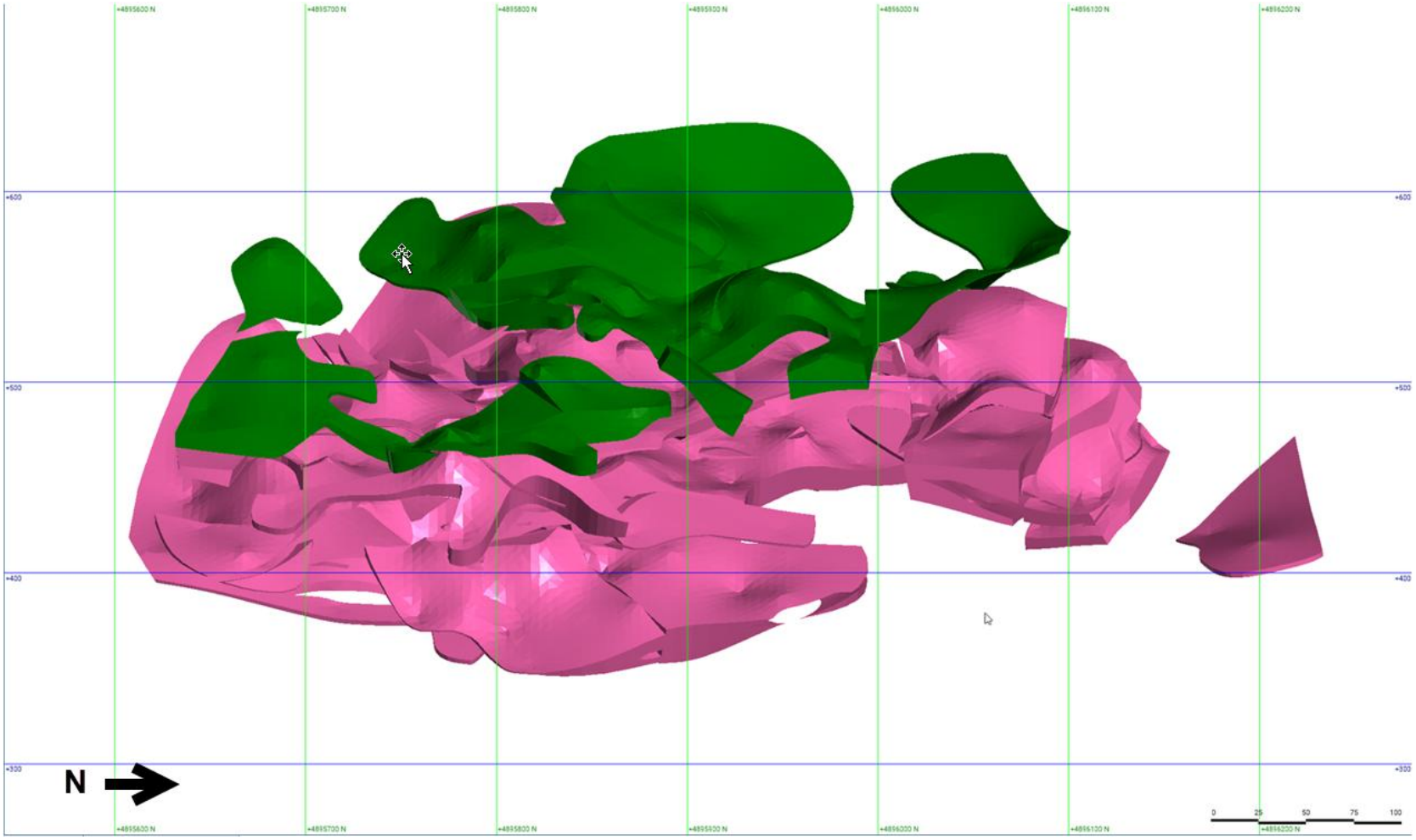
Three (3) large coherent zones were modelled using the vein modelling tool as follows:

- 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.

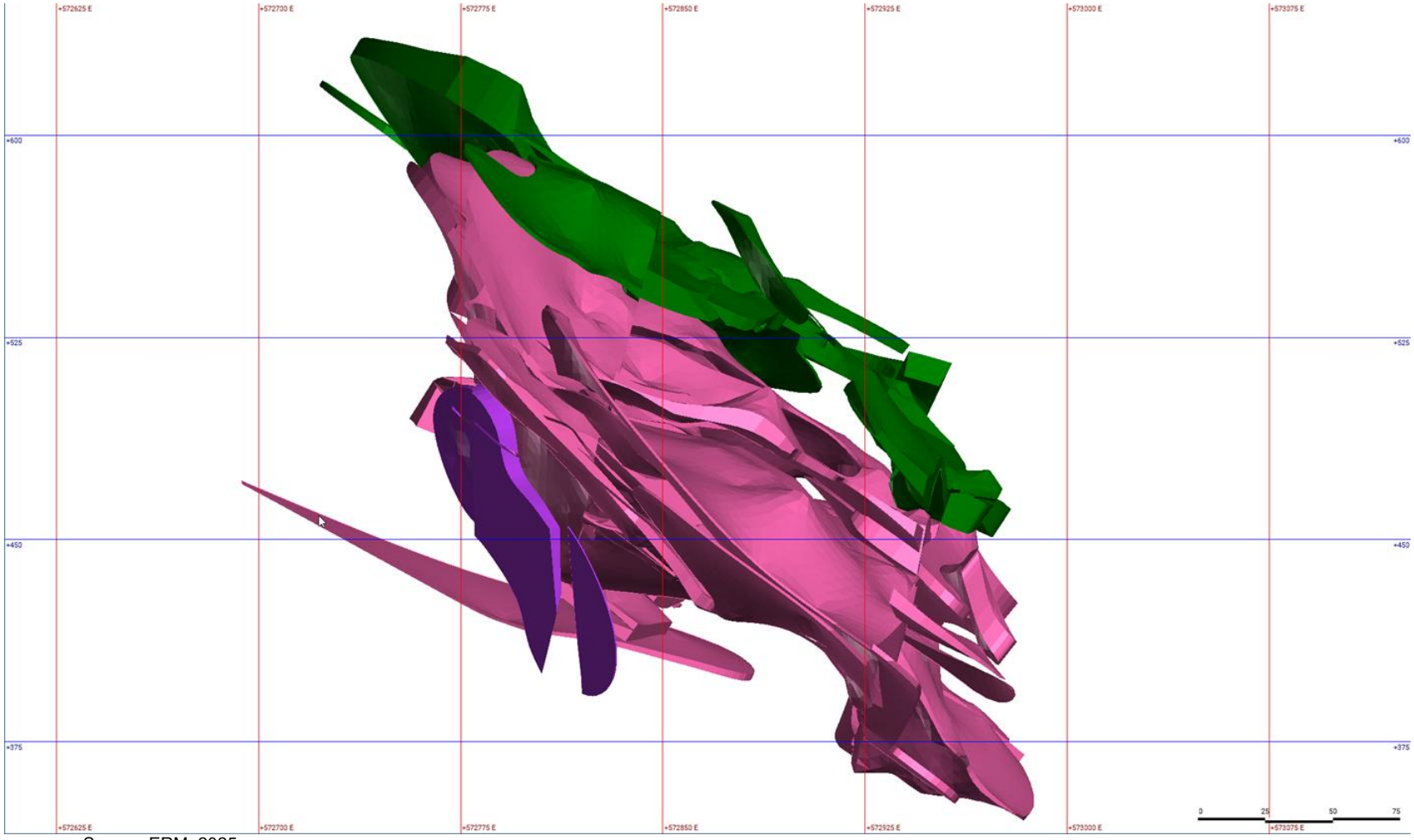
The mineralisation model is shown in Figure 14.5 to Figure 14.7.

Figure 14.5 – Long Section Showing Mineralised Domains and Drillholes, View Looking West. Hanging Wall Domain – Green, Footwall Domain – Pink



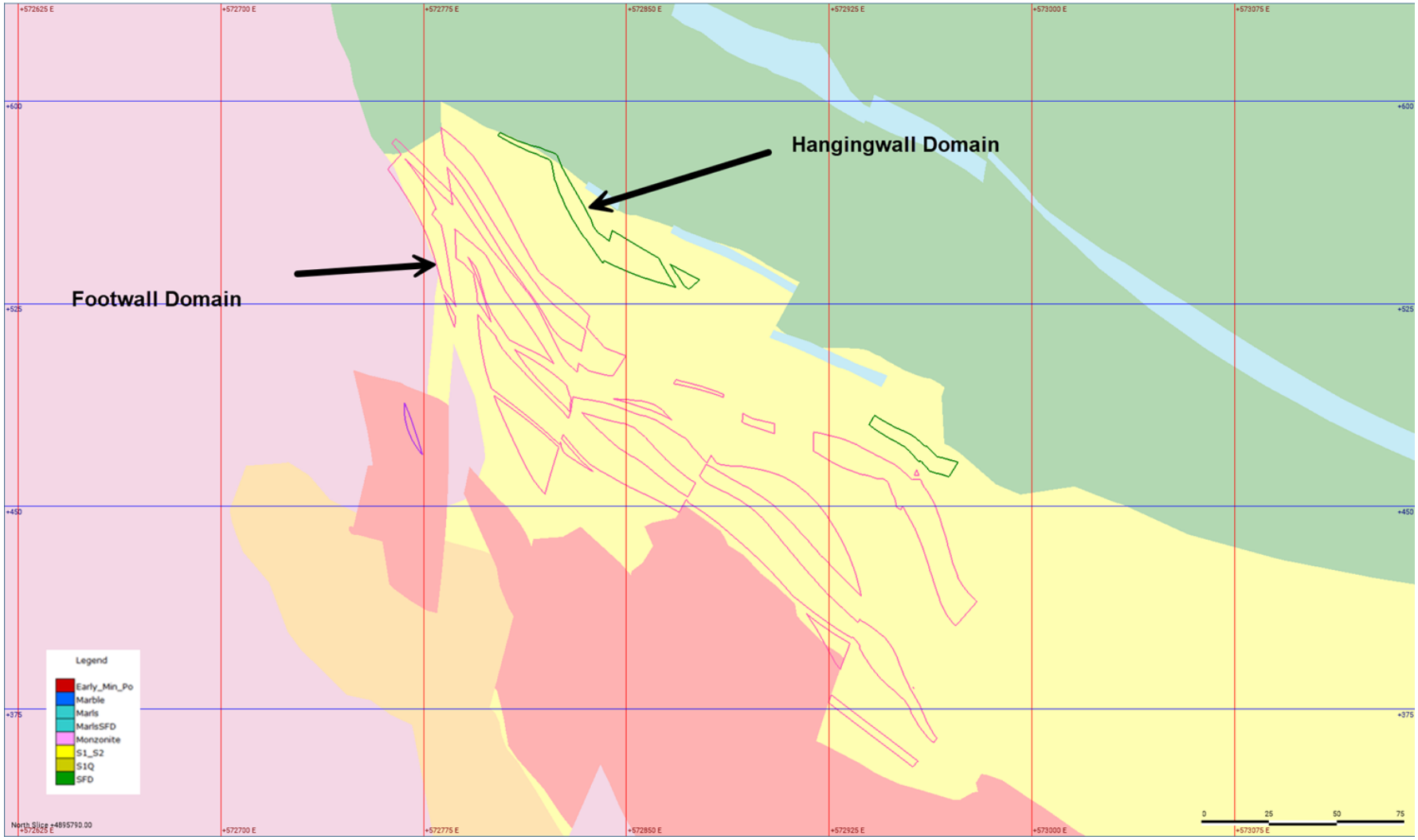
Source: ERM, 2025

Figure 14.6 – Mineralised Domains, View Looking North. Hanging Wall Domain – Green, Footwall Domains – Pink, Basal Domain – Purple



Source: ERM, 2025

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



Source: ERM, 2025

14.3.3 LITHOGEOCHEMICAL DOMAINS

No additional work was undertaken on the geometallurgical model, the following sections has been taken from the PFS report and is still current for this round of work.

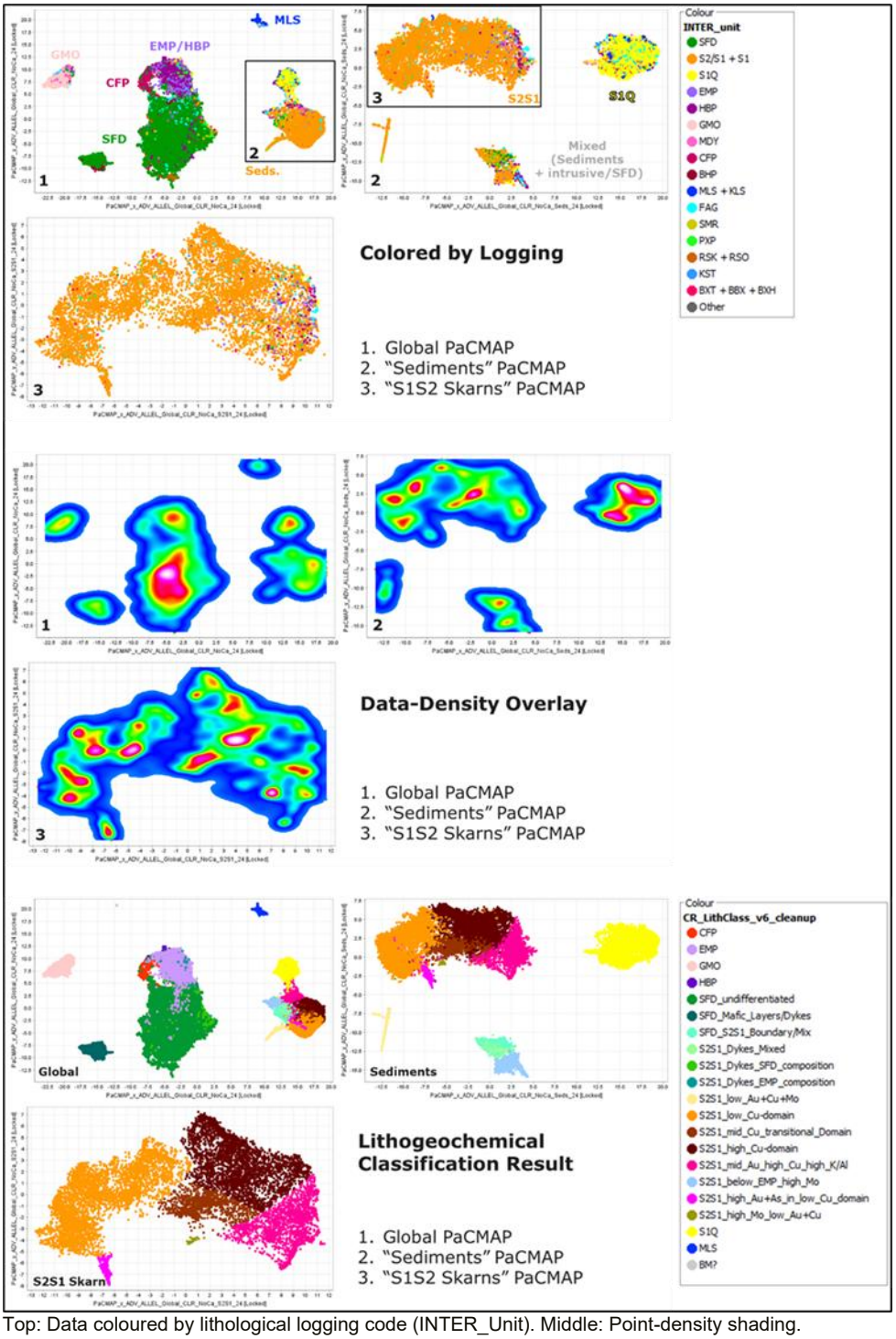
Multi-element assay data were reviewed for their suitability to generate lithogeochemical classification to inform geological modelling, resource domaining, and geometallurgical assessments. Data quality and coverage of the multi-element assay database was considered adequate for the purposes of mineral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

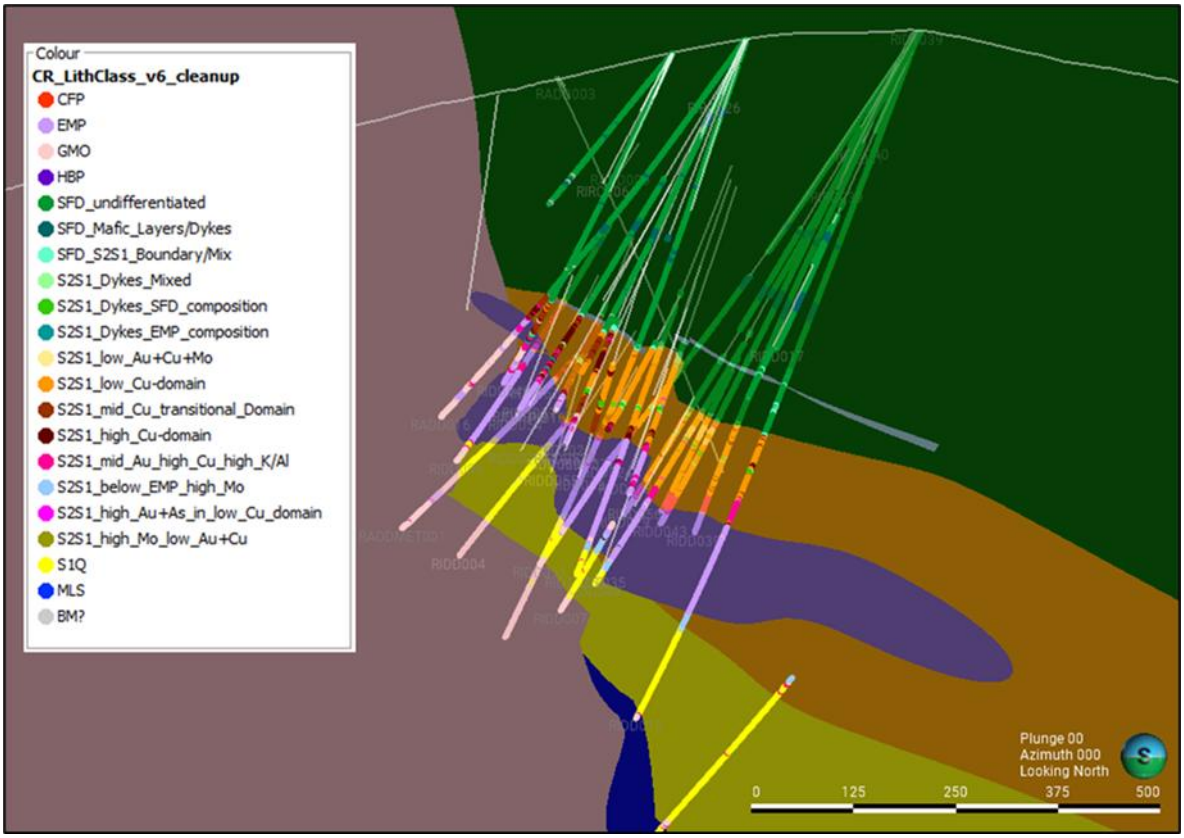
Figure 14.8 – PacMAP (Wang et al., 2021) X-Y Projection and Classification of CLR-Transformed Multi-Element Data



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.
Source: Wang et al., 2021

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

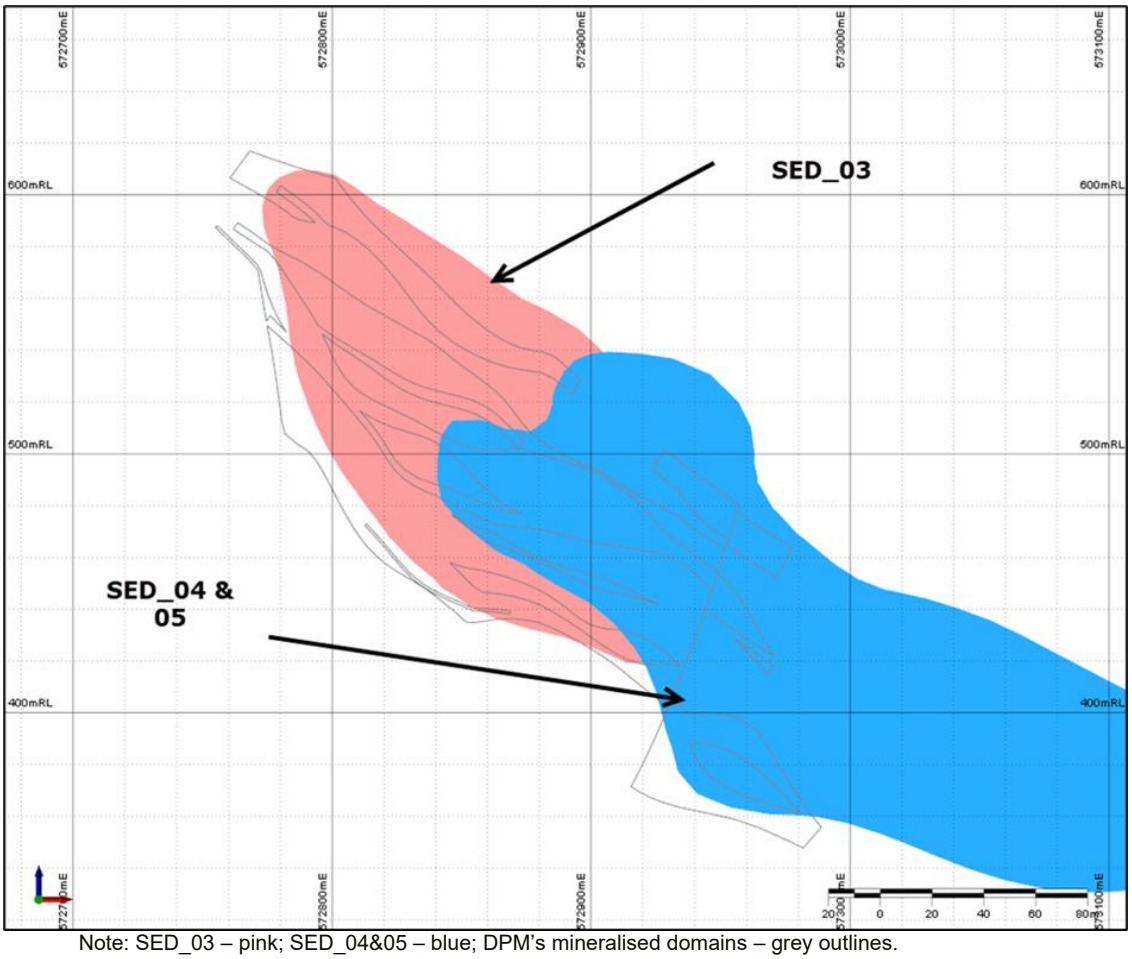
mineralisation may not be linked to the event that generated the Cu enrichment in the S2S1 unit. Very-high gold grades are generally associated with relatively low As, and zones with high values for the Au/As ratio may contain more free gold, whereas gold associated or hosted by sulphides are likely 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); hanging wall 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	1001	101^3
min_fb3_fw1_	MINZON	3001	101^3
min_fb3_fw3_	MINZON	3003	101^3
iw_fb3_fw1_	MINZON	99001	201
iw_fb3_fw2_	MINZON	99002	201
min_fb3_hw_	MINZON	3002	102
min_fb5_fw1_	MINZON	5001	101^3
iw_fb5_fw1_	MINZON	99003	201
iw_fb5_fw2_	MINZON	99004	201
iw_fb5_fw3_	MINZON	99005	201
min_fb5_fw3_	MINZON	5003	101^3
min_fb6_hw_	MINZON	6002	102
min_fb6_fw1_	MINZON	6001	101^3
iw_fb6_fw2_	MINZON	99006	201
min_fb6_fw3_	MINZON	6003	101^3
min_fb6_fw5_	MINZON	6005	101^3
min_fb7_hw_	MINZON	7002	102
min_fb7_hw1_	MINZON	7004	102
min_fb7_fw1_	MINZON	7001	101^3
min_fb7_fw3_	MINZON	7003	101^3
min_fb7_fw4_	MINZON	7004	101^3
min_fb7_fw5_	MINZON	7005	101^3
iw_fb7_fw1_	MINZON	99007	201

Wireframe	Flagging Field	Flagging code	ESTZON
iw_fb7_fw2_	MINZON	99008	201
iw_fb7_fw3_	MINZON	99009	201
iw_fb7_fw6_	MINZON	99010	201
min_fb8_fw1_	MINZON	8001	101^3
min_fb8_fw3_	MINZON	8003	101^3
iw_fb8_fw2_	MINZON	99011	201
min_fb9_hw_	MINZON	9002	102
min_fb9_fw1_	MINZON	9001	101^3
min_fb9_fw3_	MINZON	9003	101^3
min_fb9_dpmfw_	MINZON	9099	103
iw_fb9_fw2_	MINZON	99013	201
iw_fb9_fw1_	MINZON	99014	201
min_fb11_hw_	MINZON	1102	102
min_fb11_fw1_	MINZON	1101	101^3
min_fb11_fw3_	MINZON	1103	101^3
min_fb11_fw5_	MINZON	1105	101^3
min_fb11_dpmfw_	MINZON	1199	103
iw_fb11_fw2_	MINZON	99015	201
iw_fb11_fw3_	MINZON	99016	201
iw_fb11_fw6_	MINZON	99017	201
min_fb12_hw_	MINZON	1202	102
min_fb12_fw1_	MINZON	1201	101^3
min_fb12_fw3_	MINZON	1203	101^3
min_fb12_fw5_	MINZON	1205	101^3
min_fb12_dpmfw_	MINZON	1299	103
iw_fb12_fw1_	MINZON	99018	201
iw_fb12_fw2_	MINZON	99019	201
geo_fb1_pxp2_	LATE_INT	1002	200
geo_fb3_pxp1_	LATE_INT	3001	200
geo_fb3_pxp2_	LATE_INT	3002	200
geo_fb5_pxp1_	LATE_INT	5001	200
geo_fb5_pxp2_	LATE_INT	5002	200
geo_fb6_pxp1_	LATE_INT	6001	200
geo_fb6_pxp2_	LATE_INT	6002	200

Wireframe	Flagging Field	Flagging code	ESTZON
geo_fb6_pxp4_	LATE_INT	6004	200
geo_fb6_mdy3_	LATE_INT	6030	200
geo_fb7_pxp1_	LATE_INT	7001	200
geo_fb7_pxp2_	LATE_INT	7002	200
geo_fb7_pxp3_	LATE_INT	7003	200
geo_fb7_pxp4_	LATE_INT	7004	200
geo_fb7_mdy1_	LATE_INT	7010	200
geo_fb8_pxp1_	LATE_INT	8001	200
geo_fb8_pxp3_	LATE_INT	8003	200
geo_fb9_pxp1_	LATE_INT	9001	200
geo_fb9_pxp2_	LATE_INT	9002	200
geo_fb9_pxp3_	LATE_INT	9003	200
geo_fb9_mdy1_	LATE_INT	9010	200
geo_fb9_mdy2_	LATE_INT	9020	200
geo_fb11_pxp1_	LATE_INT	1101	200
geo_fb11_pxp4_	LATE_INT	1104	200
geo_fb11_pxp5_	LATE_INT	1105	200
geo_fb11_mdy1_	LATE_INT	1110	200
geo_fb11_mdy2_	LATE_INT	1120	200
geo_fb11_mdy3_	LATE_INT	1120	200
geo_fb12_pxp1_	LATE_INT	1201	200
geo_fb12_pxp2_	LATE_INT	1202	200
geo_fb12_pxp3_	LATE_INT	1203	200
geo_fb12_mdy1_	LATE_INT	1210	200
geo_fb12_mdy2_	LATE_INT	1220	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
cr_geo_early_min_po_	GEOL	7	3017
geo_s1q_	GEOL	8	308
geo_marble_	GEOL	9	309
geo_monzonite_	GEOL	10	310

Wireframe	Flagging Field	Flagging code	ESTZON
geo_metased_[2]	GEOL	11	311

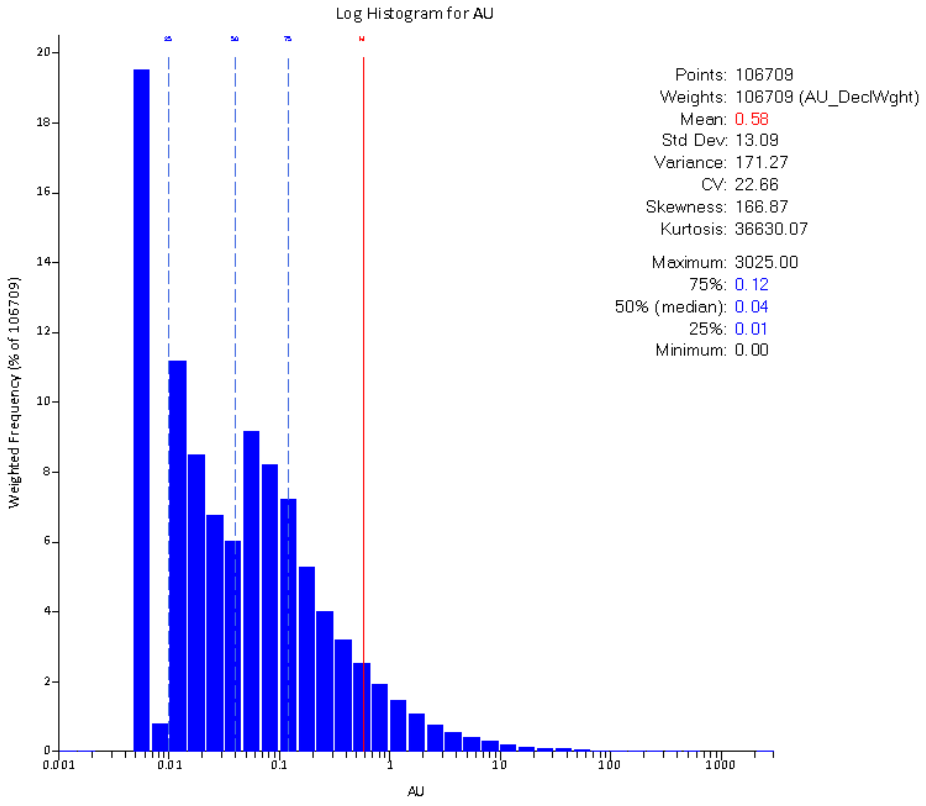
Notes:
1 ESTZON 200 is a waste domain (late-stage intrusives) but deviates from the naming convention for other waste domains due to its location largely crosscutting mineralisation.
2 While there is data in the metasediments, the block model does not extend into them.
3 ESTZON 101 is subsequently split 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.

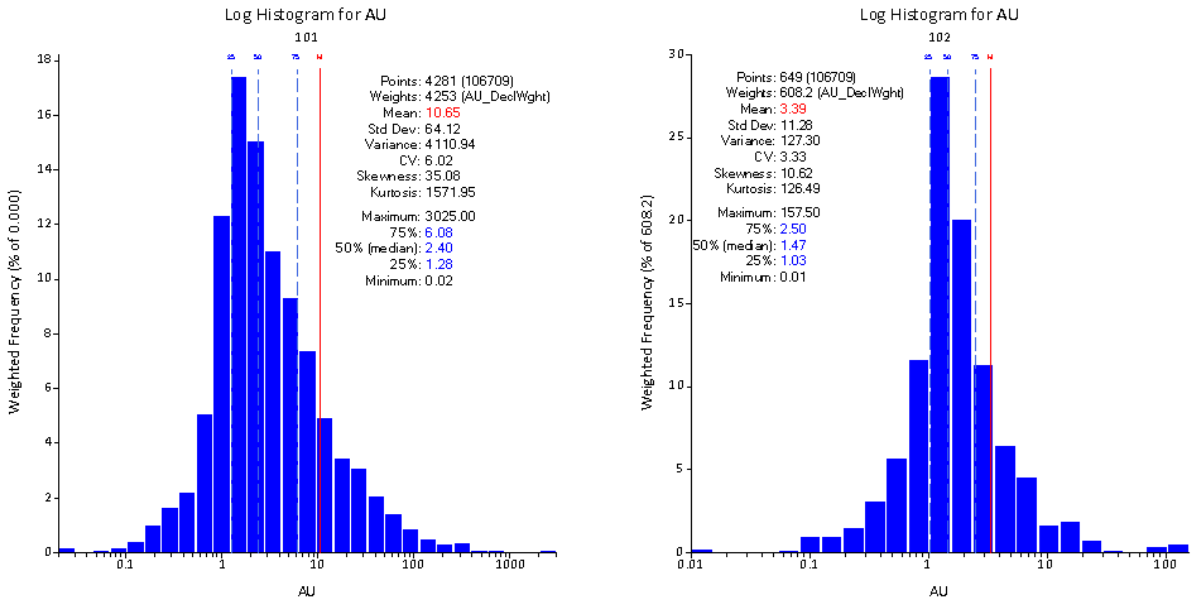
A cell declustering grid of either 10 x 10 x 5 m (for ESTZONs 101, 200 and 201) or 20 x 20 x 5 m (for all other ESTZONs) was used to account for clustering in the dataset. Statistics of raw gold data are presented in Table 14.3.

Figure 14.11 – Log Normal Histogram of Gold – Whole Dataset



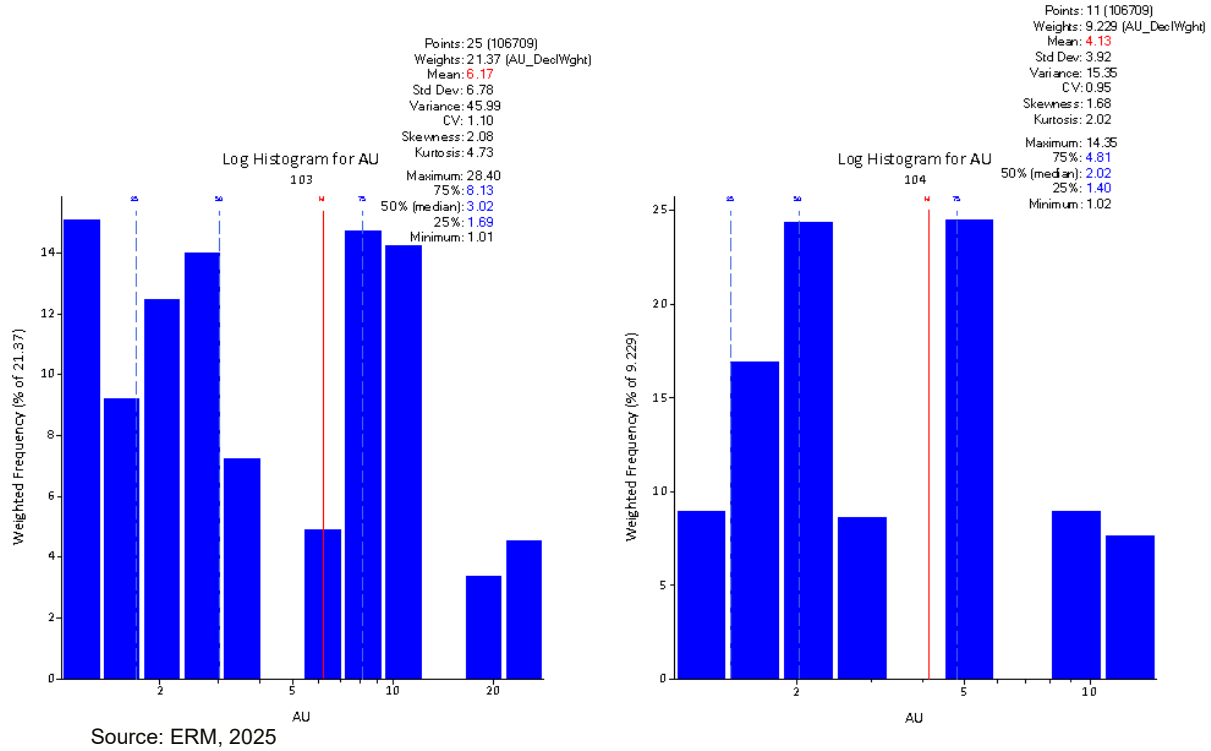
Source: ERM, 2025

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



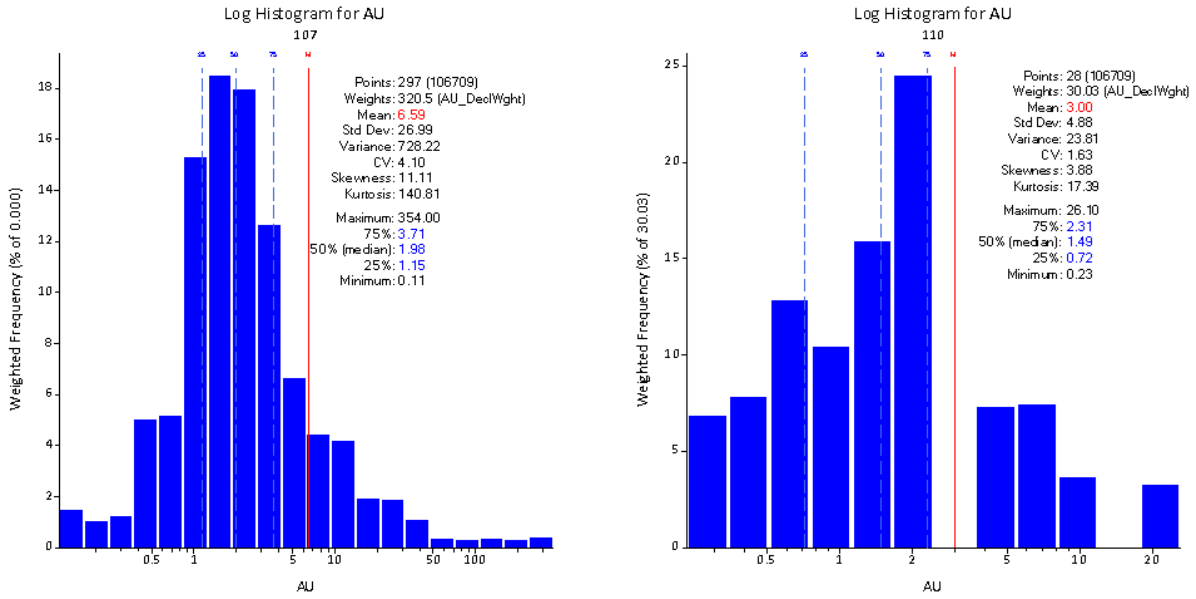
Source: ERM, 2025

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



Source: ERM, 2025

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



Source: ERM, 2025

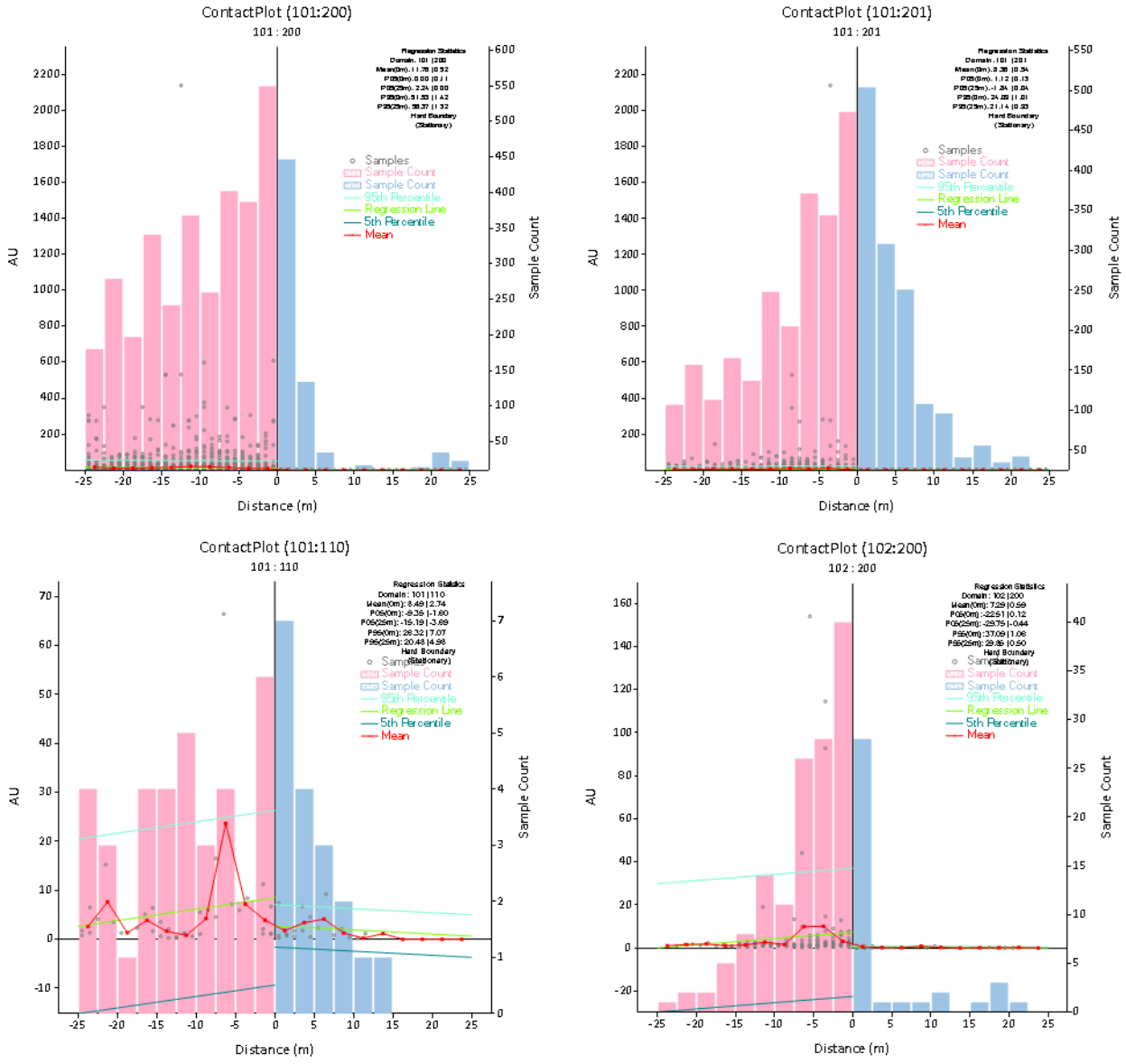
Table 14.3 – Summary Raw Statistics for Gold – Clustered and Declustered where Applicable

ESTZON	101	102	103	104	107	110	200	201
Samples	4273	638	24	11	297	28	1044	901
Minimum	0.02	0.01	1.01	1.02	0.11	0.23	0.01	0.01
Maximum	3025	157.5	28.4	14.35	354	26.1	4.89	4.98
Mean	10.98	3.32	6.2	4.38	6.64	3.07	0.3	0.49
Standard deviation	64.67	11.29	6.67	4.07	26.53	5.04	0.35	0.37
CV	5.89	3.4	1.08	0.93	3.99	1.64	1.19	0.76
Variance	4182	127.46	44.46	16.53	703.89	25.39	0.13	0.14
Skewness	33.18	10.76	2.14	1.77	10.27	3.86	4.2	3.64
Log samples	4273	638	24	11	297	28	1044	901
Log mean	1.12	0.44	1.38	1.15	0.82	0.48	-1.86	-0.98
Log variance	1.78	1.03	0.91	0.7	1.36	1.15	1.71	0.57
Geometric mean	3.06	1.56	3.96	3.15	2.26	1.61	0.16	0.38
10%	0.77	0.52	1.1	1.06	0.61	0.37	0.03	0.13
20%	1.15	0.88	1.62	1.38	1.05	0.67	0.06	0.21
30%	1.45	1.06	1.95	1.58	1.25	0.89	0.09	0.27

ESTZON	101	102	103	104	107	110	200	201
40%	1.86	1.24	2.44	2.01	1.61	1.21	0.14	0.33
50%	2.45	1.44	3.03	2.37	2.04	1.5	0.19	0.41
60%	3.4	1.73	4.85	3.84	2.41	1.76	0.26	0.5
70%	5.06	2.12	7.51	4.76	3.25	2.14	0.35	0.61
80%	8.35	2.94	8.6	4.83	4.62	3.2	0.48	0.74
90%	19	5.1	10.96	8.67	10.18	6.83	0.68	0.88
95%	36.6	7.97	18.94	11.46	20.2	8.46	0.86	0.98
97.50%	66.98	13.61	23.91	12.91	32.83	14.27	1.06	1.16
99%	140.91	26.28	26.6	13.77	75.54	21.37	1.63	1.9

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 14.15).

Figure 14.15 – Contact Analysis Results

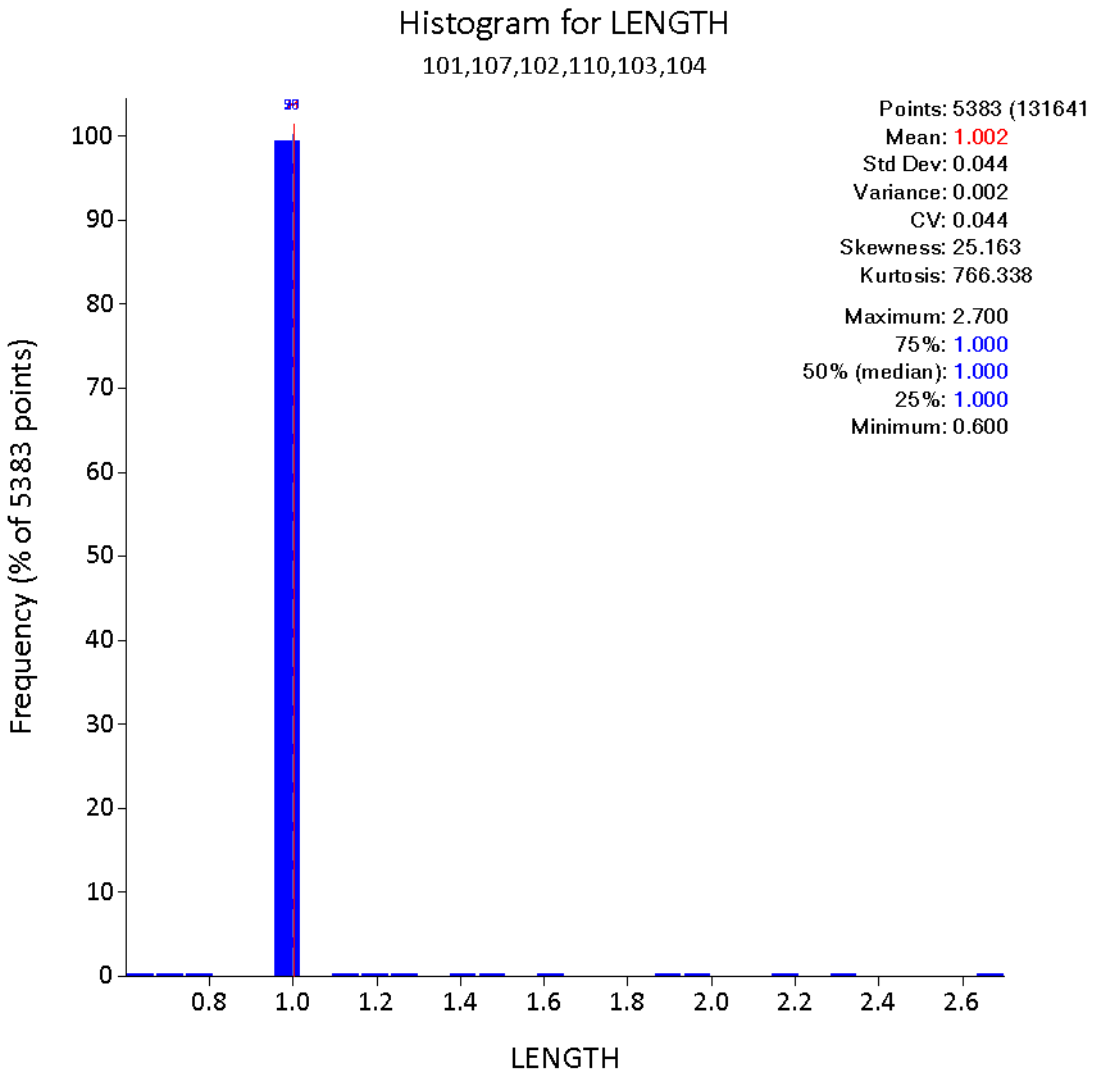


Source: ERM, 2025

14.7 Compositing

Compositing was completed to ensure comparable sample support during estimation. Sampling of drillholes has been predominantly on a 1.0 m basis; as shown in Figure 14.16; therefore 1.0 m composite length was chosen with only two residuals arising in ESTZON 103 (both 0.75 m), which were retained, as they had very similar mean grades to the composited dataset and exclusion would have resulted in the removal of a drillhole.

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



Source: ERM, 2025

14.8 Global Domain Statistics

Domain statistics of the uncut 1 m composites are presented in Table 14.4 (Gold). Uncut gold is characterized by a very high coefficient of variation.

Table 14.4 – Summary of 1 m Composite Statistics – Gold

Statistic (Gold)	ESTZON							
	101	104	107	110	103	102	200	201
Declustering Cell Size	20x20x5							

Statistic (Gold)	ESTZON							
	101	104	107	110	103	102	200	201
Samples	4281	11	297	28	25	649	1047	902
Minimum	0.02	1.02	0.11	0.23	1.01	0.01	0.01	0.01
Maximum	3025	14.35	354	26.1	28.4	157.5	4.89	4.98
Mean	10.65	4.13	6.59	3	6.17	3.39	0.29	0.49
Standard deviation	64.12	3.92	26.99	4.88	6.78	11.28	0.35	0.37
CV	6.02	0.95	4.1	1.63	1.1	3.33	1.21	0.77
Variance	4110.94	15.35	728.22	23.81	45.99	127.3	0.12	0.14
Skewness	35.08	1.68	11.11	3.88	2.08	10.62	4.32	3.93
Log samples	4281	11	297	28	25	649	1047	902
Log mean	1.1	1.1	0.8	0.46	1.35	0.49	-1.9	-0.98
Log variance	1.74	0.67	1.36	1.15	0.98	0.99	1.76	0.58
Geometric mean	3.02	2.99	2.21	1.58	3.84	1.63	0.15	0.38
10%	0.78	1.06	0.61	0.37	1.03	0.58	0.03	0.13
20%	1.15	1.38	1.03	0.65	1.35	0.95	0.05	0.2
30%	1.44	1.68	1.25	0.8	1.82	1.1	0.08	0.27
40%	1.83	2.01	1.58	1.11	2.33	1.26	0.13	0.33
50%	2.4	2.02	1.98	1.49	3.02	1.47	0.19	0.41
60%	3.33	2.96	2.38	1.74	4.8	1.77	0.26	0.49
70%	4.91	4.67	3.19	2.14	7.6	2.18	0.35	0.61
80%	8.06	4.83	4.49	3.46	9.17	3.09	0.47	0.74
90%	18.35	7.97	9.54	6.77	10.96	5.36	0.67	0.88
95%	35.82	10.91	19.33	8.32	19.69	8.59	0.85	0.98
97.50%	66.33	12.63	33.56	13.18	24.31	13.52	0.98	1.18
99%	135.39	13.66	73.76	20.93	26.77	26.01	1.59	1.91

14.9 Treatment of Outliers (Top Cuts)

The MRE is sensitive to high-grade outliers due to the existence of coarse gold particles, and the scattered distribution of extremely high grades within the domains. These outliers can skew the estimated local means, resulting in a grade bias. Imposing a top cut that is too harsh will underestimate metal in the deposit. Hence the need for sensitivity analysis to understand the impact of a range of top cuts.

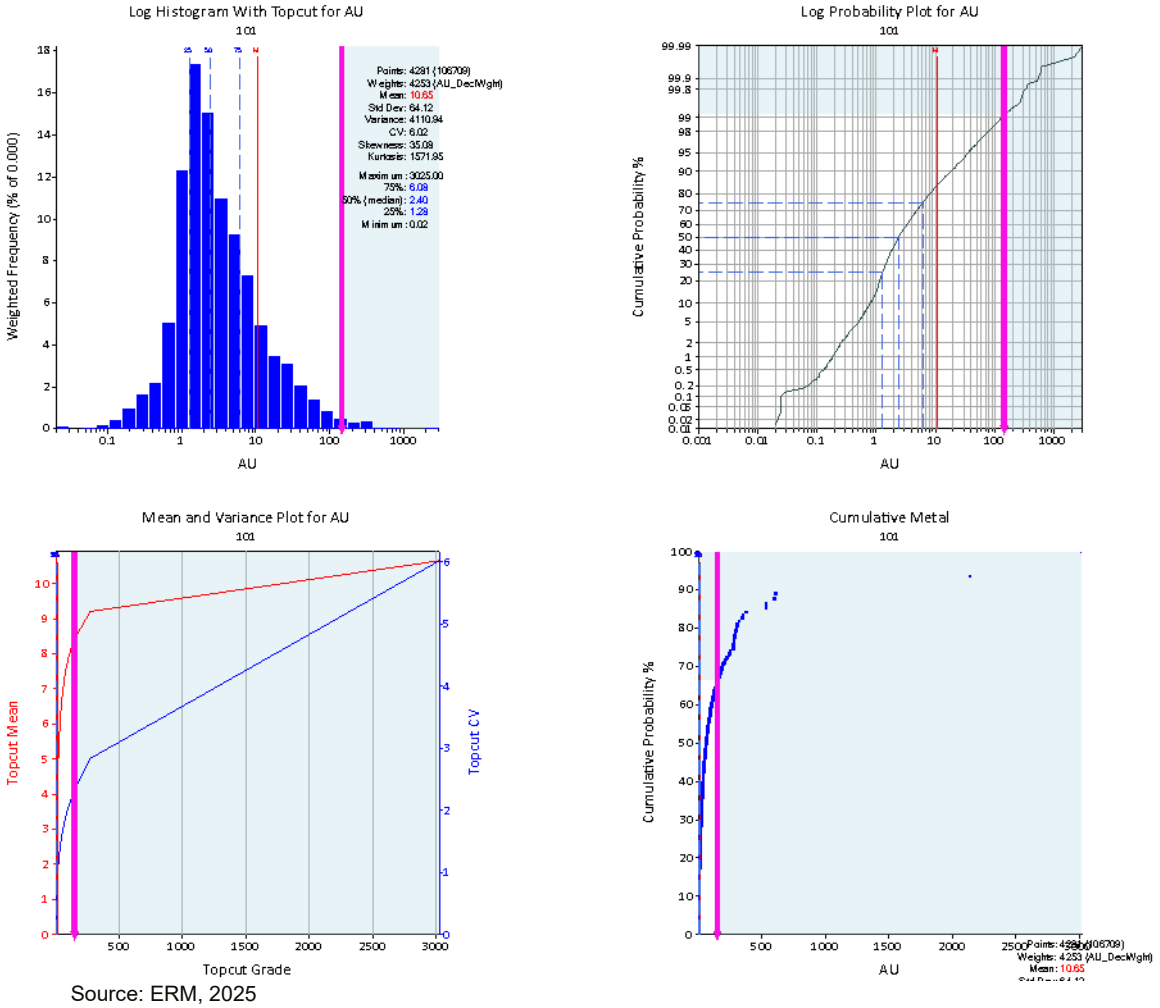
Global top cut analysis was completed in Supervisor software and outliers were reviewed spatially prior to a top cut (grade cap) being applied. Global top cut analysis comprises the simultaneous review of the grade histogram, log probability, mean-variance and cumulative metal plots, and having a balanced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

During the PFS work, 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 led to an increase the top cut, with a greater proportion of high grades being intersected. Test estimates run on the three top cut scenarios shows the chosen top cut of 144 g/t Au results in a metal estimate approximately 3% higher than a 110 g/t Au top cut and 2% lower metal than a 175 g/t Au top cut.

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 in the main mineralisation domains in Figure 14.17 to Figure 14.19 while Table 14.5 presents the top cuts applied to the mineralisation domains. Table 14.6 shows 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.

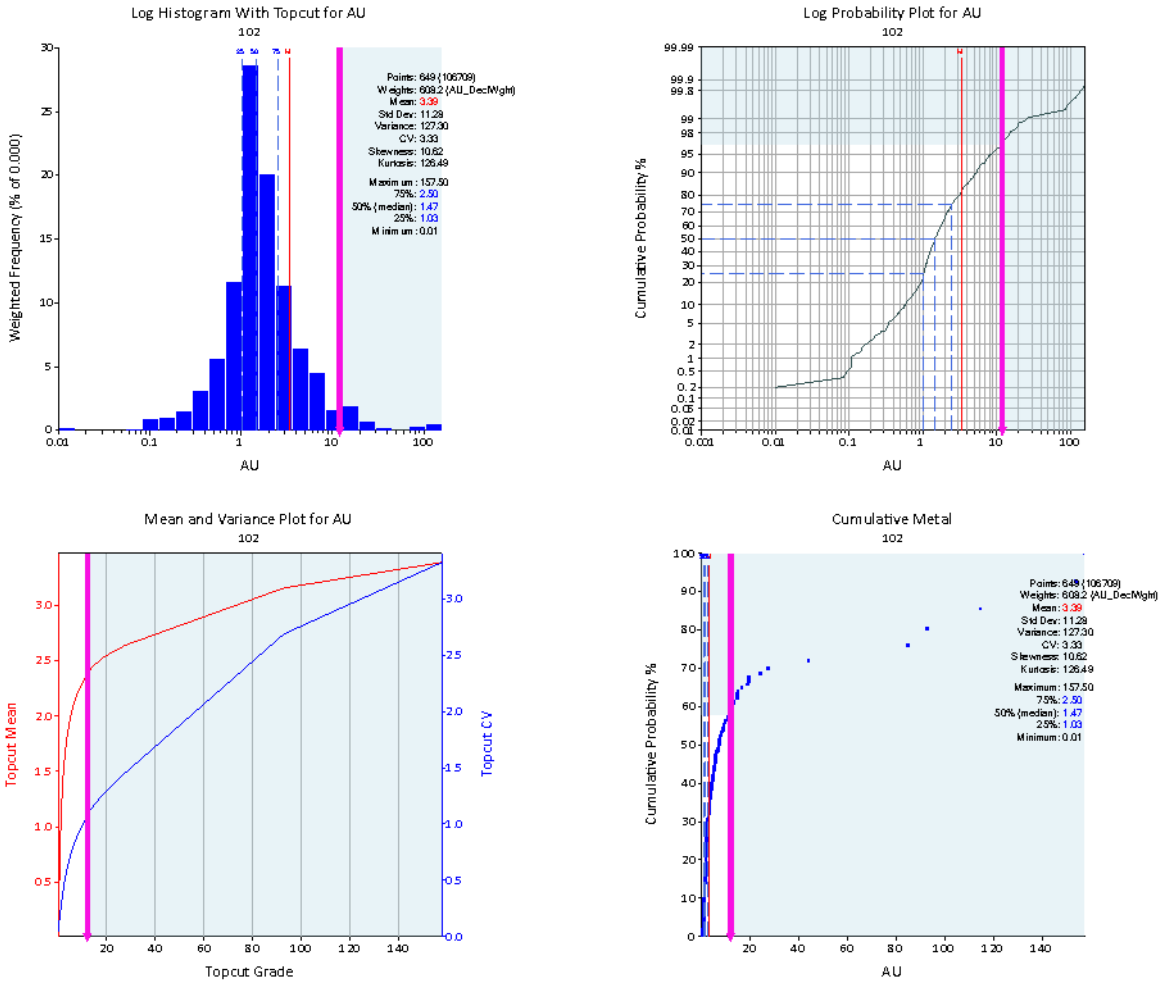
Additional capping within the estimate is also used. This is defined in the search parameters and is described further in Section 14.14.

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



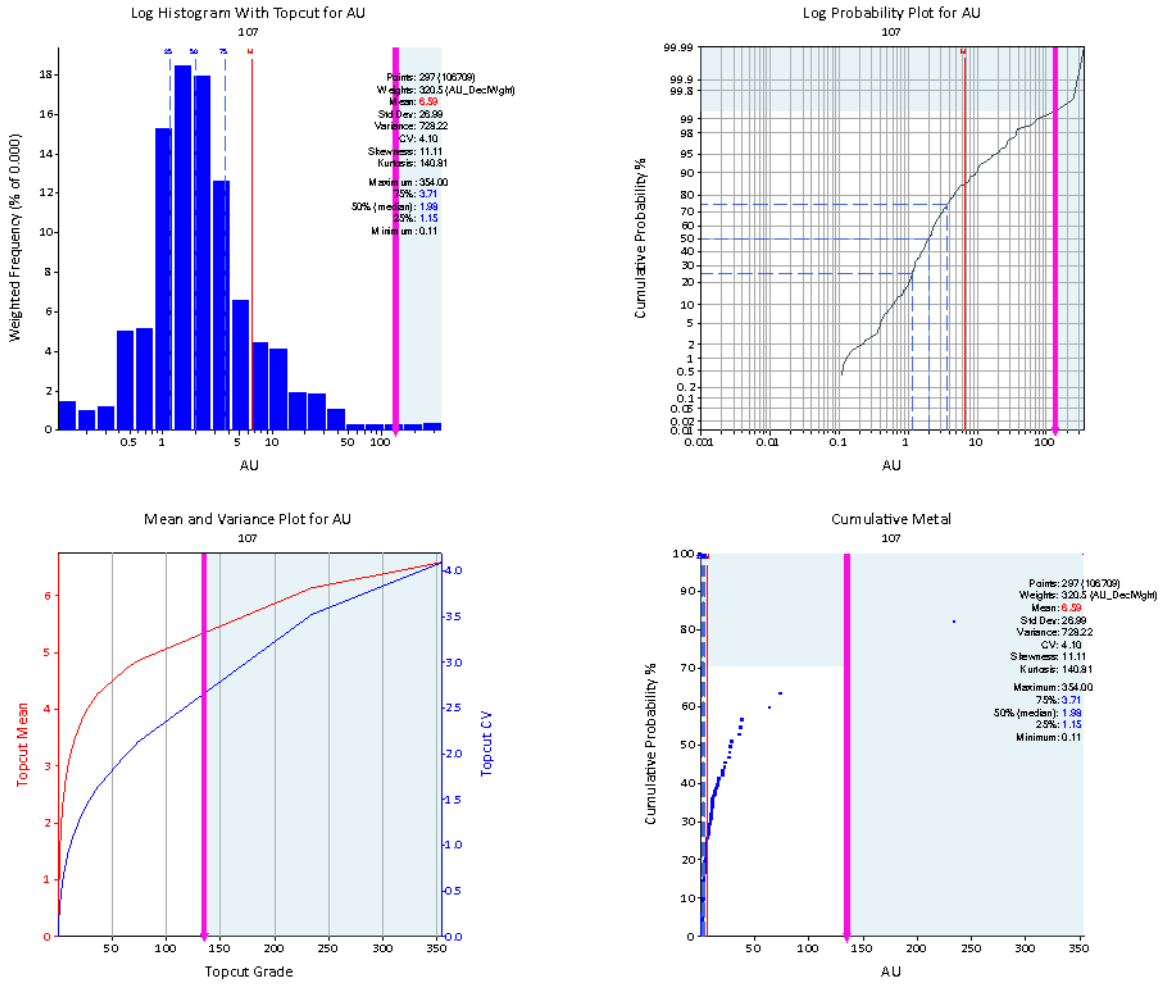
Source: ERM, 2025

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



Source: ERM, 2025

Figure 14.19 – Global Top Cut Analysis for Gold for ESTZON 107



Source: ERM, 2025

Table 14.5 – Top Cuts Used for Gold 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	4281	10.65	6.02	144	8.32	2.4	41	1.00	21.90
104	Gold	11	4.13	0.94	5	2.9	0.56	2	18.20	29.60
107	Gold	297	6.59	4.1	135	5.52	2.75	3	1.00	16.30
110	Gold	28	3	1.63	26.1	3	1.65	1	3.60	0.00
102	Gold	649	3.4	3.33	12	2.4	1.06	23	3.50	29.30
103	Gold	25	6.17	1.09	11	5.08	0.73	3	12.00	17.70
200	Gold	1053	0.53	11.32	1	0.2	1.41	33	3.10	62.50
201	Gold	902	0.53	1.93	1	0.43	0.65	43	4.80	19.00

Table 14.6 – Summary Statistic for Top Cut Composites – Gold

ESTZON	101	200	201	102	103	104	107	110
Declustering	10 x 10 x 5			20 x 20 x 5				
Samples	4281	1045	902	649	23	11	297	28
Minimum	0.02	0.01	0.01	0.01	1.01	1.02	0.11	0.23
Maximum	144	1	1	12	11	5	135	26.1
Mean	7.48	0.3	0.49	2.35	4.86	3.02	5.76	3.34
Standard deviation	18.22	0.28	0.27	2.52	3.68	1.68	16.49	4.91
CV	2.44	0.9	0.56	1.07	0.76	0.56	2.86	1.47
Variance	331.89	0.08	0.07	6.36	13.53	2.81	271.91	24.07
Skewness	4.92	1.25	0.41	2.92	0.66	0.12	7.92	3.55
Log samples	4281	1045	902	649	23	11	297	28
Log mean	1.01	-1.77	-0.92	0.45	1.29	0.94	0.84	0.6
Log variance	1.55	1.66	0.49	0.89	0.63	0.38	1.24	1.19
Geometric mean	2.74	0.17	0.4	1.57	3.63	2.57	2.31	1.82
10%	0.81	0.03	0.15	0.59	1.27	1.02	0.71	0.35
20%	1.14	0.06	0.22	1	1.74	1.29	1.11	0.72
30%	1.39	0.1	0.29	1.13	1.95	1.45	1.26	1.02
40%	1.69	0.16	0.36	1.28	2.34	1.88	1.72	1.44
50%	2.17	0.22	0.46	1.48	3.02	2.05	2.09	1.54
60%	2.86	0.29	0.55	1.77	3.58	2.94	2.41	2.09
70%	4.19	0.39	0.64	2.15	7.06	4.68	3.25	2.35
80%	6.78	0.54	0.76	2.92	8.17	4.83	4.47	4.5
90%	14.8	0.76	0.88	5.01	10.94	5	9.54	7.08
95%	31.8	0.9	0.96	8.38	11	5	19.37	7.99
97.50%	58.05	1	1	12	11	5	33.39	13.16
99%	120.03	1	1	12	11	5	106.15	20.92

14.10 Variography

Experimental semi-variograms (variograms) were generated for gold 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 backtransformed for use in the estimate. Variography was completed on ESTZON 101, 102, 107, 200, 201 for gold (presented in Table 14.7). 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:

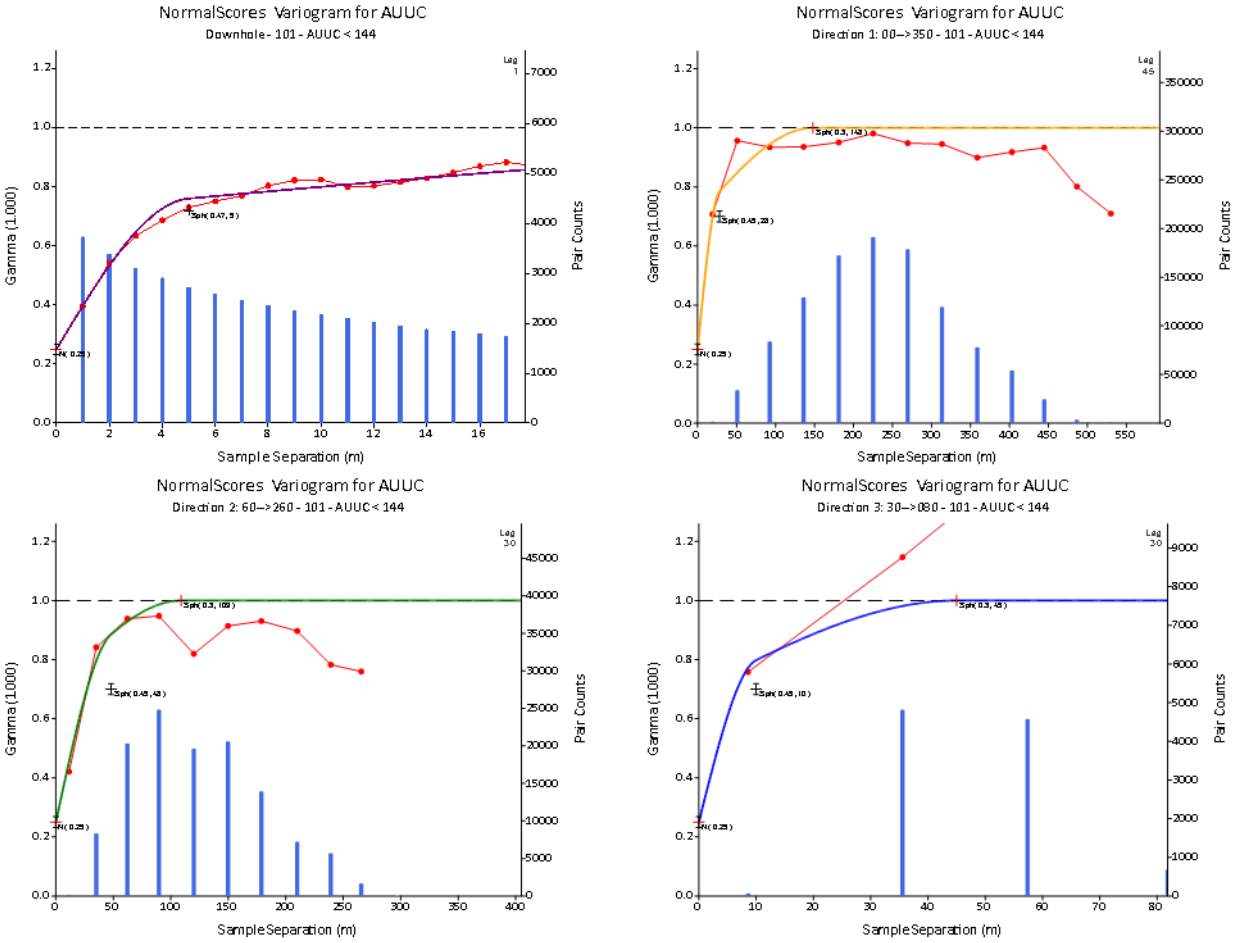
- Variograms were generated to determine the major, semi-major, and minor axes of continuity which are perpendicular to each other.
- The variogram in the downhole direction was modelled to determine the nugget to determine the close-spaced variability.
- The major, semi-major, and minor axes of continuity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

The modelled orientations were consistent with the geological understanding of the mineralisation (as depicted in Figure 14.20 to Figure 14.21). 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.

Table 14.7 – Variogram Parameters in Datamine ZYZ Rotation

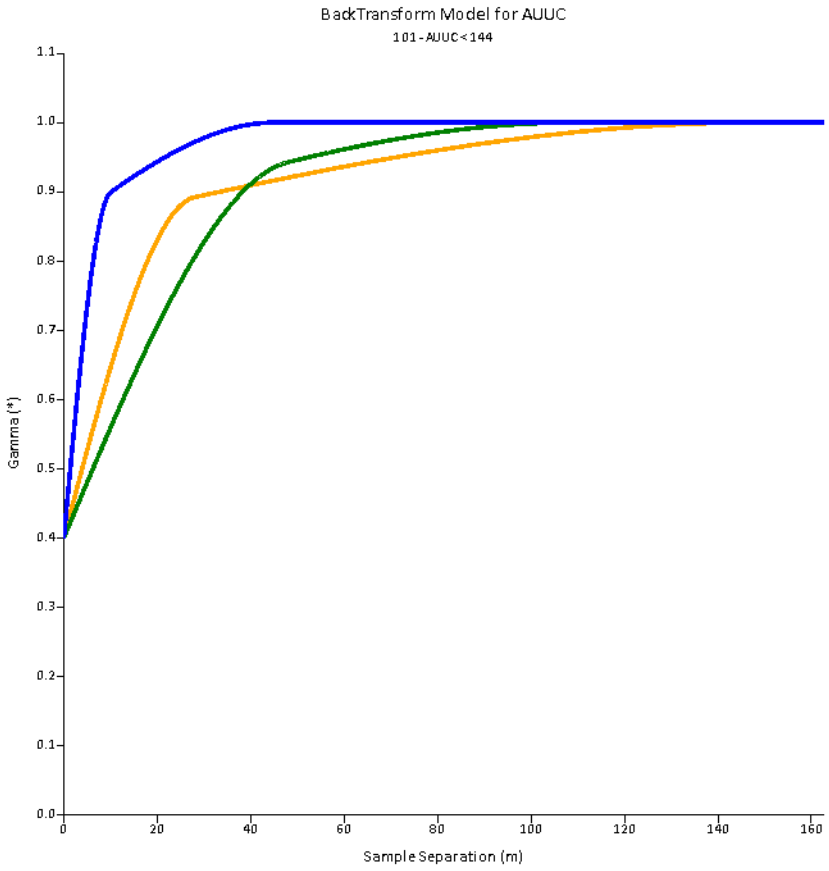
ESTZON	C0	C1	Rotation			Range			C2	Range		
			Z	X	Z	Major	Semi	Minor		Major	Semi	Minor
101	0.399	0.451	80	60	180	28	48	10	0.15	148	109	45
102	0.329	0.415	110	30	180	24	29	5	0.256	103	58	22
107	0.45	0.418	90	40	180	68	108	15	0.131	148	109	45
200	0.17	0.283	110	30	130	46	35	4	0.548	252	188	39
201	0.274	0.323	100	30	180	32	48	2	0.403	59	57	11

Figure 14.20 – Experimental Semi-Variograms for Gold in ESTZON 101



Source: ERM, 2025

Figure 14.21 – Variogram Model for Gold in ESTZON 101



Source: ERM, 2025

14.11 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 methods were used: (1) averaging the slopes of regression and kriging efficiencies across a portion of the domain to identify the optimal block size/sample search strategy; and (2) choosing a single block location representing a well-informed block, a moderately informed block and a poorly informed block and reviewing the kriging quality statistics.

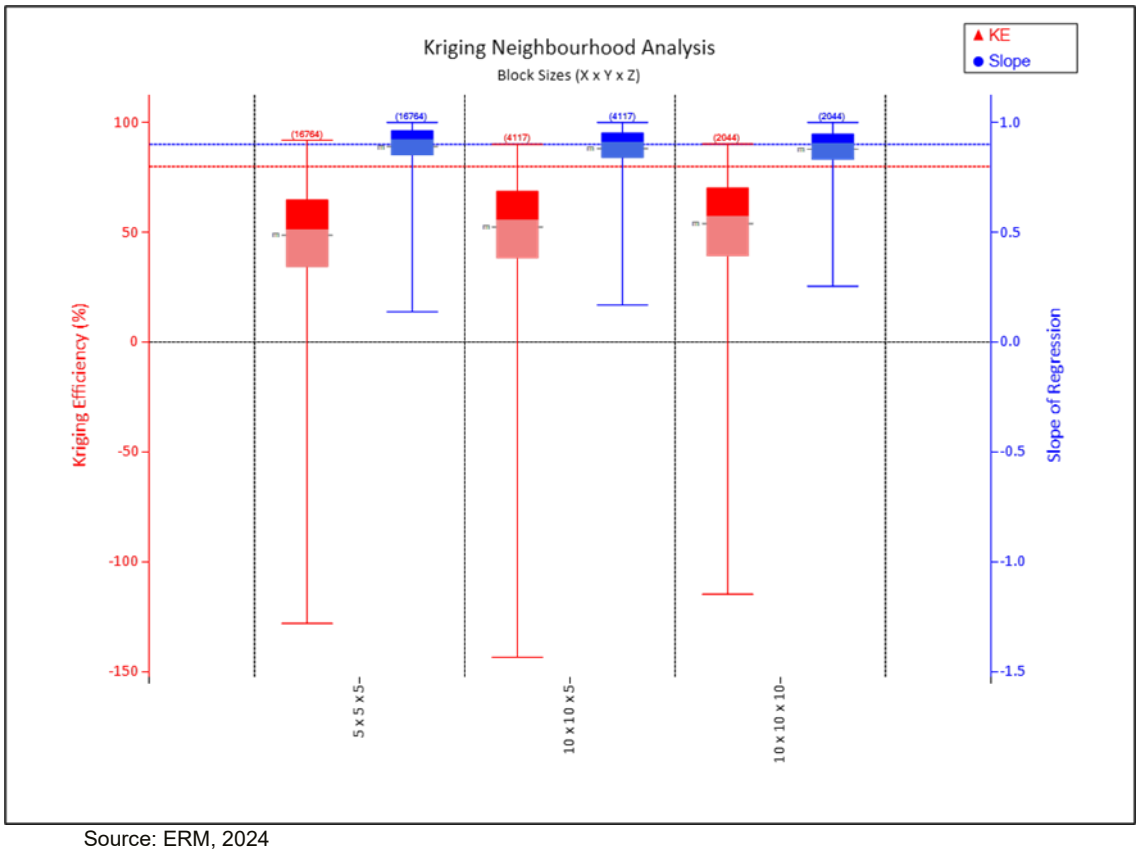
A block size of 10 m x 10 m x 5 m was chosen which is compatible with the proposed mining method and also results in good slopes of regression and kriging efficiencies when compared to other block sizes reviewed (Figure 14.22).

Minimum and maximum composite selection of 12 and 32 were chosen (Figure 14.23). 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.

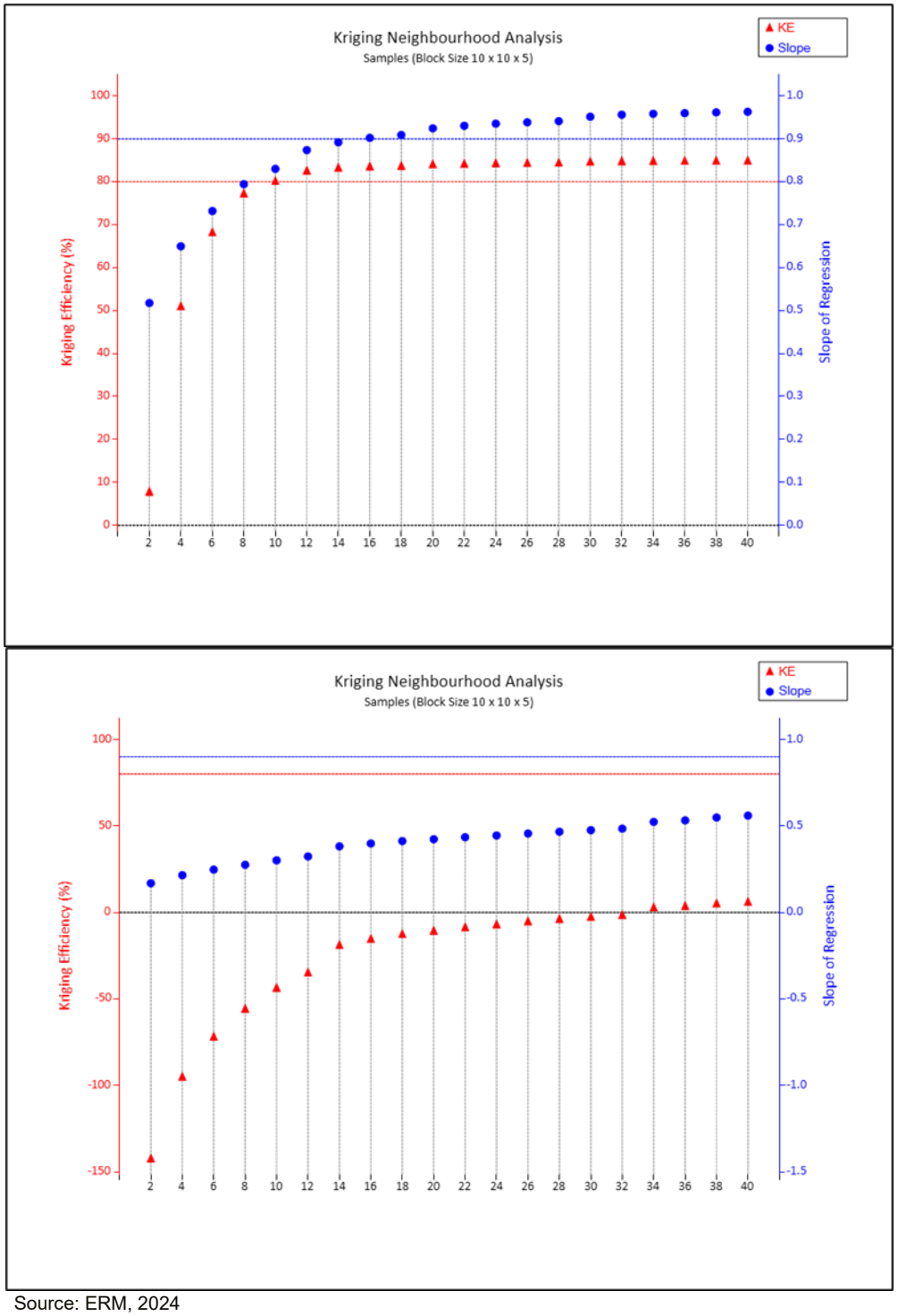
The extent of the second search pass was chosen to align broadly with the variogram ranges. The first was chosen to reflect the closer drill spacing (20 – 30 m).

Figure 14.22 – KNA Block Size Review



Source: ERM, 2024

**Figure 14.23 – KNA Sample Search Strategy Review
("Good" Block – Above, "Bad" Block – Below)**



Source: ERM, 2024

14.12 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 14.8.

Table 14.8 – Block Model Prototype

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

Table 14.9 – 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

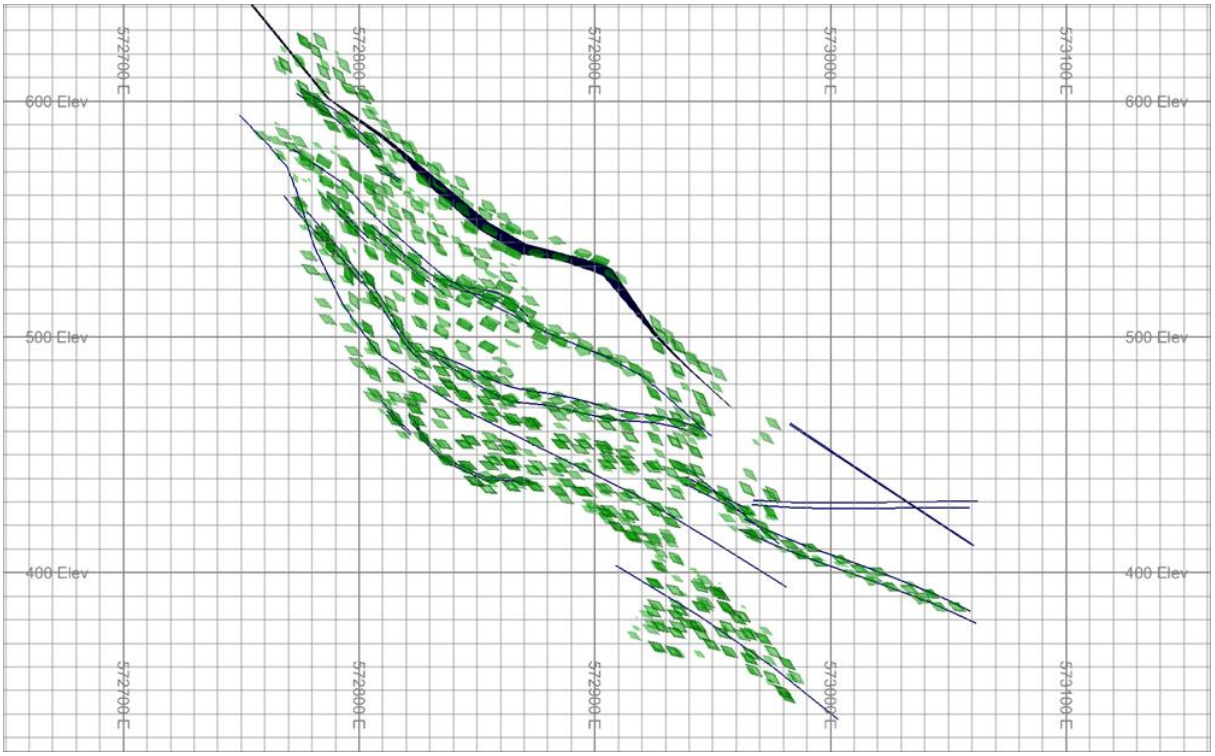
Field Description	Field Name	Type/Unit	Values/Meaning	
			9	Unclassified
Density	DENSITY	Numeric (t/m³)	2.71	Sequence felsic debris flow deposit unit
			2.65	VHM
			2.78	Marls – SFD
			Estimated Default 2.99	Marls
			Estimated Default 2.97	S1/S2
			Estimated Default 2.69	Late Sills
			2.67	EMP
			2.64	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	1	In MRE (supported by RPEEE)

Field Description	Field Name	Type/Unit	Values/Meaning	
		(Integer)	0	Outside of MRE (not classified)

14.13 Dynamic Anisotropy

Dynamic anisotropy (DA) was implemented in Studio RM to allow the search ellipsoid to be locally oriented according to local dip and dip direction of the estimation domains. Dip and dip direction were estimated into the model following the centroid plane of each domain wireframe.

Figure 14.24 – Cross Section (Looking North) Showing DA Surfaces and Resultant Model Dip and Dip Direction Displayed as Green Diamonds



Source: ERM, 2025

14.14 Sensitivity Analysis – 2024

Sensitivity analysis on assumptions and parameters used in the estimate was completed by ERM in 2024 prior to the PFS 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, characterized by grades exceeding approximately 10 g/t Au.

The conclusions of the sensitivity analysis were as follows:

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.

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).

The scenario that is currently better supported by geological evidence and logging is 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 the PFS and this MRE and are described in relevant sections:

- The undifferentiated skarn domain, and not the VHG domain, should be used as the basis for the MRE due to current geological understanding of grade continuity.
- 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 used for Au with an additional within-estimate capping strategy based on distance 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 reduce smoothing, while cognizant of the presence of coarse gold.

14.15 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 14.10 shows the percentage of blocks estimated in each search pass and Figure 14.25 shows the model colored by search pass. Searches and minimum/maximum sample numbers were supported by KNA (Section 14.11) 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 14.10 – Percentage of Blocks Estimated in Each Search Pass

	Search Pass	% Volume
Gold	1	67
	2	28
	3	4

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 neighborhood is presented in Table 14.11 Sample Search Neighbourhood for Grade Estimation. 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.

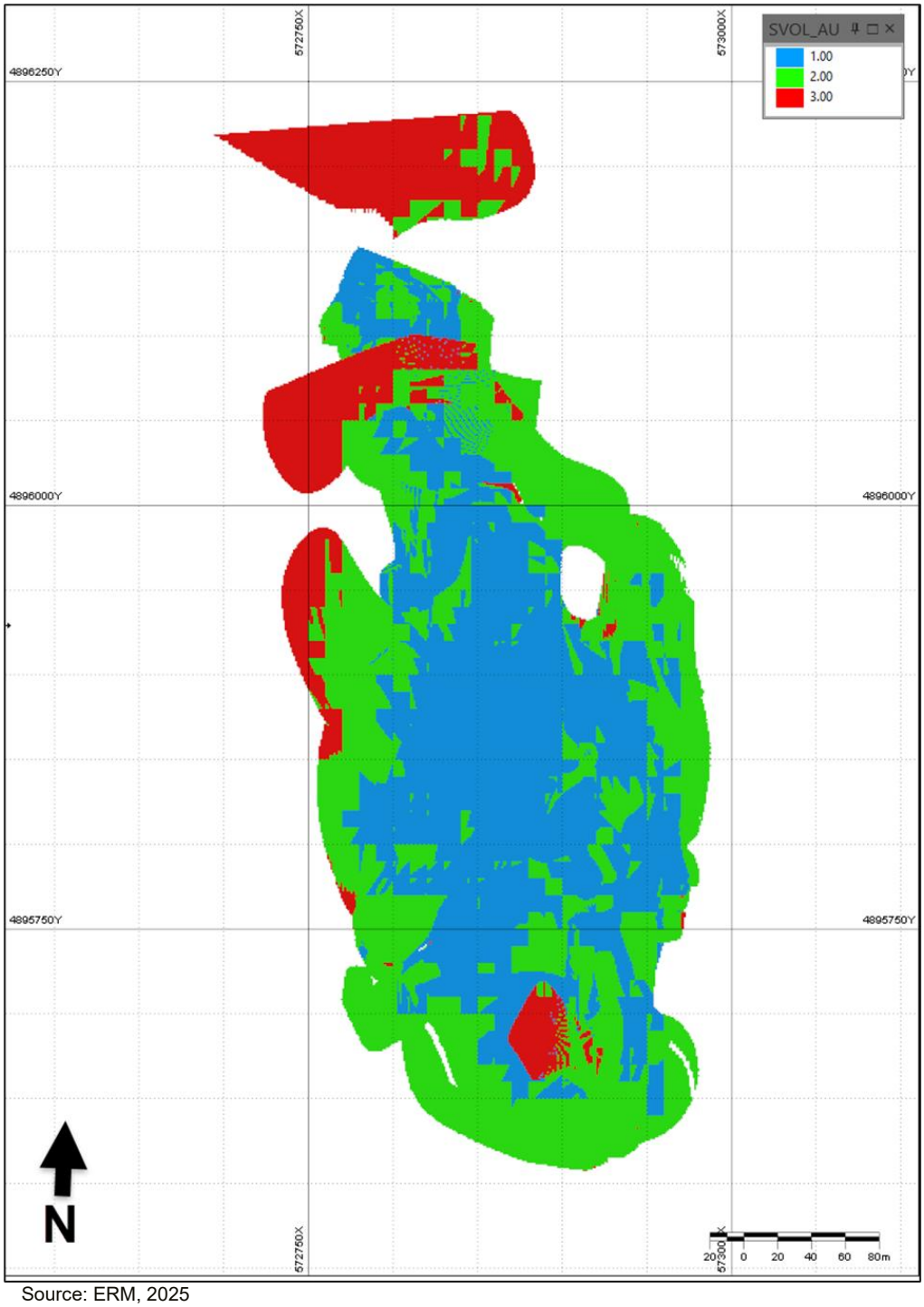
Check estimates were also run on uncut (uncapped) variables and using inverse distance weighting squared (IDW2) to assess sensitivities.

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 14.26).

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 14.26). 30 m was chosen to be well within the ranges of the 1st and 2nd directions modelled.

Figure 14.25 – Plan View of the Model, Looking Northeast, Coloured by Search Pass for Gold



Source: ERM, 2025

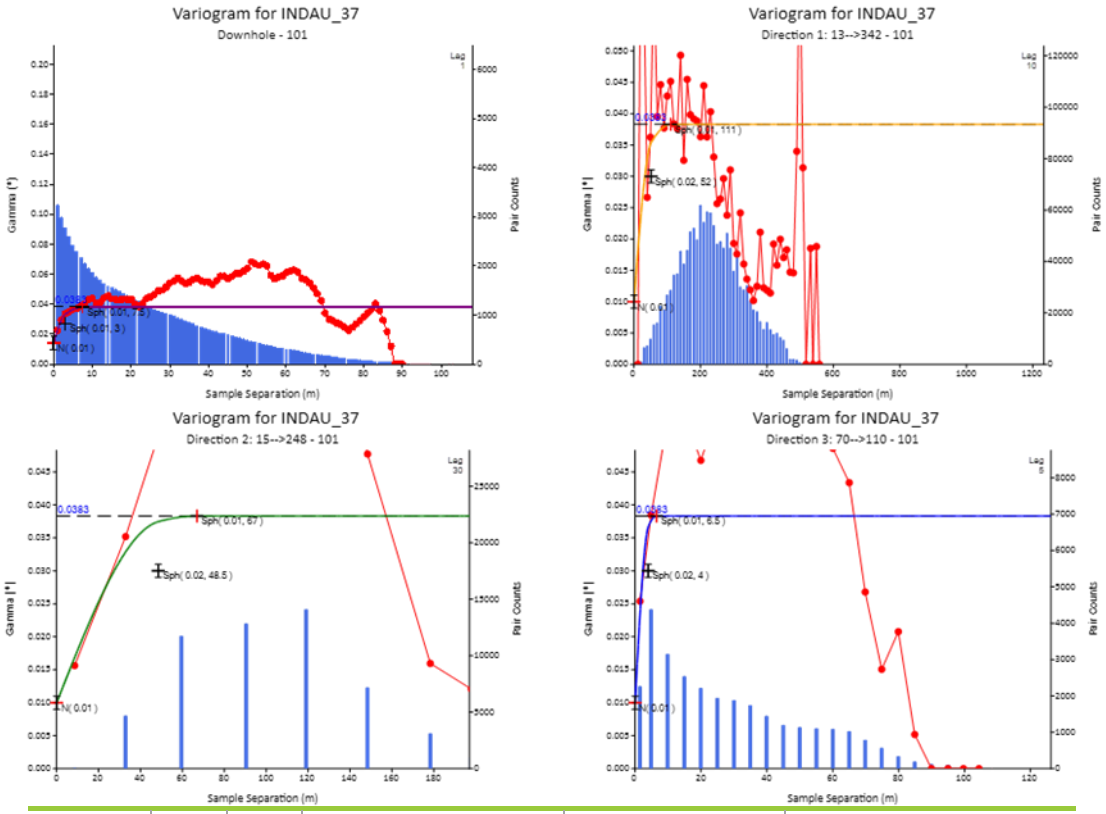
Table 14.11 – Sample Search Neighbourhood for Grade Estimation

Domains	Search Pass	Search 1	Search 2	Search 3	Minimum Composites	Maximum Composites	Maximum per DH	In-Estimate Capping
Gold* Estimate (and silver, 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	30	30	10	12	30	5	None
	2	60	60	20	12	30		
	3	300	300	100	8	30		

Domains	Search Pass	Search 1	Search 2	Search 3	Minimum Composites	Maximum Composites	Maximum per DH	In-Estimate Capping
Waste domains	1	125	100	20	20	40	5	None
	2	250	200	40	20	40		
	3	12500	10000	2000	10	40		
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

Figure 14.26 – Modelled Variogram Ranges for the 37 g/t Au Indicator

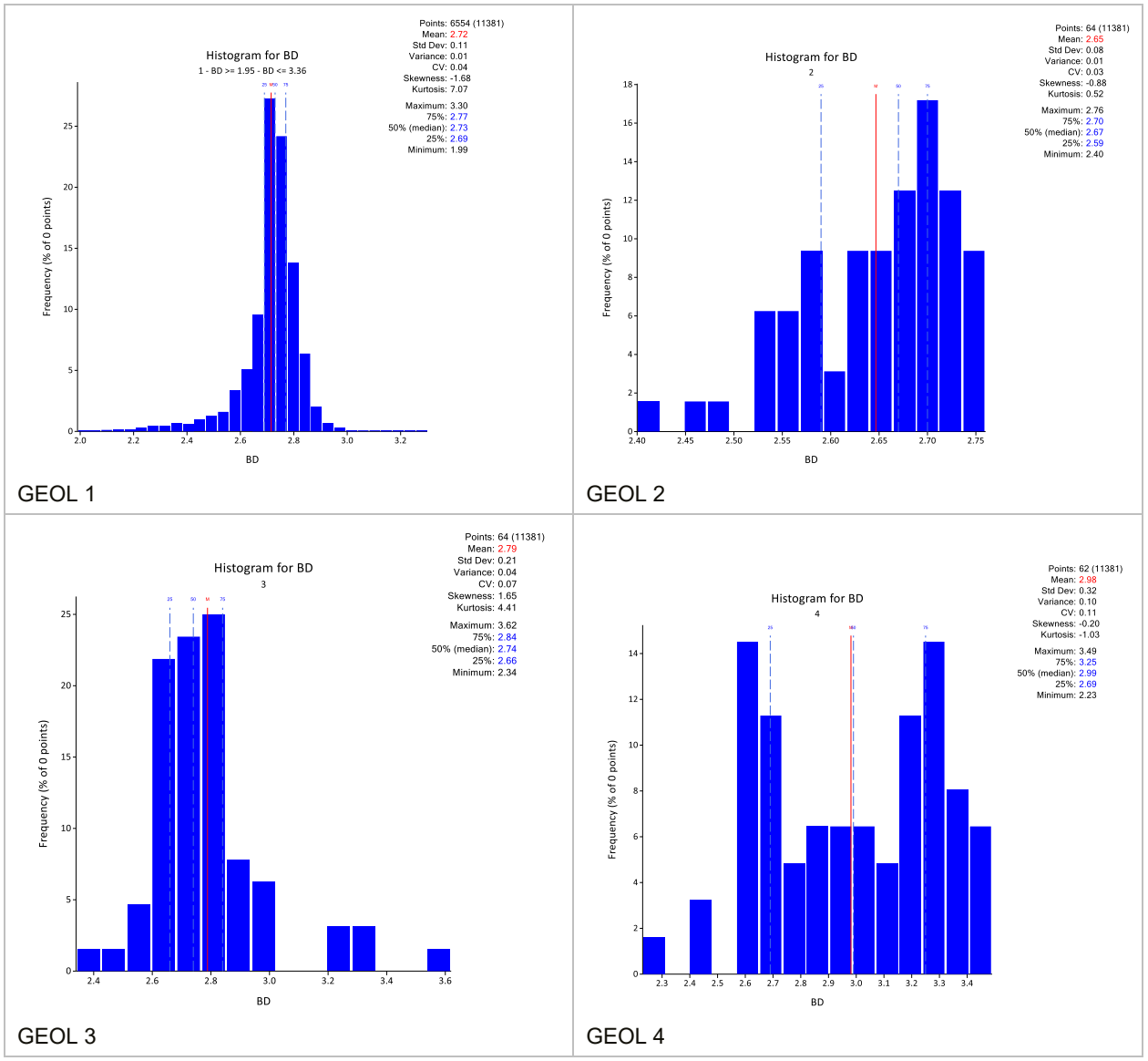


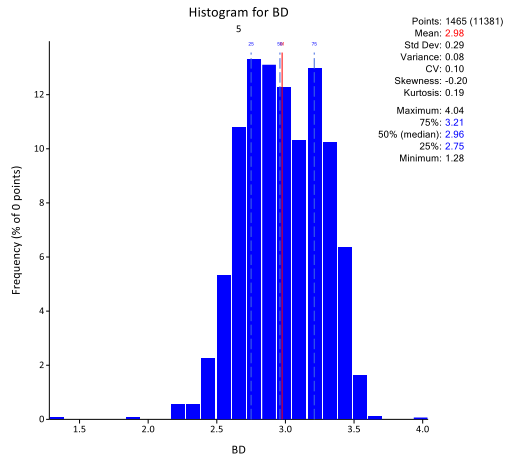
Variable	C0	C1	Range			C2	Range		
			Major	Semi	Minor		Major	Semi	Minor
IND_37	0.01	0.02	52	48.5	4	0.008	111	67	6.5

14.16 Bulk Density Estimate/Assignment

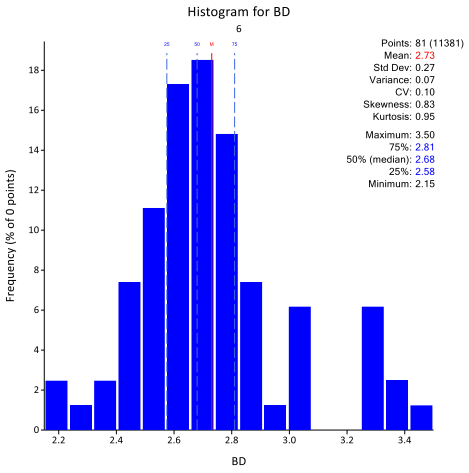
In-situ dry bulk density (BD) measurements were analysed by reviewing histograms by modelled lithology (Figure 14.27). Some zones had a narrow range of measured densities, in which case the mean BD was assigned to blocks within that lithology. Certain lithologies had a wide range of BD values, often 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 14.27 – Histograms Showing Measured BD

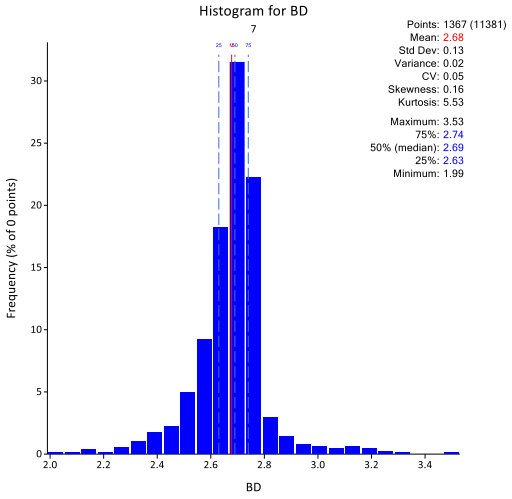




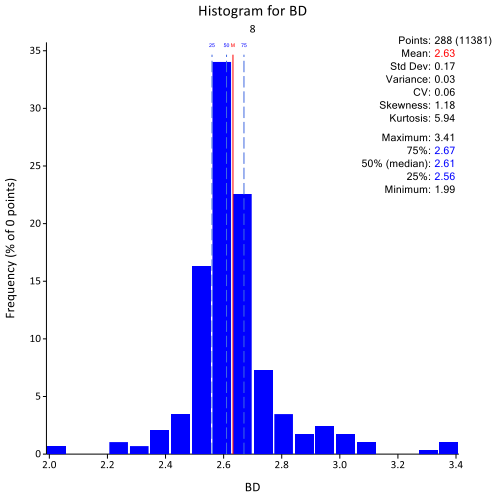
GEOL 5



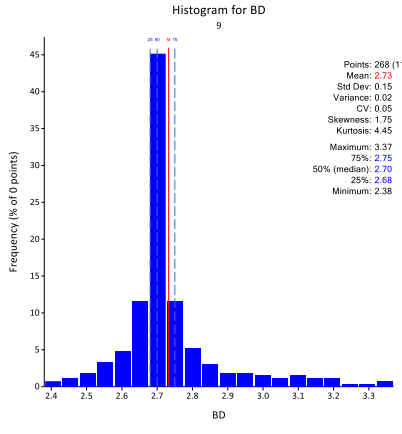
GEOL 6



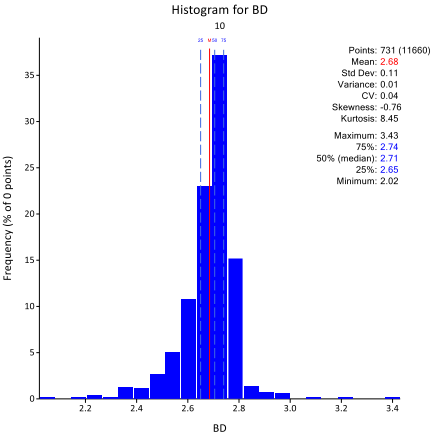
GEOL 7



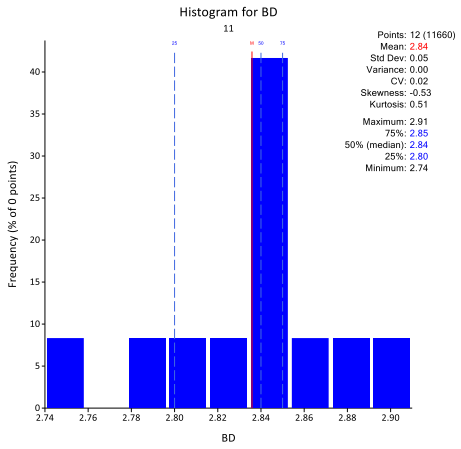
GEOL 8



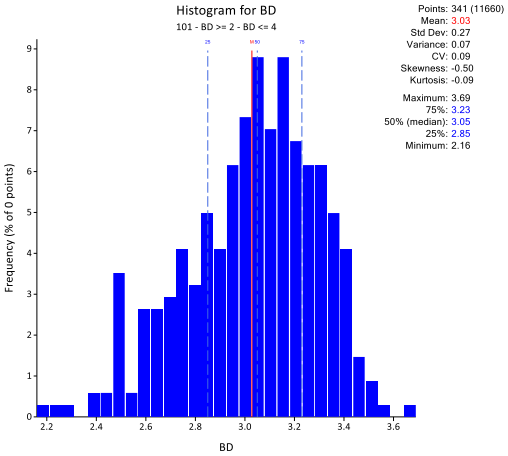
GEOL 9



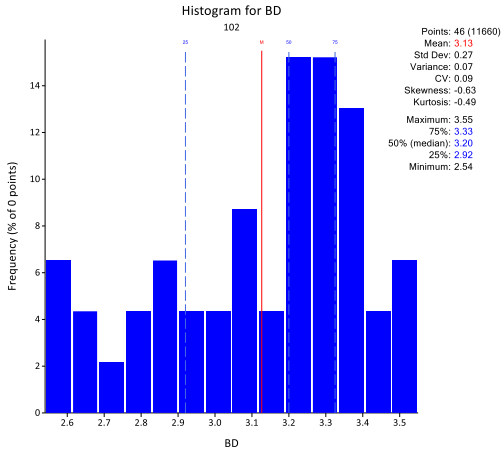
GEOL 10



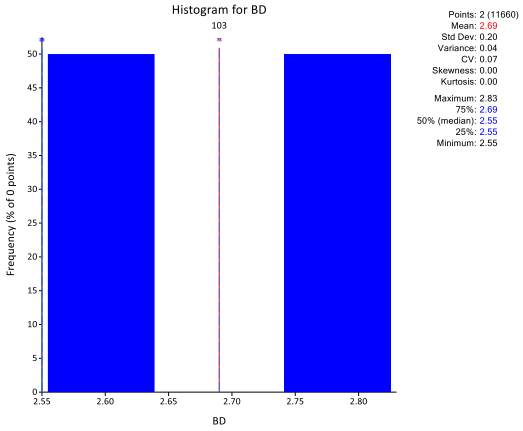
GEOL 11



ESTZON 101



ESTZON 102



ESTZON 103

Source: ERM, 2025

Table 14.12 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 14.12 – Methods to Assign BD by Lithology

Domain	Description	Method	Mean assigned if applicable (t/m³)
GEOL=1	SFD	Mean BD assigned	2.72
GEOL=2	VHM	Mean BD assigned	2.65
GEOL=3	Marls – SFD	Mean BD assigned	2.79
GEOL=4	Marls	Estimated using OK	2.99 (only assigned to small number of un-estimated blocks)
GEOL=5	S1/S2	Estimated using OK	2.99 (only assigned to small number of un-estimated blocks)
GEOL=6	Mafic sills	Estimated using OK	2.73 (only assigned to small number of un-estimated blocks)
GEOL=7	Early mineralised porphyry	Mean BD assigned	2.68
GEOL=8	Quartzite	Mean BD assigned	2.63
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 14.13.

Dynamic anisotropy was used to orient the search ellipse locally and has been described in more detail in Section 14.13. Estimated BD values were validated via statistical checks (Table 14.16), visual inspection and swath plots Figure 14.30 to Figure 14.33).

Table 14.13 – Sample Search Neighborhood for BD Estimate

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

14.17 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 14.14).
- Visual inspection of estimated grades in plan and in cross sections and comparison with the input composites (example cross sections presented in Figure 14.28 to Figure 14.29).
- Check for global bias by estimation pass and by domain – comparison of estimated and declustered composite statistics (Table 14.15).
- Check for local bias considering the supporting information – analysis of local trends in estimates using swath plots (Figure 14.30 to Figure 14.33).
- Checks to ensure the boundary conditions between estimation domains are honored.

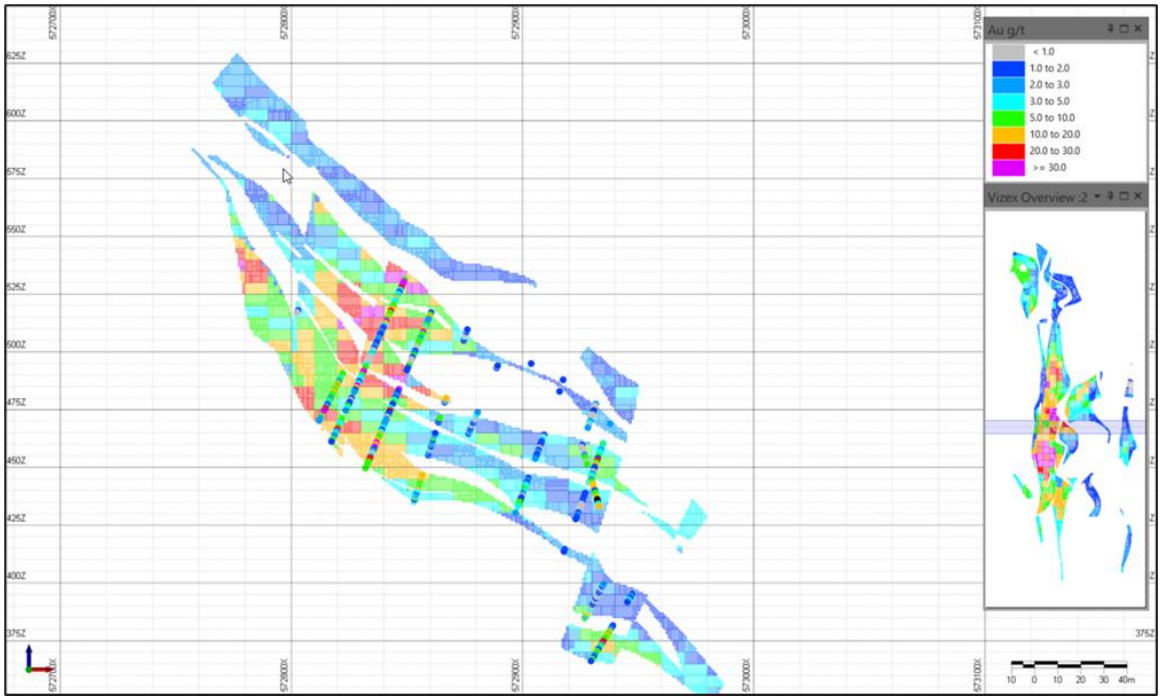
Volume checks show that the model has been built correctly based on the wireframes used. Grades are comparable on a global and domain-by-domain basis between estimation composites and blocks and are well within the 10% threshold that is considered reasonable.

Trends have been reviewed via swath plots to assess semi-local estimates and these also show good comparison for gold. 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 14.14 – Block Model vs. Wireframe Volumes

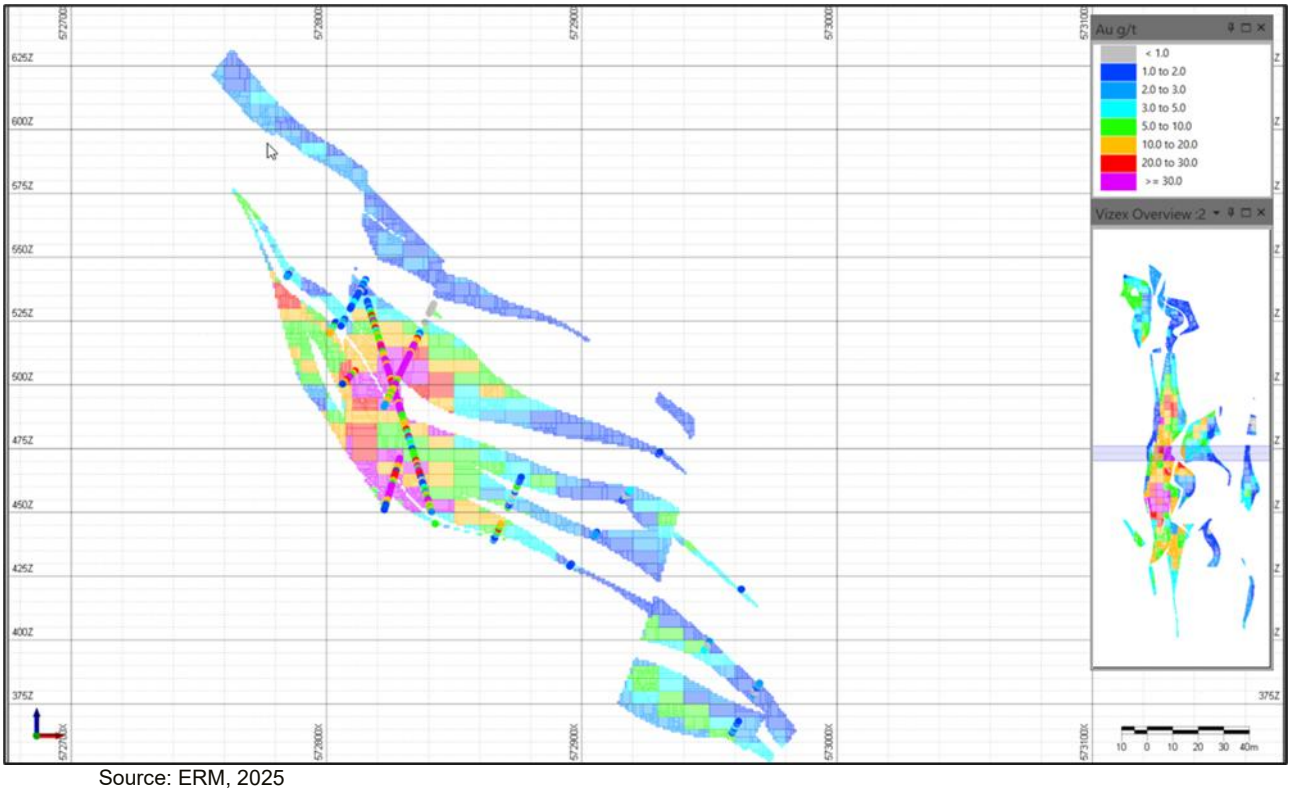
Domain	Wireframe Volume	Model Volume	% Difference
101+104+107+110	2,628,562	2,626,357	0
103	20,557	20,568	0
102	378,122	378,178	1
200	744,940	749,147	0
201	336,607	333,104	-1

Figure 14.28 – Cross Section at 4895860 M (±15 M) Looking North Showing Estimated Gold Grade and Input Composites



Source: ERM, 2025

Figure 14.29 – Cross Section at 4895880 M (±15 M) Looking North Showing Estimated Gold Grade and Input Composites



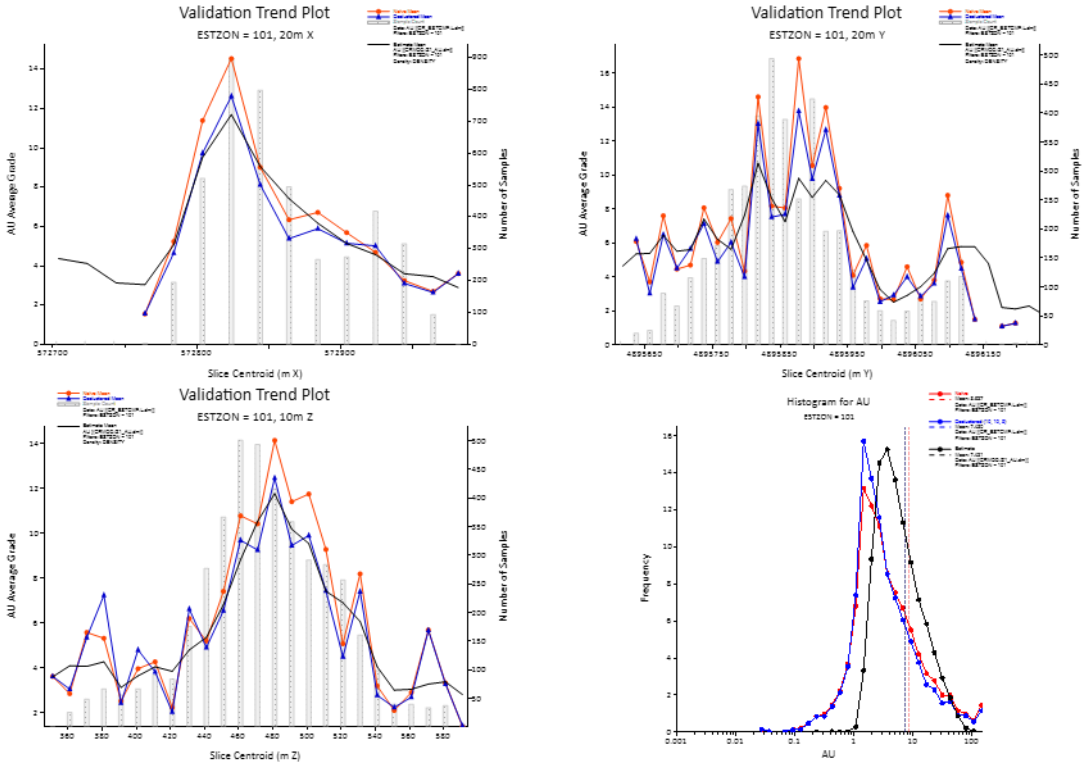
Source: ERM, 2025

Table 14.15 – Global Statistics – Comparison of Block and Composite Grades

Variable	Domain name	Domain	Declustered Composite Grade	Block Grade	% Difference	% Metal[1]
Au	ESTZON	101	7.48	7.48	0.66	88
	ESTZON	107	5.76	5.78	0.32	5
	ESTZON	102	7.48	7.43	0.66	4.7
Density	EZON_DEN	4	2.98	2.94	-1.37	
	EZON_DEN	5	2.98	2.97	-0.31	
	EZON_DEN	6	2.73	2.74	0.23	
	EZON_DEN	101	3.03	3.02	-0.36	
	EZON_DEN	102	3.13	3.14	0.47	

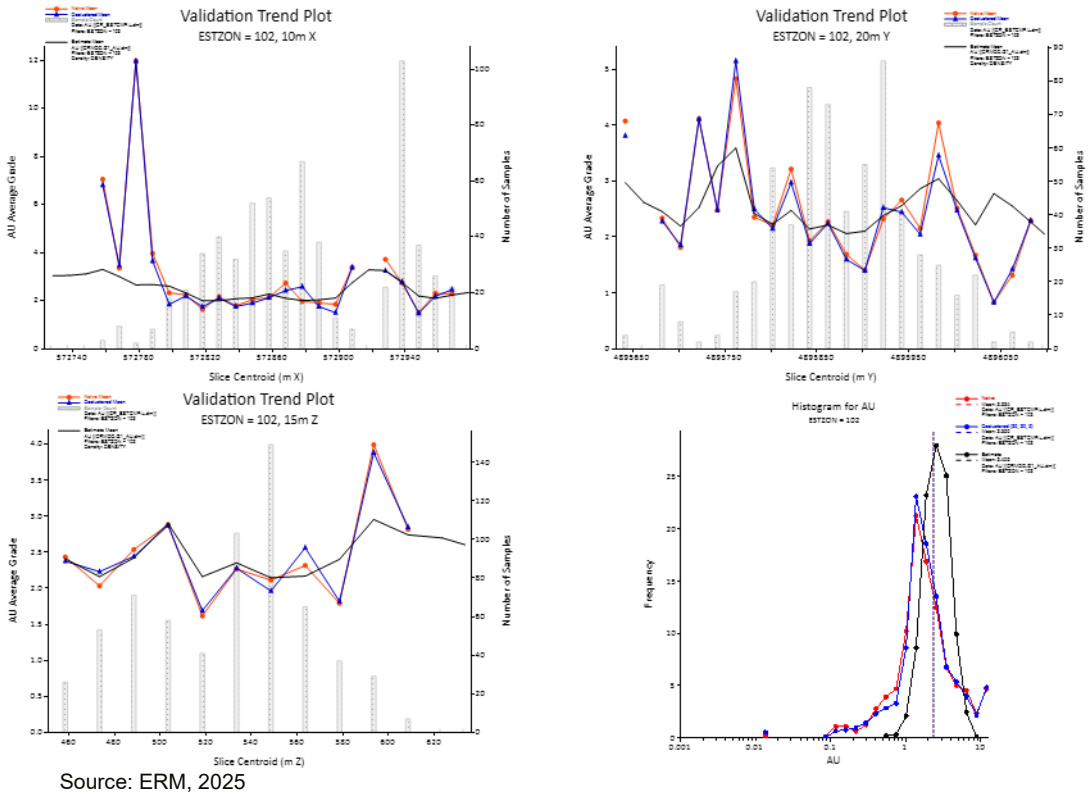
Notes:
1. % metal represents how significant a particular domain is in terms of metal content.

Figure 14.30 – Swath Plots and Log Histogram for Au ESTZON 101



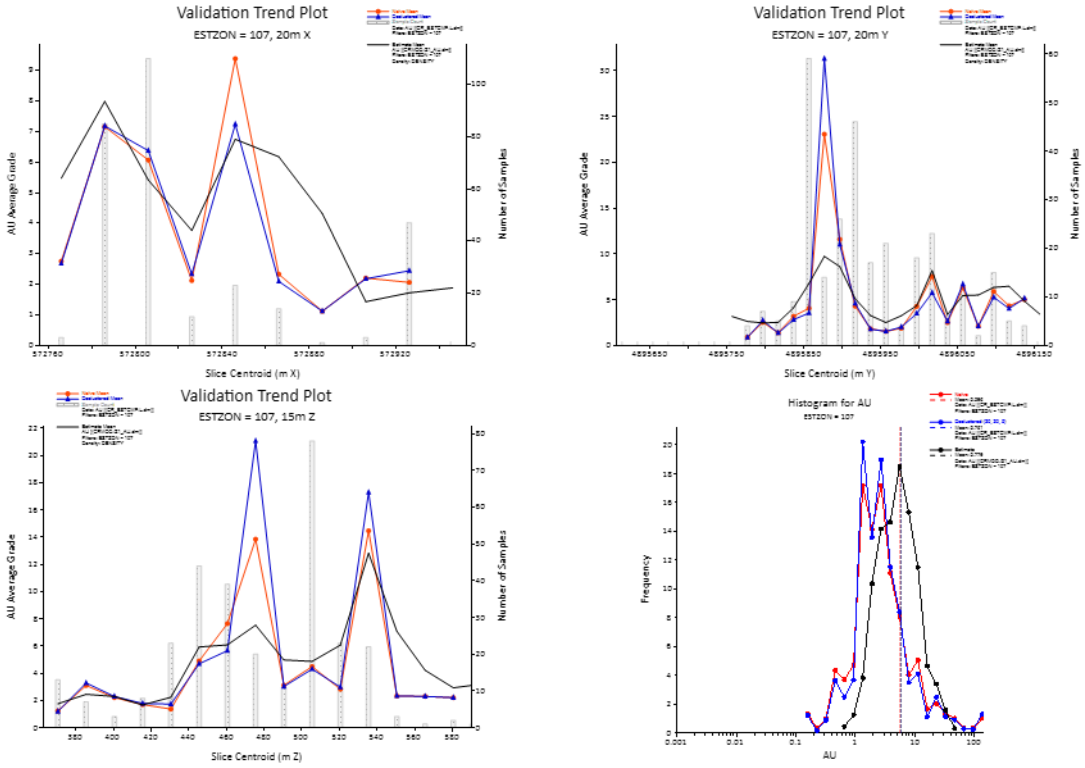
Source: ERM, 2025

Figure 14.31 – Swath Plots and Log Histogram for Au ESTZON 102



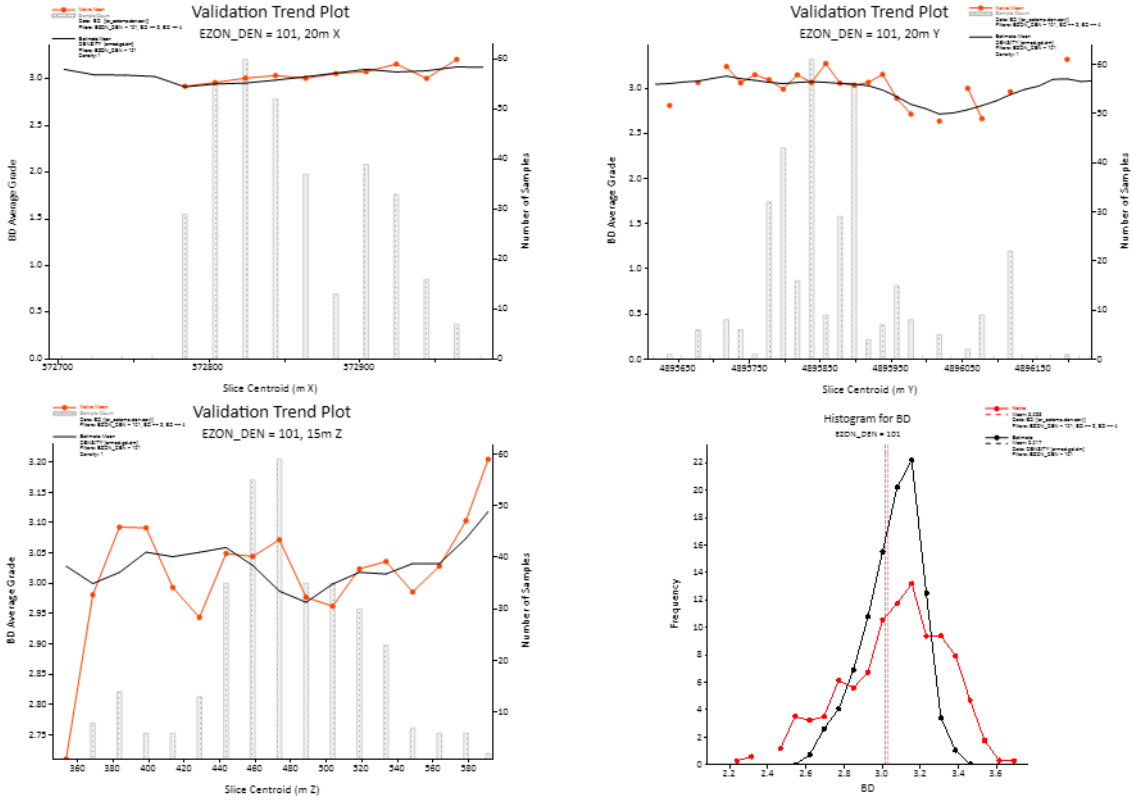
Source: ERM, 2025

Figure 14.32 – Swath Plots and Log Histogram for Au ESTZON 107



Source: ERM, 2025

Figure 14.33 – Swath Plots and Log Histogram for Density ESTZON 101



Source: ERM, 2025

14.18 Mineral Resource Classification

14.18.1 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$1,900/oz of gold and a minimum width constraint of 3 m was used to define optimized underground potentially mineable shapes using Datamine's Minable Shape Optimizer (MSO) 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 14.16 to Table 14.18. In collaboration with the QP, Malcolm Titley, and DPM's Mining Engineers, reasonable parameters were chosen for the MSO process, and these are presented in Table 14.19.

Table 14.16 – COG Calculation and Cost Assumptions for Čoka Rakita MRE

Item	Cost per tonne (US$/t)
Underground mining costs	37.85
Process costs	24.98
G&A costs	14.82
Sustaining capital	11.20
Total	**88.85**

Table 14.17 – Commercial Terms Used within the COG Calculation

Item	Unit	Cost per gold ounce
Concentrate transportation	US$/dmt	183.42
Concentrate treatment	US$/dmt	145.68
Concentrate refining	US$/oz	7.27
Concentrate penalty	US$/dmt	0
Concentrate grade – gold	g/t	115.73
Concentrate gold payable	%	98.497
Royalty	%	5
Total US$ per gold ounce		**95.72**
Total US$ per gold gram		**3.08**

Table 14.18 – COG Calculation for MRE

Item	Unit	Value
Gold price (ounce)	US$/oz	1,900
Gold price (gram)	US$/g	61.09
Revenue	US$/g	52.20
Less royalty	5% of Sales	2.61
Less per gold gram costs	US$/g	3.08
Realised revenue	US$/g	46.51
Cost per tonne to produce	US$/t	89.29
MRE cut-off grade	g/t	1.92 (rounded to 2 g/t)

Notes:
1. Processing costs: US$24.98/t Process (including Dry Tailings + Paste Fill costs).
2. Transportation cost: Transport cost assumption based on 500 km overland transport to smelter.
3. Gold concentrate is subject to 1 gram Au g/t deductible, i.e., 1 g of Au is not payable.
4. Revenue calculation assumes gold metallurgical recovery of 86.75% and gold payability of 98.5%.
5. Calculated cut-off grade for mineral resource reporting assumes mining dilution of 0%.

Table 14.19 – MSO Parameters

Parameter Name	Parameter Setting	Value	Units
Block Model Settings			
Optimisation field/default value	Au	Au	g/t Au
Density field/default value	Density	Density	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)	30 (with 10m sub shapes)	m
	V (Z axis/level height)	10	m
Stope depth (Z axis/stope width in MSO)	Minimum	3	m
	Maximum	15	m
Dilution	ELOS dilution	0	m

Parameter Name	Parameter Setting	Value	Units
Stope dip angles	Minimum	30	°
	Maximum	120	°
Stope strike angles	Minimum	-45	°
	Maximum	45	°
Maximum Stope Thickness Ratio	Top to Bottom	1.5	m
	Left to Right	1.5	m
Create sub shapes (Variable Shape Fraction)	U Min1	0	m
	U Max1	3	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	1.0	m
Material configuration	Include CLASS 9 material		
Advanced slice options	Ignore pillar requirements between full shapes		
	Ignore pillar requirements in split stope shapes		

14.18.2 MINERAL RESOURCE CLASSIFICATION

The MRE 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.

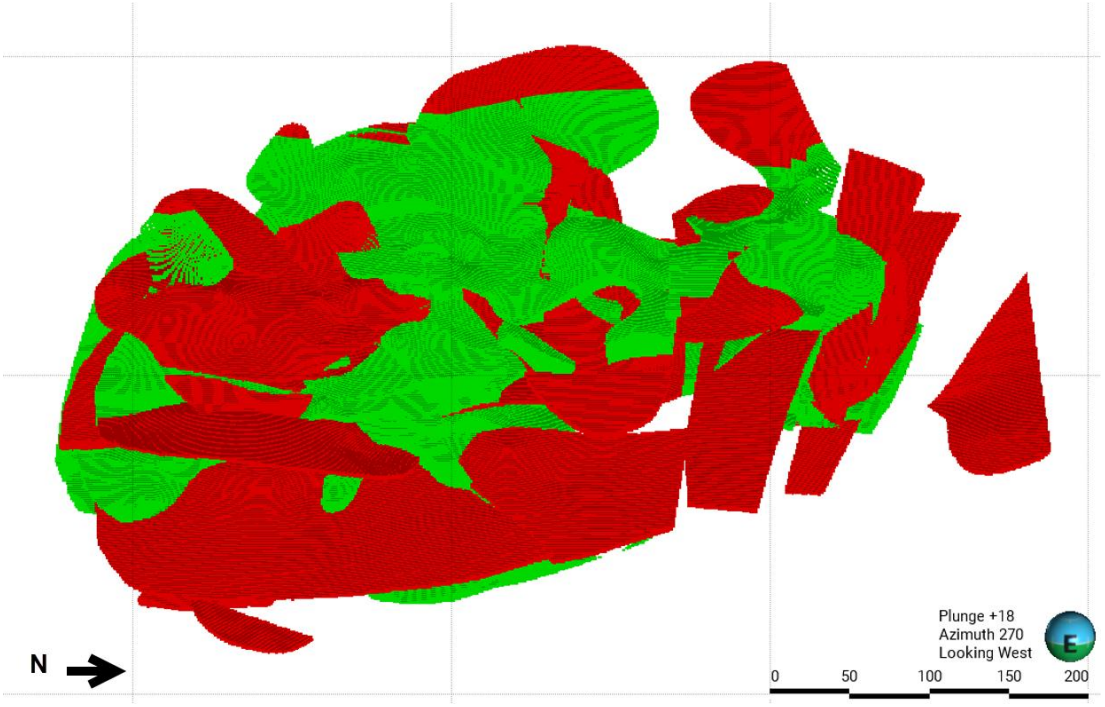
Mineral Resources for the Project were classified in accordance with the CIM definitions above, and the following was taken into consideration by the QP:

- 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 14.18.

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 20-30 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 (Figure 14.34).

Figure 14.34 – A 3D View Looking Obliquely West Showing Block Model Coloured by Resource Category (Red – Indicated, Green - Inferred)



Source: DPM, 2025

14.19 Mineral Resource Reporting

The MRE for the Čoka Rakita Gold Project is presented in Table 14.20. The Mineral Resource has been reported exclusive of Mineral Reserves.

Table 14.20 – Čoka Rakita MRE Using Underground Mining Scenario

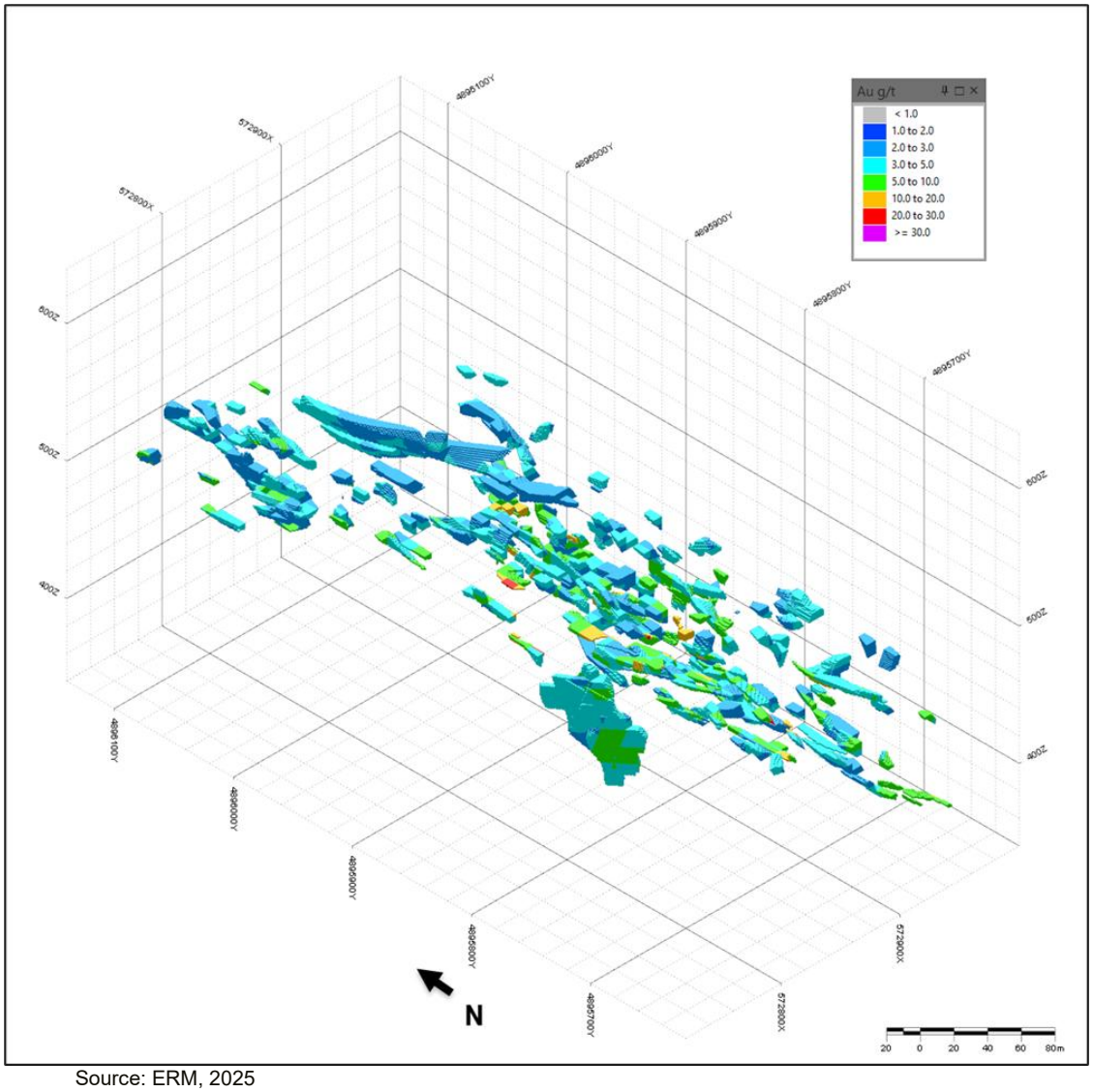
Čoka Rakita Mineral Resource Estimate Effective Date of January 17, 2025			
Category	**Tonnes (Mt)**	**Gold Grade (g/t)**	**Contained Gold (koz)**
Indicated	0.53	3.94	67
Inferred	0.09	3.60	11

Notes:

1. The cut-off grade value of 2 g/t assumes US$1,900/oz gold price, 86.75% gold recovery, 0% dilution, US$77.65/t operating cost (mining, process and G&A), US$11.20/t sustaining capital cost, as well as offsite and royalty costs.
2. 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 November 29, 2019.
3. The QP, Malcolm Titley, is not aware of any legal, political, environmental, or other risk factors that might materially affect the estimate of Mineral Resources, other than those specified in Table 14.21.
4. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
5. Mineral Resources are reported exclusive of Mineral Reserves.
6. 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 14.35.

Figure 14.35 – 3D View of the Classified Model, Supported by RPEEE, Coloured by Gold (Looking Northeast with Supporting Drillholes)



Source: ERM, 2025

14.20 Previous Mineral Resource Estimate

The decrease in Indicated Resource tonnes compared to the PFS is due to the conversion to Reserves (0.7 Mt Reserve increase), while the grade increase is driven by additional drilling. The 2024 MRE is based on an assumed gold price of US$1,700 per ounce, compared with US$1,900 per ounce used in the current MRE.

Čoka Rakita MRE – Čoka Rakita MRE (Reported Exclusive of Reserve) – Effective as of January 17, 2025

Category	Tonnes (Mt)	Gold Grade (g/t)	Contained Gold (koz)
Indicated	0.53	3.94	67
Inferred	0.09	3.60	11

Čoka Rakita MRE (Reported Exclusive of Reserve) – Effective as of August 30, 2024

Category	Tonnes (Mt)	Gold Grade (g/t)	Contained Gold (koz)
Indicated	1.45	3.30	154
Inferred	0.11	3.11	11

Note:
Refer to the Technical Report entitled *Pre-Feasibility Study. Čoka Rakita Project, Eastern Serbia. NI 43-101 Technical Report*, January 30, 2025, for information on the key assumptions and parameters relating to the previous estimates

14.21 Risk Factors that May Affect the Mineral Resource

The QP is not aware of any environmental, permitting, legal, socio-economic, marketing or political factors that could materially impact the MRE disclosed in this Report, other than those specified below and in Table 14.21:

- 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 14.21. The overall risk to the 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 14.21 – 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	Low	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 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 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 and details the key assumptions, parameters, and methods used for converting Mineral Resources to Mineral Reserves.

The Mineral Reserves for the Čoka Rakita Project were estimated by WSP using the CIM 2019 Best Practices Guidelines and are classified using the 2014 CIM Definition Standards.

The Čoka Rakita Deposit Mineral Reserves are based on the mineral resource model provided by ERM (cr_md250228.eng_5x5x5). As there are no Measured mineral resources included in the block model provided, therefore no Proven Mineral Reserves are reported in the estimate. All inferred material within the block model were set to waste.

Mineral Reserves are reported at the point of delivery to the process plant and have an effective date of January 17, 2025, which are inclusive of dilution and mining loss.

The Mineral Reserves are supported by a longitudinal long-hole stoping mine design, a detailed mine production schedule, and capital and operating cost estimates developed to a feasibility study level of engineering.

15.2 Estimation Procedure

The process involves converting Indicated Mineral Resources to Probable Mineral Reserves by identifying material that exceeds the cut-off grade and conforms to the geometrical constraints imposed by the selected mining method. This conversion also applies modifying factors such as dilution and mining recovery.

The mining plan presented is based exclusively on Mineral Reserves, with no contribution from Inferred or other non-economic material.

The conversion of Indicated Mineral Resources to Probable Mineral Reserves involves the application of following procedures:

- Review the resource block model received from ERM and set grade of all inferred mineral resources blocks to zero.
- Apply the mine design criteria to the DSO setting, including level spacing, stope dimensions, and minimum mining width.
- Apply modifying factors that include mining dilution and recovery, operating costs, gold price, and cut-off grade.

- Generate preliminary stope shapes for various cut-off grades using Deswik Stope Optimiser (DSO) software.
- Manually adjust and clean the DSO-generated shapes to create final optimised stopes that maximise value while respecting geometric constraints and input parameters.
- Design stope accesses, mine development, and infrastructure using Deswik CAD software.
- Use preliminary cost estimates to conduct an economic check on mining areas to ensure they generate value to pay for the required development and infrastructure.
- Determine the stopes mining sequence based on geotechnical guidelines.
- Apply the development and stoping production rates to create a detailed mining production schedule.
- Generate a LOM production plan and material take-offs (MTO) as a basis for cost estimation.
- Use the life of mine plan (LOMP) to prepare estimates for the initial capital (Capex), sustaining capital (Susex), and the operating costs (Opex) associated with the LOMP.
- Prepare the final Mineral Reserve statement.

15.3 Modifying Factors

15.3.1 Cut-Off Grade Estimation

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

A gold price of US$1,600/oz proposed by DPM, was used in the COG and Mineral Reserve Estimate. This price is consistent with the 10-year average gold price, based on data from the World Gold Council site and summarised in Table 15.1. The gold price used represents a conservative assumption when compared to current spot prices and medium-term gold price forecasts.

Table 15.1 – Average Gold Price

Year	Gold Price US$/oz
2015	1,160.10
2016	1,250.80
2017	1,257.20
2018	1,268.50
2019	1,392.60
2020	1,769.60
2021	1,798.60

Year	Gold Price US$/oz
2022	1,800.10
2023	1,940.50
2024	2,386.20
Average	**1,602.41**

The operating and marketing costs used in the COG estimation were derived from the economic model developed for the Pre-Feasibility Study (PFS) and validated by the Čoka Rakita NI 43-101 Pre-Feasibility Study Report, dated January 30, 2025, and are all US dollars.

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

Processing and General and Administrative (G&A) costs were considered as fixed in COG estimation, since the process plant operates at full capacity throughout the mine life. Mining costs were separated into fixed and variable components and applied to calculate three types of cut-off grades:

- **Full Cost Cut-off Grade (stopes)**: Includes all costs categories, both fixed and variable.
- **Marginal Cut-off Grade (Stopes)**: Excludes sustaining capital costs and includes mining fixed cost and stope variable mining costs.
- **Incremental Cut-off Grade (Development)**: used for development material, reflecting only the additional costs of processing material already mined during development activities.

Since the mining access and development costs are considered, marginal stopes are used to fill in gaps between stope shapes or extend strike length, they must be mined without affecting Project economics negatively.

The COG estimation parameters are shown in Table 15.2.

Table 15.2 – COG Determination Parameters

Costs Per Ore Tonne				
Category	**Units**	**Full Cost**	**Marginal Cost**	**Incremental Cost**
Mining				
Fixed	US$/t milled	11.43	11.43	0.00
Variable	US$/t milled	26.86	21.91	0.00
Processing	US$/t milled	*24.98*	*24.98*	*24.98*
G&A	US$/t milled	14.82	14.82	14.82
Sustaining Capital	US$/t milled	*11.20*	*0.00*	*0.00*
Total Cost	**US$/t milled**	**89.29**	**73.14**	**39.80**

Marketing costs		
Category	**Units**	**Cost**
Concentrate Transportation	US$/dmt	183.42
Concentrate Treatment	US$/dmt	*145.68*
Concentrate Refining	US$/oz	*7.27*
Concentrate Penalty	US$/dmt	*0.00*
Concentrate Grade Au	g/t	*115.73*
Concentrate Au Payable	%	*98.5*
Total Cost	**US$/g**	**3.08**
Total Cost	**US$/oz**	**95.72**
Mill Recovery	%	86.75
Royality	%	5

COG Calculation				
Category	**Units**	**Full Cost Stope COG**	**Marginal Cost Stope COG**	**Incremental Cost Development COG**
Gold price (ounce)	US$/oz	1600	1600	1600
Gold price (gram)	US$/g	*51.44*	*51.44*	*51.44*
Revenue*	US$/g	43.95	43.95	43.95
Less royalty 5% of sales	US$	*(2.20)*	*(2.20)*	*(2.20)*
Less per gold gram costs	US$	*(3.08)*	*(3.08)*	*(3.08)*

COG Calculation				
Category	**Units**	**Full Cost Stope COG**	**Marginal Cost Stope COG**	**Incremental Cost Development COG**
Realised revenue	US$/g	*38.68*	*38.68*	*38.68*
Cost per tonne to produce	US$/t	*89.29*	*73.14*	*39.8*
Mill Feed COG	**g/t**	**2.31**	**1.89**	**1.03**
Mining Dilution	%	*6*	*6*	*0*
Dilution Grade	g/t	-	-	-
Mine Design (in-situ) COG	**g/t**	**2.45**	**2.00**	**1.03**

*Revenue (US$/g) = Gold price (US$/g) * Mill recovery /100* Concentrate Au payable/100

For Reserve reporting, The Full Cost Stope COG of 2.45 g/t Au has been rounded to 2.50 g/t Au, and the Development to 1.00 g/t Au.

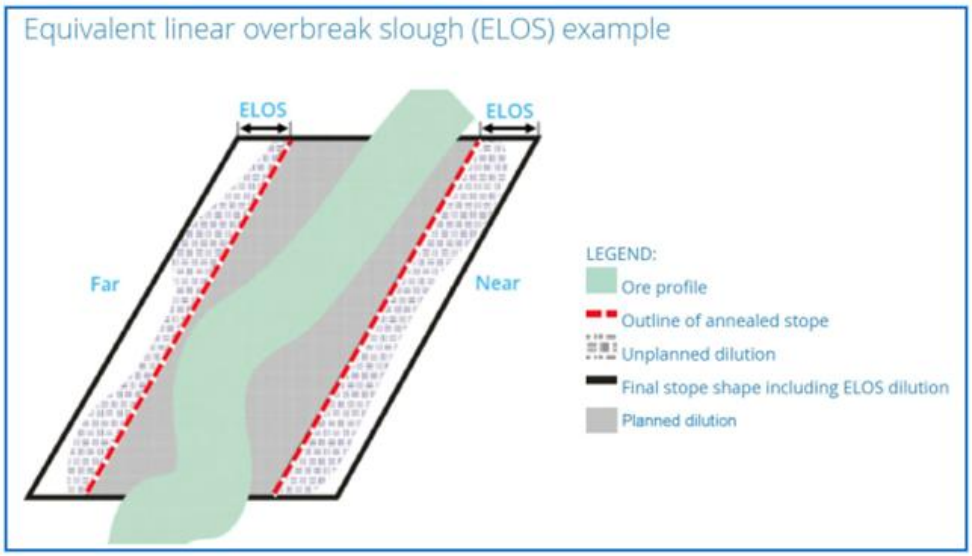
15.3.2 MINING DILUTION

Mining dilution was accounted for from three (3) principal sources: planned (internal) dilution, unplanned (external) dilution, and backfill dilution.

- Planned (Internal) Dilution: This refers to dilution from low-grade or waste material that is unavoidably included within the designed stope boundaries.
- Unplanned (External) Dilution: This includes dilution from material outside the stope due to overbreak. A base external dilution was incorporated into the DSO setup by applying an Equivalent Linear Overbreak Slough (ELOS) of 0.5 metres on the footwall and 1.0 metre on the hanging wall, as illustrated in Figure 15.1.
- Backfill Dilution: This refers to any dilution resulting from interaction between ore and backfill material during mining and was added as percentage and was estimated as 6% at zero Au grade based on benchmarking.

The tonnage and grade of both external and internal dilution were derived directly from the resource block model.

Figure 15.1 – Schematic Dilution with ELOS Application



Source: WSP, 2025

15.3.3 Mining Recovery

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

For ore development drifts, mining losses are assumed to be negligible, and a 100% recovery is applied. For the long-hole stopes, a mining recovery of 95% has been applied 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 preliminary stope shapes using Deswik Stope Optimiser Version 4 (DSO) software. The optimisation is done based on predefined design and economic criteria, including the COG, deposit geometry, and stope shape parameters. Multiple optimisation scenarios were evaluated to obtain the optimal results in terms of tonnage and grade.

DSO shapes were generated in two (2) passes—the first using the full cost cut-off grade, and the second using the marginal cut-off grade. This method ensures that high confidence DSO shapes are defined first, while allowing flexibility for potential inclusion of marginal DSO shapes under favourable economic conditions or as part of mine sequencing strategies.

The DSO parameters used in the stope optimisation process are summarised in Table 15.3.

These preliminary stope shapes served as the basis for developing detailed manual stope designs, which were refined using engineering judgment and visually checked against the geological block

model for consistency. An example of this refinement process is illustrated for Level 440 (440L) in Figure 15.2.

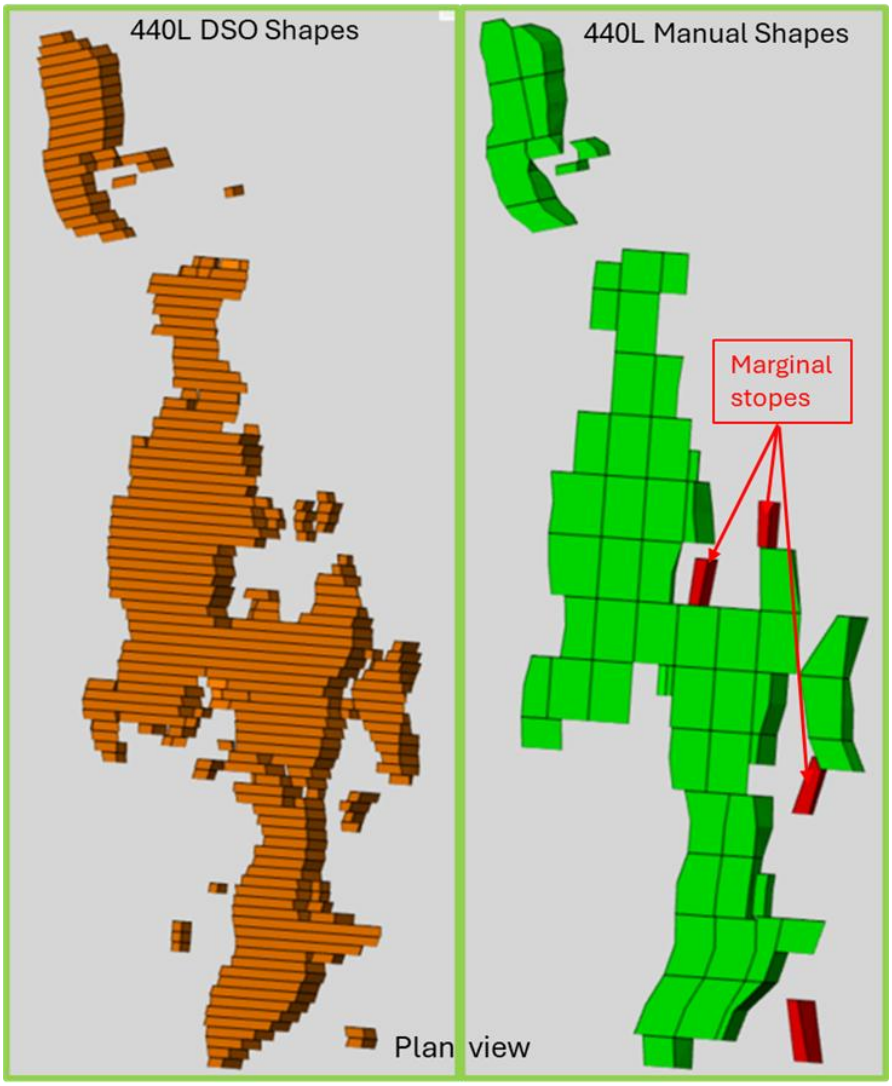
Development headings were subsequently designed and optimised, and a preliminary cost estimate was used to perform an economic assessment of the mining areas to ensure they would generate sufficient value to cover the required development and infrastructure costs.

The stopes were then sequenced based on geotechnical guidance to align with the selected mining method, long-hole longitudinal retreat. The final stope sequence was incorporated into the Deswik Scheduler, which was used to generate the production profile and define the overall LOM schedule.

Table 15.3 – DSO Parameters

Parameter	Units	Value
Optimisation field (in BM)	g/t	AU
Default density	t/m³	2.99
Stope orientation plane	NA	YZ
Level spacing	m	20
Section spacing	m	5
Stope minimum width	m	3
Stope maximum width	m	20
Stope pillar width	m	0.001
Near wall dilution	m	0.5
Far wall dilution	m	1.0
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
COG Full cost	g/t	2.5
COG Marginal cost	g/t	2.0

Figure 15.2 – 440L Plan View, DSO Vs Manual Stope Shapes



Source WSP, 2025

15.5 Mineral Reserve Statement

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

The inclusion of marginal ore material, as defined in Section 15.3, does not have a significant impact on the total reported Mineral Reserve as summarised in Table 15.34.

Table 15.4 – Impact of Marginal on Mineral Reserve

Value	Units	Stope			Development	Reserves
		Full cost	Marginal	Total		
Tonnes	t	6,504,819	146,540	**6,651,359**	**693,410**	**7,344,769**
Au	g/t	6.65	2.19	**6.55**	**5.37**	**6.44**
Contained Gold	oz	1,390,071	10,319	**1,400,390**	**119,690**	**1,520,080**

The Mineral Reserve for the Čoka Rakita Deposit is estimated at **7.344 Mt** and diluted ore grading **6.44 g/t Au**, This Mineral Reserve Estimate includes dilution, mining recovery factors, stope marginal and incremental development tonnes and is summarised in Table 15.5.

Table 15.5 – Mineral Reserve Estimate Effective as of January 17, 2025

Classification	**Tonnes** (Mt)	**Au** (g/t)	**Contained Gold** (Moz)
Proven	-	-	-
Probable	7.344	6.44	1.520
Proven + Probable	**7.344**	**6.44**	**1.520**

1. As of the Reserve effective date, 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 and 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 January 17, 2025.
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$1,600/oz for gold.
7. Mineral Reserves are reported using variable cut-off grades (Refer to Subsection 15.3.1 for COG estimation details):
   - Stope full cost cut-off grade of 2.5 g/t (in-situ).
   - Stope marginal cut of grade of 2.0 g/t (in-situ).
   - Development COG of 1.0 g/t.
8. Mineral Reserves account for a:
   - Hanging wall (HW) and footwall (FW) ELOS external dilution of 1.0 m and 0.5 m, respectively applied to the stopes at matching Au grades of the BM
   - Back fill dilution of 6% applied to the stopes at zero Au grade
   - Mining recovery of 95 % applied to the stopes and 100% applied to development tonnes
9. Contained Metal (CM) is calculated as follows: Au Contained Metal, (oz) = Tonnage (Mt) * Grade (g/t) / 31.1035 (g/oz).
10. The Mineral Reserve Estimation was completed under the supervision of Mr. Khalid Mounhir, P.Eng., Principal Mining Engineer at WSP Canada Inc., who is a QP as defined under NI 43-101.
11. The QP, Mr. Khalid Mounhir, P.Eng, is not aware of any 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 Previous Mineral Reserve Estimate

The 2025 Mineral Reserve for the Čoka Rakita Deposit, with the effective date of January 17, 2025 and estimated based on the parameters cited in this report, shows an increase of 10.7% in tonnes and 11.8% in contained ounces compared to the 2024 Mineral Reserve estimated at the effective date and with the parameters presented in the PFS NI 43-101 Technical Report, January 30, 2025.

Čoka Rakita Mineral Reserve Estimate – Effective as of January 17, 2025

Category	Tonnes (Mt)	Gold Grade (g/t)	Contained Gold (Moz)
Proven	-	-	-
Probable	7.344	6.44	1.520
Proven + Probable	**7.344**	**6.44**	**1.520**

Čoka Rakita Mineral Reserve Estimate – Effective as of August 30, 2024

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

Note:
Refer to the Technical Report entitled *Pre-Feasibility Study. Čoka Rakita Project, Eastern Serbia. NI 43-101 Technical Report*, January 30, 2025, for information on the key assumptions and parameters relating to the previous estimates

The increase is primarily due to revisions in the engineering design parameters adopted for the 2025 Reserve estimate. Key changes include the incorporation of marginal material (following the determination of a marginal cut-off grade), the inclusion of planned dilution in the stoping design, and manual adjustments to the DSO shapes to convert them to mineable stopes.

15.7 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:

- Interpretations of mineralisation geometry and continuity of mineralisation zones.
- Changes in market conditions, commodity prices and COG estimation assumptions.
- Changes in rock quality and geotechnical assumptions.
- Changes in process plant recoveries.
- Changes in hydrogeological conditions and water inflow assumptions.

- Changes in 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, there are no known 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.

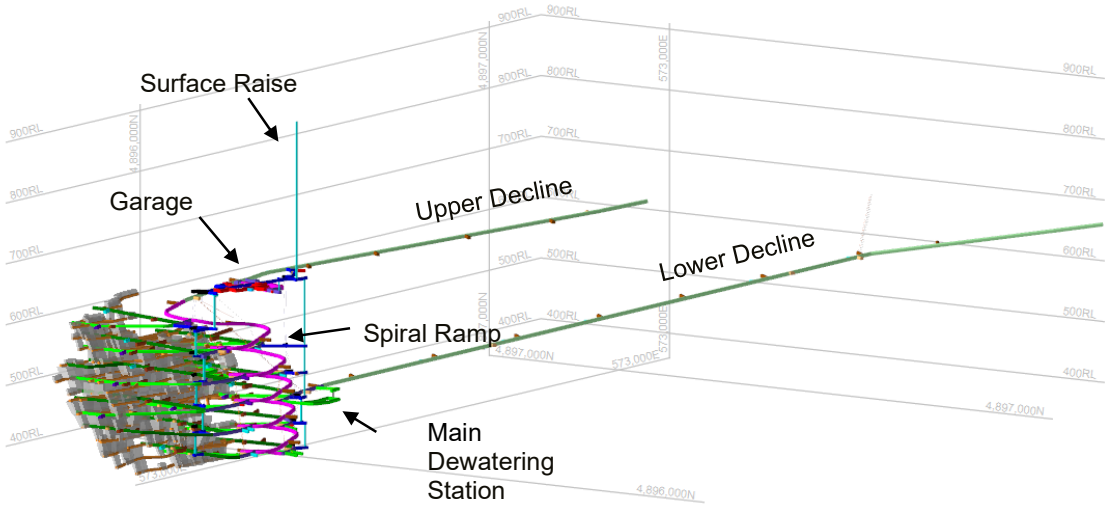
16 MINING METHODS

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° and up to 75° towards the east. The thickness of the orebody 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, 2025

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.

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.

- 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). The deposit will be accessed by driving upper and lower declines from the east side of the ore in the Hanging Wall (HW). The upper decline will give access to the RL620, and the lower decline to RL440. 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 FS level study for the Project to define the following key items:

- **Engineering Geology Model (EGM)** – Definition of geotechnical domains within proposed mine plan.
- **Stope Stability and Dilution** – Stope sizes were evaluated using empirical methods Mawdesley et al. (2001) and dilution using Clark and Pakalnis (1997).
- **Infrastructure Stability and Sequence Review** – Stability review of mine workings (e.g., stopes, drifts, maintenance bay, ventilation raises, etc.) was completed using a 3D numerical non-elastic induced stress model, fault model, and analytical and empirical assessments.
- **Crown Pillar Stability** – Scaled Span empirical assessment and non-elastic induced stress review of the crown pillar.
- **Ground Support** – Using empirical, analytical, and numerical methods.
- **Backfill Strength Estimate** – Using empirical methods (Mitchell, 1982).
- **Portal Design** – Using empirical, numerical, and analytical methods.

The data available for the FS included geotechnical core logging data, downhole televiewer surveys, laboratory testing, and acoustic emissions laboratory in situ stress estimates. The assessments and conclusions summarised in this section are presented fully in the full rock mechanics stand-alone FS design report (WSP, 2025).

16.2.1 Engineering Geology Model – Rock Mass Quality

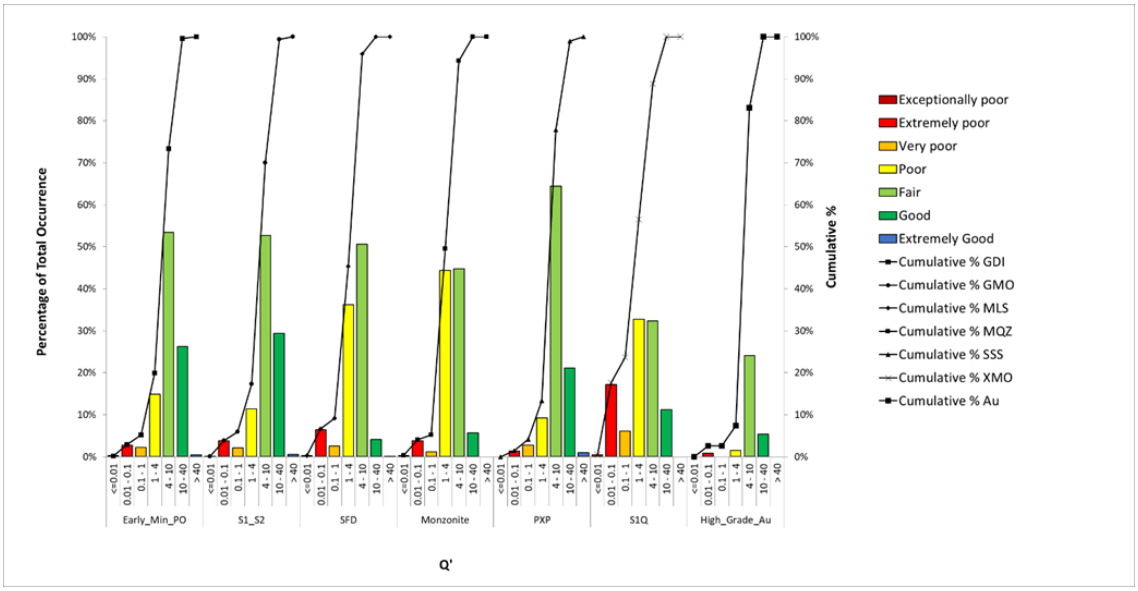
The rock mass classification {Q', GSI}, structural orientation and intact strength datasets were collected based on the logged lithology of the core. Geotechnical parameters were applied spatially to the 3D models relating the parameters to the location of the various underground mine workings. The geotechnical data contained seven (7) main logged lithologies, which had more than 100 m of geotechnical drilling. Of these lithologies - Diorite, Monzonite, Sandstone, and Breccia represent the

vast majority (i.e., 94%) of geotechnical drill core recovery; Breccia alone accounts for 54% of the geotechnical core recovery. The seven (7) lithologies assessed are Diorite (GDI), Monzonite (GMO), Marble (MLS), Quartzite (MQZ), Sandstone (SSS), Brecia (XMO) and Marl (SMR).

For the geologic formation wireframes, seven (7) were observed to intersect underground workings. Most of the underground workings are located within the S1_S2, SFD, and the Au wireframes. The geologic formation wireframes (and the predominant lithology logged within each wireframe) are S1_S2 (Sandstone), SRD (Breccia), Early_Min_PO (Diorite), Monzonite (Monzonite), PXP (Sandstone), Quartzite (S1Q), and Au (Sandstone).

Geotechnical domaining was based on the geologic formations and the input parameters were estimated accordingly. The geotechnical domains had similar classification values and ranged from Good (Barton Q'>10) for ~12% of the logged core, to Fair to Poor (Barton 10>Q'>1) for ~80% of the logged core, and Very Poor (Barton Q'<1) for ~8% of the logged core as illustrated on Figure 16.2.

Figure 16.2 – Barton Q' Distribution by Geotechnical Domain



Source: WSP, 2025

The intact strength characteristics were also similar for the geotechnical domains except for the S1_S2, PXP, and S1Q units (Table 16.1) that had relatively lower strength compared to the other formations. A review of the laboratory testing and the core quality indicates that this may be due to variable alteration/weathering resulting in lower strength values within the Sandstone domains.

Table 16.1 – Intact Strength Parameters by Geologic Formation

Geotechnical Domain (Predominant Lithology)	Mean σ_{ci} (MPa)	Mean m_i	Tension Cut-Off (MPa)
SFD (Breccia)	150	24	-5.7
S1_S2 (Sandstone)	100	12	-6.0
Monzonite (Monzonite)	140	24	-5.3
Early_Min_PO (Diorite)	170	22	-6.8
PXP (Sandstone)	100	12	-6.0
S1Q (Quartzite)	95	24	-3.6
Au (Sandstone)	100	12	-6.0

Faults with relatively poor ground character are projected to intersect the proposed mine workings. The fault model and interpretation of fault character and thickness were developed by DPM. WSP reviewed the fault model and identified 13 faults across three (3) categories of character that should either be avoided or specific mitigative actions applied to the mine plan and sequence.

Rock mass fabric (i.e., joints and veins) was also evaluated and indicated that the major open joint sets are aligned North and Southeast and are fairly consistent across the geotechnical domains. Closed structures were generally aligned with the open structure pole concentrations; however, it's influence on stability of the underground workings was estimated to be minimal as they are too infrequent and do not appear to control the rock mass behaviour under loading (based on review of the intact strength testing that broke along closed structure).

In-situ stress data was estimated at Čoka Rakita using an Acoustic Emission's laboratory test. WSP was not involved in the collection of the laboratory sample, selection of the testing area, or interpretation of the resulting stress tensor. The values estimated using this laboratory-based assessment method should be validated using an alternative method as the mining progresses. The input in-situ stress tensor (generalized from the Acoustic Emissions testing) used is:

- Sigma 3 is vertical and lithostatic.
- Sigma 1 is horizontal (East – West trending) and has a K ratio of 2.2.
- Sigma 2 is horizontal (North – South trending) and has a K ratio of 1.6.

16.2.2 Stope Stability and Dilution

The mine plan considers longitudinal long hole open stoping with cemented backfill. The proposed plan includes a diminishing sill pillar that will result in localized increased induced stress. Assessments were completed varying hanging wall/footwall angles, depth ranges, stope sizes, Q' ranges, and with or without ground support in the back and/or hanging wall. The design acceptance criteria selected is a 70% Probability of Failure (PoF) curve for the Mawdesley stability plot.

The stopes are mainly located in geotechnical domains Au and S1_S2 with a range of Q' from 4 (25th percentile) to 12.5 (75th percentile). Depending on the stope wall, a Q' design input parameter of 4 (25th percentile), 5 (35th percentile), or 7 (50th percentile) was used and varied to account for the relative risk of adverse conditions on each wall (e.g., the end walls used a Q' of 7 as only a few would experience detrimental stress conditions based upon the mine sequence). The stopes were assessed at an angle of 65° to align with the typical ore body dip. Based on the assessment, the back will require ground support in alignment with the standard support for development drifts in ore (i.e., no cable bolts). The recommended stope geometry is provided in the following Table 16.2.

Table 16.2 – Stable Stope Configurations

Mining Level	Stope Height (m)	Max Span (m)	Max Strike (m)
RL440 and Above	20	20	30
Below RL440	20	15	30

An estimate of stope dilution was completed using the stope sizes above on the Clark and Pakalnis stability charts and the following is the estimated dilution at the lowest mining domain (below 440):

- End walls – Rock mass quality Q'=7 results in <1 m of dilution and a reduction in rock mass quality to Q'=4 will increase the amount of dilution to >1 m.
- HW (65° stope dip) – Rock mass quality Q'=7 results in <0.5 m of dilution and a reduction in rock mass quality to Q'=4 will increase the amount of dilution to ~0.5 m.

16.2.3 Infrastructure Stability and Sequence Review

The infrastructure and sequence were reviewed for general stability and to help inform ground support assessments. The main method of assessment was numerical modelling with some aspects (vertical stability) also examined analytically. A non-elastic 3D numerical induced stress model was developed for the FS assessments using the PFS plan to provide insight into the challenges for the FS mine plan to identify optimisations and mitigate the design.

16.2.3.1 Mine Plan Review

The proposed mine plan was reviewed considering geotechnical ground characteristics and geotechnical induced stress assessment results to provide guidance on pillar spacing and critical infrastructure placement. The review included checking the placement of infrastructure against:

- Thirteen (13) fault wireframes considered to be geotechnically challenging for construction and operations. The 13 faults were tiered, with critical infrastructure (e.g., maintenance bay, ramp, raises, etc.) being moved to avoid the critical faults. Some long-term infrastructure remains intersected by faulting and will have an increased ground support/maintenance requirement.

- Pillar dimensions were reviewed to identify areas where long term pillars did not meet a minimum thickness of two (2) times the span of the adjacent infrastructure and temporary pillars did not meet a minimum thickness of 1.5 times the span of the temporary drifting. This review considered backfill sequencing requirements allowing drifting in proximity to backfilled stopes.
- The non-elastic 3D numerical induced stress model used a non-ideal sequence (i.e., PFS sequence) which included localized diminishing middling pillars. Generally, the non-elastic behaviour and inferred rock damage were within a few meters of the stope shapes and only become larger after a significant extraction ratio has occurred. The largest diminishing middling pillar did not enter post-peak behaviour prior to extraction; this suggests that the mine will generally remain stable excluding a <5 m damage halo around the extraction areas. Pillars will be carrying a relatively large stress load (e.g., the final middling pillar in the model contained Sigma 1 up to 48 MPa) and the blasting of the final few stopes in these pillars may cause minor to moderate seismicity which could impact nearby drift performance). A relatively large tensional zone is predicted in the ore body HW and will influence access drifting. This is accounted for in the long-term infrastructure offset distances and ground support recommendations. This large tensional zone can impact faulted rock and can result in fault damage at a distance greater than damage would be observed in the intact rock.

16.2.3.2 Sequence Review

The sequence of the mine plan was reviewed with an elastic 3D numerical stress model and recommendations were provided to the mine engineering team. It was recommended that the mine plan avoid diminishing middling pillars on each level and follow a retreat sequence from one end of the level to the other. This sequence was then reviewed by rock mechanics and mine engineering staff going over each stope and development step to confirm minimum pillar recommended thicknesses are observed, stopes and drifts are backfilled prior to adjacent mining, and that no open drifts are adjacent to each other without a sufficient rock pillar.

A temporary sill pillar planned to separate the upper deposit from the lower deposit is to be mined at the end of the mine sequence. Sill pillar stability was reviewed using the non-elastic 3D numerical induced stress modelling. The pillar core is estimated to remain stable during mining. Small areas on the pillar edges and rock near stopes may experience rock mass damage for 5 m (or up to 10 m locally in brows). Cables are recommended for the stope brows in the sill pillar.

16.2.3.3 Vertical Stability

The surface vent raise has a planned diameter of 4.1 m, accounting for 50 mm of shotcrete. The underground vent raises connect from level to level within the footprint of the spiral ramp on both the north and south sides of the mine and have a planned diameter of 2 to 4 m.

Empirical vertical stability assessments were completed using McCracken and Stacey (1989) considers a modified Barton Q value defined as QR plotted against raise diameter. A ventilation raise, with required longevity of 10 years, was considered stable if the PoF is 5% or less. During the FS study, a pilot hole was drilled at the proposed location of the surface vent raise to provide insight to the local rock mass conditions. There was limited information on the rock mass quality of the underground vent raise locations and assessments are based on rock mass conditions from the underground geotechnical program and the nearest exploration drillholes.

The following is concluded and recommended for the surface ventilation raise:

- A 4 m diameter unsupported opening is unlikely to be stable for the near surface weathered zone (up to a depth of ~ 50 m). The first 35 m is poorer quality than the 35 m to 50 m depth interval. An inclined fault zone exhibiting rubble, water flow, and lost core between 80 to 100 m depth may be unstable during raise construction and could unravel prior to finishing the raise bore and installing shotcrete. Options to address include consolidation grouting, bolting, or acceptance of the increased instability risk.
- The entire raise should be sprayed with 50 mm thick shotcrete to improve stability.

The following is concluded and recommended for the underground ventilation raises:

- Drill pilot holes and collect geotechnical data prior to excavation of each underground vent raise. This drilling can occur during operations. If poor rock mass quality is encountered in the pilot hole, relocate the underground vent raise or install wall support (e.g., 50 mm thick shotcrete). Assume up to 50% of the currently proposed raises would need 50 mm of shotcrete.

16.2.4 Crown Pillar Stability

A simple empirical crown pillar stability assessment (Scaled Span) and temporary sill pillar excavation assessment using 3D numerical stress model (FLAC3D) was completed to identify potential instability issues with the FS mine design. The crown pillar was assumed to be 260 m in thickness, 17 m in span and 50 m in length with a dip of 48° with a rock mass quality Q'=7 (Good). The Scaled Span empirical method indicated a PoF between 0.5% and 1.5% which is classified as long-term stable. The 3D non-elastic induced stress model also did not indicate any stability related issues in the crown pillar.

16.2.5 Ground Support

Ground support assessments were completed based on a range of underground excavation types (long-term, temporary, unique, intersections) and anticipated ground conditions (Q'<0.04, 0.04<Q'<1, 1 <Q'< 10, and Q'>10). Assessment methods include empirical charts (Grimstad and Barton, 1993), RocScience Unwedge, RocScience RS2, and rock behaviour interpreted from the 3D non-elastic numerical induced stress model. The ground support recommendations are based

on the SFD and S1_S2 geologic formations. Temporary headings consider split-sets (grouted and ungrouted) and long-term and unique infrastructure use grouted split sets, MD bolts, or short fully grouted cables (or equivalent). Fibre reinforced shotcrete (FRS) is only considered for a Q' < 1 or for specific critical chambers, while #6 gauge welded wire mesh (WWM) is used as surface support for other categories. Fully grouted 15.2 mm cables are required for large openings. For Q'<0.04, self drilling anchors are assumed. 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 (P)	S1_S2	> 10	Back/Shoulders: SS-47 (Ungrouted) 2.4 m long at 1.2 m x 1.2 m Ribs: SS-47 (Ungrouted) 2.4 m long at 1.5 m x 1.5 m	WWM
		1 < Q' < 10	Back/Shoulders: SS-47 (Ungrouted) 2.4 m long at 1.2 m x 1.2 m Ribs: SS-47 (Ungrouted) 2.4 m long at 1.5 m x 1.5 m	WWM
		0.04 < Q' < 1	Back/Shoulders: MD Bolts or Grouted Cables 3.0 m long at 1.2 m x 1.2 m Ribs: MD Bolts or Grouted Cables 3.0 m long at 1.2 m x 1.2 m	50 mm FRS & WWM
		<0.04	Back/Shoulders: Self Drilling Anchors 3.0 m long at 1.2 m x 1.2 m Ribs: Self Drilling Anchors 3.0 m long at 1.2 m x 1.2 m	25 mm flash FRS, 75 mm FRS second pass, WWM
Crosscut Incline and Decline (>50 m from ore body)	SFD	> 10	Back/Shoulders: SS-47 (Ungrouted) 2.4 m long at 1.2 m x 1.2 m Ribs: SS-47 (Ungrouted) 2.4 m long at 1.5 m x 1.5 m	WWM
		1 < Q' < 10	Back/Shoulders: SS-47 (Ungrouted) 2.4 m long at 1.2 m x 1.2 m Ribs: SS-47 (Ungrouted) 2.4 m long at 1.4 m x 1.4 m	WWM
		0.04 < Q' < 1	Back/Shoulders: MD Bolts or Grouted Cables 3.0 m long at 1.2 m x 1.2 m Ribs: MD Bolts or Grouted Cables 3.0 m long at 1.2 m x 1.2 m	50 mm FRS & WWM
		<0.04	Back/Shoulders: Self Drilling Anchors 3.0 m long at 1.2 m x 1.2 m Ribs: Self Drilling Anchors 3.0 m long at 1.2 m x 1.2 m	25 mm flash FRS, 75 mm FRS second pass, WWM

Development	Formation	Rock Mass Quality (Q')	Minimum Bolt Pattern Density	Surface Support
Crosscut Incline and Decline (<50 m from ore body)	SFD	> 10	Back/Shoulders: SS-47 (Grouted) 2.4 m long at 1.4 m x 1.4 m Ribs: SS-47 (Ungrouted) 2.4 m long at 1.5 m x 1.5 m	WWM
		1 < Q' < 10	Back/Shoulders: MD Bolts or Grouted Cables 2.4 m long at 1.2 m x 1.2 m Ribs: MD Bolts or Grouted Cables 2.4 m long at 1.2 m x 1.2 m	WWM
		0.04 < Q' < 1	Back/Shoulders: MD Bolts or Grouted Cables 3.0 m long at 1.2 m x 1.2 m Ribs: MD Bolts or Grouted Cables 3.0 m long at 1.2 m x 1.2 m	50 mm FRS & WWM
		<0.04	Back/Shoulders: Self Drilling Anchors 3.0 m long at 1.2 m x 1.2 m Ribs: Self Drilling Anchors 3.0 m long at 1.2 m x 1.2 m	25 mm flash FRS, 75 mm FRS second pass & WWM

Notes:

1 Temporary drifting support classes may be applied for up to 6 months in long-term infrastructure. Long-term ground support requirements (see Table 16.4) 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.

2. A FoS of 1.1 was used for design acceptance criteria of temporary drifting.

3. Crosscut incline and declines should follow Q' < 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.

4 Of note, temporary headings are mostly within the induced stress field generated by extraction resulting in less favourable stress conditions.

5 Ungrouted SS-47 bond strength is assumed at 40 kN / meter assuming excellent installation procedures.

6 Crosscut incline and decline within 50 m of the ore body will be needed prior to extracting 20% of the ore body. Initial bolting can follow the crosscut support for areas >50 m from the ore body.

Table 16.4 – Recommended Ground Support for Long-Term Headings

Development	Formation	Rock Mass Quality (Q')	Minimum Bolt Pattern Density	Surface Support
Upper and Lower Decline	SFD	> 10	Back/Shoulders: SS-47 (Grouted) 2.4 m long at 1.2 m x 1.2 m Ribs: SS-47 (Grouted) 3.0 m long at 1.2 m x 1.2 m	WWM
		1 < Q' < 10	Back/Shoulders: SS-47 (Grouted) 2.4 m long at 1.2 m x 1.2 m Ribs: SS-47 (Grouted) 3.0 m long at 1.2 m x 1.2 m	WWM
		0.04 < Q' < 1	Back/Shoulders: MD Bolts or Grouted Cables 3.0 m long at 1.2 m x 1.2 m Ribs: MD Bolts or Grouted Cables 3.0 m long at 1.2 m x 1.2 m	50 mm FRS & WWM
		<0.04	Back/Shoulders: Self Drilling Anchors 3.0 m long at 1.2 m x 1.2 m Ribs: Self Drilling Anchors 3.0 m long at 1.2 m x 1.2 m	25 mm flash FRS, 75 mm FRS second pass, WWM
Spiral ramp, Ramp level accesses, Stockpile/ Remuck bay	SFD	> 10	Back/Shoulders: SS-47 (Grouted) 2.4 m long at 1.2 m x 1.2 m Ribs: SS-47 (Ungrouted) 2.7 m long at 1.2 m x 1.2 m	WWM
		1 < Q' < 10	Back/Shoulders: SS-47 (Grouted) 2.4 m long at 1.2 m x 1.2 m Ribs: SS-47 (Ungrouted) 2.7 m long at 1.2 m x 1.2 m	WWM
		0.04 < Q' < 1	Back/Shoulders: MD Bolts or Grouted Cables 3.0 m long at 1.2 m x 1.2 m Ribs: MD Bolts or Grouted Cables 3.0 m long at 1.2 m x 1.2 m	50 mm FRS & WWM
		<0.04	Back/Shoulders: Self Drilling Anchors 3.0 m long at 1.2 m x 1.2 m Ribs: Self Drilling Anchors 3.0 m long at 1.2 m x 1.2 m	25 mm flash FRS, 75 mm FRS second pass, & WWM
Cuddies	SFD	N/A	Back/Shoulders: SS-47 (Grouted) 2.4 m long at 1.2 m x 1.2 m Ribs: SS-47 (Ungrouted) 2.4 m long at 1.4 m x 1.4 m	WWM

Notes:
1. A FoS of 1.3 was used for design acceptance criteria of long-term drifting.
2. 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 Intersections

Intersection Inscribed Circle Span	Minimum Bolt Pattern Density – Secondary Support
> 6.0 m and <= 7.0 m	Back/Shoulders: Cable Bolts (15.2 mm diameter) 5.0 m long at 2.0 m x 2.0 m
> 7.0 m and <= 12.5 m	Back/Shoulders: Cable Bolts (15.2 mm diameter) 6.0 m long at 1.5 m x 1.5 m

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

Table 16.6 – Recommended Ground Support for Unique Infrastructure

Development	Minimum Bolt Pattern Density	Surface Support
Fuel and Lube Bay (6 m W x 5 m H)	Back/Shoulders: MD Bolts or Grouted Cables 2.4 m long at 1.5 m x 1.5 m Ribs: MD Bolts or Grouted Cables 2.4 m long at 1.5 m x 1.5 m	50 mm FRS & WWM
Explosive Storage (7 m W x 6 m H)	Back/Shoulders: MD Bolts or Grouted Cables 2.4 m long at 1.5 m x 1.5 m Back/Shoulders: Grouted Cables 4.0 m long at 2.0 m x 2.0 m Ribs: MD Bolts or Grouted Cables 2.4 m long at 1.5 m x 1.5 m	50 mm FRS & WWM
Emulsion Bay (11.8 m W x 7 m H)	Back/Shoulders: MD Bolts or Grouted Cables 2.4 m long at 1.5 m x 1.5 m Back/Shoulders: Grouted Cables 7.0 m long at 2.0 m x 2.0 m Ribs: MD Bolts or Grouted Cables 2.4 m long at 1.5 m x 1.5 m Ribs: Cable Bolts 4.0 m long at 2.0 m x 2.0 m	50 mm FRS & WWM
Maintenance Bay (9 m W x 7.5m H)	Back/Shoulders: MD Bolts or Grouted Cables 2.4 m long at 1.5 m x 1.5 m Back/Shoulders: Grouted Cables 5.0m long at 2.0 m x 2.0 m Ribs: MD Bolts or Grouted Cables 2.4 m long at 1.5 m x 1.5 m Ribs: Cable Bolts 4.0 m long at 2.0 m x 2.0 m	50 mm FRS & WWM

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

16.2.6 BACKFILL STRENGTH

A backfill strength assessment was completed using the Mitchell et. Al (1982) considering backfill density of 2,000 kg/m³, UCS strength of between 0.35 and 0.5 MPa, cohesion of 30°, stope plug thickness (stope height) of 20 m, maximum stope span of 15 m, maximum height of 30 m and a stope dip varying between 65 and 90°.

The factor of safety (FOS) associated with the backfill strength and stope dips for overhand mining are as follows:

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

For underhand mining the recommended backfill strengths are:

- 1.8 MPa for a 65° and 90° stope dip = FOS of 1.5.

16.2.7 PORTAL DESIGN

An Upper and Lower Portal are required for the project to facilitate access to the underground. Decline dimensions are 5.5 m W x 6.1 m H and portals are excavated within an upwards sloping terrain. A campaign to collect site specific geomechanical information was undertaken as part of the FS study to inform of expected rock mass conditions at the portal and along the decline. A total of seven (7) boreholes were drilled at the upper portal and two (2) were drilled at the proposed lower portal location. The lithological unit for the upper and lower declines is the Monomict Breccia unit.

Geomechanical data (i.e., Rock Quality Designation (RQD), Strength, joint roughness and alteration and number of joint sets) was reviewed in 2.5 m intervals to 30 m of depth to characterise the surface profile. Depth to the soil/rock interface was validated by observing core photos for both portals.

16.2.7.1 Upper Portal

For the Upper Portal, an average Q' value of 1.4 for the first 30 m of bedrock was assessed, in alignment with "Poor" rock conditions considering the seven local boreholes. Three (3) joint sets were observed. The overburden thickness at the proposed location is 11.8 m. Based on the proposed terrace elevation, 7.9 m of bedrock remains above the portal headcover, consisting of a weathered bedrock zone and a fresh fractured bedrock zone. Two-dimensional numerical modelling was completed using input parameters informed from site investigation campaigns. The material model considered the rock and overburden as Voronoi material. The support recommendations for the first 50 m of the Upper Portal are as follows:

Table 16.7 – Upper Portal - Support Recommendations for Initial Decline

Chainage from Portal Face	Lattice Girders	Shotcrete	Ground Support
0 to 10 m	4 bar #140, bar size 20 mm, 20 cm of FRS with lattice girders on a 2.5 m spacing Note: Tunnel Sizing is 5.9 m W x 6.3 m W to accommodate lattice girders.	10 cm FRS	2.4 m long grouted steel bolts (rebar, short cables or self drilling anchors) on a 1.2 m x 1.2 m pattern. Spiling consisting of 6 m 32 mm bar (minimum yield strength 400 kN) self drilling long spiles at an angle of 10 to 15 degrees off the back alignment of the tunnel. An overlap of 2 m is recommended. Inter Spile Spacing will be 30 cm to start and may increase with improving rock mass conditions. Required from back arch and to within 1.5 m of floor.
10 to 20 m	4 bar #140, bar size 20 mm, 20 cm of FRS with lattice girders on a 2.5 m spacing Note: Tunnel Sizing is 5.9 m W x 6.3 m W to accommodate lattice girders	10 cm FRS	2.4 m long grouted steel bolts (rebar, short cables or self drilling anchors) on a 1.2 m x 1.2 m pattern
20 to 30 m	4 bar #140, bar size 20 mm, 20 cm of FRS with lattice girders on a 4 m spacing Note: Tunnel Sizing is 5.9 m W x 6.3 m W to accommodate lattice girders.	10 cm FRS	2.4 m long grouted steel bolts (rebar, short cables or self drilling anchors) on a 1.2 m x 1.2 m pattern
30 to 40 m	None Decline Sizing is 5.5 m W x 6.1 m H	5 cm FRS	2.4 m long grouted steel bolts (rebar, short cables or self drilling anchors) on a 1.2 m x 1.2 m pattern with WWM
40 to 50 m	None Decline Sizing is 5.5 m W x 6.1 m H	None	2.4 m long grouted steel bolts (rebar, short cables or self drilling anchors) on a 1.2 m x 1.2 m pattern and WWM
50 m and beyond	Typical UG Support System in alignment with Q classification. Tunnel sizing is 5.5 m W x 6.1 m H		

16.2.7.2 Lower Portal

The bedrock at the Lower Portal location was determined to be only marginally better than the Upper Portal location. The portal crown thickness based on the proposed terrace elevation is thicker at 17.5 m and the overburden depth is assessed to be less than 1 m in thickness. Similar design considerations as the Upper Portal are recommended for the Lower Portal face support for anchor length, type, and shotcrete application for 10 m on either side of the portal entrance.

Table 16.8 – Lower Portal - Support Recommendations for Initial Decline

Chainage from Portal Face	Lattice Girders	Shotcrete	Ground Support
0 to 10 m	4 bar #140, bar size 20 mm, 20 cm of FRS with lattice girders on a 2.5 m spacing Note: Tunnel Sizing is 5.9 m W x 6.3 m W to accommodate lattice girders.	10 cm FRS	2.4 m long grouted steel bolts (rebar, short cables or self drilling anchors) on a 1.2 m x 1.2 m pattern Spiling consisting of 6 m 32 mm bar (minimum yield strength 400 kN) self drilling long spiles at an angle of 10 to 15 degrees off the back alignment of the tunnel. An overlap of 2 m is recommended. Inter Spile spacing will be 30 cm to start and may increase with improving rock mass conditions. Required from back arch and to within 3 m of floor.
10 to 20 m	4 bar #140, bar size 20 mm, 20 cm of FRS with lattice girders on a 2.5 m spacing Note: Tunnel Sizing is 5.9 m W x 6.3 m W to accommodate lattice girders	10 cm FRS	2.4 m long grouted steel bolts (rebar, short cables or self drilling anchors) on a 1.2 m x 1.2 m pattern
20 to 30 m	None Decline sizing is 5.5 m W x 6.1 m H	5 cm FRS	2.4 m long grouted steel bolts (rebar, short cables or self drilling anchors) on a 1.2 m x 1.2 m pattern with WWM
30 to 50 m	None Decline sizing is 5.5 m W x 6.1 m H	Flash shotcrete may be required	2.4 m long grouted steel bolts (rebar, short cables or self drilling anchors) on a 1.2 m x 1.2 m pattern with WWM
50 m and beyond	Typical UG Support System in alignment with Q classification. Tunnel sizing is 5.5 m W x 6.1 m H		

The following is recommended for the next level of study:

- A pilot hole for the Upper Portal to assess bedrock quality and depth for the initial decline is recommended to validate study inputs. Detailed mapping of the portal face is recommended to be completed prior to commencement of the decline to validate geomechanical inputs.
- Hydrogeological conditions and regime are not well understood at the Upper and Lower Portals. Additional studies are recommended to be completed, and study findings should be incorporated into the Portal and Initial Decline designs.

16.3 Hydrogeology

Additional site-specific hydrogeological data were collected throughout the 2024 field investigation campaign by the University of Belgrade to further characterize the groundwater conditions within the areas of the proposed surface and underground mine infrastructure development in support of the FS for the Čoka Rakita Project.

As part of the FS scope of work WSP reviewed hydrogeological field data collected by the university team between September and December 2024 with an objective to update the conceptual understanding of the underground workings groundwater conditions and to expand the input dataset for an update of the PFS numerical groundwater flow model (September 19, 2024, was a cutoff date for the input parameters into the numerical groundwater flow model developed as part of the PFS completed for the project in 2024). The review included information presented in a factual report issued by the University of Belgrade in October 2024, and raw data files provided by DPM up to February 27, 2025. WSP data review was completed prior to the issue of the factual report by the University of Belgrade in March 2025, that summarized all field hydrogeological data collected at Čoka Rakita between 2023 and 2025.

The updated numerical groundwater flow model was used to update the PFS groundwater inflow predictions into the underground development modelled for the 10-year LOM in support of the underground water balance and design of the dewatering infrastructure. Due to the cutoff date for data input for the FS modelling effort, the Phase 3 PFS mine design was used for the model update. The data used for the update of the conceptual and numerical hydrogeological models are summarised in the following subsections.

16.3.1 Groundwater Monitoring Network

Standpipe monitoring wells and vibrating wire piezometers (VWP) were installed at the project site as part of the 2024 PFS and FS field investigations to obtain information on groundwater levels, gradients, and flow directions. The groundwater monitoring network within the Project area includes 16 shallow boreholes located around the surface mine infrastructure facilities and 6 deep boreholes completed in the area of the UG development. Eight (8) of the shallow boreholes are installed with standpipe piezometers, and the other eight (8) boreholes were instrumented with a single vibrating wire piezometer (VWP) each. The six (6) deep boreholes were instrumented with multilevel VWPs.

Water level data from the shallow standpipe piezometers and VWPs reviewed as part of this study cover the period from November 2024 to January 2025. Based on the collected data, the water table at the site generally follows the topography and vary between 1.5 m below ground surface in the valleys up to about 27 m below ground surface at higher elevations.

Data from the VWPs installed in the deep boreholes cover various recording periods between March 2023 and January 2025. The observed hydraulic heads in the deeper installations show a

decreasing tendency with depth with the observed vertical hydraulic gradients varying between -0.05 and -0.88 m/m.

16.3.2 HYDROSTRATIGRAPHIC UNITS

A total of eleven (11) hydrostratigraphic units (HSU) were identified within the Project area based on the local geology and the hydraulic parameters obtained from the field investigations. Their zonation has been enhanced from the PFS by incorporating the updated geological Leapfrog model local to the Project area. Table 16.9 lists the HSU zones included in the updated numerical groundwater flow model.

Table 16.9 – HSU Defined in the Numerical Groundwater Flow Model

HSU Zone	Lithology	Description
1	GDI	Diorite
2	XMO	Volcanic rocks and breccia
3	LS	Limestone and marble
4	SMR	Marl
5	SSS	Sandstone
6	MQZ and GMO	Monzonite and quartz-monzonite
7	MPH	Schist and phyllite
8	SQ1	Crystalised sandstone
9	Mineralisation	Mineralisation
10	OVB	Overburden
11	Alluvium	Stream channel sediment

16.3.3 HYDRAULIC PARAMETERS

Hydraulic conductivity values were calculated from data obtained from 196 single well response tests (139 Lugeon tests and 57 Falling Head tests) undertaken in various boreholes across the Project area during the field investigations. Most geological formations show similar hydraulic behaviour with geometric means within approximately one order of magnitude between about 2.5×10^{-8} and 1.5×10^{-7} m/s. The sandstone unit however show higher hydraulic conductivity values compared to the other tested units (i.e., volcanic breccia, diorite, monzonite, quartzite). No testing has been conducted specifically in the limestone or marble within the Project area but based on the testing performed within these units in the Timok Project area their hydraulic conductivities are expected to be as high as the sandstone, or higher.

Regional and localised faulting is prevalent throughout the Project area, and the fracture system provides the principal pathways for groundwater movement in the bedrock. A large database of

geological structures was developed by DPM based on the core data, but only the major faults tested during the field investigations with hydraulic conductivities approximately one order of magnitude higher than the surrounding rock mass have been implemented in the updated model (2×10-7 and 1×10-6 m/s).

16.3.4 RECHARGE

Infiltration of precipitation (as rain or snow) is the primary source of groundwater recharge within the Project area. Infiltration has been estimated to be approximately 14% of Mean Annual Precipitation (MAP) using the Rushton Method (WSP, 2025a).

16.3.5 HYDRAULIC GRADIENTS AND GROUNDWATER FLOW DIRECTION

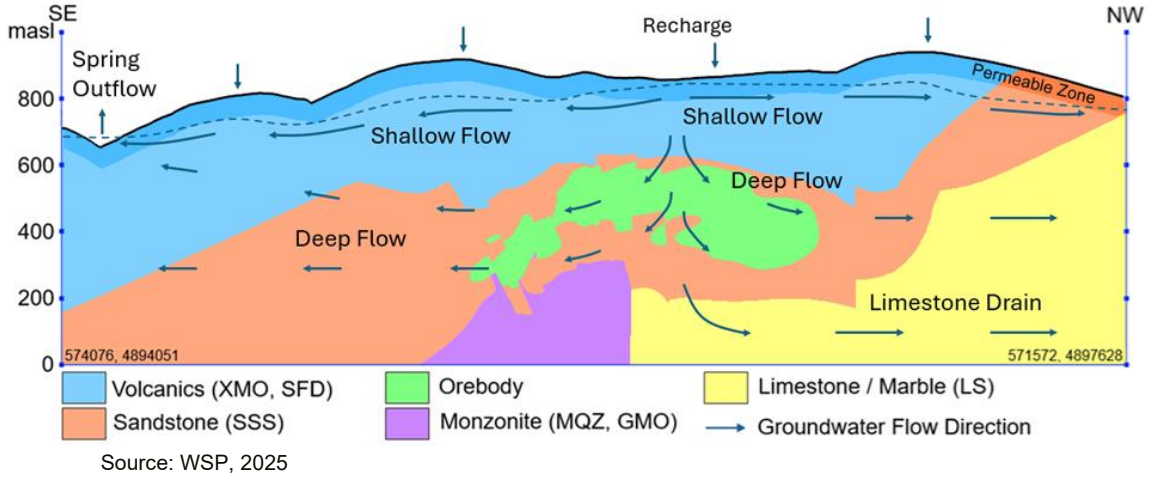
Groundwater levels within the Project area can generally be considered a subdued reflection of topography, with groundwater flow direction from the higher elevations towards the nearest stream, river, or spring. Groundwater levels vary seasonally and are heavily influenced by the amount of precipitation. The highest groundwater levels and seasonal fluctuations are typically in early spring (March and April) when precipitation rates are high and melting of the snowpack occurs.

The following presents a summary of the groundwater system observations within the Project area:

- The shallow groundwater regime is characterised by groundwater movement following topography from the summit of Čoka Rakita hill at approximately 945 masl to lower topographic elevations around 600 masl at the Jagnilo and Lipa Rivers to the northwest and east, and to about 500 masl at the Crna River to the southwest. Discharge occurs through low yielding springs and as baseflow into seasonal streams and small rivers within the Project area (UofB, 2024).
- The deep groundwater regime is characterised by relatively strong downward hydraulic gradients in the orebody area with a range from -0.05 to -0.88 m/m. Horizontal hydraulic gradients in the deeper zone are lower than horizontal gradients observed in the shallow groundwater regime due to reduced topographic influence at depth.
- Based on the information obtained from drilling and instrumentation of borehole RADDHG008, the limestone and marble unit underlaying the orebody acts as a groundwater drain. A very strong downward hydraulic gradient is the primary driver of groundwater movement towards this unit in the deep groundwater system. Groundwater flow within the limestone and marble is controlled by their position relative to the monzonites and Palaeozoic schists. The intrusion of monzonite prevents the continuity of groundwater circulation to the west from within the limestones and marbles (UofB, 2024). Discharge from the limestones and marbles is through underground outflow outside of the Project area, following the decreasing piezometric surface towards the northwest where these rocks outcrop. Significant karstic features exist where the formation outcrops in the region (UofB, 2021).

A schematic representation of the conceptual hydrogeological model of the Project area is presented in Figure 16.3.

Figure 16.3 – Schematic Conceptual Hydrogeological Model



Source: WSP, 2025

16.3.6 Numerical Groundwater Flow Model and Inflow Estimates

A numerical groundwater flow model was developed for the Project site as part of the PFS with an objective to build a model that reflected the current understanding of site hydrogeology and use the model to estimate the quantity of potential groundwater inflows to the planned underground mine development. Additional site-specific hydrogeological data collected throughout the 2024 field investigation campaign were used to update the PFS model to further characterize the groundwater conditions in the areas of the proposed mine infrastructure development and update the inflow estimates into the underground development in support of the FS for the Project.

A 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 discretize the local features around the mine in parallel with the regional extent of the catchments that contribute to the overall groundwater balance.

The updated numerical groundwater flow model was used to simulate three different scenarios for groundwater inflows into the underground development including:

- Base Case Scenario that represents the current understanding of Site conditions during the FS.

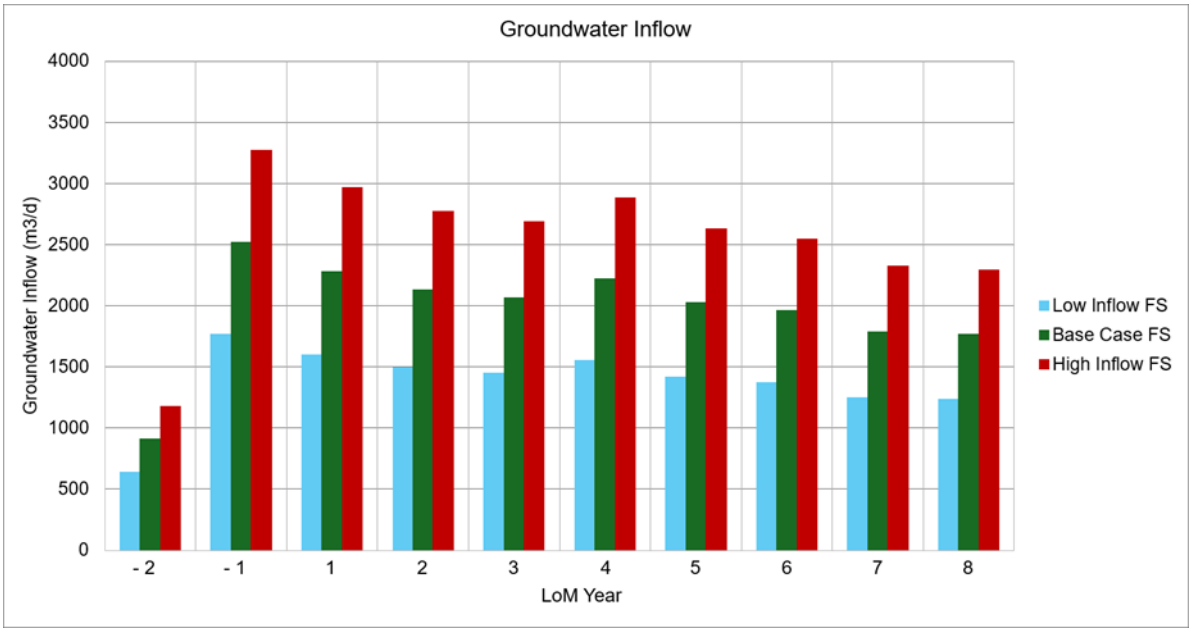
- High Inflow Scenario that simulates increased transmissivity, storage and recharge in the model's HSU, representing higher than expected groundwater inflow.
- Low Inflow Scenario that simulates decreased transmissivity, storage and recharge, representing lower than expected groundwater inflow.

Each scenario required modifications to the base-case pre-mining model to produce a unique pre mining model for each scenario. The computed mean annual groundwater inflow into the mine is presented for all three cases in Table 16.10 and Figure 16.4.

Table 16.10 – Simulated Groundwater Inflows (annual mean)

LOM Year	Base-case PFS (m^3/d)	Low Inflow FS (m^3/d)	Base-case FS (m^3/d)	High Inflow FS (m^3/d)
-2	1187	637	909	1180
-1	1721	1770	2520	3271
1	1739	1599	2281	2964
2	1585	1493	2131	2770
3	1546	1447	2066	2685
4	1532	1554	2218	2883
5	1504	1418	2024	2630
6	1527	1372	1959	2546
7		1252	1787	2323
8		1236	1765	2294

Figure 16.4 – Simulated Groundwater Inflows to UG Development



Source: WSP, 2025

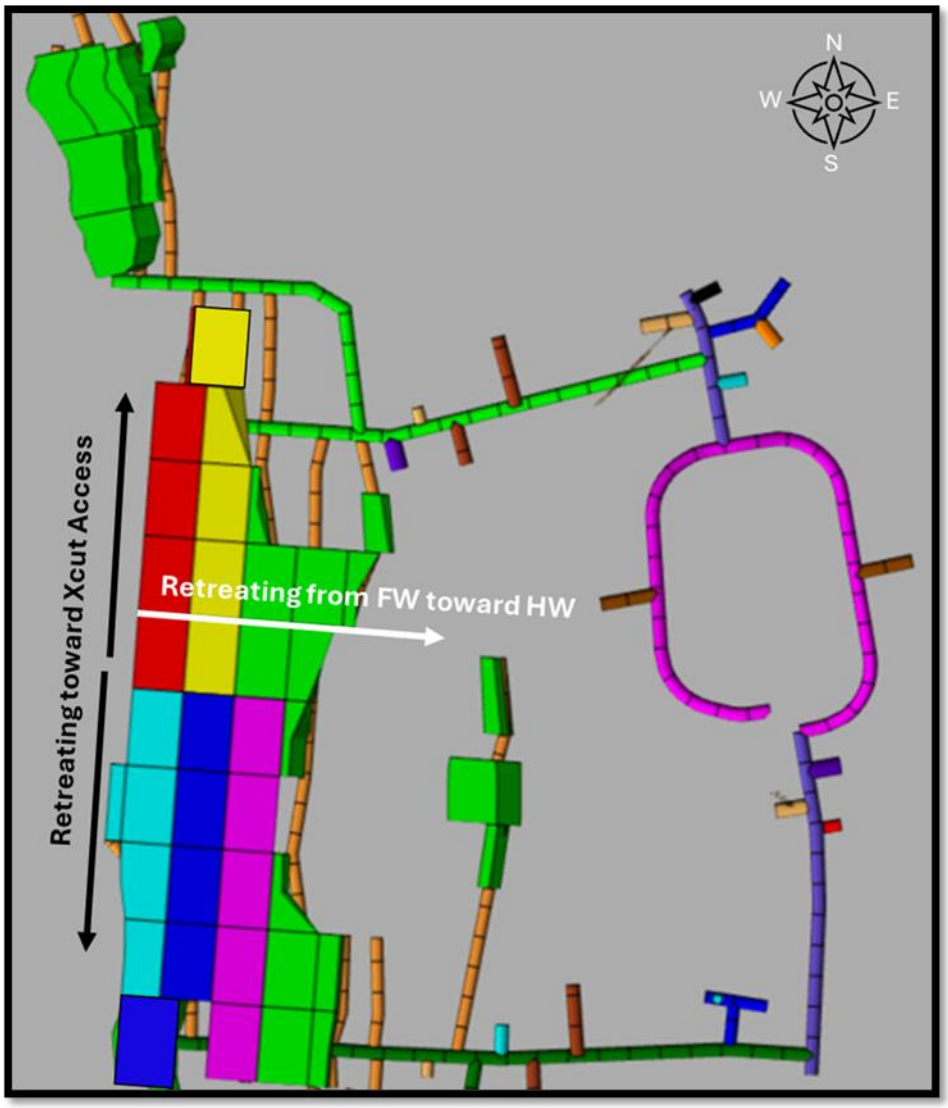
16.4 Mining Method

16.4.1 SUBLEVEL STOPING

The orebody will be mined by sublevel long-hole open stoping with long hole drills. 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 ore sill.

Figure 16.5 illustrates the strategy for developing the sublevels, using a plan view of the RL490 - RL510 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 an ore sill on the upper sublevel for drilling and loading explosives and an ore sill on the lower sublevel for mucking blasted ore. These ore sills 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 on RL420.

Figure 16.5 – Sequence Advancement on Typical Level



Source: WSP, 2025

The longitudinal stopes will be mined in a retreating fashion, advancing in series, one after the other, towards the crosscuts.

- Dimensions of the stopes in the upper horizon (RL440 and above) will be 20 m wide (from Footwall to Hanging wall) by 20 m high which corresponding to the sublevel interval and will have a strike length of 30m.
- Dimensions of the stopes in the lower horizon (Below RL 440) are like those in the upper mine; however, the width of the stope is to be 15m.

There will be instances where stopes need to be mined against previously mined stopes that have at least two (2) faces of paste backfill. Drilling of these stopes should be properly planned out during the short-term planning phase with a detailed fan pattern and strategy to minimise dilution.

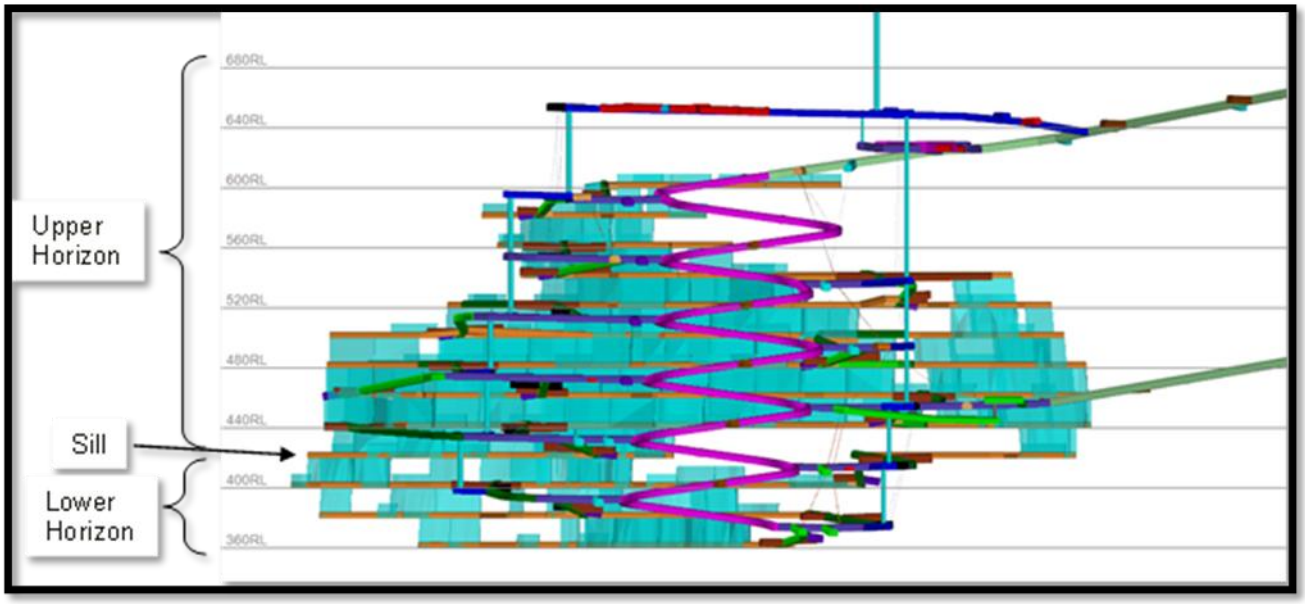
The QP is of the opinion that sublevel stoping remains an appropriate method for mining the Čoka Rakita deposit.

16.4.2 SEQUENCING

The mine is divided into two (2) horizons: an upper horizon from RL440 to RL620 and a lower horizon from RL360 to RL440 with the sill pillar at RL420 as presented in Figure 16.6.

The mining sequence will start from the bottom of the upper horizon at RL440, and the sublevels will be mined in ascending sequence, working upwards. Once RL560 has been reached, mining of the lower horizon can begin at RL360 and will continue bottom up.

Figure 16.6 – East to West View of the Mining Horizons



Source: WSP, 2025

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 16.6. Stoping operations will generally occur on more than one sublevel at a time to ensure enough areas are active to meet production targets.

16.4.3 DRILLING

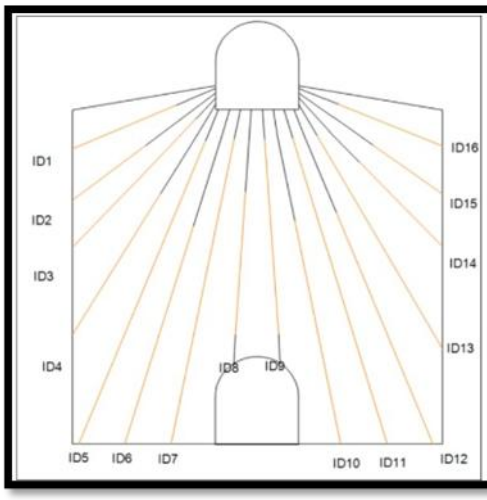
16.4.3.1 Production

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 required for service holes and longholes.

The slot which provides the free face for blasting the rings will be a conventional 2.5m by 2.5m drop raise with five (5) 203mm diameter reamed holes.

Production holes will be drilled in rings angled downward or upward in fan formation from a single pivot point at the centre line of the upper ore sill and the lower ore sill for uppers. The holes will have a diameter of 89 mm, and the toe spacing between them will be 2.5 m. The burden between rings will be 2.5 m. Figure 16.7 illustrates a typical longhole drilling layout in cross-section.

Figure 16.7 – Drilling and Loading Layout of a Typical Longhole Ring



HOLE ID	Diameter (mm)	Length (m)	Collar (m)	Toe collar (m)	Charge length (m)	explosive quantity (kg)
1	89	8,4	2,3	0	6,1	47,35
2	89	9,7	4,8	0	4,9	38,04
3	89	12,3	1,6	0	10,7	83,06
4	89	15,9	6	0	9,9	76,85
5	89	20,3	1,8	0	18,5	143,61
6	89	19,7	7	0	12,7	98,58
7	89	19,25	1,6	0	17,65	137,01
8	89	14	4,6	1,2	8,2	63,65
9	89	14	1,6	1,2	11,2	86,94
10	89	19,25	6,3	1,2	11,75	91,21
11	89	19,7	1,6	0	18,1	140,50
12	89	20,3	6,1	0	14,2	110,23
13	89	15,9	2	0	13,9	107,90
14	89	12,3	5	0	7,3	56,67
15	89	9,7	5	0	4,7	36,48
16	89	8,4	2,3	0	6,1	47,35
	Total	239,1			Total	1318,0725

Source: WSP, 2025

16.4.3.2 Development

The development heading size varies throughout the mine; however, the blasting pattern remains similar for all headings. Drilling is done by a 2-boom development jumbo drilling 45 mm diameter blast holes with reamed slot holes of 89 mm in diameter to create the free face.

Ground Support – will be by 2-boom development jumbo utilizing FS-46 split sets grouted and ungrouted based on geomechanical recommendations. Welded wire mesh is to be used in conjunction with the split sets. Depending on ground conditions, grouted splits will be used and will be grouted with a separate crew after the bolts are installed. Additional secondary support will also

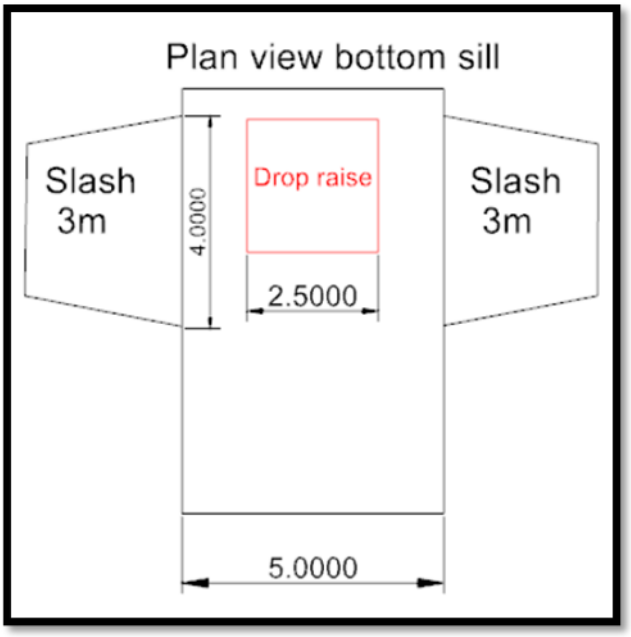
be applied in the form of shotcrete as directed by the geomechanical engineer. Additional details surrounding ground support can be found in Section 16.2.5 of the Report.

Underground Mine Roadbed - Aggregates from the nearby Žagubica quarry (local quarry) will be used to construct and maintain the underground mine roads. Total roadbed thickness throughout the mine is to be 400 mm, consisting of a base depth aggregate of 325 mm (0-100 mm, Class II) and a top layer of 75 mm (0-31.5 mm, Class I. Placement and maintenance of material will be by wheel loader or LHD.

16.4.4 BLASTING

The production holes for stope blasting will be loaded with bulk emulsion explosives. The mine equipment fleet will include two (2) mobile emulsion chargers. Each hole will be primed with a booster and initiated with a detonator, which can be electronic or a Nonel. The design has been optimized from the PFS by adjusting the number of holes, spacing and burden to achieve an appropriate powder factor. The lower ore sill will also be slashed on both sides of the breakthrough area of the slot raise of approximately 3 m in depth. Slashing the walls of the lower drive near the slot will provide a greater initial void to assist with the first blast. Figure 16.8 illustrates the lower ore sill drop raise set-up.

Figure 16.8 – Lower Ore Sill Drop Raise & Wall Slashes



Source: WSP, 2025

16.4.5 MUCKING

After each production blast, an LHD with a 15-tonne tramming capacity will muck the broken ore from the lower ore sill of the stope to a nearby remuck bay. A portion of the material may be extracted with the operator seated in the LHD while the brow is closed. However, most of it must be mucked by radio remote control, with the operator controlling the LHD from a safe operating position in the ore sill.

After filling its bucket, the LHD will tram the material out of the stoping area through the ore sill to the crosscut remuck. Once a haul truck has arrived at the truck turn around, the LHD can commence loading the truck perpendicularly until it's fully loaded, if the truck is not there the ore can be temporarily stockpiled in one of the unused ore sills or the remuck. A truck turn around was designed to reduce truck tram time as well as to increase efficiency of the LHD and truck such that they will not have to perform multiple repositioning's to dump the ore into the truck. After discharging the load, the LHD will return to the stope to extract another bucket.

LHDs can also be used for various tasks in occasional instances but in low utilization, such as placing CRF or rockfill, cleaning of sumps, removing cuttings from raise development or service holes as required.

16.4.6 HAULAGE

The 45-tonne underground mine haul trucks will haul the material to the surface ore stockpile via the spiral ramp and the upper decline. Production tonnes from all sublevels of the mine will be trammed using the upper decline. The lower decline will be utilised for truck re-entry once their load has been dumped on surface. What should be noted is that in additional to truck re-entry, the lower decline could also be used as a haulage route if necessary. Subsequently, a surface wheel loader will rehandle the material, transferring it from the surface stockpile to the chute of the ROM bin, which feeds the primary crusher. After the trucks dump their payload at surface, they will continue their route following the mine haul road and re-enter the mine via the lower decline. The proposed travel route is to assist with reducing ramp congestion.

16.4.7 BACKFILL

Backfill is required to maximise mineral extraction and maintain stope stability. Three (3) types of backfill will be used at Čoka Rakita: unconsolidated waste (rock fill); cemented rock fill (CRF); and cemented paste fill (CPF).

Cemented Rock Fill – Development waste material is used for rock fill and CRF. Transportation of the rock fill from development areas to the crosscuts will be made by trucks that will dump it in remuck bays dimensioned for that effect. A surface plant will produce the cement slurry in batches, which is transferred underground via transmixer. The slurry is then mixed with waste rock in a CRF mixing bay and placed in the stope by an LHD. The CRF mixing bays are intended to be designed

by the short-term planning team to make use of dead ends and unused drifts as well as making allowance for the equipment to be near empty stopes.

Cemented Paste Backfill – A paste backfill plant will be built on surface and will be using the available tailings mixed with a cement binder. The cemented paste backfill will be delivered by a pipeline in the upper decline and distributed throughout the mine by an underground distribution system of boreholes and pipelines.

Both the paste fill and cemented rock fill require adequate time to cure and attain their target strength before an adjacent secondary stope can be mined.

Before production activities can commence in the upper sublevel of a stope backfilled with paste fill, a layer of rock fill must be spread over the paste fill to create a suitable surface for operating rubber-tired equipment.

Rock Fill – Rock fill and CRF will be used where the mining sequence allows it. Where possible, unconsolidated rock fill will be used to fill empty stopes, to avoid costs incurred by trucking waste rock to surface.

16.4.8 GROUND SUPPORT

Ground support for the underground mine has been separated into three (3) distinct categories based on heading type which are: Temporary Headings, Long Term Headings and Unique Infrastructure.

Temporary Headings are inclusive of: Sub-level ore drifts, crosscut inclines/declines (>50 m from ore body) and crosscut inclines/declines (<50 m of ore body).

Long Term Headings are inclusive of: Upper and lower declines, Spiral Ramp, Level accesses, remuck bays and Cuddies (General Bays).

Unique Infrastructure is inclusive of: Fuel Bay, Explosive storage, Mobile shop.

Refer to Section 16.2.5 for complete details Ground Support Details.

16.4.8.1 Shotcrete

Shotcrete will be used as secondary support as directed by the geomechanical engineer based on ground conditions. Shotcrete is to be brought down directly from the surface Batch Plant which will be used for building shotcrete barricades when backfilling stopes and building bulkheads as required.

Shotcrete will be used to support the ventilation raises as required and deemed appropriate by the geomechanical engineer.

Shotcrete will also be utilised for the initial 80 m of both declines to increase the structural integrity of the surrounding poor ground.

16.5 Mine Design

16.5.1 Stope Design

Preliminary stope shapes were created using Deswik Stope Optimizer Version 4 (DSO) software with respect to the geotechnical and economic criteria, the DSO parameters can be found in Section 15. The DSO shapes were then used as a basis for manual stope optimisation. The stopes were reshaped and visually checked against the geological block model (BM) for consistency.

General development to access the stopes and major infrastructure was subsequently designed and optimised.

The stope dimensions and design parameters were established through consultation with the WSP geotechnical and drill blast specialists and are summarised in Table 16.11. Stopes were then sequenced following the geotechnical recommendations and production activities were scheduled using Deswik software modules.

The stoping operation starts with floor cleaning, slashing the undercut drift and drilling of the drop raise and then the parallel rings. Blasting of the slot to create the void is next followed by the blasting of the production rings into the open slot. Mucking of the stope then ensues on the lower production ore sill and once that is complete, backfilling follows. If the stope is filled by paste backfill and or CRF then a shotcrete barricade will need installation prior to filling the stope. A cure time of 28 days for paste fill and 21 days for CRF should be respected before blasting adjacent or above the filled stope.

Table 16.11 – Mine Design Parameters – Stopes

Stope Design	Parameter
Mining Method	Sublevel Stoping
Unplanned Dilution (Backfill dilution)	6%
Mining Recovery	95%
Stope Dimensions	
Width (above RL440) (below RL440)	20 m 15 m
Height	20 m
Length	30 m

Stope Design	Parameter
Longhole Drilling	
Drilling Layout	Downhole Fans
Hole Diameter	89 mm
Holes per Ring	16 ea. (typical)
Metres Drilled per Ring	238m (typical)
Burden between Rings	2.5 m
Toe Spacing between Holes in Ring	2.5 m
Drilling Factor (includes slot)	10.72 t/m-drilled
Drop Raise	5 Reamed holes (203mm)
Blasting	
Explosive	Bulk Emulsion
Powder Factor	1.19 kg/m³
Backfill	
Stopes	Paste Fill
Selected Stopes	CRF and Rock Fill

Source: WSP/DPM, 2025

16.5.2 DEVELOPMENT DESIGN

16.5.2.1 Horizontal Development

There are two access points, consisting of an upper and a lower decline with an internal spiral ramp. It is assumed that a mining crew will develop both the upper and lower declines in parallel and until the spiral ramp has broken through utilizing their own equipment.

Both declines and the spiral ramp are driven at a maximum gradient of -15%. The mining area will be accessed by driving the crosscut from the level access and then sills along the ore.

It is assumed that a mining crew will complete all aforementioned lateral development as well as all auxiliary infrastructure excavations (development remucks, sumps, electrical stations, explosive storage area, main dewatering station, refuge station). The mining crew will also excavate a stub portion of other areas as it drives past so services or installs are not affected when the development returns.

The declines will have a 5.5 m x 6.0 m (width x height) arched cross-section with 2.25 m radius, rounded shoulders, minimising friction loss in the ventilation flow. The spiral ramp, level access headings, and crosscuts will be driven at 5.5 m x 5.5 m, providing adequate clearance for operating

the 45-tonne mine trucks. The ore sills will have a 4.5 m x 5.0 m profile, sufficient for the longhole drill rigs and LHDs, even with vent ducting and other services installed overhead.

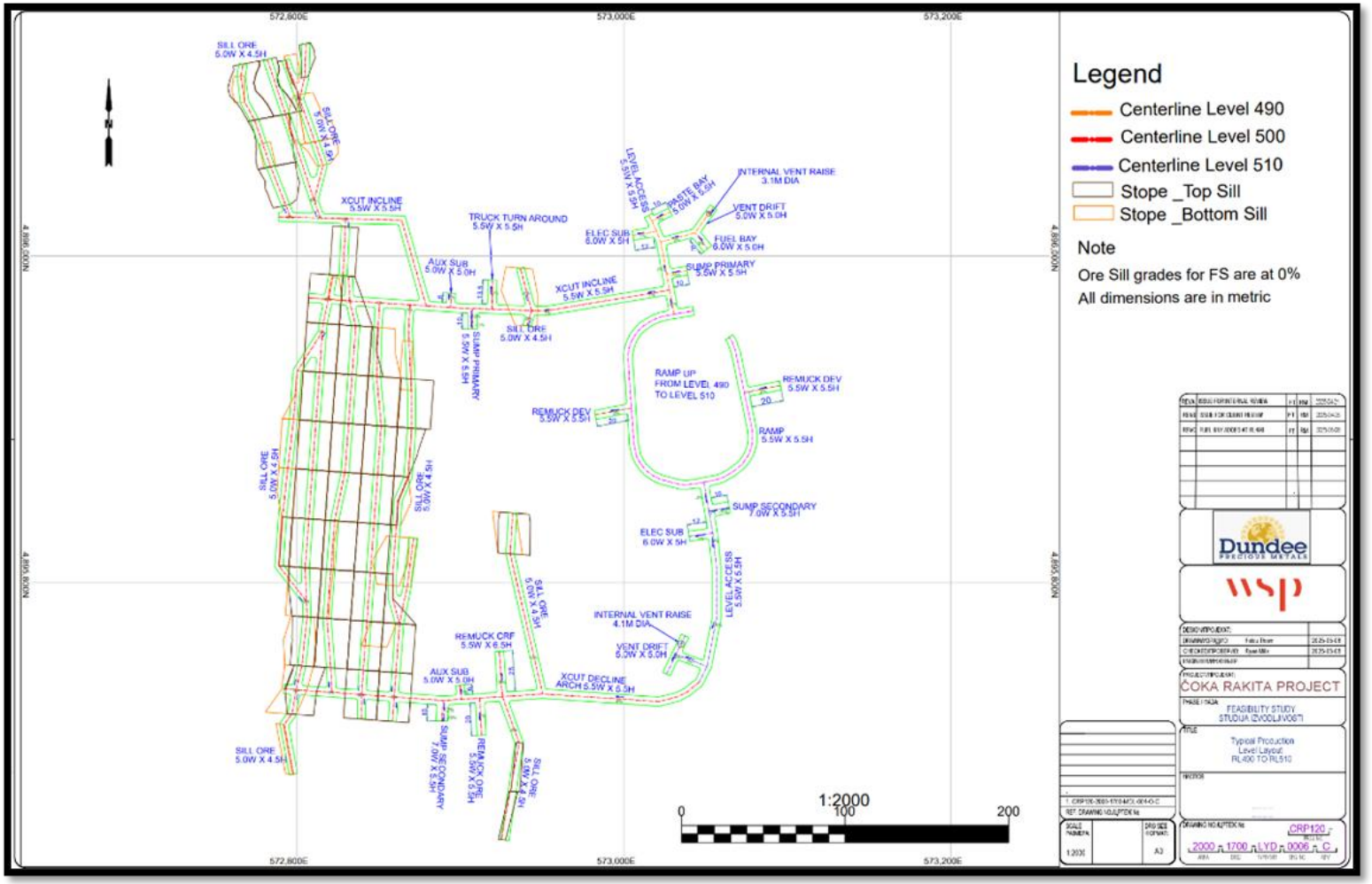
Services will be installed along the walls of most development, while ventilation will be hung from the back.

Roadbed is to be placed by service crews and will be placed in lifts such that the less coarse material can be installed and graded appropriately to allow for proper drainage and traction. The cross-section designs of the excavations also allow for ditching on both sides of the drift to promote natural drainage where appropriate.

The ground support requirements were established in collaboration with WSP's geotechnical specialists and presented in Section 16.2.5. It should be noted that additional support in terms of shotcrete is required for the initial 80 m of both declines to accommodate a thicker layer of shotcrete of 152 mm. The section below does not show this.

Figure 16.9 is a plan schematic showing a typical level. Each typical level consists of various auxiliary infrastructure such as electrical stations, paste backfill bays, sumps, truck turn arounds, remucks, ventilation accesses and ventilation raises.

Figure 16.9 – Typical Production Level Layout – RL490 – RL510



Source: WSP, 2025

16.5.2.2 Vertical Development

All vertical ventilation raise development is assumed to be done by contractor. WSP recommends that a vertical contractor be pursued.

Surface Ventilation Raise: The surface ventilation raise is to have a diameter of 4.1 m, drilled by surface raisebore and approximately 280 m deep. Geological guidance by the geomechanical group from WSP indicated that the first 30 m of raise is in poor condition and would need mitigation. The geomechanical group also indicated that the final 20 m of the surface ventilation raise was also situated in poor ground. To mitigate the concerns in the lower portion of the raise, the raise area was redesigned by reconfiguring the access drift from RL620 to RL640 which had limited impact to any additional development which reduces the risk of any failure. The raise can be accessed from the bottom to perform any maintenance should the raise slough at all. Secondary support of the raise is to be shotcrete only.

Northern Raises: The Northern internal raise will have a diameter of 3.1 m and be drilled by raisebore. The Northern raise connects the surface raise on RL640 to RL450 (bottom of the lower decline) which establishes the ventilation circuit and is approximately 190 m long. Only secondary support of shotcrete is to be used with a thickness of 0.05 m. It should be noted that DPM has agreed to the location of the Northern internal raise and has accepted the risk of any outcome because it intersects some faults. DPM has also opted out of using any primary support in the raise but will use shotcrete as secondary support in areas that are identified as poor.

The smaller Northern raises to the sublevels below RL450 also have a diameter of 3.1 m and are to have only secondary support in the form of 0.05 m thick shotcrete. No secondary egress has been allocated for these raises. The bottoms of each of the Northern raises are to have drop board regulators to allow cleaning of the raises should any sloughing occur during the LOM.

Southern Raises: The Southern raises were placed closer to the ore sills per the outcome of the ventilation trade off study (TOS) which aimed to reduce ducting. During the ToS process other options and evaluations were conducted which led to all the Southern raise sizes to be increased which assisted with reduced power consumption. The raises were increased from 3.1m in the PFS to 4.1 m in the FS. The increase in design diameter still allowed and accounted for a secondary egress in the form of a ladderway system to be installed. All raises are to be drilled by raise bore.

Garage Egress Raise: This raise is situated at the back end of the garage and is to be drilled by raise bore from RL640L. The diameter of the raise is to be 2.1 m in diameter and approximately 20 m long. The raise in this location will also act as a flow through for the garage and assists with any concern of potential fire as all smoke will flow through the raise and up out of the surface raise.

Main Dewatering Station Raise: This raise is required for flow through ventilation of the main dewatering station. The collar area is to be the pump room, and the breakthrough is situated in the clean sump area of the main dewatering station. The raise is to be drilled by raisebore, and the dimensions will be approximately 8 m in length with a 3.1 m diameter to allow for proper air flow and pump arrangement.

16.5.2.3 Service Boreholes

Service boreholes will be drilled typically by production drill, diamond drill or other. Service boreholes include:

Dewatering: Drill holes will be drilled (without casing) to promote natural drainage to reduce the number of pumps required. Additional dewatering holes will be required for pumping up to the main dewatering station from RL360 to RL450 which will be required to be cased.

Electrical and Communications: Drill holes are required to connect one main substation to another eliminating the need to be installed on the spiral ramp. Casing of the holes is not required.

Paste backfill: Drill holes are required to connect the paste backfill underground distribution system from one paste backfill bay to another. Each bay is to have receiving and distribution holes, and holes will be cased. There is to be one redundant capped line in each station.

Ventilation: Drill holes are required where excavations meet the criteria of containing mechanical equipment and that are longer than 6m. The holes will be used in lieu of auxiliary mechanical ventilation. Casing of the ventilation holes is not a requirement.

16.5.2.4 Development Rounds

A typical development heading is 5.5 m x 5.5 m, which applies to the spiral ramp, level access development, and crosscuts. The principle is the same for headings with different cross-sections, although the number of holes in the round will vary.

Table 16.12 presents the parameters for the drilling and blasting design of a 5.5 m x 5.5 m drift round. The design specifies drilling 45 mm diameter holes using 4,305 mm drifter rods. Three (3) central holes in the cut are reamed to 89 mm. The holes forming the arched back are loaded with cartridge type pre-splitting emulsion explosive, designed for perimeter control blasting to minimise rock fracturing. The remaining 45 mm holes are loaded with bulk emulsion explosives, and the three (3) reamed holes are left uncharged. Each loaded hole is primed with a booster and charged at the toe with a stick of detonator-sensitive emulsion explosive. The loaded holes are initiated with Nonel type LP detonators. The calculated advance per round is 3.8 m.

Table 16.12 – Drilling and Blasting Design for 5.5 m x 5.5 m Drift Round

5.5 m x 5.5 m Drift Round	Value
Drilling Supplies	
Button bit	45 mm
Reaming bit	89 mm
Drifter rod	4,305 mm
Drill Holes	
45 mm holes	50 ea.
Reamed holes	3 ea.
Hole depth	4,050 mm
Blasting	
Total loaded	50 holes
Bulk emulsion	38 holes
Roof perimeter – pre-splitting emulsion cartridge	12 holes
Toe charge – detonator-sensitive emulsion cartridge	50 holes
Booster 25g	50 holes
Nonel-type LP detonator 6.0 m	50 holes
Average break per round	3.8 m

Source: WSP, 2025

16.5.2.5 Mine Underground Facilities and Auxiliary Excavations

The underground mine design is also inclusive of various underground main infrastructure facilities and auxiliary infrastructure.

The underground shop area situated on RL620L is comprised of:

- Maintenance Service Aisle.
- Repair and Rebuild Service Aisle.
- Tire Change Bay.
- UG Shop Lube Bay.
- Waste Oil Collection Bay.
- Compressor and Receiver Bay.
- Warehouse and Storage.
- UG Shop Office.
- Latrine.

Located on RL640 just above the garage contains the following excavations:

- Washbay.
- Detonator Storage Bay.
- Explosive Storage Bay.
- Emulsion Storage Bay.

Other auxiliary infrastructure which are found throughout the mine and most levels consists of the:

- Main Electrical Stations.
- Auxiliary Substations.
- Electrical Cut-Out and Distribution Equipment (DBE).
- Fuel Bay.
- Refuge Station.
- Ramp Sumps.
- Level Sumps.
- Secondary Sumps.
- Transfer Station Pumps.
- Paste Backfill Bays.
- Main Dewatering Station.

Section 16.6 provides details about the underground mine design infrastructure.

16.6 Underground Mine Mobile Equipment

The Čoka Rakita Mine will be a mechanised operation utilising rubber-tired diesel equipment for all mining activities. Table 16.13 lists the mobile equipment planned for the mine, indicating the maximum number of units of each type required. Table 16.14 presents a schedule of annual equipment requirements over the LOM. The number of units needed per year for each equipment type is based on their estimated productivities and the development and production targets outlined in the LOMP.

The QP, Khalid Mounhir, is of the opinion that the number of units in the planned equipment fleet, as well as their types, makes, and models, are appropriate for the projected production rate, mining method, and development requirements at Čoka Rakita.

Table 16.13 – 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 long-holes up to 54m in depth	2
Production Drill, Top Hammer	Electro-hydraulic top hammer; capable for vertical and inclined fans; 102 m long-holes up to 38m in depth	1
Emulsion Charger	Required for production drills as required	3
Mucking and Haulage	Tunnels up to 65 m² cross section	2
Mine Truck 45 t Payload	45 tonne payload	6
LHD 15 t Tramming Capacity	15 tonne 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, 3,658 mm Basic	1
Service Machine - Wheel Loader	Equipped with quick coupler for forks and bucket	2
Utility vehicle c/w cassettes	Modular vehicle Boom Cassette Water Tank Cassette Fuel Tank Cassette	2 2 1 1
Mobile Rock Breaker	5,000 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, 2025

Table 16.14 – Mine Mobile Equipment – Units Required by Year

Equipment Type / Unit	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
Drilling and Blasting											
Development Jumbo	0	2	3	3	2	1	1	1	1	1	1
Production Drill, Top Hammer	0	1	2	2	2	2	2	2	2	2	2
Production Drill, Top Hammer	0	1	1	1	1	1	1	1	1	1	1
Emulsion Charger	0	1	1	2	2	1	1	1	1	1	1
Mucking and Haulage											
Mine Truck 45 t Payload	0	1	5	6	6	5	5	5	5	5	4
LHD 15 t Tramming Capacity	0	1	5	5	5	5	5	5	4	5	3

Ground Support											
Cable Bolter	0	1	1	1	1	1	1	1	1	1	1
Shotcrete Sprayer	0	1	1	1	1	1	1	1	1	1	1
Transmixer	0	2	2	2	2	2	2	2	2	2	2
Service											
Telehandler	0	1	1	1	1	1	1	1	1	1	1
Grader	0	1	1	1	1	1	1	1	1	1	1
Service Machine – Wheel Loader	0	2	2	2	2	2	2	2	2	2	2
Utility Vehicle	0	2	2	2	2	2	2	2	2	2	2
Mobile Rock Breaker	0	1	1	1	1	1	1	1	1	1	1
Diamond Drill Rigs	0	1	1	1	1	1	1	1	1	1	1
Pickup Truck	5	10	15	15	15	15	15	15	15	15	15
Mine Rescue Vehicle (Ambulance)	0	1	1	1	1	1	1	1	1	1	1

Source: DPM and WSP, 2025

16.7 Mine Portal 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 RL510. 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 underground facilities and the quantities at each level for the mine are listed in Table 16.15.

Table 16.15 – Underground Infrastructure List

Description	Upper Decline	Lower Decline	620 - 640	600 - 610	580 - 590	560 - 570	540 - 550	520 - 530	500 - 510	480 - 490	460 - 470	440 - 450	420 - 430	400 - 410	380 - 390	360 - 370
Portable Refuge Sation							1				1			1		
Escapeway			1	1			1		1		1	1	1	1	1	1
Fire Protection			1													
Level Sump	4	6	1	1	1	2	2	3	4	2	2	3	3	2		
Secondary Sump										1	2				2	1
Dewatering Transfer Station														1		
Main Dewatering Station												1				
Electrical Main Switchroom			3	1		1	1	1	1	1	1	3	1	1	1	1
Electrical Auxiliary Substation				1	1	2	2	2	2	2	2	2	2	2	2	2
Distribution Equipment (DBE)	2	3														
Temporary Electrical	1	2														
Dewatering Borehole					2		2	2	2	2	2			2		2
Electrical Borehole						2	2	2	2	2	2	4	2	2	2	2
Paste Backfill Borehole				2		4		4		2	2	4		4	4	
Communication Borehole						1	1	1	1	1	1	1	1	1	1	1
Fuel Bay										1						
Maintenance Bay (Maint Bay)			1													
Wash Bay			1													
Preventative Maintenance Bay (Prev Maint Bay)			1													

Description	Upper Decline	Lower Decline	620 - 640	600 - 610	580 - 590	560 - 570	540 - 550	520 - 530	500 - 510	480 - 490	460 - 470	440 - 450	420 - 430	400 - 410	380 - 390	360 - 370
Warehouse			1													
Office/Lunchroom			1													
Lube Bay			1													
Waste Oil Bay			1													
Tire Changing Bay			1													
Latrine			1													
Emulsion Storage Bay			1													
Explosive Storage Bay			1													
Detonator Storage Bay			1													
Paste Backfill Receiving Station				1		2		2		1	1	2		2	2	
Paste Backfill Distribution Station			3	1		2		1		1	2	2	1			

16.7.1 MINE DEWATERING SYSTEM

The mine dewatering system consists of a network of sumps that collect water generated from the mine activities, such as seepage, drilling, backfilling and directs them via gravity or pumps to a central main dewatering system. Sumps are located at each level with the main dewatering station placed at RL450. The pumps at the main dewatering system are sized for the peak flow scenarios with a holding capacity in the sump to achieve a minimum 8 hour hold up time in the event of an emergency, such as a site-wide power failure.

The dewatering system has been designed to maximize gravity draining via ditches and boreholes, in order to optimise energy consumption within the underground facility. Where gravity drainage is not possible, submersible pumps have been installed to transfer the water to either a sump where water can be diverted to the main sump by gravity or directly pumped to the transfer station or main sump for eventual transfer to the clarifier or CW pond via the lower decline.

16.7.1.1 Level and Ramp Sumps

Level sumps are located at each level, except for the lower two (2) levels (RL380 and RL360). Level sumps will either gravity drain via a borehole or ditches along the development, or are equipped with sump pumps, complete with piping and instruments to drain the sump to the main dewatering pump station. Each sump requiring a pump will be equipped with a monorail to facilitate maintenance of the pump.

Ramp sumps are typically situated in the level access drifts, with two identical sumps on either side of the drift. The secondary sumps act as collector sumps that the level sumps drain/feed into. They either gravity drain down to the next secondary sump below or are equipped with a sump pump in each sump.

The level and ramp sumps have a typical size of 5.5 m W x 5.0 m H x 10 m L to accommodate the water generated from development and production operations.

16.7.1.2 Transfer Station Pumps

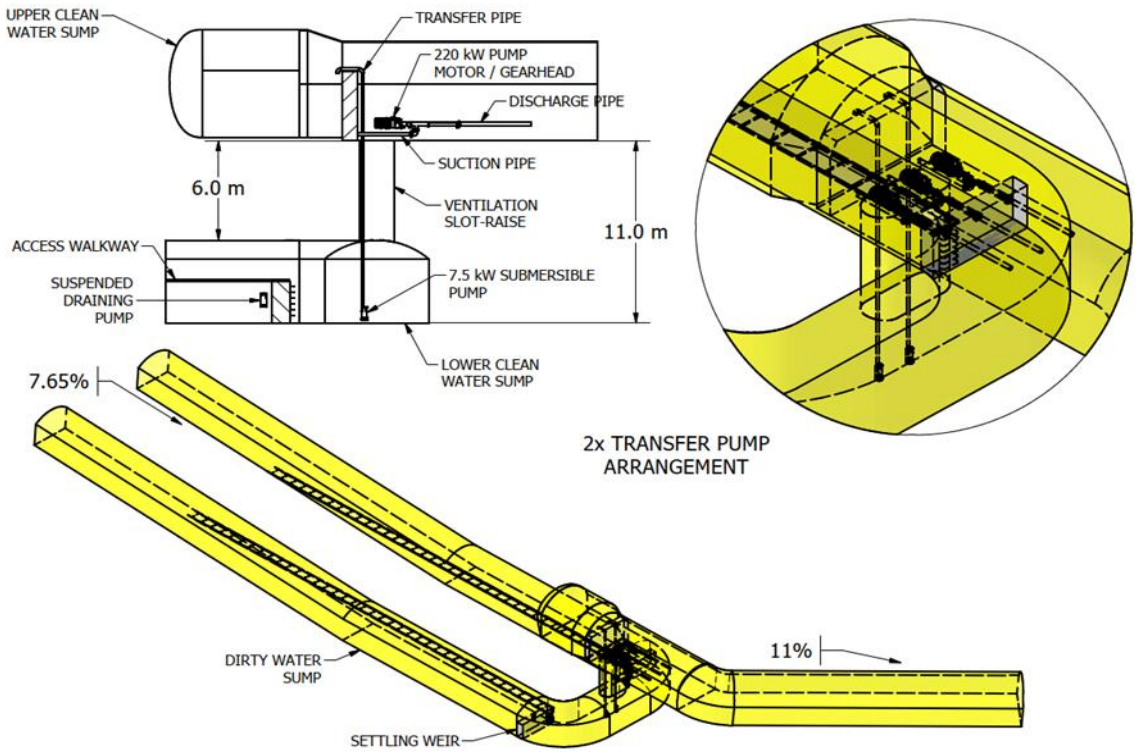
One (1) Transfer Station has been located at RL410, which is intended to act as an intermediary collection point for sumps below the main dewatering station and primarily in operation Year 3 onwards as per the mine production plan. The transfer station consists of two pump boxes with progressive cavity pumps that receive water from sumps below RL450. The pumps transfer water to the main dewatering station via hard pipe through boreholes and into the main dewatering sump.

Each pump box is sized to provide sufficient suction pressure for the pumps. The pumps are operated on level control in the tank. The system is design with 100% redundancy with one (1) pump box in operation with the second pump box as an installed spare.

16.7.1.3 Main Dewatering Station

The main dewatering sump consists of two dirty water sumps that is tied to a common clean water sump. A concrete settling weir separates the dirty section from the clean area. Two sets of submersible pumps transfer the water to an upper clean water sump. The upper clean water sump is intended to provide sufficient suction head for three (3) main dewatering progressive cavity pumps to transfer the water from underground to the contact water pond at surface. Figure 16.10 illustrates the configuration of the main dewatering station.

Figure 16.10 – Main Dewatering Station Configuration



Source: WSP, 2025

Water is pumped from the lower clean sump to the upper clean sump that acts to provide sufficient head in maintaining a continuous operation of the main dewatering pumps in the pump room and is intended to always operate full. Individual pump suctions piping/flanges in the concrete weir feed the main dewatering pumps. Based on the dewatering requirements, progressive cavity pumps have been selected to transfer the mine water via hard piping through boreholes placed on the lower decline to the surface near the process plant, from where it is piped to either the clarifier or the CW pond for further processing. The operating philosophy of the pumps is two (2) operating at 50% of design capacity with an installed spare. Should there be an emergency requirement, the third pump may be brought online for a total of 150% of design capacity.

16.7.1.4 Redundancy in the Dewatering System

The dewatering system is critical for the continuous and safe operation of the underground mine. As such there are several redundant features provided in the overall dewatering system to ensure dewatering facilities are always available.

- The capacity of the main dewatering sumps exceeds the 8-hour requirement as per Serbian regulation requirements.
- The main dewatering pumps, including the submersible pumps in the clean sump, are on emergency power from the generator at the process plant on surface. While level sump pumps will lose power, the intent is to ensure all gravity drained water can be pumped to avoid flooding sections that are critical.
- All permanent pumps within the dewatering system have an installed spare. For the main dewatering pump station, the pumping system is designed in a lead-lag setup to handle the various flow scenarios.
- For the main discharge line to the surface a redundant line has been put in place. This would allow operations to swap the line in the event of damage or plugging of the main line. The line can also be manually hooked up to one of the main dewatering pumps to add additional pump-out capacity via twinning to surface.

16.7.2 SERVICE SUPPORT INFRASTRUCTURE

16.7.2.1 Fuel Bay

A fuel bay has been located at RL490 close to the spiral ramp for ease of access for underground mobile fleet. 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 Paus utility vehicle with the fuel cassette. The fuel bay consists of one (1) SatStat with a 1,000 L capacity and pneumatically driven delivery system. The SatStat has an integrated containment system, sized for 150% of tank capacity, and sits below the fueling cube.

The entrance to the fuel bay consists of a Lattice wall with double equipment doors. A fire extinguisher, spill kit and portable eye wash station have been provided in the fuel bay.

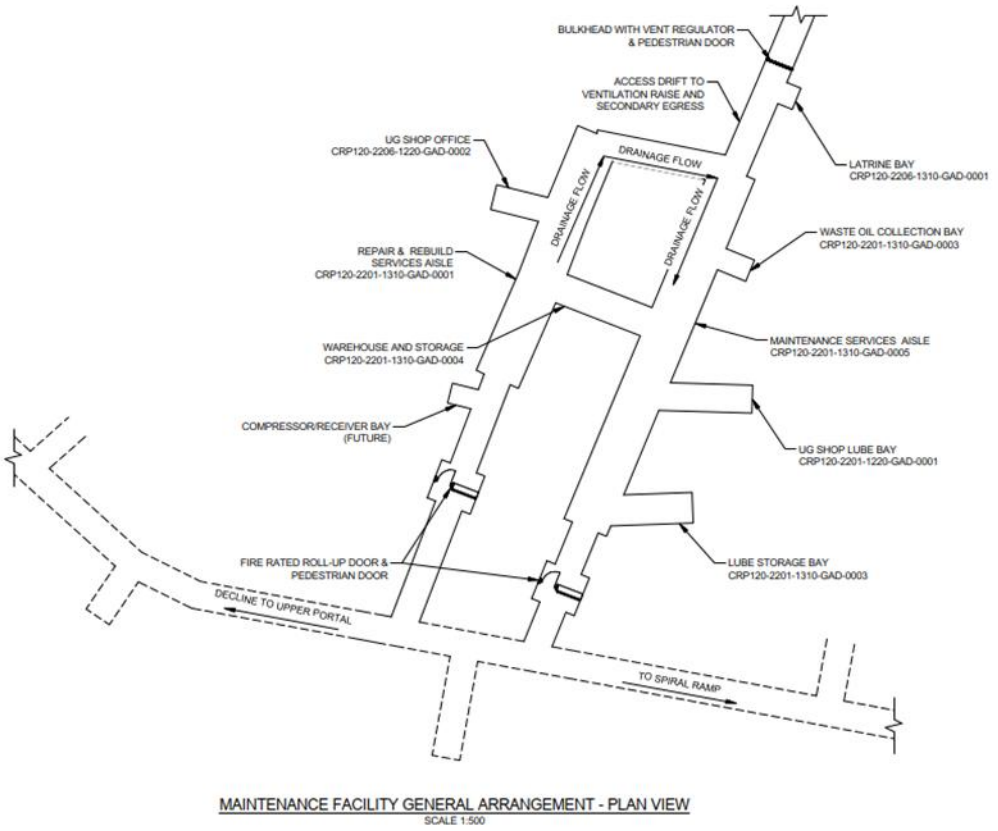
16.7.2.2 Maintenance Shop and Auxiliary Facilities

A maintenance facility is designed at RL620 for supporting underground development and mining activities, as depicted in Figure 16.11. The overall facility consists of the following:

- Maintenance aisle allowing for a single pass flow through the overall maintenance facility with one access from the upper portal decline and the second leading to the spiral ramp.
- Access drift to ventilation raise and secondary egress.

- Fire-rated roll-up doors and pedestrian doors on both entrances from the decline.
- Bulkhead with vent regulator and pedestrian door on the access drift to ventilation raise.
- Bays for the following:
  - Compressor and Receiver Bay.
  - Warehouse and Storage.
  - Repair and Rebuild Services Aisle.
  - UG Shop Office.
  - Waste Oil Collection Bay.
  - Maintenance Service Aisle.
  - UG Shop Lube Bay.
  - Lube Storage Bay.
  - Latrine Bay.

Figure 16.11 – RL620 Maintenance Shop and Auxiliary Facilities



Source: WSP, 2025

16.7.2.3 Maintenance Services Aisle

The maintenance bay is purposed for the maintenance of slower moving equipment and equipment that is intended to stay underground such as the production and development drills. The area is intended for routine maintenance. The dimensions of the maintenance bay are approximately 9 m W x 7.5 m H x 75.8 m L. There is sufficient room for repairs and disassembly lay down areas if required. The area is also fitted with a 10-tonne crane fitted with a motorized bridge, trolley and hoist. The crane services a total distance of 65.7 m along the Maintenance Services Bay. The Maintenance Service Bay is supported by bays dedicated for Lube Storage, Tire and Lube Services and Warehouse and Storage Bays. There is a compressed air and service water hose reel mounted on the wall along with an eyewash station, workbench and area to store toolboxes.

16.7.2.4 Repair and Rebuild Services Aisle

The Repair and Rebuild Services Aisle is meant to conduct emergency repairs or non-routine maintenance on the UG mobile fleet. The dimensions of the Repair and Rebuild Services Aisle are 9 m W x 7.5 m H x 43 m L. The available height under the crane is 5.14 m, which is sufficient for removal of components for the largest anticipated mobile equipment in the UG fleet. A 10-tonne crane, fitted with a motorized bridge, trolley and hoist and a span of 43 m can service the entire length of the aisle. The aisle is supported with bays dedicated for offices, tool crib and eating mezzanine, along with access to the Warehouse and Storage Bay. There is a compressed air and service water hose reel mounted on the wall along with an eyewash station, workbench and area to store toolboxes.

16.7.2.5 Warehouse and Storage

The Warehouse and Storage Bay can be accessed from both the Maintenance Service and Repair and Rebuild Aisle. Shelving and pallets storage are planned along both walls of the aisle with access in between for forklift and personnel traffic. The intent for this area is to store tires for underground mobile equipment. The warehouse dimensions are 6 m W x 6 m H x 18.2 m L. There is also room provided along the wall for installation of hanging hooks for lifting gear.

16.7.2.6 Office/Lunchroom

The UG Shop Office Area is located in the Repair and Rebuild Services Aisle with an overall dimension of 5 m W x 5 m H x 10 m L. The area has one (1) computer desk, a dining area for four (4) personnel and a counter complete with a microwave oven and storage space.

16.7.2.7 Lube Storage Bay

The Lube Storage Bay is situated in the Maintenance Services Bay. The dimensions of the bay are 5.5 m W x 5 m H x 15.75 m L. There are two (2) self-contained SatStats for storage and dispensing of motor, gear, antifreeze and hydraulic oil. The SatStats are self-contained with individual pneumatic dispensers and sit on a 200 mm concrete slab.

There is space allocated to store grease drums on the back side of the bay. The Lube Storage Bay is designed to accommodate a steel lattice swing door.

16.7.2.8 Waste Oil Bay

The waste oil bay is located in the Maintenance Services Bay. Due to Serbian regulations a total of 1,000 L of lube oil is the maximum that can be placed in a single underground storage area. Creating the stand-alone waste oil storage area allows the overall shop design to remain below 1,000 L, which is the maximum quantity that an individual storage area can contain lube oil. The dimensions of the waste oil bay are 4.5 m W x 5 m H x 6 m L. The excavation contains a self-contained waste

oil SatStat of 1,000 L complete with hose and reel attachment. The Waste Oil Bay is designed with a steel lattice swing door.

16.7.2.9 Latrine

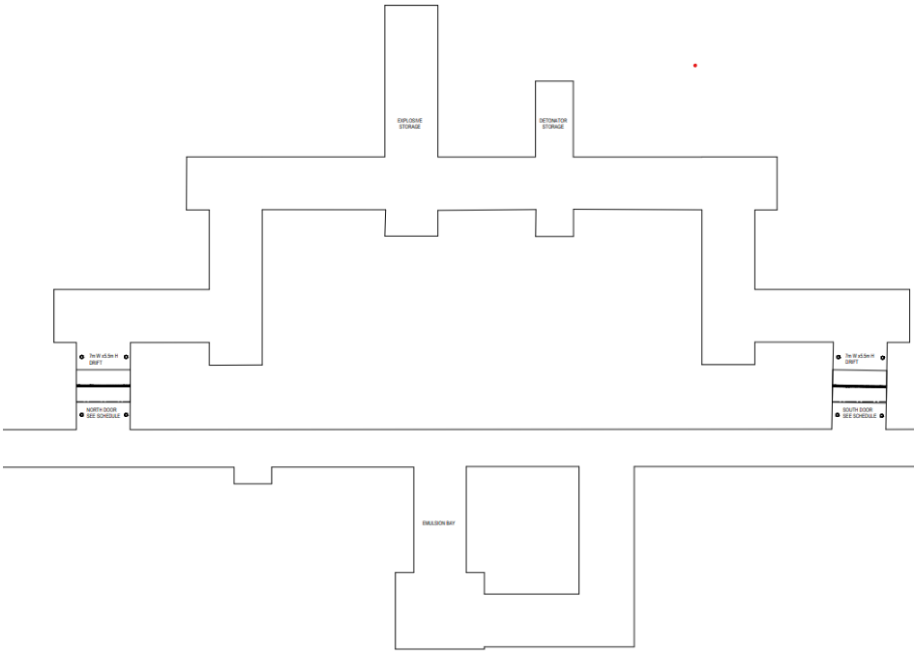
One (1) latrine is located along the Maintenance Services Aisle close to the bulkhead. The dimensions of the latrine area are 4.5 m W x 5 m H x 3 m L which can accommodate a single latrine and a handwash station for hygiene. The latrine is to be protected with a concrete barrier that will limit access from large mobile equipment in the aisle.

16.7.3 EXPLOSIVE MAGAZINES

The mine explosive magazines consist of three (3) differently labelled areas which are the emulsion storage bay, explosive storage bay and detonator storage bay situated on RL640 in the return air ventilation access drift. The explosive and detonator bays are designed to meet Serbian proximity regulations. Two (2) steel doors on both the north and south corridors provide means to secure access to the detonator and explosive bays.

Explosive materials are to be transported to site via the preferred vendor of DPM. There would 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 utility truck.

Figure 16.12 – Explosive Magazines Layout



Source, WSP 2025

16.7.3.1 *Explosive Storage Bay*

The explosive storage bay is to be a dedicated area for explosives that are not emulsion. The dimensions of the explosive storage bay are 7 m W x 5.5 m H x 20 m L. The area is sized to accommodate a utility vehicle with a boom attachment and telehandler which will transport explosives from surface to the underground explosive storage bay. There is a 3 m deep excavation across the storage area to take the initial shock of an air blast should an explosion occur. The entrance to the explosive storage bay will be of metal lattice with a double door which can be locked. A security camera is to be placed at the entrance for around the clock monitoring. A ventilation fan will supply air into the explosive storage area.

16.7.3.2 *Detonator Storage Bay*

The detonator storage bay will be a dedicated area for explosive detonators. The dimensions of the detonator storage bay are 5 m W x 5 m H x 10 m L. Access to the detonator storage area consist of metal lattice with a double door which can be locked and a concrete barricade to prevent vehicles from accessing the area and preventing any mobile incidents. A security camera would be placed outside the entrance for around the clock monitoring. Detonators will be placed on a shelf and off the floor of the excavation such that the detonators can be handled with care. There is a 3 m deep excavation across the storage area to take the initial shock of an air blast should an explosion occur.

16.7.3.3 *Emulsion Storage Bay*

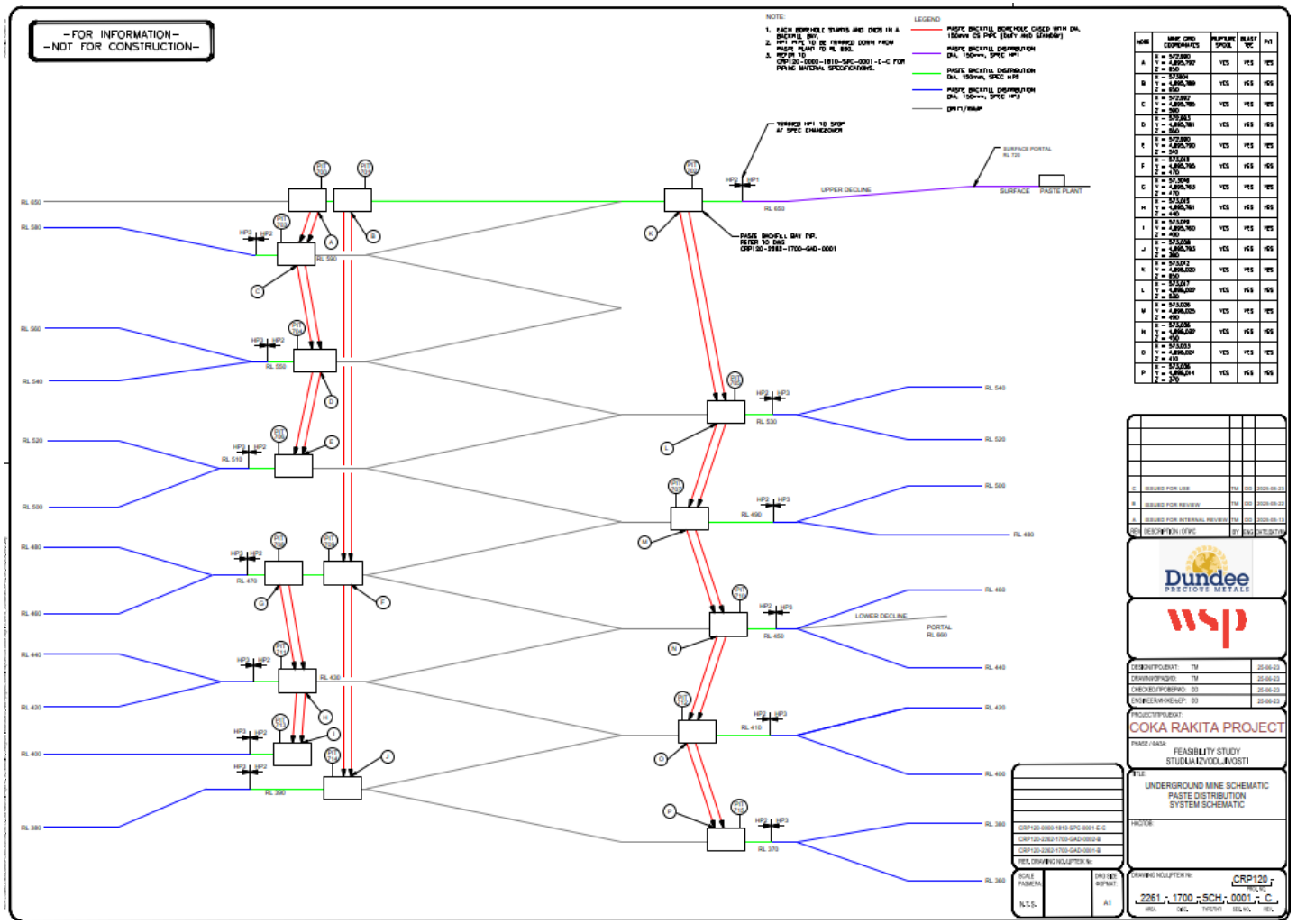
A third explosives storage chamber is required in addition to the detonator storage bay and the explosive storage bay which is the Emulsion storage bay. The emulsion storage bay is located along the return air access drift and not situated in the main explosives corridor. The general dimensions of the emulsion storage bay are 7 m W x 5.5 m H x 20 m L, but the width of the final 10 m will require additional wall slashing for enlargement which will allow for proper placement and storage of the emulsion vessels atop a containment bunk. The entrance to the explosive storage bay will be of metal lattice with a double door which can be locked and is sized appropriately to allow access for a mobile explosive charger. A security camera is placed at the entrance for around the clock monitoring.

16.7.4 Paste Backfill Underground Distribution System Infrastructure

The paste backfill underground distribution system (UDS) infrastructure consists of trunk delivery lines, backfill receiving and distribution bays, level piping, and stope crosscut piping. These components are designed to limit system pressures to acceptable levels and facilitate the safe operation of the system during any unanticipated or abnormal operating scenarios.

A schematic of the UDS is presented in Figure 16.13. There are no paste backfill lines located in any mine ramps and all piping in the UDS is 150 mm carbon steel and 150 mm HDPE in stope crosscuts.

Figure 16.13 – Paste Backfill UDS Schematic



Source: WSP, 2025

16.7.4.1 Trunk Delivery Line

A trunk line is defined as being the part of the UDS through which paste must travel to reach the mine regardless of stope being filled. The trunk line consists of the line exiting the paste plant, travelling down the upper decline, and boreholes and pipe components between distribution bays on the levels. The trunk line consists of 150 mm Schedules 120 and Schedule 80 pipe of API 5L Grade X52 or equivalent grade steel. The transition point from Schedule 120 to Schedule 80 pipe was selected to be where pressure losses reduce system pressures enough that risk of pipe rupture is less of a concern. The Schedule 120 pipe is used from the paste plant and in the upper part of the decline where the possibility of pipe rupture needs to be minimized since any closure of the decline to repair the pipe would have a great impact on operations.

The trunk line system from the paste plant down the upper decline to the RL650 level is twinned, being comprised of a duty and a standby pipeline, again to minimize the possibility of mine operation disruption should a paste line failure occur.

The UDS boreholes contain Schedule 80 pipe casings to provide a long operating life without excessive cost and account for possible poor host rock quality. The UDS boreholes are twinned to provide system redundancy and reduce the risk of extended downtime in the case of a system blockage resulting in loss of borehole functionality or other causes of borehole failure. Paste receiving and distribution bays are designed to allow for rapid manual change over from one borehole to the other with relatively little effort required.

The trunk line pipe is robustly supported to account for high system pressures and is braced to account for dynamic loads imparted by system flushing and other dynamic effects.

16.7.4.2 Level Piping

From the receiving and distribution bays, the distribution piping proceeds along the mine levels to the stope crosscuts. WSP has performed hydraulic flow modelling to determine the approximate locations where system pressures will be low enough to allow the transition from Schedule 80 to Schedule 40 pipe. This transition point determines where the system changes from “trunk” to “level” pipe per WSP nomenclature. The level pipe continues along the level to the stope crosscuts where manual change overs allow mine crews to configure the system to direct paste to the desired stope.

Level piping is typically supported in a similar manner to trunk line piping.

16.7.4.3 Stope Crosscut Piping

Level piping is directed to the stope access drifts (crosscuts) where system pressures become low enough to transition to stope piping. Typically, WSP approximates this as a maximum of 50 m from the discharge point of the system into the stope. Stope crosscut piping material is High Density

Polyethylene (HDPE) DR9 PE4710 pipe and can be reused for another stope after the current stope fill sequence is complete. Pipe lengths are re-used in this manner until wear from paste service or damage from dragging them along the mine floor make the pipe unusable, at which time it is discarded and replaced.

Stope crosscut pipe is supported by chains mounted to the back for easy installation and removal.

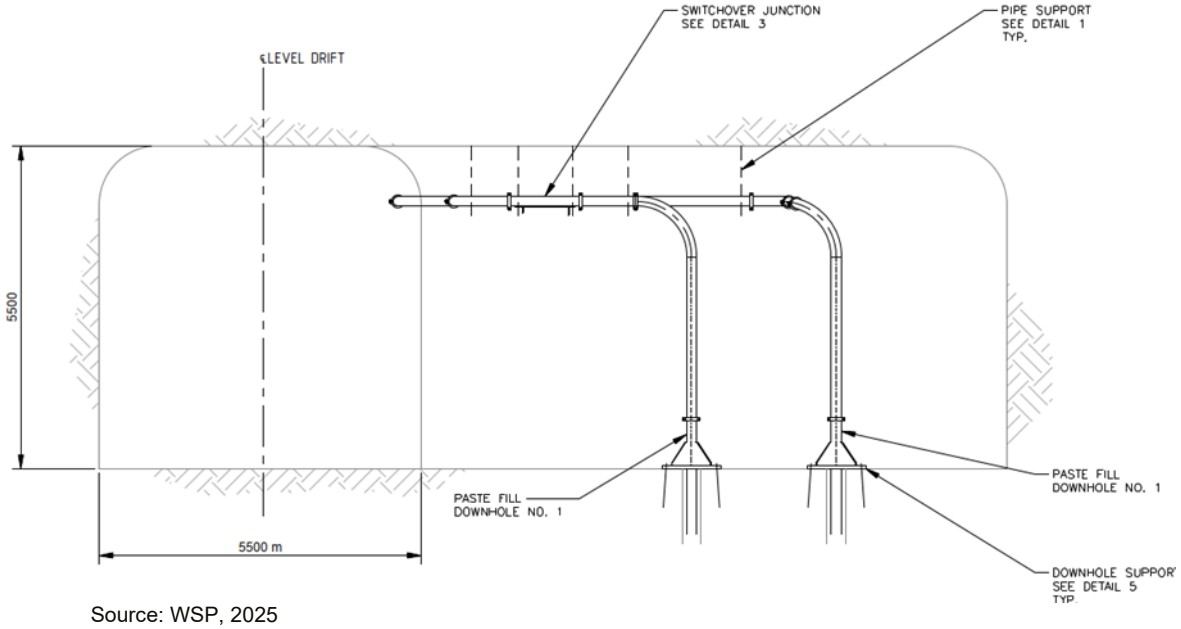
16.7.4.4 Receiving Bays

Paste receiving bays are where the trunk line enters boreholes. Receiving bays differ from transfer bays in that there aren't any incoming boreholes or outgoing pipeline from the bay. These receiving bays deliver paste to the mine's lower level receiving and distribution bays.

There are only three (3) receiving bays in the UDS and are located on the RL650 level. The bays are strategically located to supply the Eastern upper stopes, Eastern lower stopes, and the Western portion of the orebody. All receiving bays are equipped with pressure transducers which allow the system operators to monitor UDS pressures and diagnose blockage formation or pipeline ruptures.

A typical general arrangement cross section of a receiving bay is presented in Figure 16.14.

Figure 16.14 – Paste Backfill Receiving Bay



Source: WSP, 2025

16.7.4.5 Distribution Bays

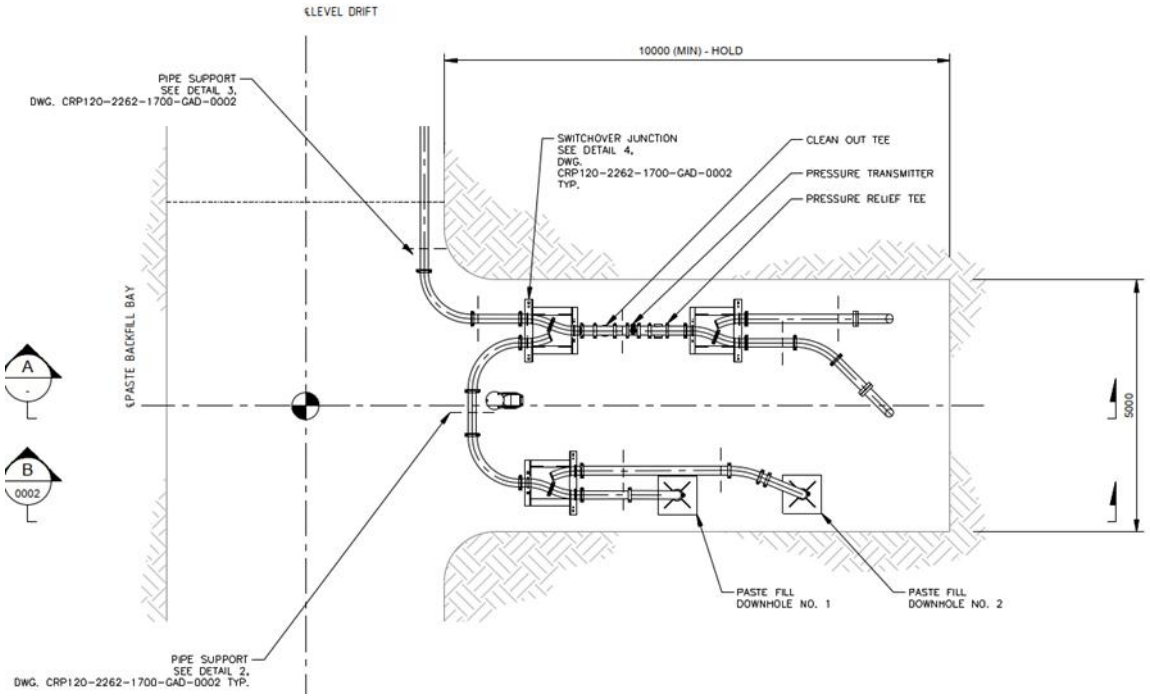
The remainder, and the majority of paste backfill bays in the mine are distribution bays Distribution bays, located on the mining levels, are strategically located such that the paste boreholes breakthrough onto the planned mining levels. Distribution bays are designed to utilize manually

operated pipe changeover modules for mine operators to direct the paste flow to the levels or to the next borehole in the system. Change over modules are designed for quick changeovers with minimal labour effort and downtime required.

All distribution bays are equipped with a camera (CCTV) and a pressure transducer for system monitoring, a rupture spool for system overpressure protection, and a clean out Tee to assist with clearing paste from a blocked system.

A typical general arrangement plan view of a distribution bay is presented in Figure 16.15.

Figure 16.15 – Paste Backfill Distribution Bay



Source: WSP 2025

16.7.5 UG Mine Utilities, Alarms and Communication

16.7.5.1 Compressed Air

The underground facility will include a compressed air network that will provide air for the following services:

- Maintenance facilities for servicing and repairs.
- Line blowouts, as required at each level.
- Air supply for UG Mobile equipment.
- Hard piped primary air supply to each portable refuge station.

- Air supply for operation of dewatering pumps.

The compressed air network originates at the Maintenance facility at the Compressor and Receiver Bay and is hard piped down the spiral ramp with branches at each production level. Two (2) compressor skids, complete with compressor and receiver, are provided to meet the air requirements.

16.7.5.2 Process Water

Clarified water from surface will be stored in water tanks close to the upper portal and hard piped down a borehole daylighting in the lower decline. This will be gravity fed for UG usage via hard piping and will provide water for development and production requirements, as well as meet firewater needs at the facility. A normally closed isolation valve separates the headers from the upper and lower declines and may be opened in the event of loss of water from the lower decline. A series of Pressure Reducing Valve (PRV) stations at each level maintain the pressure of water.

16.7.5.3 Firewater

Firewater branches off the process water network and supplies water to the following facilities in the event of an area fire:

- Emulsion, explosive and detonator storage areas.
- Maintenance facilities and Lube Storage Bay.

In the event of a firewater demand SV-700 on the upper decline process water header will actuate to close position, ensuring sufficient water is diverted to the affected area(s).

16.7.5.4 UG Fuel

One (1) fuel bay has been located at RL490 to service UG Mobile Fleet. The fuel bay's dimensions are 6 m W x 5.5 m H x 15 m L and is designed with a double equipment lattice wall door.

16.7.5.5 Alarm System (Stench Gas)

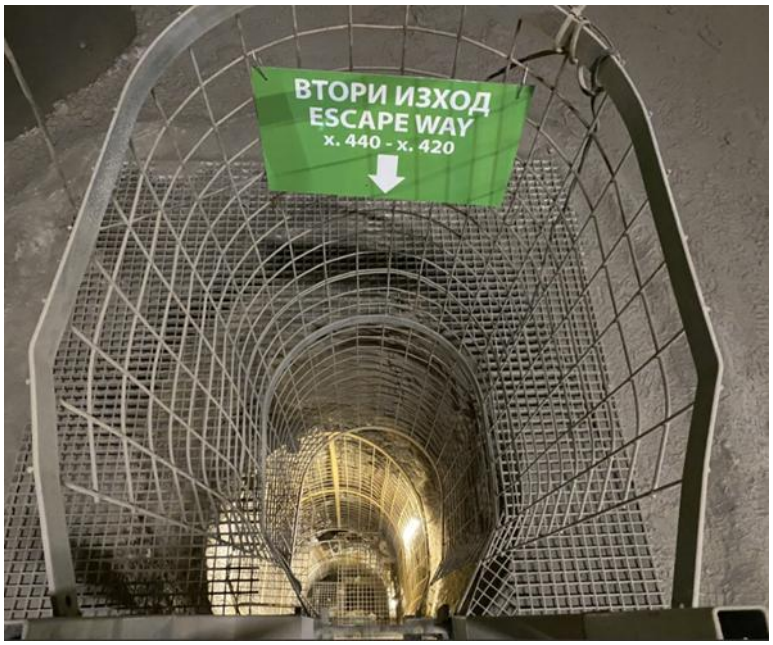
A stench gas alarm system is provided to alarm UG personnel of an event that requires stoppage of work and/or evacuation. The stench gas will be delivered via the ventilation system through both upper and lower declines.

16.8 Mine Safety Infrastructure

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

The ventilation return air raises connecting the sublevels and at the back end of the maintenance bay will be equipped with ladderways and lighting, providing an emergency escape alternative to the spiral ramp. No cables or other piping will be installed in this raise.

Figure 16.16 – Internal Escapeway



Source: DPM, 2024

Each decline will act as means of secondary egress from the mine if one of the declines cannot be used.

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 refuge stations chambers will be positioned along the spiral ramp at levels RL550, RL470 and RL410, each designed and equipped to accommodate up to eight (8) people for 36 hours. The quantity of three (3) portable refuge stations was determined by using Serbian mine regulation which state that a 600 m to 700 m separation range shall be used.

The upper and lower declines do not require a portable refuge chamber because the self-rescuers have a 30 min capacity which would give mine personnel sufficient time to get to surface or to the next nearby portable refuge.

16.9 Mine Personnel

Table 16.16 presents the annual staffing requirements for the underground mine for Year -1, Year -2 and Year 1 as well as a summary of the total mining personnel at peak for comparison. This roster includes provisions for absences, vacations, and temporary vacancies. The underground mine portion only (Mining, Technical Services) will reach its peak workforce of 239 employees in Year 1 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. According to the mine plan, all development and production activities will be carried out by DPM employees, except raise boring.

The QP is of the opinion that the proposed personnel structure is appropriate for the requirements of an underground mining operation of the scale of Čoka Rakita.

For future resource evaluations, WSP recommends assessing the potential benefits of engaging a vertical development contractor, similarly how the lateral development contractor was able to provide an estimated resource number for the duration they are expected to be on site.

Table 16.16 – Personnel Requirements of Owner and Contractor at Year -2, Year-1 and Year 1*

Dept.	Job Position	Total # of Owner's Personnel	Y-2	Y-1	Y1	Dev. Contractor Personnel
Mining	TOTAL	239	25	152	239	103
Maintenance	Mech. Maint. Underground	57	4	30	57	21
Maintenance	Elec. & Automation Maint. Underground	16	1	10	16	12
Maintenance	Total Maintenance (UG Total)	73	5	40	73	33
GRAND TOTAL	**GRAND TOTAL (**UG - Mining/Tech Services and Maintenance)**	**312**	**30**	**192**	**312**	**136**

Notes:
* The mine personnel list is not official at the time this Report is written because it is being reviewed internally by DPM.
** Underground mine personnel list does not include administrative and surface personnel.

16.10 Pre-Production Schedule

The pre-production phase extends until the start 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 surface stockpile.
- Procurement of essential mining equipment, considering the lead time required for delivery from manufacturers.
- Recruitment and training of personnel for mine development.

Development in the preproduction phase will be done with a contractor and will start with excavating of the upper and lower decline portals with a scheduled advance rate of 3.0 m/day. After the excavation of both portals, the focus will shift to developing the upper and lower declines along with the Surface and North ventilation raises. Lateral headings advance with a 5.0 m/day rate.

The spiral ramp will continue to advance at 5.0 m/day simultaneously from the upper and lower decline along with establishing the south ventilation raise (egress raises) until breakthrough on the RL510.

Major underground infrastructure such as the garage, explosive and emulsion storage and the main dewatering station are all excavated during the pre-production period with their respective completion dates summarised in Table 16.17.

Table 16.17 – Main Underground Infrastructure Completion Dates

Item	Finish Date	
Surface Ventilation Raise	Start of Q4	Year -2
North Ventilation Raise	Middle of Q2	Year -1
Garage	Start of Q1	Year 1
Emulsion and Explosive Storage	End of Q1	Year -1
Main Dewatering Station	Start of Q1	Year 1
Ramp Breakthrough	Middle of Q4	Year -1

Towards the end of the second year (Year –1) and with the introduction of the mine crew, the focus will shift towards developing the RL440 sublevel and will consist of developing crosscuts and sills required for accessing the stoping, resulting in generating about 83 kt of ore from development and stoping. Key stoping activity dates are summarised in Table 16.18.

Table 16.18 – Stope Activity Start Dates

Activity	Start Date	
First Stope Drilling	End of Q3	Year -1
First Stope Mucking	Start of Q4	Year -1
First Stope CRF	Start of Q4	Year -1
First Stope Paste Backfill (RL460)	End of Q2	Year 1

Development in waste and ore will continue during Year 1 and stope production will ramp up during the year, accounting for 82% of the ore mined. The pre-production phase will conclude with the mine attaining commercial production (75% of planned capacity) in Q1.

16.11 Life of Mine Plan

16.11.1 Production Plan

As indicated in Table 16.19, the LOM production plan extends over nine (9) years, during which the mine will produce ore at full capacity for seven (7) years. Table 16.20 provides a breakdown of the LOM production by level. Figure 16.17 illustrates the locations of the levels. The mine plan is based on probable reserves and incorporate a Backfill dilution factor of 6% and a mining recovery of 95%.

During the first year, and most of the second year of the pre-production period (Year -2 and Year-1), the mine will focus solely on waste development. Toward the end of Year -1, ore production will commence, targeting high-grade stopes from the 440L sublevel in line with DPM's strategic plan.

In Year 1, ore production will ramp up with 75% of the planned capacity - beginning at Q1. Throughout the year, ore output will average approximately 80% of the mine's full capacity, at around 684 kilo tonnes (kt). Of this, approximately 28% will originate from development headings, while the remainder will come from stope production.

From Year 2 through Year 8, the mine will operate at its full planned ore production capacity of 850 kt per year. Ore will continue to be sourced from a combination of development and stope production. Starting in Year 5, production will gradually shift to include lower horizon levels, beginning at RL360 and progressing upward to RL420, which represents the sill pillar and will be the final sublevel to be mined.

In Year 9, the final year of the LOM plan, production will taper off slightly, maintaining a level of around 67% of full capacity, as mining operations begin to wind down.

Table 16.19 – LOM Production Plan

Description	Unit	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Total / Average
Development Ore	tonnes	27,611	195,728	83,471	37,967	114,620	131,159	63,217	10,288	12,042	17,307	693,410
Grade	g/t	8.40	8.25	6.16	4.16	4.12	3.38	2.51	4.21	2.70	3.18	5.37
Contained Gold	oz	7,456	51,899	16,532	5,083	15,170	14,234	5,109	1,393	1,047	1,767	119,690
Stope Ore	tonnes	56,001	502,618	771,527	817,033	740,308	723,841	791,783	844,712	842,958	560,579	6 651,359
Grade	g/t	11.42	9.60	11.21	9.14	6.43	6.08	4.25	3.82	4.82	3.86	6.55
Contained Gold	oz	20,552	155,138	277,997	240,036	153,160	141,384	108,082	103,739	130,736	69,566	1,400,390
Total Ore Mined	**tonnes**	**83,612**	**698,345**	**854,997**	**855,000**	**854,928**	**855,000**	**855,000**	**855,000**	**855,000**	**577,886**	**7,344,769**
Grade	g/t	10.42	9.22	10.71	8.92	6.12	5.66	4.12	3.82	4.79	3.84	6.44
Contained Gold	oz	28,008	207,036	294,529	245,120	168,330	155,618	113,192	105,131	131,782	71,333	1,520,080

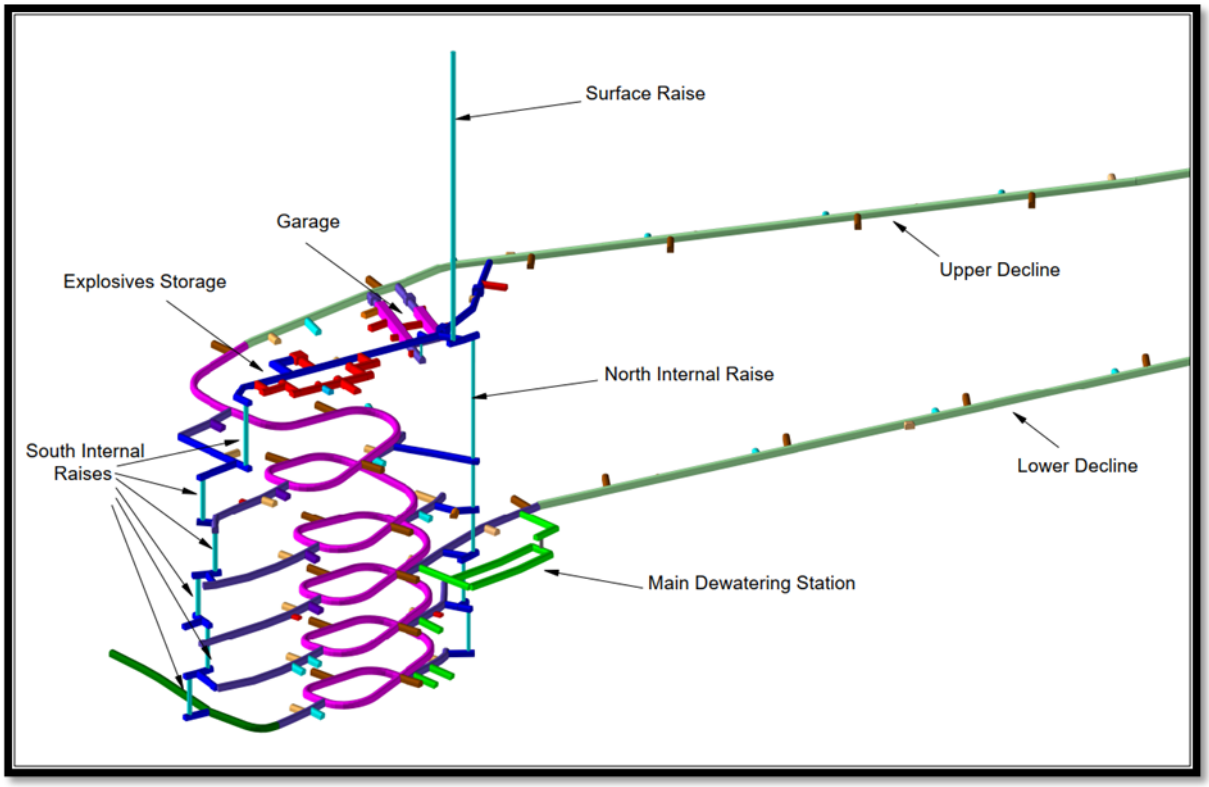
Source: WSP, 2025

Table 16.20 – LOM Mined Ore Tonnes by Level

Description	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Totals (t)
Level 360	-	-	-	2,101		27,054	1,797	32,329	52,042	128,797	244,121
Level 380	-	-	-	-	9,882	17,745	13,177	41,913	31,521	38,250	152,490
Level 400	-	-	-	-	-	30,118	46,051	61,691	33,245	33,955	205,060
Level 420	-	-	-	-	20,818	39,016	44,286	24,724	181,046	185,655	495,544
Level 440	62,673	293,752	248,936	-	101,906	186,611	193,318	203,694	149,539	34,441	1,474,870
Level 460	20,940	156,827	382,245	340,341	175,788	63,317	14,902	40,484	268,896	112,615	1,576,355
Level 480	-	176,555	149,046	419,034	253,172	109,102	-	34,804	83,570	44,173	1,269,457
Level 500	-	65,484	59,990	72,999	147,712	171,300	286,233	80,855	-	-	884,573
Level 520	-	5,727	14,780	11,000	105,062	138,250	141,006	168,286	-	-	584,112
Level 540	-	-	-	9,524	28,219	27,854	20,371	116,254	36,009	-	238,231
Level 560	-	-	-	-	12,369	16,522	45,733	33,798	17,285	-	125,707
Level 580	-	-	-	-	-	14,388	28,546	11,661	1,847	-	56,442
Level 600	-	-	-	-	-	13,722	19,581	4,506	-	-	37,808
Total Mined Ore Tonnes	**83,612**	**698,345**	**854,997**	**855,000**	**854,928**	**855,000**	**855,000**	**855,000**	**855,000**	**577,886**	**7,344,769**

Source: WSP, 2025

Figure 16.17 – West Isometric View of the Mine Levels



Source: WSP, 2025

16.11.2 DEVELOPMENT PROCEDURE

Mining at Čoka Rakita is planned to be developed with two (2) separate mining crews. One crew for the development of the lower decline and another crew for the development of the upper decline. The mining crews will use conventional drilling and blasting techniques to excavate all lateral development inclusive of the declines, spiral ramp, level accesses, crosscuts, auxiliary infrastructure, garage area, main dewatering station and explosive storage areas. The mining crews will start developing at the same time, in parallel and will demob at the start of Year 1. Development jumbos will drill the drift rounds. Once a round is blasted, an LHD will muck the broken rock, tram it away from the face, and dump it at the nearest remuck bay. An LHD will later rehandle the muck stockpiled in the remuck bay, loading it onto mine trucks for haulage to surface. However, development waste may also be used as rock fill for backfilling secondary stopes.

DPM plans on using the same 2-boom jumbos that drill the development rounds to scale and install ground support (Split Sets and welded wire mesh). This multi-functional use of the jumbos is practiced at DPM's Chelopech Mine and is a common approach at underground mines in Australia and Asia. Using a jumbo for drilling, scaling, and installing ground support can reduce non-

productive time in the drift-round cycle. Instead of three (3) pieces of equipment requiring time in the cycle for travelling between headings, moving in, setting up, tearing down, and moving out, this non-productive time is limited to just one machine.

In areas of poor ground, grouted Split Sets will be utilized and a separate grouting crew will grout the hollow cores of the ground support. In areas of intersections or larger spans cable bolts will be installed and grouted via the cable bolter rig.

Most activities in Y-2 to Y-1 will be by specific mining crews. Once the owners crew starts, the same methodology will apply as above. All material movement will be by telehandler, wheel loaders or boom trucks.

16.11.3 DEVELOPMENT PLAN

Table 16.21 presents the LOM development plan for Čoka Rakita. Lateral mine development is expected to be most intensive during the early years of operation, gradually decreasing over time. In the initial four years of the LOM, which include two years of pre-production and two years of production ramp-up, horizontal advance is anticipated to average approximately 4,970 meters per year. For the subsequent four-year period, the rate of development is projected to decrease to an average of 2,290 meters per year. From Years 7 to 9, the development rate is expected to further decline, averaging around 330 meters annually.

Most major infrastructure will be completed during the pre-production period (refer to Section 16.7), and capital development will conclude by Year 3. Most of the operating development is expected to be completed by Year 5, except for some sill meters in the lower mine sublevels. These will continue through to the end of the LOM due to sequencing constraints related to pillar issues between stopes and sills.

Table 16.21 – LOM Development Plan

Development Metres by Excavation Type	Y-2	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Total
HORIZONTAL DEVELOPMENT CAPEX:												
Decline Upper	836											**836**
Decline Lower	1,379											**1,379**
Ramp	132	1,247		446								**1,824**
Level Access Drift	244	673	63	212								**1,192**
Ventilation Drift	427	436		54	24							**941**
Ventilation Door	16	-										**16**
Emulsion Bay Vent Drift		43										**43**
DEV REMUCK												
Remuck CRF	281	180	-	60	-	-						**521**
GARAGE DEVELOPMENT												
Compressor Bay		8	-									**8**
Tire Change Bay		17	-									**17**
Office			10									**10**
Latrine			3									**3**
Lube Bay		15										**15**
Preventative Maintenance Bay		76										**76**
Waste Oil		6										**6**
Wash Bay		20										**20**
Warehouse		-	18									**18**
Maintenance Bay		6	46									**52**
ELECTRICAL DEVELOPMENT												
Main Electrical Station	60	92		36								**189**
Temporary Substation	14											**14**
Auxiliary Substation	2	14										**17**
Ground Fault Panel	9	6										**15**
DEV EXPLOSIVE STORAGE												
Detonator Storage		10										**10**
Emulsion Bay		7										**7**
Emulsion Bay (Over-excavation)		14										**14**
Explosive Storage Bay	59	154										**213**

Development Metres by Excavation Type	Y-2	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Total
SERVICE BAY DEVELOPMENT												
Fuel Bay		13										**13**
Paste Backfill Bay	10	64										**74**
Refuge Station		14		7								**21**
DEWATERING DEVELOPMENT												
Secondary Sump 6.0W x 5.0W				61								**61**
Secondary Sump 7.0W x 5.5W	10	41										**52**
Primary Sump	124	63		11								**199**
Main Dewatering Station		119	20									**139**
Settling Sump		158	37									**195**
Total Horizontal Capex Development	3,604	3,496	197	886	24							**8,207**
HORIZONTAL OPEX DEVELOPMENT:												
Remuck Ore		26	77	127	77	25						**331**
Remuck Dev		30	96	141	88	75						**430**
Truck Turn Around Bay		15	45	45	15							**121**
Paste Backfill Bay		27	69	20	20							**136**
Secondary Sump 7.0W x 5.5W			31	44	40							**115**
Primary Sump			32	32	42	10						**117**
Auxiliary Substation			43	49	36	6						**135**
Sill Ore		781	3,488	2,274	1,032	2,381	2,469	1,320	255	264	480	**14,743**
Crosscut Incline		411	799	679	616	300	111					**2,917**
Crosscut Decline		464	1,075	777	352	65	48					**2,782**
Total Horizontal Opex Development		1,753	5,755	4,189	2,318	2,863	2,628	1,320	255	264	480	**21,826**
Total Horizontal Development	**3,604**	**5,249**	**5,951**	**5,075**	**2,342**	**2,863**	**2,628**	**1,320**	**255**	**264**	**480**	30,033
VERTICAL DEVELOPMENT												
SERVICE HOLE DEVELOPMENT												
Communications		375		108								**483**
Electrical		1,229		215								**1,444**
Dewatering				150								**150**
Paste Backfill		356	651	140	211							**1,358**
Ventilation	210	525	255	330	210	30						**1,560**

Development Metres by Excavation Type	Y-2	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Total
VENTILATION RAISE DEVELOPMENT												
Surface Ventilation Raise	258											**258**
Southern Ventilation Raise		181	11		32							**224**
Northern Ventilation Raise		189		70								**259**
Garage Ventilation Raise			16									**16**
Vertical Capex Development	**468**	**2,855**	**933**	**1,013**	**453**	**30**						**5,752**
Total Development	**4,072**	**8,104**	**6,884**	**6,088**	**2,795**	**2,893**	**2,628**	**1,320**	**255**	**264**	**480**	**35,785**

16.12 Electrical Design

Power for the underground mine loads will be supplied from the surface substation 6720-SUB-003 main 10 kV 6720-SWG-002 switchgear. This substation and associated distribution network steps down the site transmission voltage to the distribution voltage of 10 kV and delivers it to the upper decline via the upper portal and to the lower decline via surface borehole.

- One 10kV feeder cable will be routed down from a dedicated breaker on busbar A on 6720-SWG-002 to the Upper Decline Substation 2232-BD-0700 via appropriately rated MV junction boxes housed in electrical cutouts.
- The second 10kV feeder cable will be routed down from a second dedicated breaker on 6720-SWG-002 switchgear on busbar B via a surface borehole to a remuck bay near Lower Decline Substation 2232-BD-0708.

These two (2) mine power feeders will provide a redundant connection to ensure reliability. These power feeders will connect to UG main electrical substations through a ring (radial) system topology. These two feeds will be indirectly connected to an emergency generator on the surface. Further development and consideration are required for coordination and control between operation of critical and non-critical loads in the mine when emergency power is required.

There is a total of fifteen (15) main UG substations situated in various areas in the mine to service development, production, dewatering and maintenance areas. Eleven (11) main substations for production, one (1) for upper decline, one (1) for lower decline, one (1) substation for Maintenance area, one (1) for main pumping station area. The 10 kV ring cable from the various main UG electrical substation will terminate at the incoming feed-through terminals and further routed through ramps/drifts or through strategically located boreholes to the next main UG electrical substation. If a failure occurs at any point in the network, power can be redirected through the ring main, restoring power to other systems. This is especially important for critical ventilation and dewatering loads.

A dedicated main earthing grid cable will run along side the main power 10kV ring cable to ensure earthing continuity is maintained through out the mine. All major electrical and mechanical equipment including but not limited to electrical room earth buses, power transformers, Low Voltage (LV) distribution skids, RMUs, standalone Variable Speed Drives (VSDs) or Starter, motor frames, equipment enclosures, cranes, tanks, bins, fences, pump housing, ladderways, metal supports, skid and various other non-current carrying steel structures shall be directly or indirectly connected to this earthing grid. The mine central earth loop will be extended and bonded directly to the surface main earthing grid at two locations inside the respective RMUs in the upper and lower declines

16.12.1 Main Underground Substation

Each main UG electrical substation will be constructed with a concrete floor, fire rated wall, chain-link access doors, raised pads, lighting, cable trays, and Earthing bar. The roof and walls of the

electrical chamber will be shotcrete and tested for water tightness. Electrical equipment will be mounted on rack structures constructed using steel channels. A chain link fence access door to be sized to allow for movement of largest piece of equipment. The chamber will have free air flow ventilation without any fan or air condition (AC) unit. All equipment will be placed with sufficient clearance to allow for ease and safe transportation, handling, operation and maintenance. Ancillary loads will be serviced from a local skid-mounted 30kVA 400/230V step down transformer and associated lighting panel board. Low voltage critical loads (including communication and process control cabinets) to be serviced from local 10 kVA Uninterruptible Power Supply (UPS). Two (2) dry chemical, Class C fire extinguishers along with Fire Alarm Panels will be installed at each location.

All Fifteen (15) main UG substations will house either an extensible or non extensible skid-mounted 10 kV Ring Main Unit (RMU) switchgear to establish a ring power system topology. Thirteen (13) units will be non-extensible and will include two (2) motor operated load break switch with short bushing and one (1) motor operated circuit breaker with 51/50N/51N protection all with MODBUS TCP/IP communication capabilities. Three (3) units will be designated as extensible with each skid mounted unit comprising of two (2) extensible RMUs which will include two (2) motor operated load break switch with short bushing and one (1) motor operated circuit breaker with 51/50N/51N protection all with MODBUS TCP/IP communication capabilities.

All Fifteen (15) main UG substations will house step down transformers, thirteen (13) will be 1000kVA 10/1 kV and two (2) main UG substation 630 kVA 10/1 kV IT-unearthed dry type transformers with necessary load break switch.

Fourteen (14) main UG substations for production, declines and maintenance will house skid mounted 630A, 1000V main distribution boards each with plug in motorized main incomer circuit breaker and an LSI trip unit as well as shunt trip, close coil, digital meter and MODBUS TCP/IP communication capabilities. Feeder circuit quantities and size will vary for each unit. All feeder circuits will be fixed circuit breakers with LI trip capabilities.

The main UG substation for the 450L Pumping Station will include a 630A, 1000V rated main distribution board with dual feed plug-in motorized main incomer breakers. Incomer breakers will include trip units as well as shunt trip, close coil, digital meter and MODBUS TCP/IP communication capabilities. Both breakers will be mechanically and electrically interlocked to ensure only one feed is servicing the pump station. Normal power will be supplied from 2231-SWG-709 RMU on 450L. The dedicated 1kV emergency feed to be supplied from the lower decline. In addition, the main UG substation for the 450L Pumping Station will house three (3) 315kVA 1000/690V IT-unearthed dry type transformer feeding three (3) 220 kW VSD units for the main dewatering pumps.

16.12.2 UG Auxiliary Substation

Each production level main UG electrical substations typically services either two (2) or three (3) auxiliary electrical substations downstream. This design allows local 1000 V or 690 V power distribution to be used to further support in the development of infrastructure and mining activity.

Only the maintenance shop main UG electrical substation will service four (4) auxiliary electrical substations (maintenance auxiliary electrical substation #1, maintenance auxiliary electrical substation #2, explosive material drift auxiliary electrical substation and wash bay drift auxiliary electrical substation).

Each auxiliary electrical room will house a skid mounted 400A, 1000V Distribution Board with a plug-in main breaker, (with the exception of the explosive material drift auxiliary electrical substation and wash bay drift auxiliary electrical substation which will house 200A, 1000V distribution board with plug-in main breakers), LI trip unit and shunt trip as well as various feeder breakers starters, lighting transformers, panel boards, free standing, or wall mounted VSDs, communication panel, control panel, earthing electrodes, earthing bar, tray system. Electrical equipment will be mounted on rack structures constructed using steel channels. At least 20% of the space available will be set aside for future expansion. Ancillary loads will be serviced from a local skid-mounted 7.5kVA 400/230V step down transformer and associated lighting panel board. Two (2) dry chemical, Class C fire extinguishers shall be installed in every Electrical Rooms.

16.12.3 Surface Main Exhaust Fans Substation

Two (2) 500 kW 690V Surface main exhaust fans and associated ancillary loads will be supplied from two redundant sources. First feed will originate from 6750-SUB-003 main 10.5kV substation. The second source will be supplied from an offsite admin building switchgear unit.

Redundancy of equipment is integral to the design to system reliability for critical loads that make up the surface main exhaust fans facility. The two power supplies will feed into one (1) skid mounted unit with two (2) 10 kV extensible 10 kV Ring Main Units. These are further fed to redundant two (2) 1600 kVA 10/0.69 kV TNS, rigidly earthed dry type power transformers each with load break switches.

The stepped down power feeds are supplied to an E-house which will house a one (1) Skid mounted unit with two (2) 1600A, 1000V Distribution Boards, identified as Bus A and Bus B respectively with tie breaker LSIG linking both. Each bus will have a main 600AF/1600AT breaker + LSIG trip unit LSIG trip unit suitable for upstream TNS system as well as power meters. These three (3) breakers will have MODBUS TCP/IP communication capability. All will be mechanically and electrically interlocked. Two (2) branch 800AF/800AT breakers will supply power to two (2) Active front end heavy duty 690, 500kW IP42 VSDs units. One (1) feeder 400AF/400AT breaker will supply one (1) 400A, 400V Electrical Panel Board with various feeders via one (1) 160 kVA DYN5 0.69/0.4kV TNS

System IP42 dry type transformer. The remainder feeders will be spares. This 400A, 400V panel will supply power to all ancillary loads of the e-house (HVAC, indoor lighting, outdoor lighting, emergency lighting and receptacles) as well as 10kVA UPS system 400/230V system and associated panel board, welding receptacles, PCS panel, Communication cabinet, and Fire Alarm Panel. The prefabricated building will be suitable for the equipment designed plus an additional 20% spare space. Installation of the e-house will be on elevated open steel supports. There will be floor openings for switchboards, VSDs and electrical distribution boards. In addition, E-house will be designed for one (1) hour fire rating including wall and floor penetration.

16.13 Communication

Underground communications solution will be designed to provide voice, video, and process connectivity across the mine. It will provide operations with an overview and central control of 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 whilst enhancing health, safety, and security of the personnel.

The network architecture comprises of both an automation and business network. A single media ring topology shall be adopted for the automation and business network. Single Mode fiber shall be used for Automation and Business Network Backbones. Redundant fiber routes shall be used for Business Network Backbone. For surface level automation and business network, redundant fiber routes will be used to connect e-house at Exhaust fan Surface facility.

16.13.1 Automation Network

The automation network will consist of the following:

- Sixteen (16) Process Control Stations (PCS) cabinets will be installed for both Surface Main Exhaust Fans E-house and all UG main substations.
- Twelve (12) PCS cabinets will be installed located in various areas in the field throughout the mine for local field instruments.

All PCS cabinets will be connected to local UPS distribution panels. PCS servers will also be redundant. Cabinets and workstations will be rack mounted. The PCS network shall be part of the plant backbone system and will use a combination of ring and star topology to ensure redundancy of the communication network.

All field instrumentation and applicable control stations will be fed back to the local PCS cabinets via CAT6 Cable (Maximum 100 m length). Multimode fibre optic cable could be used only if required for specific application. The following are general requirements for instrumentation.

- Modbus TCP/IP is the main protocol for control system communication with electrical systems.

- 24 VDC for Digital Inputs for Dry Contacts such as Field Devices and Push Buttons.
- 24 VDC for Digital Outputs for Devices such as Solenoids, Interposing Relays and Pilot Lights.
- 24 VDC for Motor Control Circuits and Hardwired circuits.
- 24 VDC for control interlocks for panels with more than one power source.
- 24 VDC Analog Instruments (power) for two-wired transmitters.
- 24 VDC Analog Instruments (power) for four-wired transmitters.
- 4-20 mA isolated Analog input (signal) transmitters.
- 4-20 mA isolated Analog output (signal) transmitters.
- 4-20 mA temperature input transmitters (for control loop).
- RTD or TC temperature input monitoring circuits.
- Pressure transmitter range should be so that is 30% greater than the Maximum Operating Pressure or Exceeds the system design pressure, whichever is greater.
- RTD elements with electronic transmitters are the preferred method to be used for temperature measurement with 3- wire used for most applications and 4-wire when used for flow compensation. A Thermowell is required for all temperature measurement elements located in system processes.
- Level transmitters are preferred over level switches with “Smart” level switches (4 – 20 mA) with diagnostic capabilities being considered, before implementing discrete level switches.
- On/Off Vales with Electrical actuators shall be considered where instrument air is not available.
- For Solenoid Valves, Coil are preferred to be 24 VDC “Low-Power” type.

Communication Cabinets will be installed in all main underground substations as well as all auxiliary underground substations. Each communication cabinet shall house a dedicated industrial PCS LAN and switch and business network LAN and switch. Each Access switch will relate to core switches for each LAN. Cabinets and workstations will be rack mounted. Refer to drawing CRP120-2200-1510-LYD-0001-E-C-TYPICAL Communication Panel Layout a typical communication panel layouts to be used in Main substations and associated Bill of material.

All Electrical equipment such as skid mounted- distribution boards, UPS, RMUs and switchgear will be connected back to industrial PCS switch using MODBUS TCP/IP protocol.

16.13.2 BUSINESS NETWORK

The scope of supply for underground mine for the voice/data communication architecture is based on 5G LTE system over leaky feeder for this study.

LTE Network over leaky feeder typically consists of Remote Radio Units/heads or Small Cells, antennas and/or radiating cables with accessories, Leaky Feeder controllers, bi-directional amplifiers, splitters, repeaters, antennas, terminators, cables, LTE core on-premises, push-to talk over cellular server, headsets with all accessories, LTE radio Antenna, PoC servers and Microwave Backhaul System. Quantities of each will be determined by supplier.

The UG business network will support telephone, data, IT services and business application and consist of the following:

- Ethernet connection to a Wireless (Wi-Fi) Access Point from the switch.
- Ethernet connection to Emergency Alarm System.
- Ethernet connection to a Remote Radio Unit for LTE communication.
- Ethernet connection to Fire Alarm System (if applicable).
- Ethernet connection to Real time location system.

16.13.3 FIBRE OPTIC BACK BONE

The underground fibre optic back bone will be a 48 / 72 strand single mode armoured cable. 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 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 60^0C.

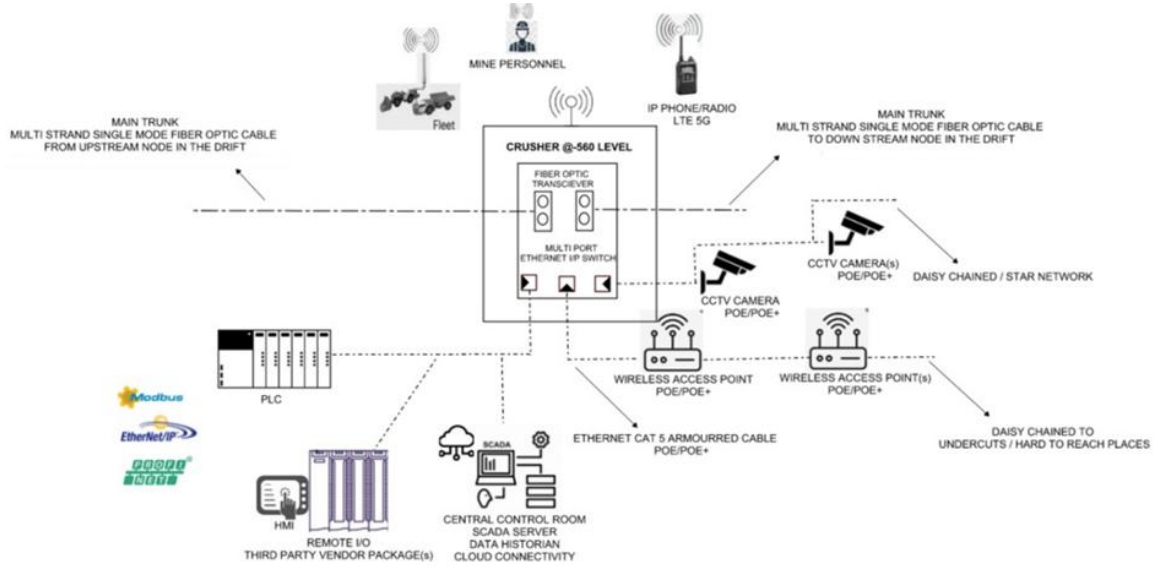
Fiber optic backbone will traverse through the switch cabinet installed at various critical infrastructure points and production areas throughout the mine to integrate the following services as presented in Table 16.22.

Table 16.22 – 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 16.18 illustrates a typical a switch cabinet integrating all the services listed above.

Figure 16.18 – Typical a Switch Cabinet Integration



Source: WSP, 2024

16.13.4 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 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. It is assumed that all engines selected will be in alignment with Stage 5 Emissions Regulations pending availability. Stage 5 is EU regulation for engines (emission) and must comply with articles 40, 41, 49 and 50 of the Rulebook on technical norms for diesel powered equipment used in underground mining in non-methane conditions (Sl. List SFRJ 66/70) in terms of Diesel PM and CO, CO2, NOx, SO_2 and aldehydes emission. WSP progressed the equipment fleet and applied a utilisation factor to capture the worst-case scenario with the resulting airflow demand of 282 m³/s is required to be supplied to the Project to meet the LOM air supply demands

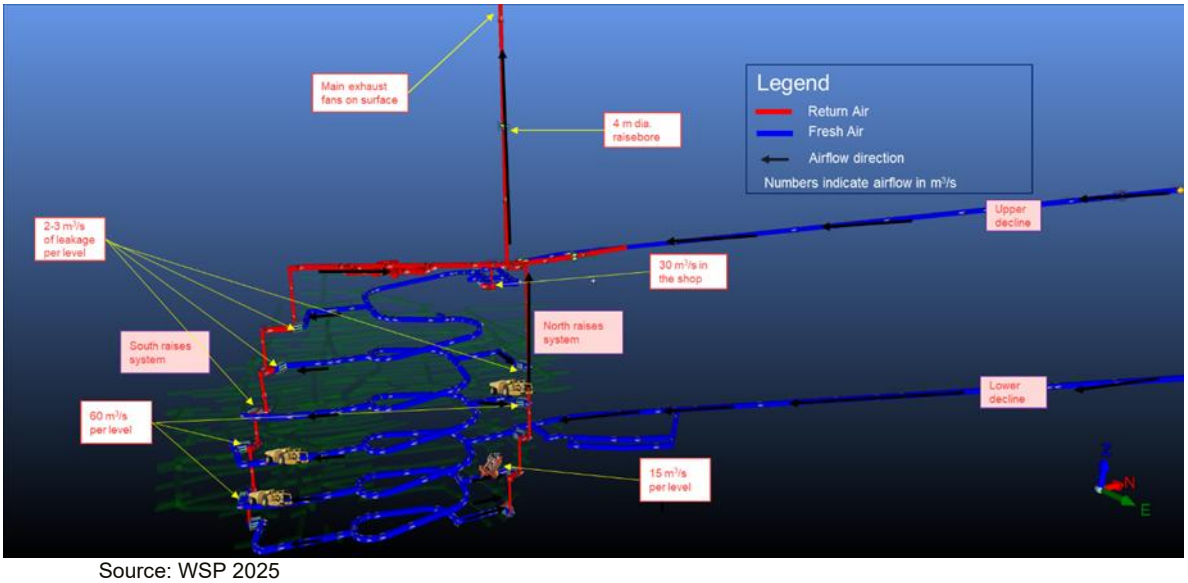
Other design criteria are defined as follows:

- 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 (only where personnel/equipment are operating)
- Foul air from production levels shall not mix with air from the ramp.
- Optimise blast clearing times.
- 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 where personnel and equipment are operating and to minimise the number of raises to surface, a design to have one dedicated Return Air Raise (RAR) with two (2) portals as intakes was selected as shown in Figure 16.19. To ensure that the design does not mix the foul air with fresh air from the ramp, two (2) sets of internal raises are required; one for the North side of the levels and another for the South side of the levels. For the control of the airflow in the ramp, the ventilation network for Čoka Rakita has the intake from the upper and lower decline entering the spiral ramp. Air is then exhausted through the network of ventilation raises with the use of airflow regulators.

Figure 16.19 – Isometric View of Airflow Split Per System of North and South Raises



Source: WSP 2025

There are two (2) main fans each with an estimated motor power of 575 kW installed on surface at the main exhaust raise in a parallel horizontal arrangement. For safety purposes in case of a fire, main fans must have reversal capabilities of at least 50% and reversing of the fans shall not take more than 10 minutes as per Serbian regulations. The design collar pressure of the main fans is estimated at 2,450 Pa and a total airflow of 282 m³/s.

It is assumed that as a worst case in terms of airflow requirements on production levels, there will be a maximum of three (3) trucks operating on the production levels and the three (3) remaining trucks will be operating in the ramp system.

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 4.1 m diameter design) and the internal raises (currently 4.1 m (South) and 3.1 m North) diameter designs). The escapeway raise is planned to be installed in the Southern

ventilation raises. The air velocity in the internal raises should consider a maximum velocity of 6 m/s when workers are present.

Airflow regulators will be used to control the air distribution amongst the different production levels. They are controlled from surface and are used to clear blasts by isolating the affected production levels. To ensure adequate airflow on all levels, airflow sensors with gas sensors are installed on every level access.

The system will be equipped with a state-of-the-art Ventilation-On-Demand (VOD) system to modulate the speed of the main fans, auxiliary fans and automated regulators based on airflow requirements determined considering the planned production, equipment fleet and utilisation factor. The system will detect the entry and exit of trucks on the production level to start/change fan speed as required and open/close airflow regulators.

In the spiral ramp, between levels, the control system will detect low airflows and make fan adjustments to maintain adequate airflow. Jet-fans will pressurize the upper or lower decline to draw more air in either one to control the airflow in the spiral ramp. Jet-fans are selected as opposed to using a booster fan and a door so that vehicular traffic is not impacted.

Automated double doors will also block the airflow where vehicles must pass. Double doors will be installed on RL640. The first door is located just past the washbay in the ventilation access drift and the second is located just past the paste backfill bay. The two (2) doors will remain closed under normal operation and can be activated with a pull cord system for vehicle passage. An interlock will be set so that at least one of the two is always in the closed position. For the mechanical shop, the airflow will be managed with an airflow regulator and a personnel door for egress at the outlet of the shops exist. At each entry to the garage, one (1) fire roll up door will be installed to isolate the shop in case of a fire. Under normal operation the roll up doors will remain opened. To have access to the powder mag in the return air system, double doors hydraulically actuated will be used for vehicle passage. The doors will be opened/closed using a pull cord system. A personnel door will also be installed in the wall for personnel passage.

There are six (6) different types of walls for airflow regulators and doors which are:

Walls with airflow regulators equipped with double personnel doors for egress. This type of wall will be used for all ventilation walls connected to the South raise and the RL620 shop.

- The ventilation access drift which is parallel to the mechanical shop will be equipped with automated double doors for vehicle passage. The type of door will be opposed swing type. A double personnel door will be installed on the side of the doors for personnel passage.
- The entry doors of the mechanical shop will be a roll-up door type.

- RL450 will be equipped with a wall with airflow regulators and closed drop board regulators for LHD passage in case material needs to be removed at the bottom of the raise. The drop board regulator will be sized for LH515i passage.
- RL370 and RL410 will be equipped with a wall with an airflow regulator and closed drop board regulators for LHD passage in case material needs to be removed at the bottom of the raise. The drop board regulator will be sized for LH515i passage. The wall will also have a double personnel door for egress.
- Walls with airflow regulators only are for the North raise for level RL530 and RL490.

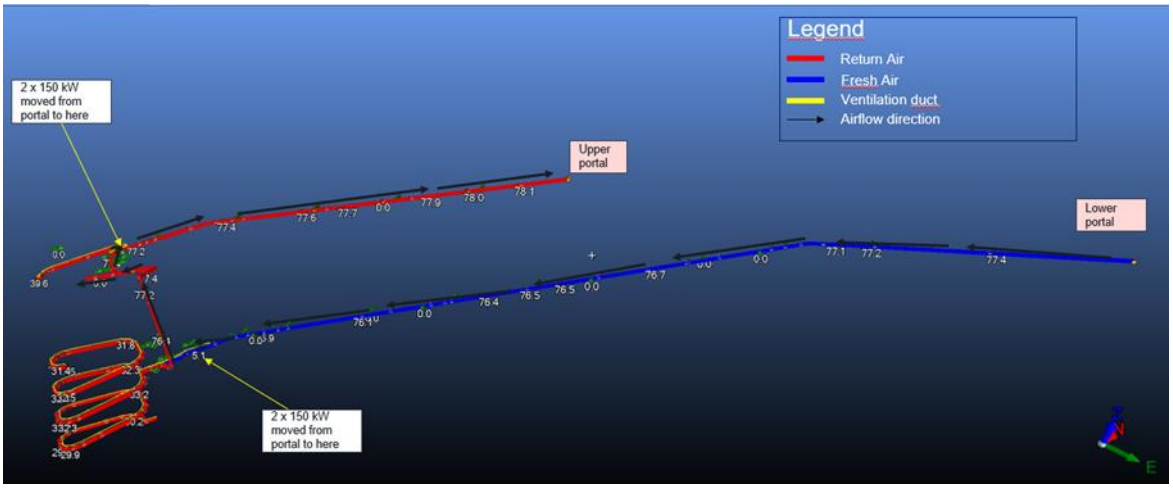
The ventilation walls shall withstand a maximum pressure of 2.5 kPa.

16.14.3 AUXILIARY VENTILATION DESIGN

For the ramp development, auxiliary fans for the upper and lower decline would each have two (2) lines of 1.4 m diameter flexible duct each supplied by auxiliary fans.

Once the main surface raise and Northern internal ventilation raise are developed, a ventilation circuit will be created by installing the fans for the upper decline in a bulkhead as shown in Figure 16.20. 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 auxiliary fans.

Figure 16.20 – Ramp Development Following Northern Internal Raise Connection



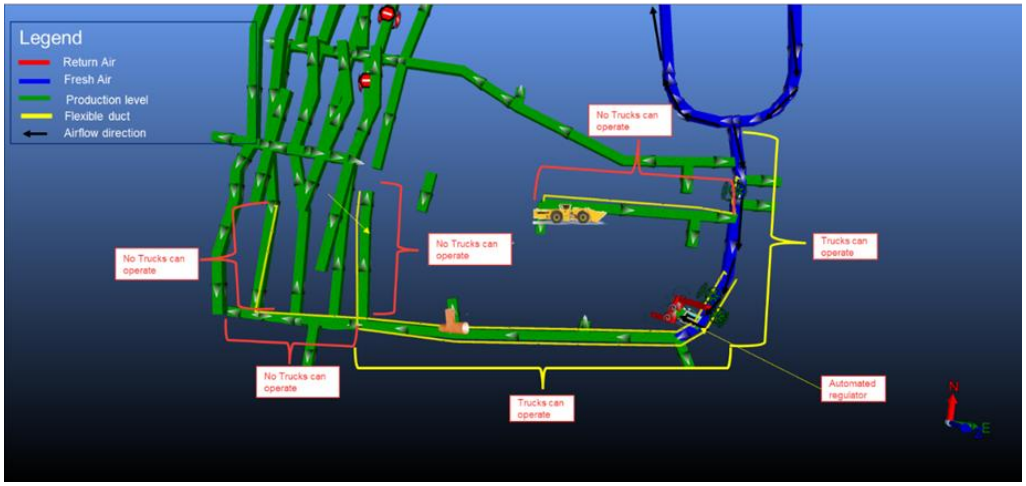
Source: WSP 2025

For the development of production levels, it is assumed that the trucks will be loaded in the ramp. When the breakthrough of the North internal ventilation raise is complete, then the mine truck can be loaded in the access. The ventilation supply requirement for the LHD is 17 m³/s. When the LHD is operating, no other vehicles shall operate within the heading.

On a production level, the mine truck is to be loaded at near the truck load out located in a dedicated truck turn around area. The ventilation duct supplies 52 m³/s air for an LHD and a truck using twin flexible ducts of 1.2 m diameter each. In the case that only one flexible duct with a 1.2 m diameter is used to ventilate the heading, the truck must remain within the main fresh air circuit (i.e. between the regulator and the Return Air Raise {RAR}).

The limitation of where the truck can operate on a level is shown in Figure 16.21. One (1) of the two (2) ventilation ducts will supply 17 m³/s at the working face to ventilate the ore sill for the LHD. Leakage in the duct is assumed at 100 mm²/m² which is equivalent to approximately 4% per 100m. The maximum length of the ventilation duct is 250 m to the truck turn around and 350 m total. An additional 5 m³/s of fresh air must circulate where the auxiliary fan is supplying to, to avoid air re-circulation. The larger auxiliary fans for production headings will be equipped with the two-speed starters. Two-speed fans were selected after performing a high-level evaluation when compared to VSD (Variable Speed Drive). To allow for four (4) active levels while mucking and one (1) level drilling/bolting/loading. It is possible to utilise two-speed auxiliary fans with VOD (Ventilation on Demand) using truck detection on the level. Through DPM guidance, the decision was to use two-speed fans over VSDs because of familiarity where they are currently seeing success at neighbouring mines. The two-speeds fans can operate at 100% or 66%.

Figure 16.21 – Operational Truck Limitation on Production Level



Source: WSP 2025

The airflow requirement for activities when diesel engines are not operating (such as, bolting, drilling or loading the face) is 12 m³/s at the working face to maintain an airspeed requirement of 0.5 m/s. Small auxiliary fans will be used to ventilate these headings, which will provide the airflow to 100 m. These fans are equipped with On/Off starters and do not have remote control capabilities like the other fans.

When the sills connect on either side of the level, a temporary ventilation curtain will need to be installed adequately to prevent short circuiting of the ventilation system. If the ventilation curtain is not in place, uncontrolled flow through ventilation will occur which can create a lack of airflow in the sill or other areas of the mine. It is therefore decided to maintain the auxiliary ventilation throughout using a ventilation curtain. Once a stope is blasted, a ventilation curtain will need to be installed on the upper level of the stope and before commencing mucking activities. When the stope is ready to be backfilled, the ventilation curtain should then be installed at the lower level. The ventilation curtain can be removed once the stope is fully backfilled. The temporary ventilation curtains should achieve a minimum hydraulic resistance of 5 N s2/m8 not to exceed more than 2 m³/s of leakage; it should be noted that lack of maintenance of ventilation curtains have been identified as one of the highest risks for the ventilation system integrity. It is recommended that sufficient resources be allocated to install, repair and maintain the curtains.

16.15 Mine Backfill

The primary mine backfill is Cemented Paste Fill (CPF) formulated from mill tailings in the surface Paste Plant.

The Underground Distribution System (UDS) is comprised of underground boreholes, trunk line piping, level piping, and stope crosscut piping. The design begins at the discharge of a positive displacement (PD) pump located at the Paste Plant and ends with paste delivery into the mine's primary and secondary stopes.

The CPF will be pumped using a positive displacement pump from the paste plant via a twinned buried pipeline (operating and standby) approximately 285 m long to the upper decline portal. The CPF piping continues down the upper decline to its intersection with the orebody on RL650 level. The surface and decline pipelines are the only twinned lines in the UDS. All remaining pipelines in the orebody will be single duty lines. Boreholes within the orebody will be cased and twinned with duty and standby boreholes.

Boreholes cascade downward from level to level between paste backfill bays, varying in dip, but all as close to 70% as practical.

The paste backfill bays are designed to be approximately 10 m long to allow sufficient room for two borehole collars and breakthroughs (5 m apart), pipe fittings, and instrumentation. The bays (5 m width and 5.5 m height) are designed to be aligned with the size of the level drifts from which they branch.

The CPF design criteria and material mass balance was 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 is rock fill and will use the development waste rock generated in the last quarter of Year -1 and onwards. Cemented rock fill (CRF) will be used starting in the last quarter of Year -1 and the use of uncemented rock fill (RF) will begin starting in the first quarter of Year 1.

The CRF will be placed into stopes prior to the addition of CPF, and into stopes that will be exposed by adjacent mining. The maximum proportion of CRF in a stope will be approximately 55% of the overall stope volume, during these formative years of operation.

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 also serves the purpose of mixing shotcrete for underground use.

The Batch Plant consists of a high shear colloidal mixer that creates a homogeneous binder slurry at a 0.85:1 water to cement ratio dosed with a hydration retardant admixture, all of which is discharged in an “on demand” basis into a mobile transmixer.

The transmixer will transport 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 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 UDS design are:

- Trunk line piping from the Paste Plant to the three (3) receiving bays on RL650 level is 150 mm CS Schedule 120 and Schedule 80 pipe with flanged connections on the surface portion and Victaulic style HP70ES couplings on the underground portion.
- All boreholes are “twinned” at 5 m spacing, meaning that there is one duty and one standby borehole in case the duty borehole becomes inoperative. All boreholes are “cased,” meaning

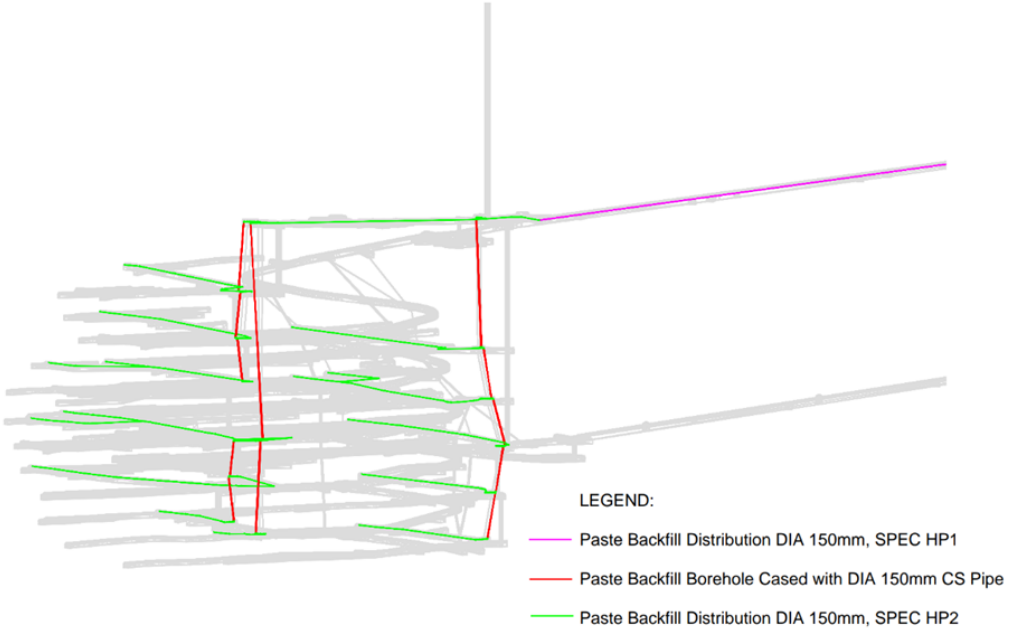
that oversized 230 mm holes are drilled, and 150 mm carbon steel (CS) Schedule 80 pipe installed inside them to mitigate the predicted questionable rock quality characteristics.

- All boreholes originate or terminate in paste backfill bays to allow the drilling equipment's operation without interfering with mine vehicle traffic.
- Backfill receiving and distribution functions are in the same bay.
- CS pipe sections of 4-m lengths are connected using high-pressure Victaulic HP70ES style couplings.
- 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 crosscut piping connected to the level piping is constructed of 150 mm DR9 HDPE pipe due to low internal pressures.

The mine paste fill underground distribution system schematic is presented in Figure 16.13. Figure 16.22 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 16.14.

Figure 16.22 – Backfill Underground Distribution System on a Mine Longitudinal Section



Source: WSP 2025

16.15.2 KEY DESIGN COMPONENTS OF THE SHOTCRETE/CRF SLURRY BATCH PLANT

The Batch Plant for the preparation of Shotcrete and CRF slurry will be of modular construction so it can be built, shipped, and erected quickly on a relatively small footprint.

Vendors typically supply distinctly purposed modular plants which serve to either produce Shotcrete or CRF slurry. In this case the Vendors have been asked to supply a hybrid modular plant capable of producing both Shotcrete and CRF slurry and thereby offering the potential of common use of some of the individual plant's components.

It is important to note that different Vendors have different components contained within their plant's modules so what is presented are the components that are believed to be the most likely to be included in Vendor supplied modules for a functional plant.

The components in the Vendors' modules will consist of:

- Vendor Supplied Cement Slurry Mixing System (potentially a Colloidal Mixer).
- Shotcrete Mixer.
- Plasticizer Reagent Tote.
- Plasticizer Metering Pump.
- Retardant Reagent Tote.
- Retardant Metering Pump.
- Clarified Water Tank.
- Clarified Water Pump.
- Agitated Discharge Water Tank.
- Air Compressor.
- Air Dryer.
- Air Receiver.
- Electrical and Instrumentation Room.
- Two (2) Cement Silos (with associated silo equipment and a Passive Dust Collection System).
- Two (2) to four (4) Cement Screw Feeders.
- Binder Weigh Hopper.
- Shotcrete Aggregate Feed Bin.
- Aggregate Conveyor with Weigh Element.

WSP has successfully used similar style CRF plants on other mine backfill projects.

17 RECOVERY METHODS

17.1 Introduction

The Čoka Rakita process plant has a nameplate capacity of 850,000 metric tonnes per year. The flowsheet consists, primarily, of crushing, grinding, gravity concentration and flotation unit operations. Three saleable products are produced i.e., a flotation concentrate, a gravity concentrate and doré bars. Tailings are filtered and either stored on a dry-stacked Tailings Management Facility or repulped for paste backfill underground.

17.2 Process Plant

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.

17.2.1 Trade-Off Studies

During the Pre-Feasibility Study (PFS), process development trade-off studies (ToS) were undertaken to evaluate flowsheet options for gravity concentrate treatment, comminution, and flotation.

- Gravity Concentrate Treatment:

  The initial assumption was to sell primary concentrate in its current state to a third party after dewatering and bagging. However, the presence of coarse gold renders sampling and analysis of the primary concentrate problematic. To overcome this “nugget effect” downstream beneficiation of the primary concentrate was explored. Secondary gravity concentration followed by bagging of the secondary gravity tails and direct smelting of the secondary gravity concentrate was recommended and adopted.

- Comminution ToS:

The comminution ToS used variability testing results to size comminution equipment and evaluated various combinations of new and existing grinding equipment. The selected configuration features reusing the SAG mill from Ada Tepe as an AG mill including the pebble crusher from Ada Tepe as the primary grinding circuit. For the secondary grinding circuit one of the two Vertimills available from Ada Tepe will be reused. The existing cyclopacs from Ada Tepe will be re-used to close out both grinding circuits.

- Flotation ToS:

  A circuit consisting of new Jameson cells to a circuit consisting of both new and refurbished SFR cells from Ada Tepe. Due to its simplicity and costs advantages, the Jameson cells option was adopted for the PFS and carried forward into the FS phase of the Project.

17.2.2 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 FS the value of 600 was retained as this aligns with typical industry practice.
- Based on testwork results an empirical correlation was derived for gravity recovery as a function of gold head grade, as follows:
  - Gravity Recovery = 39.6 x Gold Head Grade ^ 0.133.

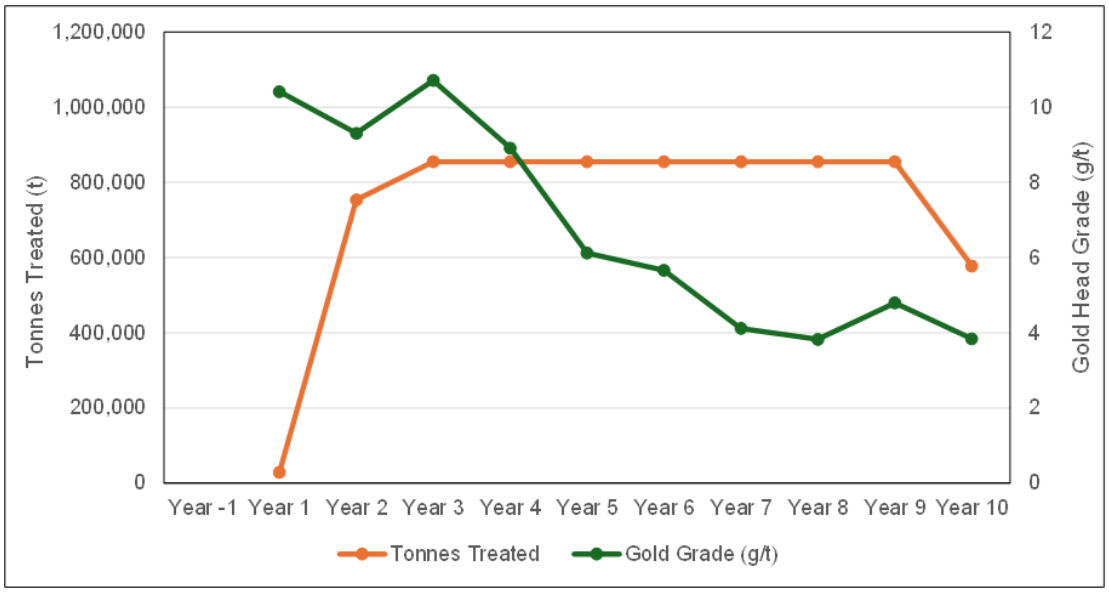
  Note that this correlation is similar to what was used before in the PFS phase, but it has been updated to better fit the increased gravity recoveries recorded during the Phase 3 testwork campaign.
- Based on industry experience a downgrade factor was applied to scale the laboratory gravity recoveries up to full-scale plant recoveries. This downgrade factor was 20% at the nominal head grade and 10% at a head grade of 10 g/t. The empirical correlation used for this downgrade factor is:
  - Downgrade factor = -2.7624 x Gold Head Grade +37.62.
- Gold stage recovery from primary to secondary gravity concentrates are fixed at 63% and the stage mass pull is 5% (unchanged from PFS).
- A constant flotation stage recovery of 79% based on testwork results from all three (3) phases.
- Mass pull to final flotation concentrate = -0.2008 x Gold Head Grade + 5.860
- For the first month of operation, recoveries are expected to be about 5% below par due to operator inexperience and optimisation / commissioning issues. From the second month and

continuing up to the end of the 12th month of operations this inefficiency penalty reduces to only 0.4% below expected.

With the above assumptions, a production profile was determined and is presented in Table 17.1.

Figure 17.1 shows the ROM feed profiles over the LOM. It was assumed that all material mined in Year -1 would be stockpiled and treatment would commence in Year 1 only.

Figure 17.1 – Plant Feed Tonnage and Gold Grade Profiles



Source: DRA, 2025

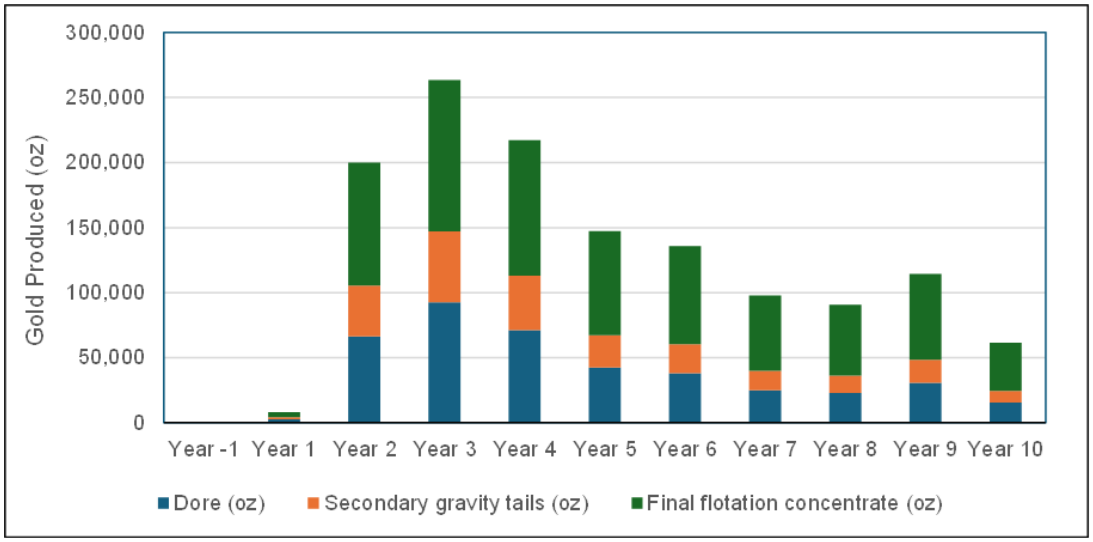
Table 17.1 – Gold Production Profile by Year of Mining

Description	Unit	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10
Tonnes Treated	t	28,333	753,624	854,997	855,000	854,928	855,000	855,000	855,000	855,000	577,886
ROM Au Grade	g/t	10.42	9.31	10.71	8.92	6.12	5.66	4.12	3.82	4.79	3.84
Gravity Product Au Distribution	%	17.3	17.3	18.5	17.1	14.8	14.4	13.0	12.8	13.6	12.8
Gravity Product Au Content	oz	1,645	39,006	54,410	41,804	24,887	22,401	14,765	13,430	17,985	9,122
Doré Au Distribution	%	29.5	29.4	31.5	29.0	25.2	24.5	22.2	21.8	23.2	21.8
Doré Au Content	oz	2,801	66,416	92,643	71,179	42,375	38,142	25,140	22,868	30,624	15,533
Flotation Conc Au Distribution	%	39.9	41.9	39.6	42.6	47.4	48.3	51.1	51.7	49.9	51.7
Flotation Conc Au Content	oz	3,787	94,509	116,506	104,388	79,844	75,109	57,897	54,378	65,707	36,876
Overall Au Recovery	%	86.7	88.6	89.5	88.7	87.4	87.2	86.4	86.3	86.7	86.3
Overall Au Produced	oz	8,232	199,931	263,559	217,371	147,106	135,652	97,802	90,676	114,316	61,531

Note: Totals may not add due to rounding.
Source: DRA, 2025

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 two (2) years 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

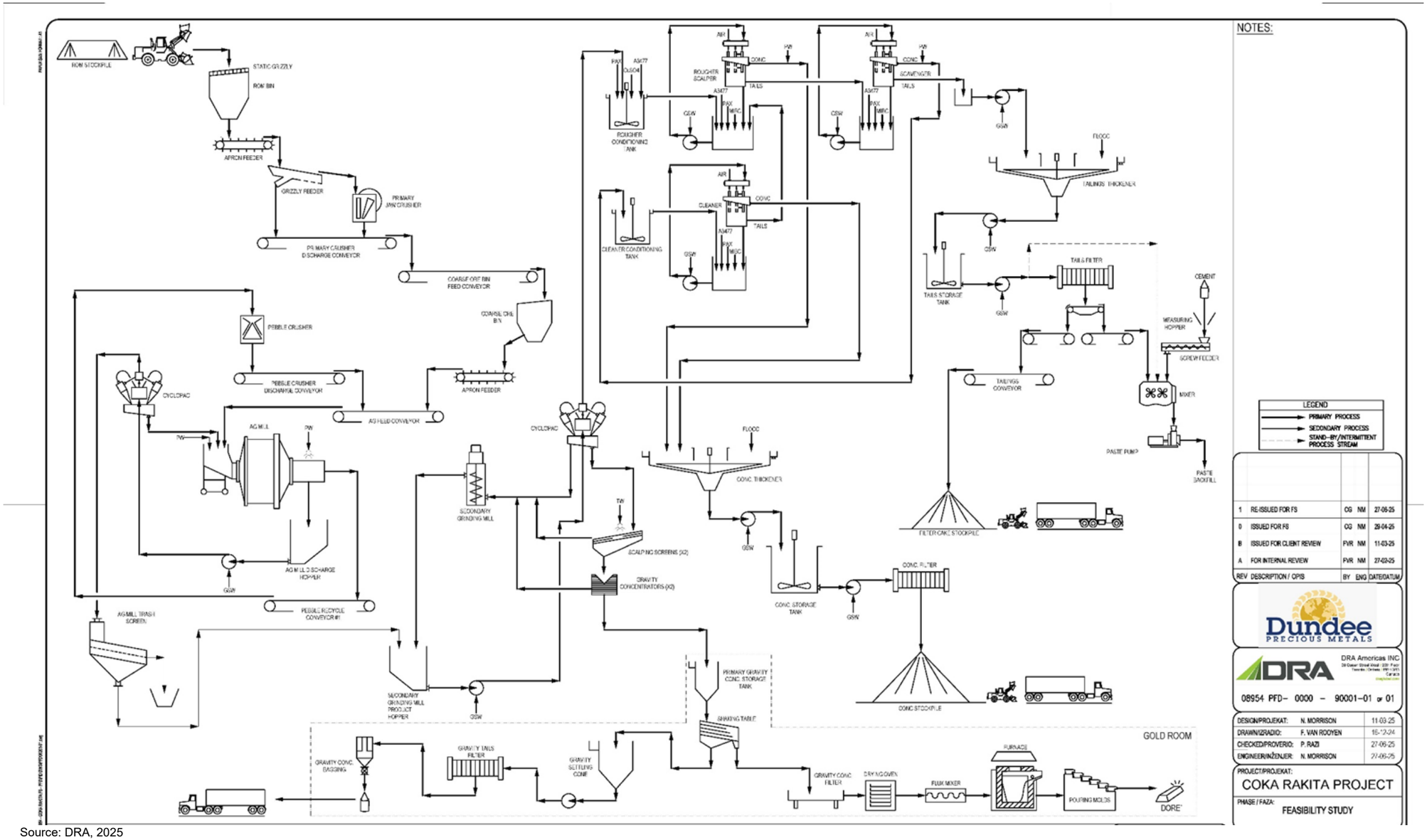


Source: DRA, 2025

17.2.3 PROCESS FLOWSHEET

The Čoka Rakita deposit has high gold-grade, gold-copper skarn type mineralisation roughly 250 m below surface. Based on its relatively modest copper grades, copper is not expected to contribute significantly to the revenue stream. Preliminary metallurgical test results indicate that the mineralisation is amenable to flotation and gravity concentration and has the potential to produce 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 a paste backfill sections. The proposed process plant will be producing doré bars, a gravity concentrate and a bulk flotation concentrate. Tailings will be disposed of underground as paste backfill or aboveground 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, 2025

17.2.4 PROCESS DESIGN CRITERIA

Table 17.2 summarises the Process Design Criteria (PDC) to nominally process 850,000 dry tpa of ROM material 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 tph	97	111.6
ROM Throughput-Plant	dry tpa	850,000	977,500
Feed Grades			
ROM Material Head Grade	Au g/t	6.38	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	10	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 tph	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	
Total Ports on Cluster	#	6	
Number of Operating Cyclones	#	1 or 2	
Cyclone Diameter	mm	508	
Secondary Grinding			
Fresh 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	
Total Ports on Cluster	#	12	
Number of Operating Cyclones	#	7	
Cyclone Diameter	mm	254	
Gravity Concentration			
Scalping Screens Dimensions		1.83 m L × 0.92 m	
Slurry Flow to Screens (combined)	t slurry/h	170	205
# of Parallel Gravity Circuits	#	2	
Slurry Flow to Concentrators	m³/h/unit	60	70
Flotation Design: Jameson Cells			
Rougher Scalper Model	-	E3432/8	
Rougher Scalper - Fresh Feed Slurry Flow	m³/h	367	
Rougher Scavenger Model	-	E3432/8	
Rougher Scavenger - Fresh Feed Slurry Flow	m³/h	366	
Cleaner Model	-	E1714/2	
Cleaner - Fresh Feed Slurry Flow	m³/h	62	
Secondary Gravity Concentrate Handling			
Primary Gravity Concentrate Produced	t/d	1.71	2.35
Operating hours per weekday	h	5	6
Number of Days Operated Weekly	#	5	5
Shaking Table Required Feed Rate	m³/t/h	0.34	0.39
Secondary Gravity Concentrate Produced	t/d	0.086	0.117

Description	Unit	Nominal	Design
Flotation Concentrate Handling			
Thickener Fresh Feed Slurry Flow	m³/h	28.1	40.4
Thickener Underflow	m³/h	3.7	5.3
# of Flotation Concentrate Storage Tanks	-	1	
Concentrate Storage Tank – Storage Time	h	24	
Concentrate Storage Tank – Live Volume	m³	95.7	137.6
Tailings Handling			
Tailings Thickener Fresh Feed Flow	m³/h	359.7	413.7
Tailings Thickener Underflow	m³/h	108.7	125.1
Number of Tailings Filter Presses	-	2	

Source: DRA, 2025

17.2.5 PROCESS DESCRIPTION

The Čoka Rakita process plant is designed to process 850,000 dry metric tonnes of ore per year. ROM material is crushed by a primary jaw crusher and then stored in a coarse ore bin which provides 24-hour surge capacity. The crushed ore undergoes two-stage grinding, first with an autogenous mill (including a pebble crusher) and then a vertical stirred mill, achieving 80% passing 53 µm. Both grinding circuits are closed out by dedicated hydrocyclones. Part of the secondary cyclone underflow feeds a two-stage gravity concentration circuit, producing high-grade concentrate for smelting. 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, 87.5% of the gold is recovered: 25.5% as doré, 15% as gravity concentrate, and 47% in the 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 excess contact water treated before discharge.

ROM material is transported from the 3-day mining 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 autogenous (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. Under nominal conditions the AG mill is expected to draw only 1.6 MW of its installed motor capacity of 3 MW. When treating ore at the 85th percentile of hardness the power draw is expected to increase to 1.75 MW. 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 766 kW nominally and 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 (+2 mm), sending them back to the pump box. Screen undersized material proceeds to the 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 half-hourly to a holding tank in the secure gold room. The holding tank is sized to accommodate 1 day of arisings which allows for accumulating primary concentrate over the weekend when the goldroom is not manned.

Concentrate from the holding tank is processed on a shaking table for about five (5) hours daily, producing around 86 kg of final concentrate at a grade of 48 kg/t gold. The table is sized to operate for approximately 26 hours each week, providing sufficient spare capacity to draw the bin down rapidly on Mondays and to provide flexibility for maintenance and cleaning tasks. 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 1.5 kg/t gold in a mass of about 1.63 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 concentrate 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 55% of their flow.

The scavenger concentrate is sent to a cleaner conditioning tank, which overflows 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 recycle tank. This final concentrate contains 47% of the ROM gold at 65.5 g/t and represents 4.6% of the ROM feed by mass. Reagents used include 0.2 kg/t PAX, 0.09 kg/t MIBC, 0.25 kg/t copper sulfate, 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, at a slurry density of 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 Cake with a 13% moisture content.

When needed, tailings cakes are sent to a paste mixer, where it is blended with thickener underflow and approximately 5% cement (by dry solids mass) to create a paste for backfilling underground. Around 46% of the yearly tailings are used as paste backfill underground, while the remaining 54% is transported and stacked in a designated DTSF.

17.2.5.1 Reagents and Consumables

The following reagents are used throughout the process plant:

- Flocculant – Concentrate and tailings thickeners and the clarifier.
- Coagulant – Concentrate thickener.
- Copper sulphate – Flotation reagent.
- Potassium amyl xanthate (PAX) – Flotation collector.
- Cement Binder – For Paste Plant.
- Methyl isobutyl carbinol (MIBC) – Flotation frother.
- A3477 – Flotation collector.

- Borax, Silica sand and Lime – Smelting fluxes.
- Hypochlorite – To produce tap water through chlorination.
- Plasticizer – Shotcrete admixture.
- Retardant – Shotcrete and CRF Slurry admixture.
- Fibre stands – Shotcrete recipe additive.

Reagent mixing will be completed in a designated area within the plant. The design of this area includes safety features such as bunding, with dedicated sump pumps and ventilation fans. The layout and general arrangement of the reagent area accounts for the need to prevent contact of incompatible reagent types. Separate onsite long-term reagent supply storage is provided at a safe distance away from the process plant.

Reagents are made up with clarified 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
Flocculant – Clarifier	g/m³	3	3
Coagulant – Concentrate Thickener	g/t concentrate	20	20
Collector (A3447)	g/t ROM	160	200
Collector (PAX)	g/t ROM	160	200
Activator (Copper sulphate)	g/t ROM	500	500
Frother (MIBC)	g/t ROM	33	41.3
Grinding Steel			
VertiMill Media Consumption	kg/t ROM	0.4	0.4

Source: DRA 2025

17.2.5.2 Utilities and Services

Water Services

Raw / fresh water will be harvested from surface run-off and stored in the NCW pond. This water will be used to replenish other water sources when needed. Given the low moisture content of the products leaving the plant relative to the large influx from mine dewatering; 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 clarifier overflow will be routed to the Clarified Water System as make-up to minimise the volume that has to discharge to the environment. A small portion of the clarified water will be chlorinated and used as tap water for domestic purposes. Drinking water will be produced on site through purifying a portion of the tap water.

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.

Electrical Services

Electricity will be sourced from the local grid via buried power lines. The total installed power for the surface facilities is estimated at 12 MW.

17.3 Process Control

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

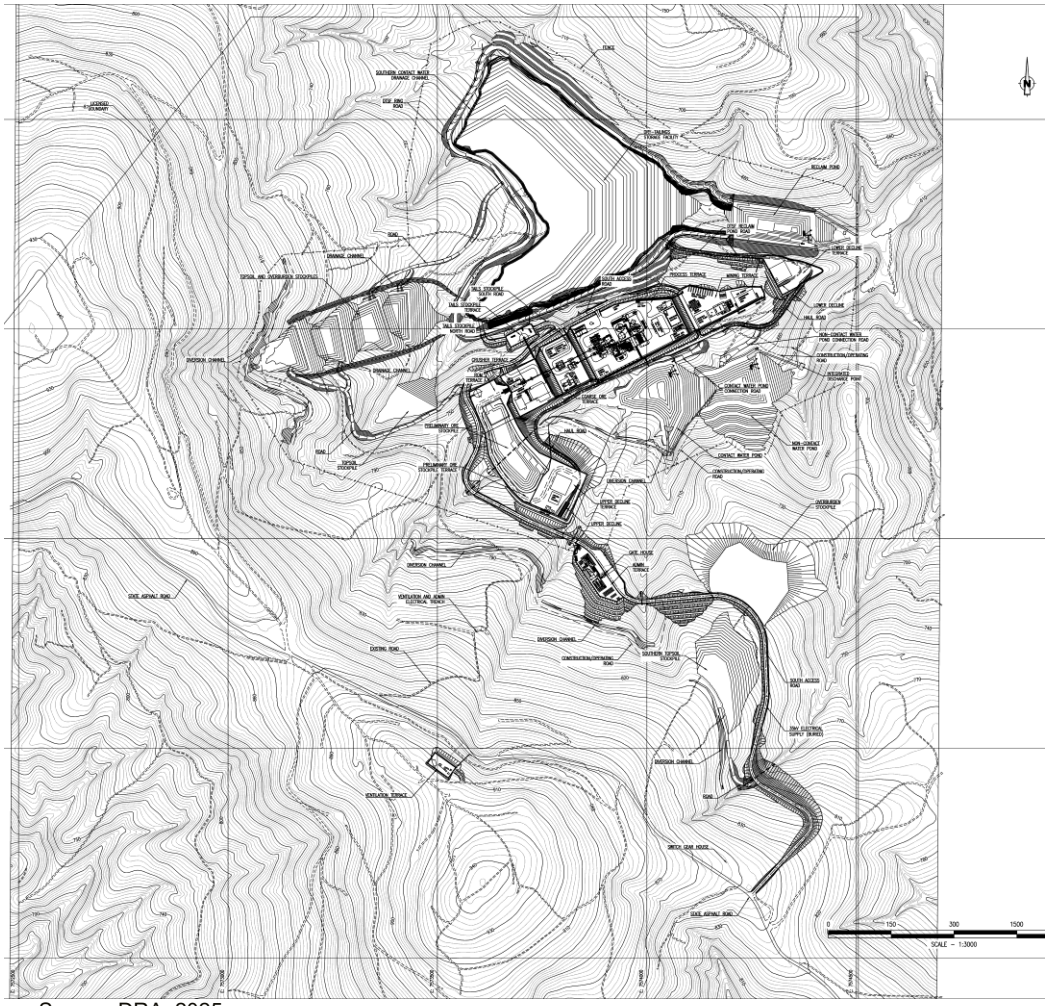
18.1 General

This Project Infrastructure section describes the details associated with the primary processing infrastructure and the supplemental infrastructure required for the successful development and operation of the Čoka Rakita facilities. The infrastructure described in this section includes:

- Roads.
- Process terraces.
- Mining services area terrace (including fuel depot).
- Mine and processing plant supporting infrastructure terrace.
- Administration terrace.
- Mine waste rock management.
- Overburden and topsoil management.
- Dry Tailings Storage Facility (DTSF).
- Water management.
- Ventilation shaft and equipment.
- Power supply and distribution.
- Automation.
- Communication.
- Services and utilities.

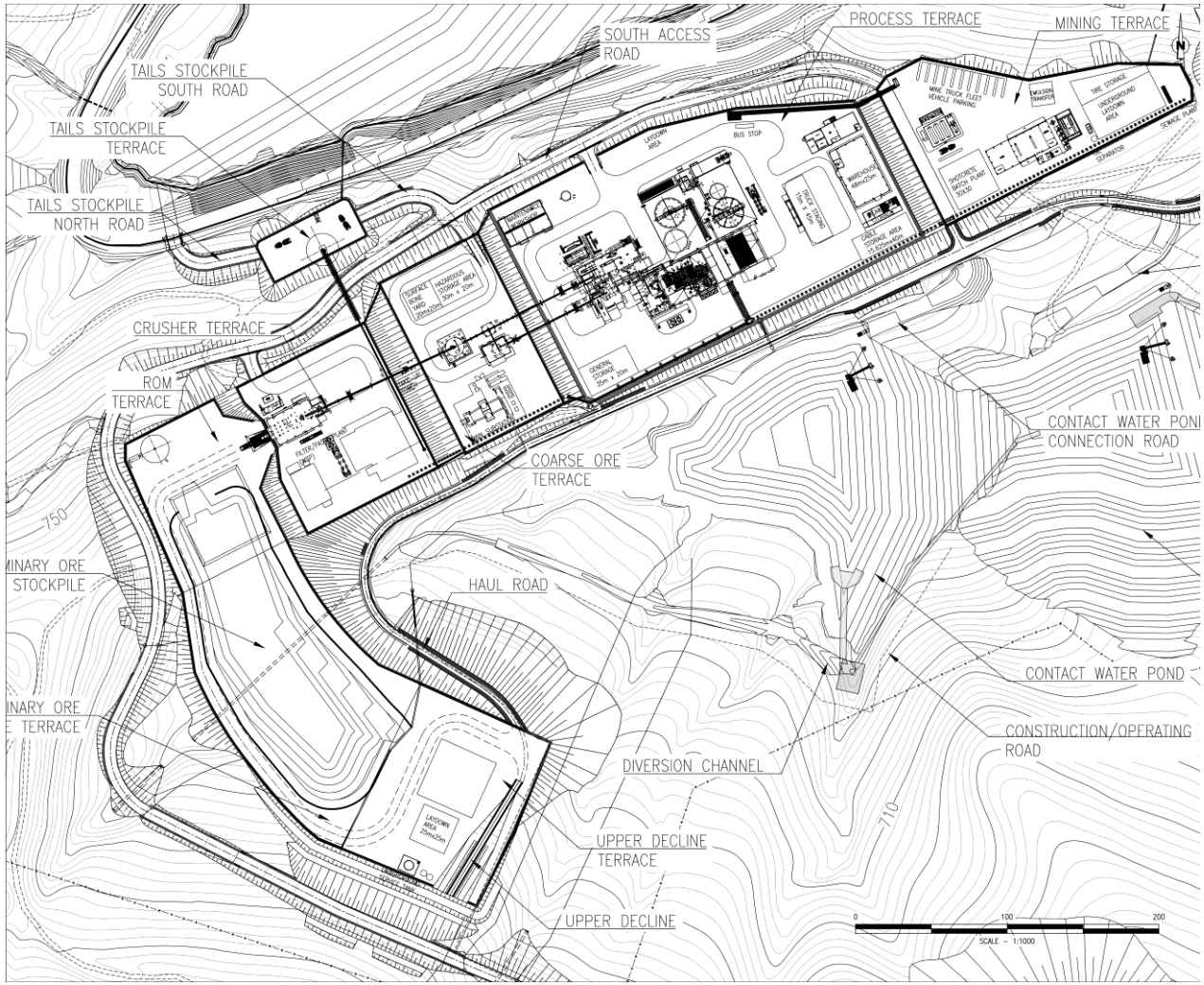
The proposed overall site layout, with site access, is shown in Figure 18.1, while Figure 18.2 illustrates the proposed details of the project and plant layout.

Figure 18.1 – General Site Layout



Source: DRA, 2025

Figure 18.2 – Process Plant Layout



Source: DRA, 2025

18.2 Main Access Road

The Project site will be accessed via a new road, which will connect the existing public road, near the southeastern corner of the site perimeter, to the site terraces. This road will be generally orientated in a north-south direction, servicing the administration terrace, and the gated access to the process plant and mining operations.

The main access road will feature a guard house near the administration terrace. The guard house will control access to the Mine Dry and Mine Offices at the administration area, as well as access via the travel way to the mining area, process plant area and Dry Tailings Storage Facility (DTSF).

The road is approximately 2,600 m long, constructed from cut and fill materials. Volumes are indicated in Table 18.2.

18.3 Site Roads

A series of roads will be constructed to provide access to various areas surrounding the project site. Access roads and haul roads will be sized with turning radii to accommodate the underground mining trucks exiting the mine and travelling to the stockpiles, the mobile equipment operating at site, haul trucks for tailings handling, and various delivery trucks. All two-way (dual carriage) access roads will be sized to 8 m wide, and all one-way (single carriage) haul roads will be sized to 4 m wide. The total length of the site roads is summarised in Table 18.1.

The construction of the new roads consists of the following:

- Access to process plant terraces.
- South haul road.
- Stockpile access roads.
- DTSF access roads.
- Tailings material transport roads.
- Access to Contact Water Pond.
- Access to Non-Contact Water Pond.
- Ventilation Facility Access.

Table 18.1 – Estimated Length of Roads

Item	Approximate Length (m)
Main (South) Access Road with Entrances	2,571
Haul Road with Entrances	954
Tails Stockpile-North	68
Tails Stockpile-South	69
Ventilation Facility Access Road	40
DTSF Ring Road	2,300
DTSF Reclaim Pond	234
Sub-Total: Haul Roads	*1,090*
Sub-Total: Access Road	*2,845*
Sub-Total: DTSF Ring Road	*2,300*
Total Length of Roads	**6,235**

The road on the south side of the process plant terrace will be used as the haul road for mining trucks travelling between the upper decline (and ROM Pad) and the lower decline. Since this road will be managed and controlled by mining personnel, it will also be used by other mining vehicles, for example such as materials and 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 for process plant and general surface personnel to the individual process plant terraces. It will be developed at full width down to the mining services terrace below the process plant.

18.4 Terraces

The project site requires terraced areas to accommodate the project infrastructure and equipment. The current terraces were designed through an iterative process between project element layout and suitable topography and terrain.

The terraces are designed with grading and a drainage system that will allow the runoff water to reach the collection ponds and natural drains, in accordance with the site water management requirements.

Table 18.2 and Figure 18.3 present the earthworks balance where topsoil will be stored to the west of the DTSF and at a higher elevation than the DTSF. The remaining cut and fill quantities are almost balanced, assuming the excavated weathered rock (poor quality rock) and good quality rock can be used for the majority of the project fill requirements. This should be further reviewed and optimised

in the next phase of the project, once additional geotechnical information becomes available; and recommendations are updated.

The process plant terrace and elevations are designed for optimal functionality, accessibility and elevations for cut and fill construction.

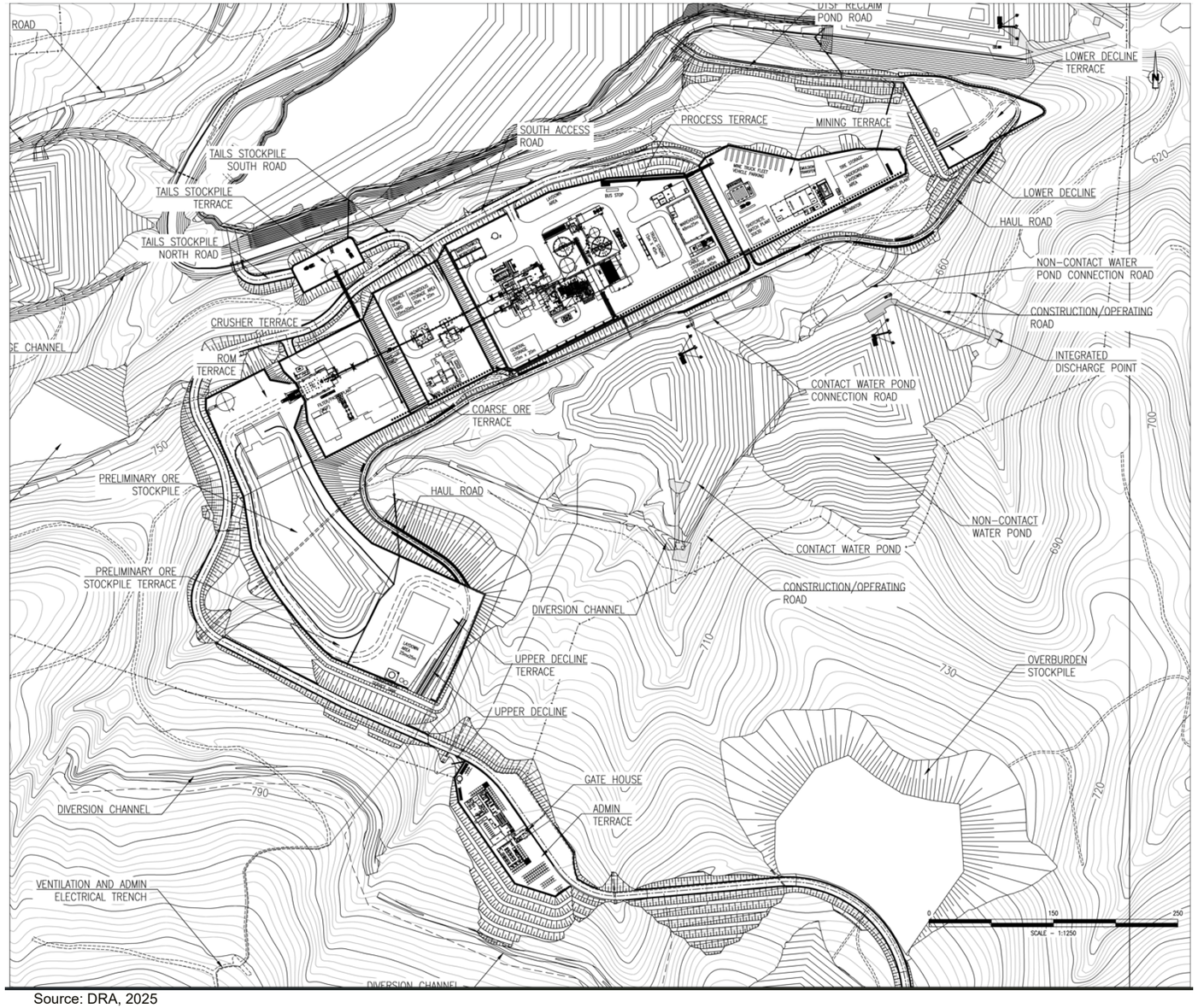
Table 18.2 – Site Earthworks Volume

Item	Type	Excavation in Topsoil (0.3m depth) (m^3)	Excavation in Overburden, Weathered Rock or Good Rock (m^3)	Total Excavation (from Design) (m^3)	Total Fill (from Design) (m^3)
South Access Road	Road	20,235	122,873	143,107	136,679
Haul Road	Road	4,951	9,576	14,526	31,981
Tailings Stockpile Access Road - N	Road	379	2,598	2,977	376
Tailings Stockpile Access Road - S	Road	245	1,252	1,497	330
Ventilation Road	Road	3	3	6	581
DTSF Reclaim Pond Rad	Road	1,804	14,325	16,130	1,813
Admin Terrace	Terrace	3,973	66,523	70,496	1,453
Lower Decline Portal	Terrace	2,791	83,205	85,996	7
Mining Terrace	Terrace	3,392	55,678	59,070	4,427
Process Plant Terrace	Terrace	15,262	246,735	261,997	45,746
Coarse Ore Bin Terrace	Terrace	8,903	145,855	154,758	326
Crusher Terrace	Terrace	5,935	91,645	97,580	24,916
ROM Terrace	Terrace	3,816	72,897	76,713	13,287
Preliminary Ore Stockpile Transition Terrace	Terrace	424	7,138	7,562	204,749
Upper Decline Portal	Terrace	4,663	74,544	79,208	106,959
Tailing Conveyor	Terrace	1,039	12,772	13,811	253
Ventilation Terrace	Terrace	1,059	870	1,928	4,121
DTSF	DTSF	55,184	312,711	367,895	36,322
DTSF Under Drains	DTSF	10,738	60,850	71,588	355,432
Infrastructure Terrace	Terrace	-	-	-	-
Preliminary Ore Stockpile (POS)	Stockpile	-	-	-	3,452
Contact Water Pond	Pond	9,391	14,876	24,307	101,714
Non-Contact Water Pond	Pond	12,068	41,292	87,296	81,088

Item	Type	Excavation in Topsoil (0.3m depth) (m³)	Excavation in Overburden, Weathered Rock or Good Rock (m³)	Total Excavation (from Design) (m³)	Total Fill (from Design) (m³)
Reclaim Pond	Pond	9,842	29,892	79,340	44,351
Terrace Stockpile	Stockpile	16,305	42,787	59,092	12,835
Initial Topsoil Stockpile	Stockpile	-	651	651	353
Ring Road (North)	Road	12,618	77,408	112,237	45,516
Ring Road (South)	Road	11,038	52,376	92,135	65,143
Total		**230,657**	**1,112,146**	**1,938,743**	**1,324,208**

The above quantities pertain to the surface cut and fill activities; but, do not include any waste rock (PAG) from the decline developments.

Figure 18.3 – On-Site Terraces



Source: DRA, 2025

18.5 Camp Site Accommodations

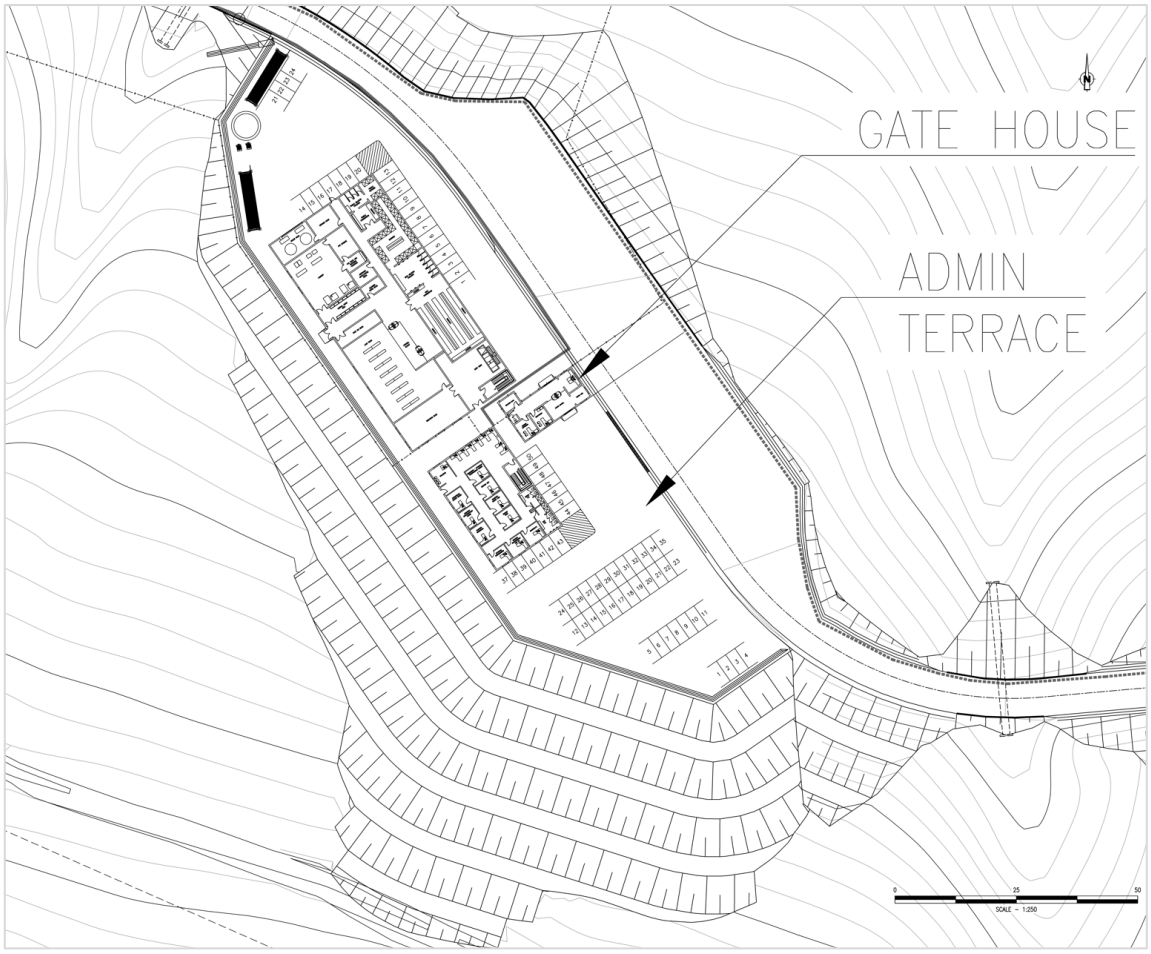
No accommodation camp will be present on-site. Sufficiently established towns are located within reasonable distance and travel time from the project site. Operational staff will be required to relocate to a local town of their choice.

18.6 Infrastructure and Equipment

18.6.1 Administration Area Terrace

Figure 18.4 illustrates the layout of the Administration Terrace, which includes the Guard House, Main Administration Building, and Mine Offices and Mine Dry. It also features designated parking areas, water storage systems, and sewage treatment facilities.

Figure 18.4 – Administration Area Terrace



Source: DRA, 2025

18.6.1.1 Guard House

The Guard House building is sized at 17 m long by 10 m wide to meet the functionality in accordance with the approved Infrastructure List. This Guard House is located on the main access road, to control access to the administration area and project site. The building will be a modular structure and will comprise offices, turnstiles, induction area, waiting area, and ablution.

18.6.1.2 Main Administration Building

The Main Administration Building will be a 25 m long x 16.5 m wide modular double story building structure featuring offices, reception, ablution facilities and open space working areas. The building will be located on the administration terrace near the main entrance guard house on the main access road.

18.6.1.3 Mine Dry and Mine Administration Offices

The Mine Dry and Mine Administration Offices Building will be a 52 m long x 26 m wide modular double story building structure. The Mine Dry will be on the ground floor and contain the lamp room and underground tag control room, boot-wash, laundry, showers, change rooms, ablutions, storage, first aid and induction room. The Mine Administration Offices will be on the second floor and feature the kitchen and canteen for the mine, offices, boardrooms, ablution facilities and open space working areas. This building will be located on the administration terrace near the main entrance guard house.

18.6.1.4 Parking

A demarcated parking area, consisting of 51 parking spaces, sized 5000 mm long x 2300 mm wide is positioned optimally on the administration terrace, outside of the controlled perimeter, to serve as a parking area for staff working in the main administration building and guests.

18.6.1.5 Purified Water

The Water Purification Plant is required to purify tap water to drinking water standards for use at the cafeteria in meal preparation. The Water Purification Plant and Purified Water Tank are located at the northern point of the administration terrace. The Water Purification Plant will receive a steady supply of water from the Tap Water Tank in Section 18.6.7.2 and will treat a maximum 20 m³ of water per day. The Purified Water Tank is sized at 74 m³ of storage capacity, in order to store four (4) days consumption volume, to mitigate any long-term snow events or breakdowns of equipment which might interrupt the supply of tap water.

18.6.1.6 Sewage Treatment

A centralised packaged Sewage Treatment Plant is located at the administration terrace. This treatment facility will treat all sewage from the ablution facilities located in the main administration

offices, mine administration Office, and Mine Dry. This Sewage Treatment Plant will have a maximum treatment capacity of 85 m³/day. The clean discharge, from this sewage treatment plant, will be combined with other treated water sources and be discharged to the environment.

18.6.2 CRUSHER TERRACE

Figure 18.5 depicts the Crusher Terrace and the various components and infrastructure within the Crusher Terrace, detailing the function and layout of each facility.

18.6.3 PRIMARY CRUSHER

The Primary Crusher area will consist of a new retaining wall to create the tip area, a ROM Tip with jockey slab and static grizzly.

The retaining wall will be a Mechanically Stabilized Earth (MSE) wall with pre-cast concrete panel face.

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 43 m long x 25 m wide and will receive new insulated cladding and roof sheeting.

The Primary Crusher Building will be comprised of a ROM Bin, crusher, vibrating grizzly feeder, compressor, and lubrication units. The ROM bin and jaw crusher will be new equipment, whereas the remaining equipment and structures will all be sourced from Ada Tepe. The dust collection system will be sourced from Ada Tepe and be positioned adjacent to the Primary Crusher Building inside a new structure.

The crushed material exits the crusher building to the ore storage facility via a conveyor.

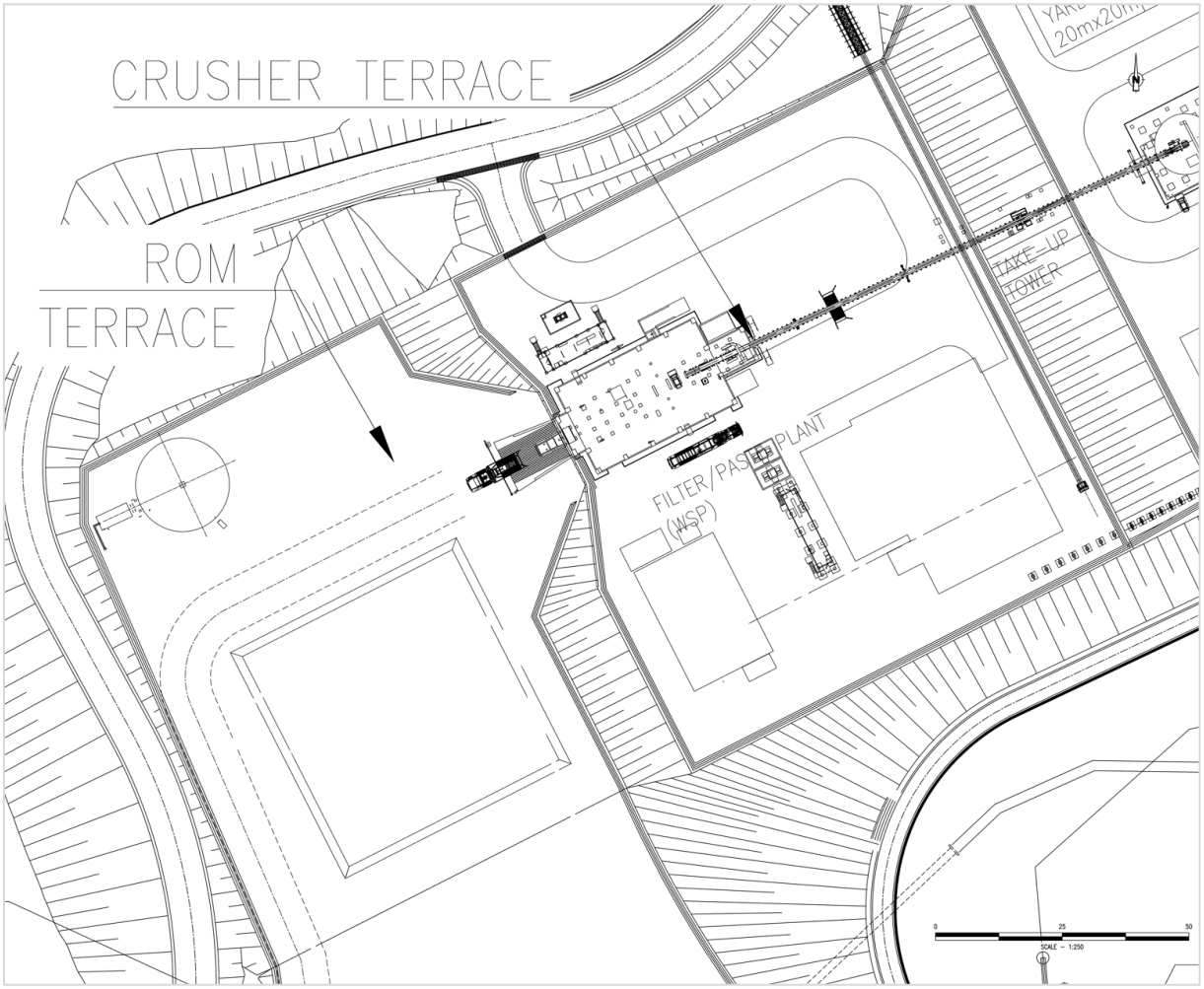
18.6.3.1 Paste Backfill Plant

The Paste Backfill Plant will be contained in a new structural steel building with insulated cladding and roof sheeting. The building size will be 27 m long x 18 m wide and will be located on the crusher terrace, south of the primary crusher building.

18.6.3.2 Tailings Filtration Plant

The Tailings Filter Plant will be contained in a new structural steel building with insulated cladding and roof sheeting. The building size will be 38.5 m long x 28 m wide and will be located on the crusher terrace, southeast of the primary crusher building.

Figure 18.5 – Crusher Terrace



Source: DRA, 2025

18.6.3.3 Tailings Conveyance to DTSF

Filtered Tailings will be conveyed to the DTSF area by conveyor, to minimise traffic congestion in the process plant. The Tailings Filter Discharge Conveyor will receive filter cake material from the conveyor within the Tailings Filtration Plant (4280-CV-204 - Filter Cake Conveyor to Terrace Conveyor) and convey the material along the primary crusher terrace towards the north. This conveyor will be a new horizontal conveyor with a length of approximately 149 m long. It crosses over the north access road and discharge onto the tailings stockpile terrace, created by a steep embankment/retaining wall. This stockpile will be dedicated to the tailings filter cake material. The material handling and transport, from the stockpile to the DTSF, will be done by surface mobile equipment

18.6.3.4 Primary Crusher Discharge Conveyor

The Primary Crusher Discharge Conveyor is 27 m in horizontal length with a vertical lift of 2.35 m. It conveys the crushed material from the primary crusher to the conveyor that transports the coarse ore onto the coarse ore bin feed conveyor. It will be relocated from Ada Tepe with no modifications required to the installation. This conveyor will be re-installed as per existing drawings and details. The head end of this conveyor will have a magnet to remove any tramp iron. Minor modifications will be required to the existing magnet support structure, to allow the magnet to traverse sideways to discharge the tramp iron.

18.6.3.5 Coarse Ore Bin Feed Conveyor

The Coarse Ore Bin Feed Conveyor transports crushed, ROM material from the primary crusher discharge conveyor, on the crusher terrace, to the coarse ore bin on the coarse ore stockpile terrace. The Coarse Ore Bin Feed Conveyor will be relocated from Ada Tepe, with required modifications so that it will be suitable for use at Čoka Rakita. The orientation of the conveyor exiting the crusher building has been adjusted in the new Čoka Rakita layout. The overall length of this conveyor has been reduced by approximately 65 m, from 173 m to 108 m. The vertical lift of the conveyor has also reduced from 20 m to 12 m.

The existing conveyor mechanical equipment will therefore be suitable for re-installation. Whilst most of the structure of this conveyor will be re-used, modifications will be required due to the minor change in the conveyor angle and conditions of the Project site. These modifications to be performed at a conveyor contractors fabrication shop facility, to reduce the number of on-site modifications, and then transported to Čoka Rakita.

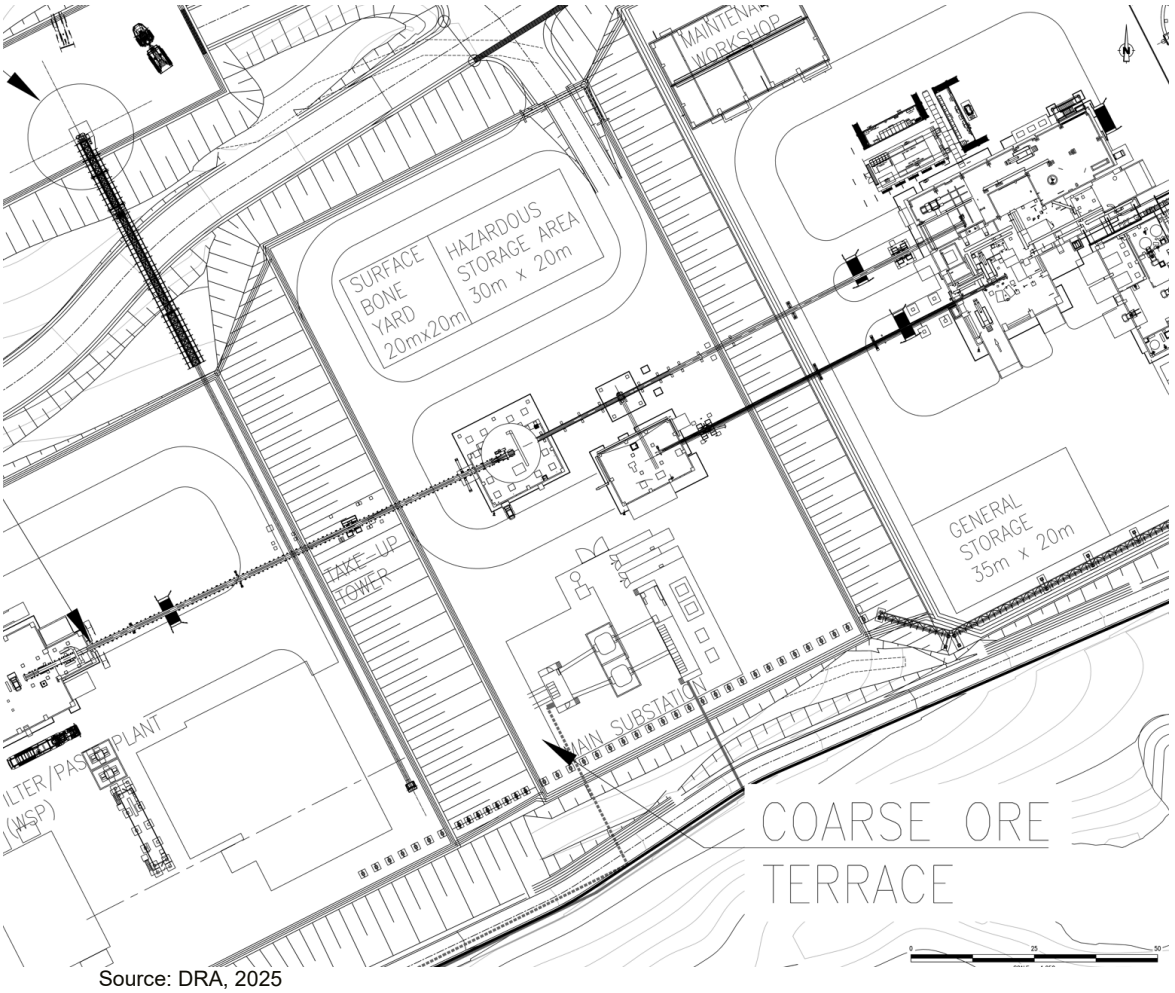
The take-up tower structure will not be relocated from Ada Tepe, it will be newly manufactured, specifically for the Čoka Rakita design. This structure will be positioned between the two terraces

and serve as an access staircase between the crusher terrace and the process terrace. It will also provide access to the top of the coarse ore bin via the conveyor walkway.

18.6.4 COARSE ORE STOCKPILE TERRACE

Figure 18.6 illustrates the Coarse Ore Stockpile Terrace, which outlines the facilities located within the terrace, their layout, and their functions.

Figure 18.6 – Coarse Ore Stockpile Terrace



Source: DRA, 2025

18.6.4.1 Surface Bone Yard

The Surface Bone Yard is a 20 m long x 20 m wide storage area located on the northern side of the coarse ore stockpile terrace, next to the hazardous storage area. This Bone Yard is an allocated space to store damaged equipment and parts in a collective area, for salvage or combined safe

removal. This is the preferred location, as it does not interfere with any other infrastructure, is close to the hazardous waste area, and is serviceable by a ring road around the area.

18.6.4.2 Hazardous Storage Area

The Hazardous Storage Area is a 30 m long x 20 m wide storage area allocated on the northern side of the coarse ore stockpile terrace, next to the entrance of the terrace. This is the preferred location, as it does not interfere with any other infrastructure and is easily accessible by a ring road around the area.

18.6.4.3 Coarse Ore Bin

The Coarse Ore Bin will be a new steel structure located on the coarse ore stockpile terrace. It will receive ore from the primary crusher, via conveyor and feed it via conveyor, to the Autogenous Grinding (AG) Mill. This bin will have a capacity volume of 2,531 metric tonnes and 1,581 m³ in accordance with the process design criteria. It will be 25.37 m high with a diameter of 12.0 m. It will be a mass flow bin, with one load-out point. The Coarse Ore Bin will be fitted with the existing dust collector system from Ada Tepe.

18.6.4.4 AG Mill Feed Conveyor

The AG Mill Feed Conveyor is fed from the coarse ore bin and feeds ore to the AG Mill for further processing. This conveyor will be relocated from Ada Tepe and modified to meet the design needs for the Čoka Rakita process operation. The overall length of this conveyor has been reduced by approximately 7.5m, from 117.5 m to 110 m, while the vertical lift of the conveyor has been reduced by 6 m, from10 m to 4 m.

The existing conveyor mechanical equipment will therefore be suitable for re-installation. Whilst most of the structure of this conveyor will be re-used, modifications will be required due to the minor change in the conveyor angle.

18.6.4.5 Pebble Crusher Building

The Pebble Crusher equipment structure will be relocated from Ada Tepe and modified to meet the design needs for the Čoka Rakita process operations. Pebbles from the AG mill will be conveyed to the pebble crusher; while crushed pebbles will be fed back onto the AG Mill Feed Conveyor via the pebble crusher discharge conveyor. The Pebble Crusher building is 17 m long x 12 m wide and will receive new insulated cladding and roof sheeting.

18.6.4.6 Pebble Crusher Feed Conveyor

The Pebble Crusher Feed Conveyor returns oversized material to the pebble crusher from the AG Mill for secondary crushing. The intention of this conveyor is to utilise most of the equipment from

the existing conveyor (0200-CV-002) from Ada Tepe but also add some of the structural segments, idlers and belting from another conveyor from Ada Tepe 0200-CV-001 to make up the full new length required. Conveyor contractor to confirm in detailed engineering, if trestle heights and connections will need to be modified to suit the new conveyor angle.

This new Čoka Rakita conveyor length is 78 m with a vertical lift of 10.5 m. The existing conveyors from Ada Tepe are 64 m and 16.5 m long respectively.

18.6.4.7 Pebble Crusher Discharge Conveyor

The Pebble Crusher Discharge Conveyor is a short run conveyor of approximately 19 m and will transfer crushed pebbles from the pebble crusher building onto the AG Mill feed conveyor. For this purpose, a small transfer structure is required at the low point of the AG Mill feed conveyor near the coarse ore bin.

18.6.5 PROCESS PLANT TERRACE

Figure 18.7 illustrates the Process Plant Terrace which outlines the facilities located within the terrace, their layout, and their functions.

18.6.5.1 Walkways and Bus Stop Area

The bus drop-off point is on the northeastern corner of the process plant terrace, right adjacent to the Plant Offices, Laboratory and Control Room. From here a winterised waiting area and covered walkway is required for personnel to walk across to the mine services area terrace. The Walkway and Bus Stop Area will be a newly constructed steel structure and will be approximately 115 m long, with waiting area and staircase included.

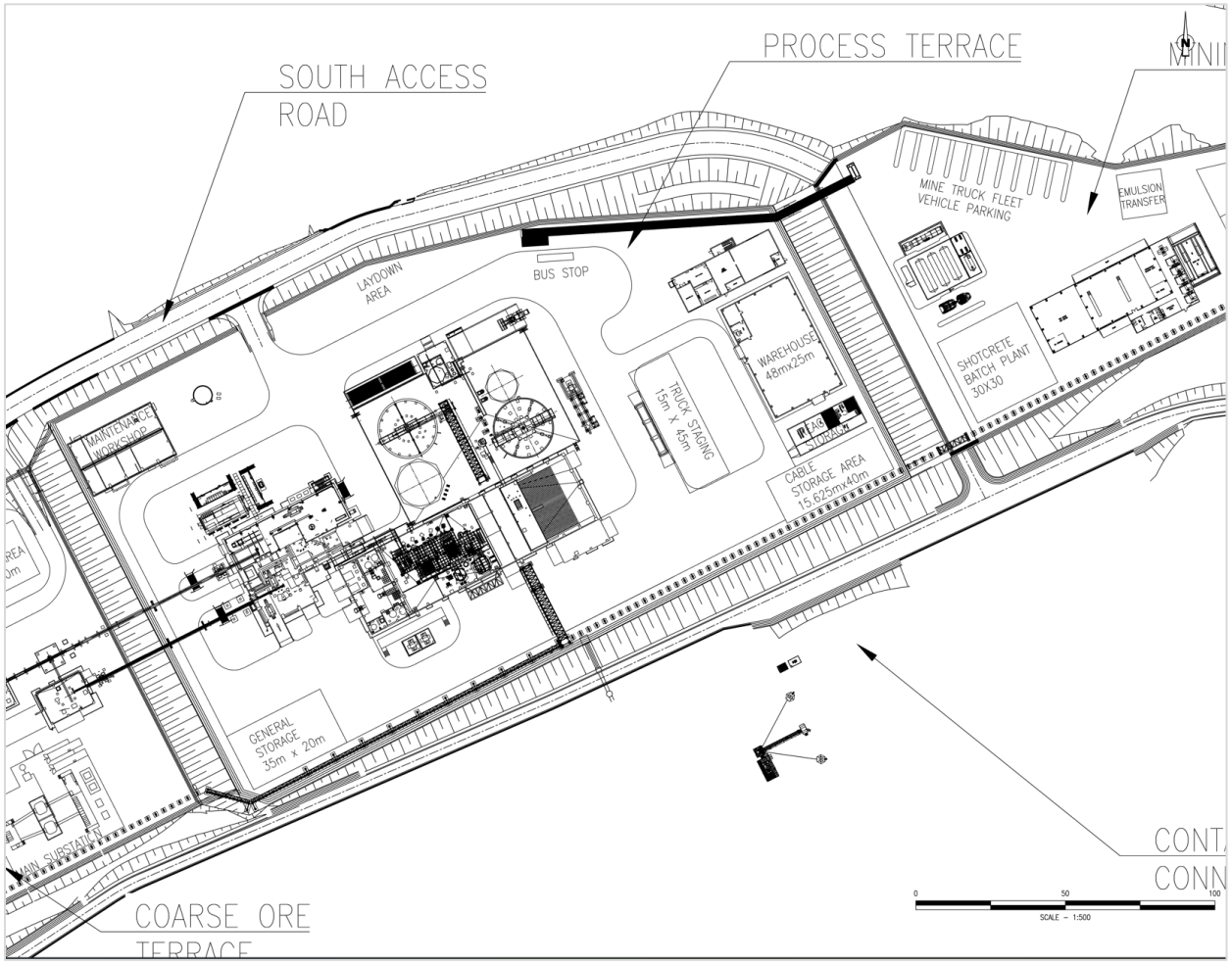
18.6.5.2 Plant Offices, Laboratory and Control Room

The Plant Offices, Laboratory and Control Room building is a double story, modular building located on the northeastern corner of the process plant terrace, with a footprint of 37 m long x 13 m wide. The ground floor will feature the laboratory portion as it will contain heavy equipment, as well as the Mine Rescue, First Aid and Ambulance Bay which require ground floor access. The first floor will be the offices for the process plant, as well as the control room.

18.6.5.3 Warehouse and Laydown Area

The Warehouse and Laydown Area building is on the eastern side of the process plant terrace and will be a new insulated steel structure. The building will be 41 m long x 25 m wide and will feature one office and one lockable storage room.

Figure 18.7 – Process Plant Terrace



Source: DRA, 2025

18.6.5.4 *New Reagents Storage*

The Reagents Storage building is to be located on the eastern side of the process plant terrace, south of the warehouse building and laydown area and will be a new insulated steel structure. The building will be 25 m long x 10 m wide and will feature a liquid reagents storage room, PAX storage room and $CuSO_4$ Flocculant and Coagulant storage room.

18.6.5.5 *Cable Storage Yard*

The Cable Storage Yard will be a 40 m long x 16 m wide fenced storage yard to limit access to the cables and maintain stock control. The Cable Storage Yard is to be located on the eastern side of the process plant terrace, south of the reagents storage building.

18.6.5.6 *Reagents Building*

The Reagents building will be relocated from Ada Tepe and adjusted to suit the conditions of the project site. The Reagents building forms part of the larger Flotation and Concentrate Handling building, which is located on the process plant terrace, south of the AG Mill. The Reagents building is 15 m long x 26 m wide and will receive new insulated cladding and roof sheeting.

18.6.5.7 *Flotation Building*

The Flotation building will be relocated from Ada Tepe and adjusted to suit the conditions of the project site. The Flotation building forms part of the larger Reagents, Flotation and Concentrate Handling building, which is located on the southern side of the process plant terrace. The Flotation portion of the building is 52 m long x 26 m wide and will receive new insulated cladding and roof sheeting. The Flotation cells from Ada Tepe will not be reused, as they will be replaced by Jameson Cells. The existing building size is adequate to accommodate the new equipment.

18.6.5.8 *Concentrate Handling Building*

The Concentrate Handling building will be relocated from Ada Tepe and adjusted to suit the conditions of the project site. The Concentrate Handling building forms part of the larger Reagents, Flotation and Concentrate Handling building, which is located on the southern side of the process plant terrace. The Concentrate Handling portion of the building is 19 m long x 26 m wide and will receive new insulated cladding and roof sheeting. The existing handling system from Ada Tepe will not be large enough to meet the functional requirements of the project. Subsequently the building will require a new addition, to expand upon the original Ada Tepe building size and accommodate the new concentrate loading requirements. The new addition will be 18 m long x 8 m wide. This new area will allow for trucks to drive through and be loaded with concentrate, while the concentrate storage bunkers will be located inside the existing building. Loading the concentrate into the trucks will be performed by a front-end loader.

18.6.5.9 *General Yard*

The General Yard is a storage area located in a previously unutilised space on the southwestern corner of the process plant terrace. The yard is 35 m long by 20 m wide.

18.6.5.10 *Tailings Thickener*

The Tailings Thickener is 20 m in diameter and will be relocated from Ada Tepe. The structural shell/body of the thickener will be replaced. The Tailings Thickener will be located in the central portion of the process plant terrace on a bunded concrete area, which is to the east of the AG and Regrind Mill building and north of the Flotation and Concentrate Handling building. The bunded concrete area is sized to 110% containment volume of the Tailings Thickener.

18.6.5.11 *Drive-In Sump*

The Drive-In Sump is a 25 m long x 10 m wide x 2.1 m deep concrete structure and is located to the northeastern corner of the tailings thickener, bunded concrete area. The entrance is sloped from the east to allow the removal of grit by machine.

18.6.5.12 *Flocculant Plant Building*

The Flocculant Plant is an existing structure that will be relocated from Ada Tepe and adjusted to suit the conditions of the project site. The Flocculant Plant building is 13 m long x 11 m wide and will receive new insulated cladding and roof sheeting. The Flocculant Plant will be located to the north of the tailings thickener bunded concrete area.

18.6.5.13 *Waste Water Treatment Plant*

In the first stage of water cleaning a skimmer and/or clarifier will be used to remove organics (oils) and suspended solids, primarily from mine dewatering water and the tailings reclaim pond. A portion of this water will be re-used in the process plant as clarified water, for duties requiring good quality water such as reagent mixing and gland seals. The balance of water, not used in the process plant, will be treated in the WWTP, for tap water use or release to the environment. The WWTP will include, as a minimum, a reverse-osmosis (RO) step to remove dissolved contaminants and a chlorination step to render the water safe for tap water use. The clarifier will be fed directly from mine dewatering, the CW pond and the reclaim pond. It is expected to clarify and settle the full 2,628 ktpa (300 m³/h) to an acceptable standard for use in the process plant.

As part of the treatment process, the brine effluent from the RO plant will be evaporated and crystallised, for removal from site.

The process plant is expected to use 1,467 ktpa (167.5 m³/h), with the balance of 1,161 ktpa (132.5 m³/h) of clarified water going through additional treatment stages, for use as tap water or released to the environment. The WWTP will be located to the north-west of the tailings thickener.

18.6.5.14 Clarified Water and Process Water Tank

The Clarified Water and Process Water Tank are located in the process plant, near the thickener. inside the thickener concrete bunded area. The Clarified Water Tank will have a storage capacity of 807.6 m³, and the Process Water Tank will have a storage capacity of 934 m³.

18.6.5.15 AG Mill and Secondary Regrind Mill Building

The AG Mill and Secondary Regrind Mill building will be relocated from Ada Tepe and will require significant modifications to suit the functional changes and conditions of the project site. The building is located on the central part of the process plant terrace, to the northwest of the Reagents, Flotation and Concentrate Handling buildings. The building is a L-Shape building with a maximum length of 48 m and a maximum width of 33 m and will receive new insulated cladding and roof sheeting.

This building will house the AG Mill and Regrind Mill which will come from Ada Tepe.

18.6.5.16 Gold Room

The Gold Room will be located in the northern section of the AG Mill and Secondary Regrind Mill building. It will be constructed as a fully secure facility.

18.6.5.17 Maintenance Workshop

The Maintenance Workshop will be an existing building from Ada Tepe, namely the Warehouse building, that will be relocated on the northwestern corner of the process plant terrace. All cladding and roof covering will be new insulated panels. The building footprint is 31 m x 19 m.

18.6.5.18 Tap Water Tank

The Tap Water Tank is located to the north of the process plant terrace near the road entrance, on the open space next to the plant workshop. The tank will supply tap water to the maintenance workshop, process plant area, plant offices, laboratory and control room, warehouse and laydown area, reagents storage and mine truck shop. The Tap Water Tank will have a storage capacity of 134 m³.

18.6.5.19 Truck Staging and Weigh Bridge

The existing Weigh Bridge at Ada Tepe will be relocated to Čoka Rakita and will be positioned on the eastern side of the process plant terrace. The area between the Weigh Bridge and the warehouse and laydown area building will be allocated as a Truck Staging Area (approximately 30 m long x 15 m wide).

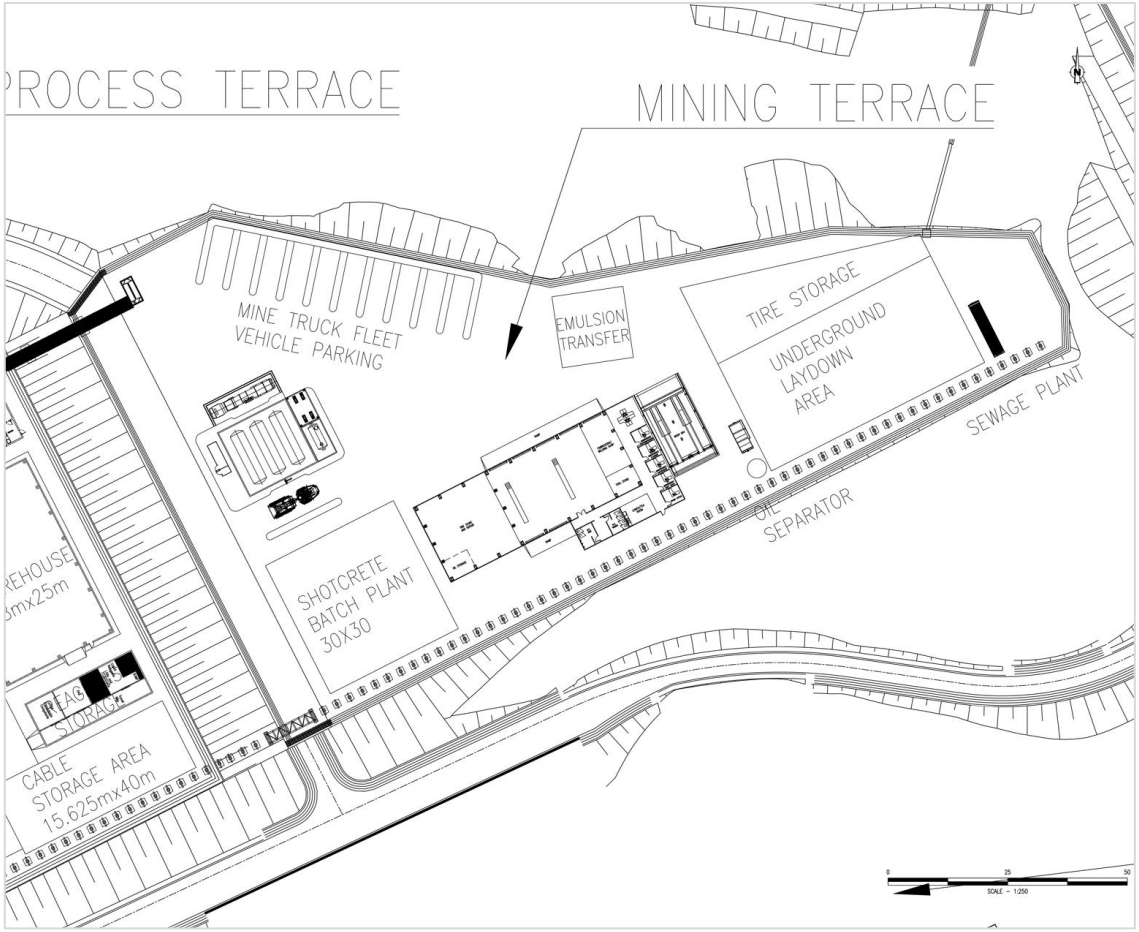
18.6.5.20 Air Compressors and Receivers

A new structural steel building (10 m long x 6 m wide) will be installed in the Reagents building, to house the compressors required for the process plant. The existing Plant and Instrument Air compressors from Ada Tepe will be relocated to this facility. The flotation air blowers from Ada Tepe are not required for Čoka Rakita.

18.6.6 Mine Services Area Terrace

Figure 18.8 illustrates the Mine Services Area Terrace which outlines the facilities located within the terrace, their layout, and their functions.

Figure 18.8 – Mine Services Area Terrace



Source: DRA, 2025

18.6.6.1 Sewage Treatment

A packaged sewage treatment plant is located at the lowest point of the collective process plant terraces. This treatment facility will treat all sewage from the ablution facilities located in the filtered

tailings building; plant offices, laboratory and control room; and mine truck shop. This Sewage Treatment Plant will have a maximum treatment capacity of 15m³/d. The clean discharge, from this sewage treatment plant, will be combined with other treated water sources and be discharged to the environment.

18.6.6.2 Mine Truck Shop and Wash Bay

The Mine Truck Shop building is sized at 52 m long by 24 m wide as per the PFS design and is located on the mine services area terrace, near the lower decline, for operational efficiency. The building will be a new steel structure with insulated roof and side covering. The building will comprise maintenance bays for repair of equipment, a tire store and repair area, a fabrication/welding shop area, a tool store area, an oil storage area, offices, ablution facilities, lunchroom for staff and a wash bay. The wash bay will also be covered to prevent freezing of wash-water during winter months. It will be equipped with a pressure washing system relocated from Ada Tepe and a new oil/water separator.

18.6.6.3 Shotcrete Batch Plant

A 30 m x 30 m area is allocated for the shotcrete plant on the lower terrace. The position is strategical for optimal operation efficacy as it is near the lower decline and ease of delivery for materials via large trucks.

18.6.6.4 Diesel Storage Facility

The Diesel Storage and Filling station comprises three (3) storage tanks, each with a volume of 54 m³ totalling 162 m³, which is equal to eight (8) days of fuel consumption by operations. The proposed storage tanks come as self-bunded units, complete with dispensing station each. The diesel storage tanks are positioned on a bunded concrete area. The vehicle filling area will be a ramped and bunded concrete area to limit environmental concerns with spillage during refuelling. This location is preferable due to the close proximity to the lower decline portal.

18.6.6.5 Emulsion Transfer Area

Emulsion will be stored in an underground facility and not on surface. For this reason, an area is required, on surface, where the emulsion can be received from delivery vehicles and transferred to mining vehicles for transport to the underground storage facility. The Emulsion Transfer Area is located on the lower terrace, north of the Underground Laydown area and east of the Mine Truck Fleet Vehicle Parking. This location is preferable due to the close proximity to the lower decline portal to transport the emulsion underground.

18.6.6.6 Mine Truck Fleet Vehicle Parking

The Mine Truck Fleet Vehicle Parking is a 50 m long x 12 m wide area allocated for the staging of mining vehicles to park. The area is on the northwestern side of the mine services area terrace, near the diesel storage and filling area.

18.6.6.7 Underground Laydown Area

The Underground Laydown Area is a 50 m x 25 m demarcated area to be used by mining personnel. It is positioned on the eastern side of the mine services area terrace, near the lower decline portal.

18.6.7 SITE WIDE INFRASTRUCTURE

18.6.7.1 ROM Pad

The ROM Pad is the terraced area above the Crusher building, where ore material will be stockpiled and handled (from the upper decline and lower decline), prior to loading into the ROM tip. Initially this area will be covered as part of the 85 kt Pre-Production Ore Stockpile.

18.6.7.2 Fire and Tap Water Tank

A storage tank for Fire and Tap Water is required at the ROM Stockpile terrace to house a designated Fire Water storage capacity and supply Tap Water to the administration terrace area's facilities as well as the tailings filtration plant. The tank will have a capacity of 342 m³ for use as Tap Water, while the bottom portion of the tank will have a designated Fire Water storage capacity of 1,222 m³. The total capacity of the tank will be 1,564 m³.

18.6.7.3 Clarified Water Tank

A Clarified Water Storage Tank is required above the upper decline portal to assist with water supply to underground mining activities. The capacity of the tank is 170 m³.

18.6.7.4 Mining Equipment Boneyard

An Underground Boneyard area of 25 m x 25 m has been located on the upper decline terrace, north of the upper decline portal entrance. This Bone Yard is an allocated space for the mining team to temporarily store damaged underground equipment and parts in a collective area, for salvage or combined safe removal.

18.6.7.5 Pre-Production Ore Stockpile

The pre-production ore stockpile terrace will be developed by filling in the ravine, between the upper decline terrace and ROM terrace. This fill be developed using engineered fill, weathered rock and overburden, as identified in the WSP stability memo. This stockpile terrace is required to allow sufficient laydown space for the Pre-Production Stockpile (approximately 85 kt), as required and

sized by DPM and WSP. This location was identified as the ideal location for this stockpile, as it is close to the upper decline portal and the ROM Tip.

18.6.7.6 Pipe Racks

The Pipe Rack structure will be located on the southern side of the collective process plant terraces, with a cross section bridging out north towards the Flotation plant. The bridge of the Pipe Rack runs through the Flotation and Concentrate Handling building and terminates at the Flocculant Plant. This portion of the pipe rack will need to cater for the services for the processing plant.

18.6.7.7 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.

18.6.7.8 Additional Stockpiles

Three stockpile locations have been identified along the main access road. The stockpiles are to be used for NAG overburden material from cut and fill operations during construction, should the need to stockpile surplus material arise.

18.6.7.9 Reclaim Pond

Contact water seepage from the DTSF, along with CW which is unable to directly distributed to the Con Water will be captured in the Reclaim Pond. From the Reclaim Pond, the water will be pumped to the Clarifier or the CW Pond. The water will be treated as required and used in the process or discharged to the environment, as per 18.6.5.13.

This pond will be equipped with two (2) pumps on a floating barge.

18.6.7.10 Contact Water Pond

The CW Pond will be located to the south of the process plant terraces and will be a reservoir of contact water to be treated for use or released to the environment, as per Section 18.6.5.13. The intention of this pond is to be maintained at minimum liquid level during operations.

This pond will be equipped with two (2) pumps on a floating barge.

18.6.7.11 Non-Contact Water Pond

The NCW pond will be located to the south of the process plant terraces, downhill of the CW pond. This pond will contain non-contact surface run-off water and treated water to a level of quality capable of being released to the environment. The pond will be capable of discharging water to the integrated discharge facility from the overflow discharge and from a lower elevation discharge point.

The pond will be typically full, with overflow cascading to the integrated discharge facility and onto to the environment. The pond will, also, be equipped with two (2) pumps on a floating barge, capable of supplying water up to the process plant, if required. This is in accordance with Section 18.6.5.13.

18.6.7.12 Fencing

The Project site will be fenced to prohibit access of the public to the works and hazardous environments. The areas near the process plant and high security areas of 103 ha will be fenced with 1,800 mm high fencing. An estimated 7,800 m of fencing has been identified.

18.6.7.13 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 bifurcated unit. To meet the locational needs of the underground mine, this facility is located above the main spiral ramp, southwest from the process plant, across the existing public road. The Ventilation Facility will have its own access road and terrace.

18.6.7.14 Site Earthworks Balance

The Site Earthworks Balance involves a comprehensive assessment of all cuts and fill activities, to achieve an optimal material equilibrium across the site. This process ensures that the volumes of topsoil, subsoil, and overburden excavated (cut) are efficiently reused in areas requiring fill, minimizing the need for importing or exporting material. By accurately quantifying material types and volumes, the balance study identifies whether on-site resources are sufficient or if external borrow sources or spoil areas are required. The objective is to achieve a cost-effective, environmentally responsible, and logistically practical balance between material excavation and placement, supporting the overall efficiency and sustainability of the Project's earthworks operations.

The following assumptions have been used to develop the earthworks balance of the Project:

- Engineering fills for terraces and roads are 2H:1V using the blending of Overburden (50%) and granular material (50%), as basis for the Feasibility Study phase. This proportion needs to be validated by geotechnical analysis, to ensure it will meet the required geomechanical properties for detail design and construction.
- Granular material for Road structures, 50% of road embankment and finish grading on terraces is coming from Quarries, 50% of terraces embankments will be granular material from site.
- Only Preliminary ore stockpile terrace fill will be using Overburden and Geogrid, as recommended by Geotechnical recommendations from WSP.

18.7 Filter and Paste Plants

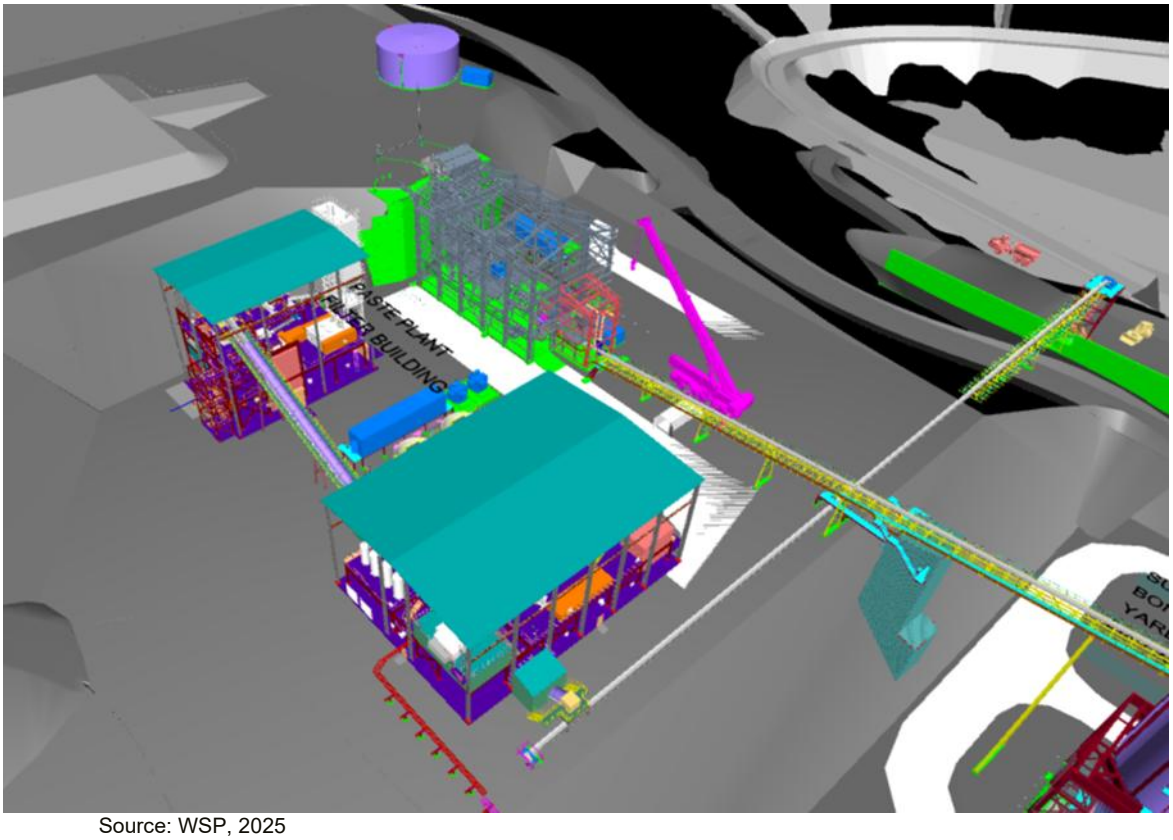
18.7.1 PROCESS

18.7.1.1 Process Design

There are two (2) distinct buildings that contain the process related equipment for the Tailings Filter Plant and Paste Backfill Plant. The plants are located to the south of the Primary Crusher, as illustrated in Figure 18.9.

The Tailings Filter Plant is required to reduce the moisture content of the tailings slurry, so that Filter Cake can be sent to the Paste Backfill Plant for blending and discharge to the underground mine as Paste Backfill; or to be discharged as Filter Cake for disposal to the DTSF. A system of conveyors has been designed to distribute the filtered tailings to the Paste Backfill Plant or to the Filtered Tailings Stockpile at the north side of the crusher terrace and south of the DTSF, as illustrated in Figure 18.9.

Figure 18.9 – Filter Plant (foreground) and Paste Plant on Primary Crusher Terrace



Source: WSP, 2025

The Filtration and Paste production processes, and the types, numbers, and sizes of the various pieces of process equipment required were determined from information provided by:

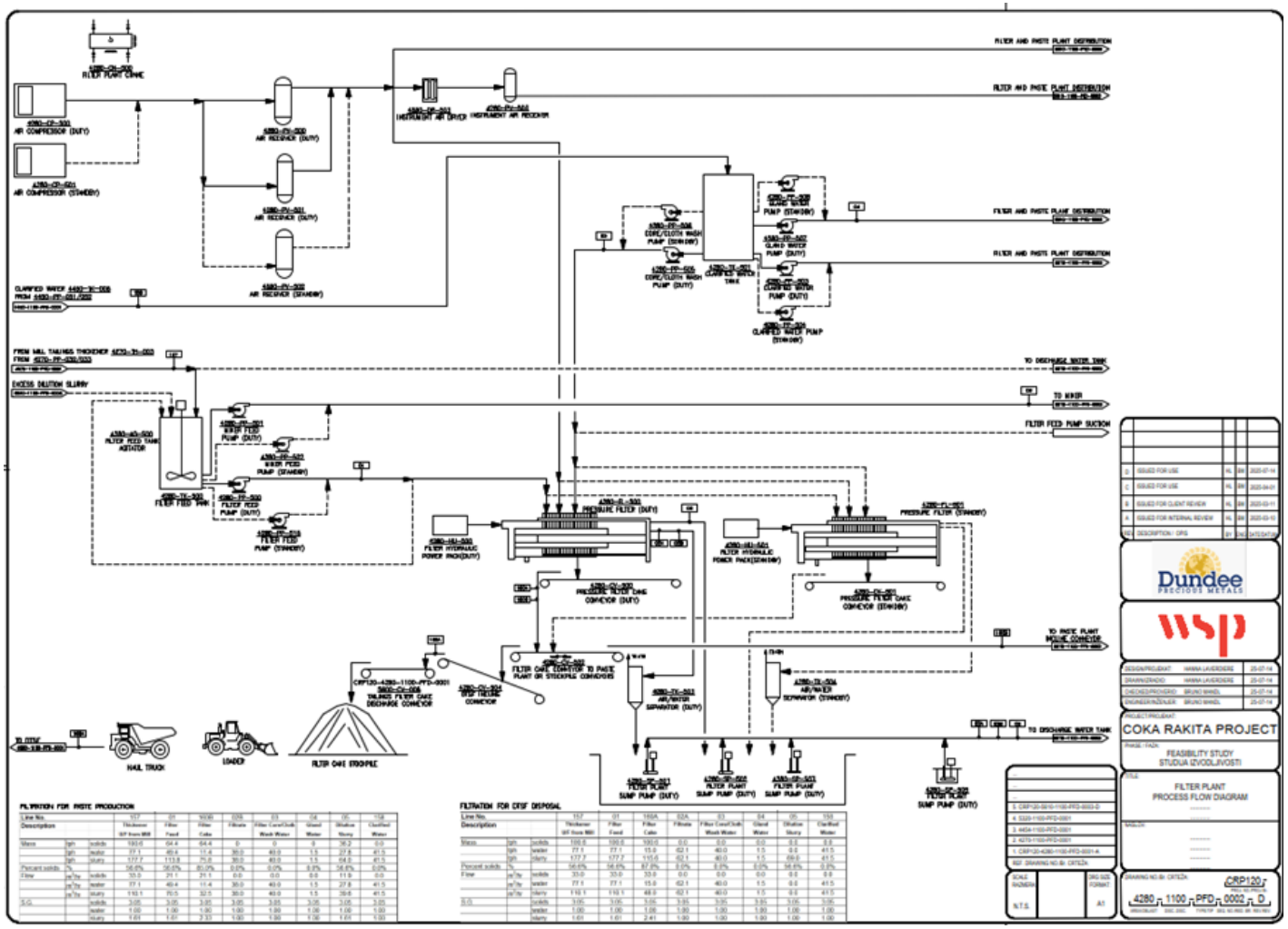
- DPM on laboratory testing results on the mill tailings.
- The LOMP from the WSP underground mine design team.
- DPM mill LOM production plan.
- DPM, DRA, and other groups in WSP working on the Project

The process flow diagrams (PFD) associated with the Filter Cake and Paste Backfill production process are presented in Figures 18.10 and 18.11.

18.7.1.2 Process Equipment Data Sheets

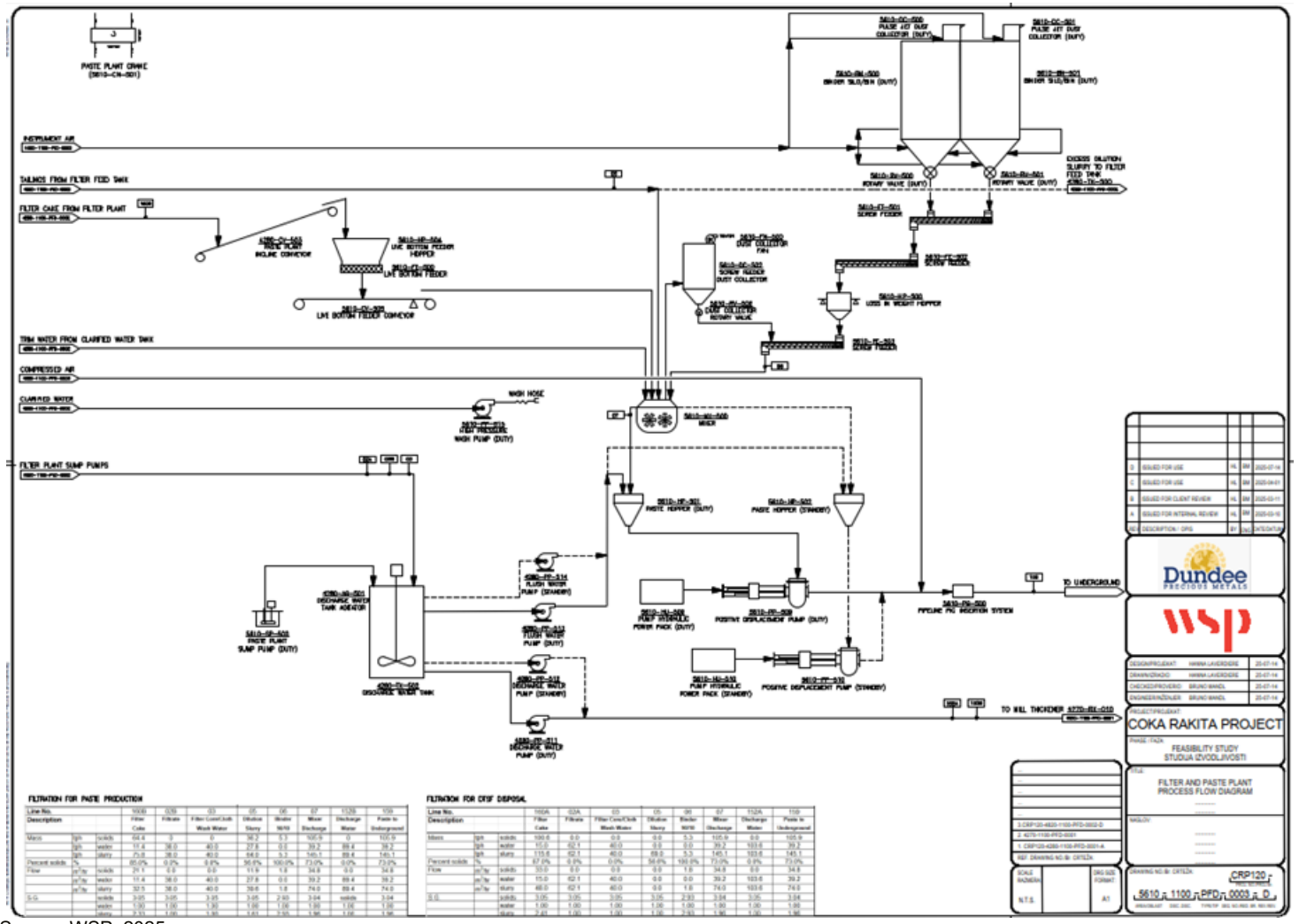
Equipment data sheets listing major mechanical equipment's design requirements were developed based on the plant's process design criteria. The equipment for which data sheets were prepared are presented in Section 18.7.6 of the Mechanical Cost Information.

Figure 18.10 – Filter Plant Process Flow Diagram



Source: WSP, 2025

Figure 18.11 – Paste Plant Process Flow Diagram



Source: WSP, 2025

18.7.2 MECHANICAL

18.7.2.1 Mechanical Design

WSP used the Project mechanical and piping design criteria documents, to inform all design decisions and ensure compatibility with other Project facilities. WSP provided input into these documents where required, to satisfy specific requirements of paste and slurry handling equipment in both the Filter and Paste Plants.

18.7.2.2 2D and 3D Models

2D general arrangement plan and sectional "cuts" of the plants were prepared for both the Filter Plant and Paste Plant to depict the interrelation of the plants' components.

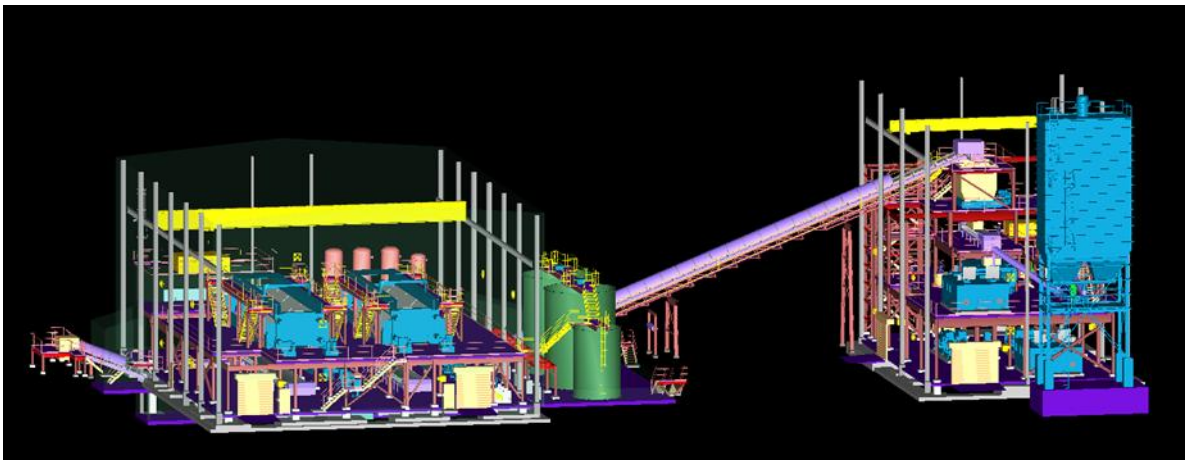
The plants are designed with thought given to such elements as:

- 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 unconfined compressive strength (UCS) room on the same floor and in close proximity to the paste mixer for UCS cylinder casting and breaking activities.
- Lifting wells and exterior overhead doors 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 delivering Filter Cake to the Paste Plant or the DTSF below the ground floor elevation of the plant.
- Providing adequate binder storage capacity to allow uninterrupted operation of the Paste Plant by such things as weather events that could affect delivery of binder.
- Providing emergency egress stairs in case of an unexpected building evacuation.
- Providing comfort facilities for the plant personnel in the Filter Plant such as a washroom and breakroom, as well as an area to monitor key operating parameters of the plants.

A series of screen shots from the Navisworks models are presented in Figure 18.12 to Figure 18.14.

Figure 18.12 is a view looking southeast, with the Filter Plant on the Left and Paste Plant on the right.

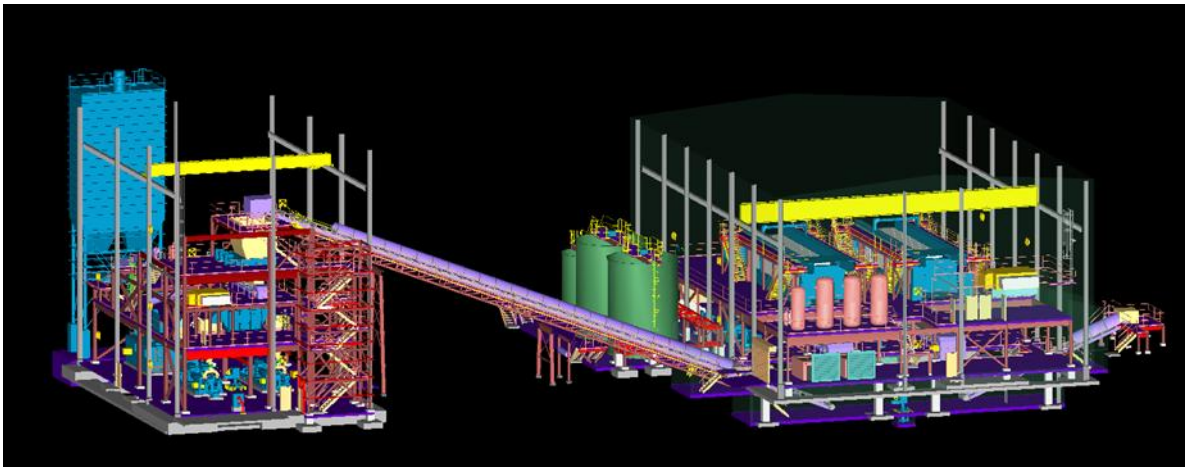
Figure 18.12 – Filter Plant (left) and Paste Plant (right), Looking Southeast



Source: WSP, 2025

Figure 18.13 is a view looking northeast, with the Paste Plant on the Left and Filter Plant on the right.

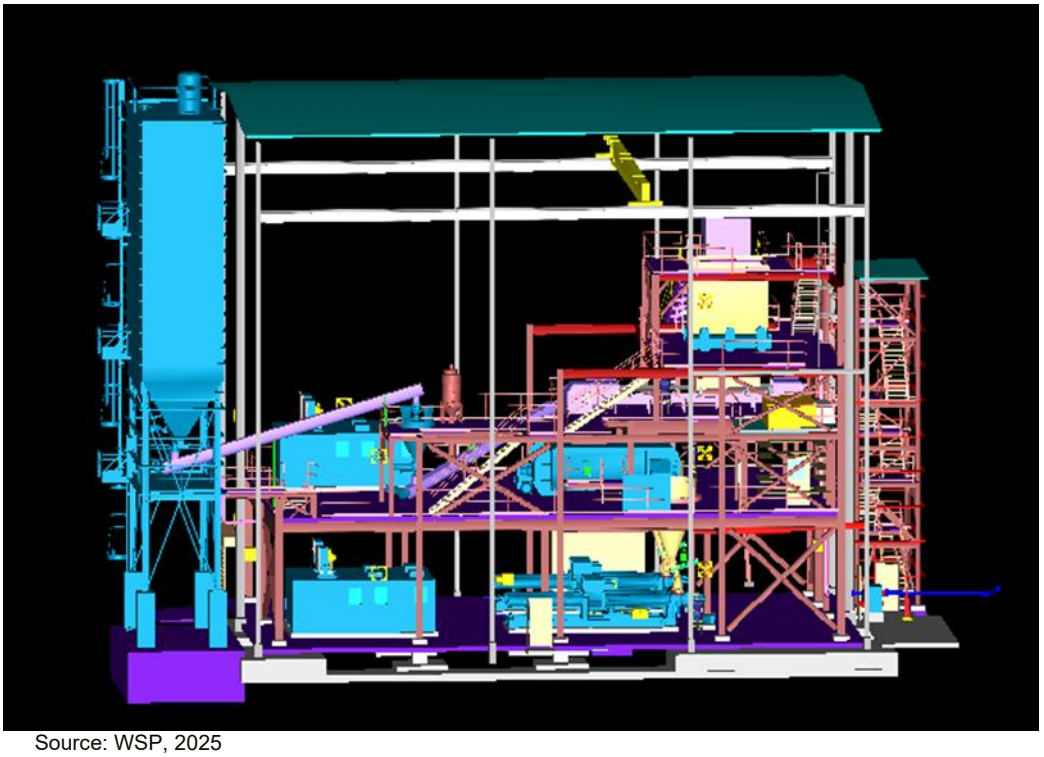
Figure 18.13 – Filter Plant (right) and Paste Plant (left), Looking North



Source: WSP, 2025

Figure 18.14 is a view looking northeast of the Paste Plant.

Figure 18.14 – Paste Plant Looking Northeast



Source: WSP, 2025

18.7.2.3 Paste Plant - Piping and Instrumentation Diagrams

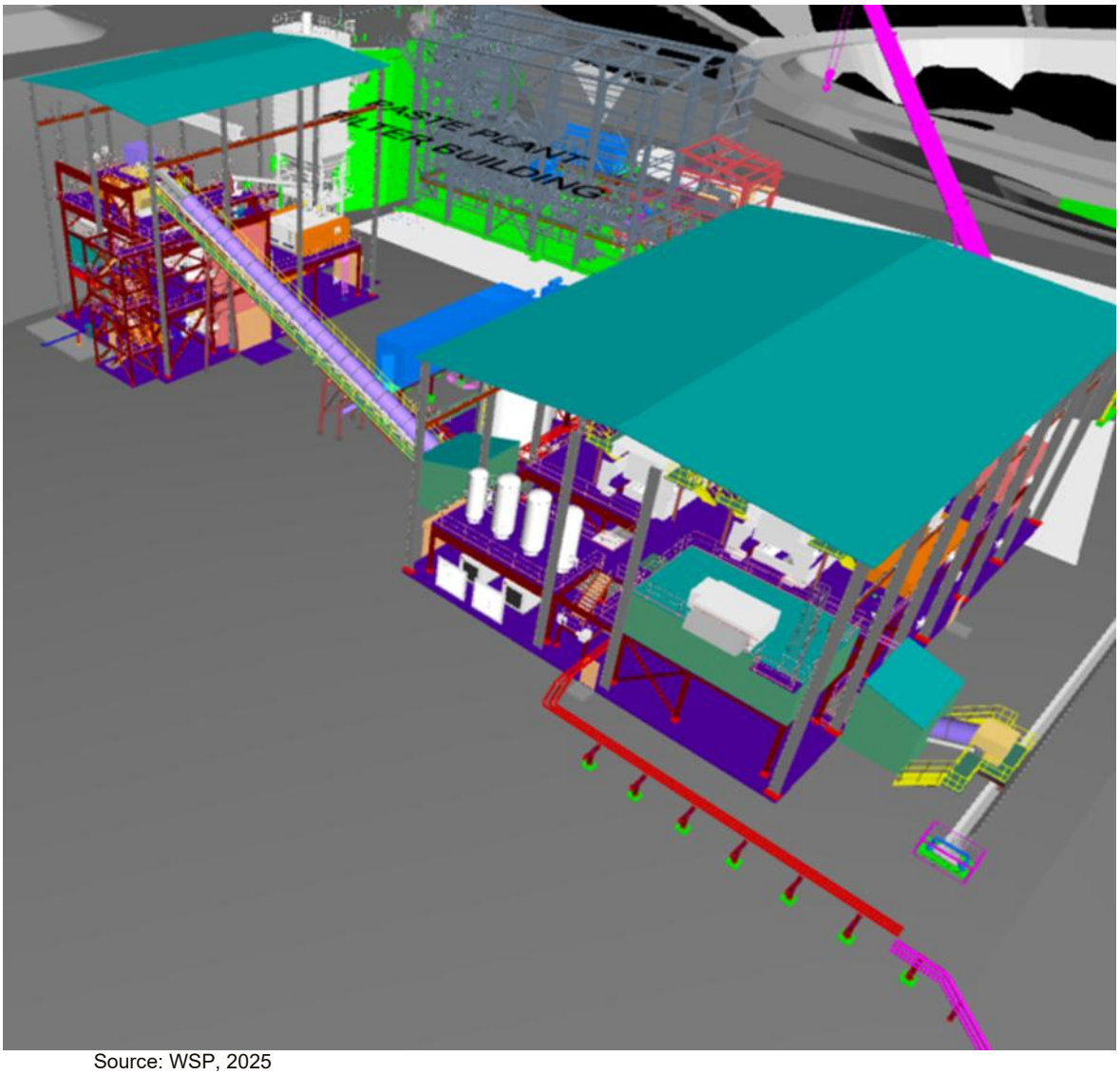
The piping and instrumentation diagrams (P&ID) were prepared for all piping and instrumentation in the plant; and form the basis for the generation of the line list, valve list, and instrument list.

The most important control functions, related to the consistency and quality of the paste leaving the plant for delivery underground, as well as monitoring and controlling the pipeline discharge pressure, are shown on the Paste Plant P&ID(s).

18.7.2.4 Piping Routes

All incoming and outgoing slurry or water arrives or leaves the Filter Plant from pipelines on a trestle originating at the Mill as illustrated in Figure 18.15.

Figure 18.15 – Filter and Paste Plants – Pipelines from Mill on Trestle



Source: WSP, 2025

The plants' major pipelines originate or terminate at the storage tanks located between the Filter and Paste Plants.

18.7.2.5 Filter and Paste Plants - Line and Valve Lists

Line and valve lists were prepared for all pipes and vales within the plants. Lists were populated using data produced by the database driven intelligent P&IDs and updated with size and specification data which was then fed back into the P&ID database. In this phase of the Project all lines were sized according to flow rate, and specifications were selected based on service and expected pumping pressures.

Valves were selected based on allowable specifications from the Project pipe specifications document unless WSP determined that a specialty specification was required.

Two (2) key examples of specialty valves are the paste line flush isolation valves and paste hopper sample port valves. In both cases the presence of cement means that the valves must be able to handle the binding of solids and in the case of the flush isolation, the valves must also be rated for high pressure applications.

18.7.2.6 Fire Suppression System

Fire suppression systems for the Paste Pumps and Pressure Filters hydraulic power packs were requested from Vendors in the Request for Quotation packages. For both the Filter and Paste plants, portable fire extinguishers will be installed.

18.7.2.7 Material Take-Offs

Material take-offs were prepared for the piping elements in both the Filter and Paste Plants. The methodology for the preparation of these take-offs was to leverage the intelligent functionality of the Plant3D software package to produce a raw data set take-off. From the raw data set a specification verification was performed to ensure that all items were compliant with the Project pipe specifications and then the data was transferred to an excel spreadsheet for use in cost estimates.

18.7.3 STRUCTURAL

All earthworks required for and in the area where the Filter and Paste Plants will be established as a function of the site earthworks and terrace development.

18.7.3.1 Structural Design

The Feasibility level Filter and Paste Plant Structural design for most of the equipment, piping, service platforms, and access routes within Filter and Paste Plants was performed in accordance with the Structural Design Criteria for Čoka Rakita.

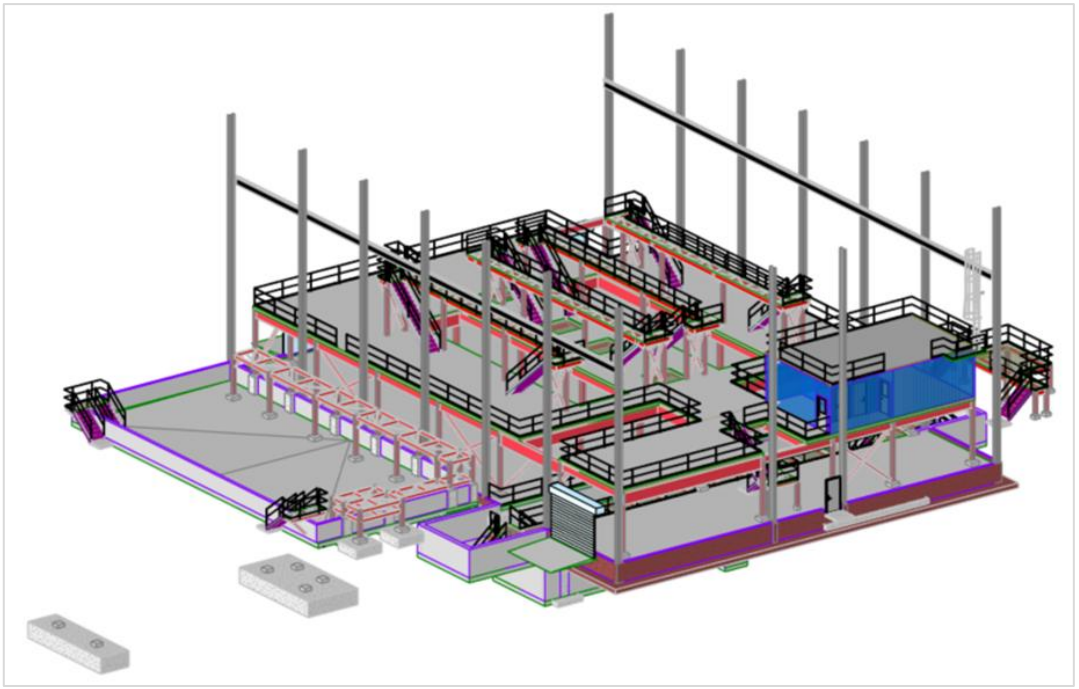
The main structural materials are structural steel, reinforced concrete foundation, and slab on grade. All foundations are designed to bear on engineered fills.

18.7.3.2 2D Drawings and 3D Models

The 2D drawings and 3D models were prepared for internal steel (steel required for equipment, piping and electrical support). Building steel and associated concrete pedestals are shown for completeness only and have not been designed by WSP.

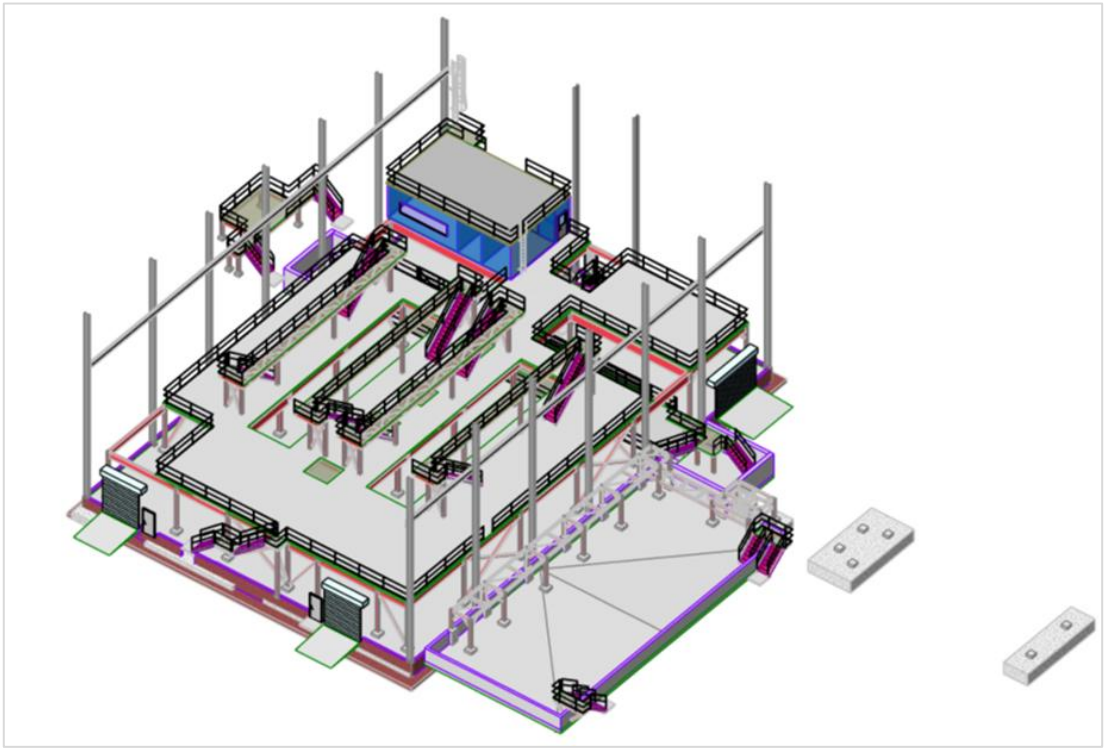
Figure 18.16 to Figure 18.19 present some Filter and Paste Plant structural images.

Figure 18.16 – Filter Plant – View 1



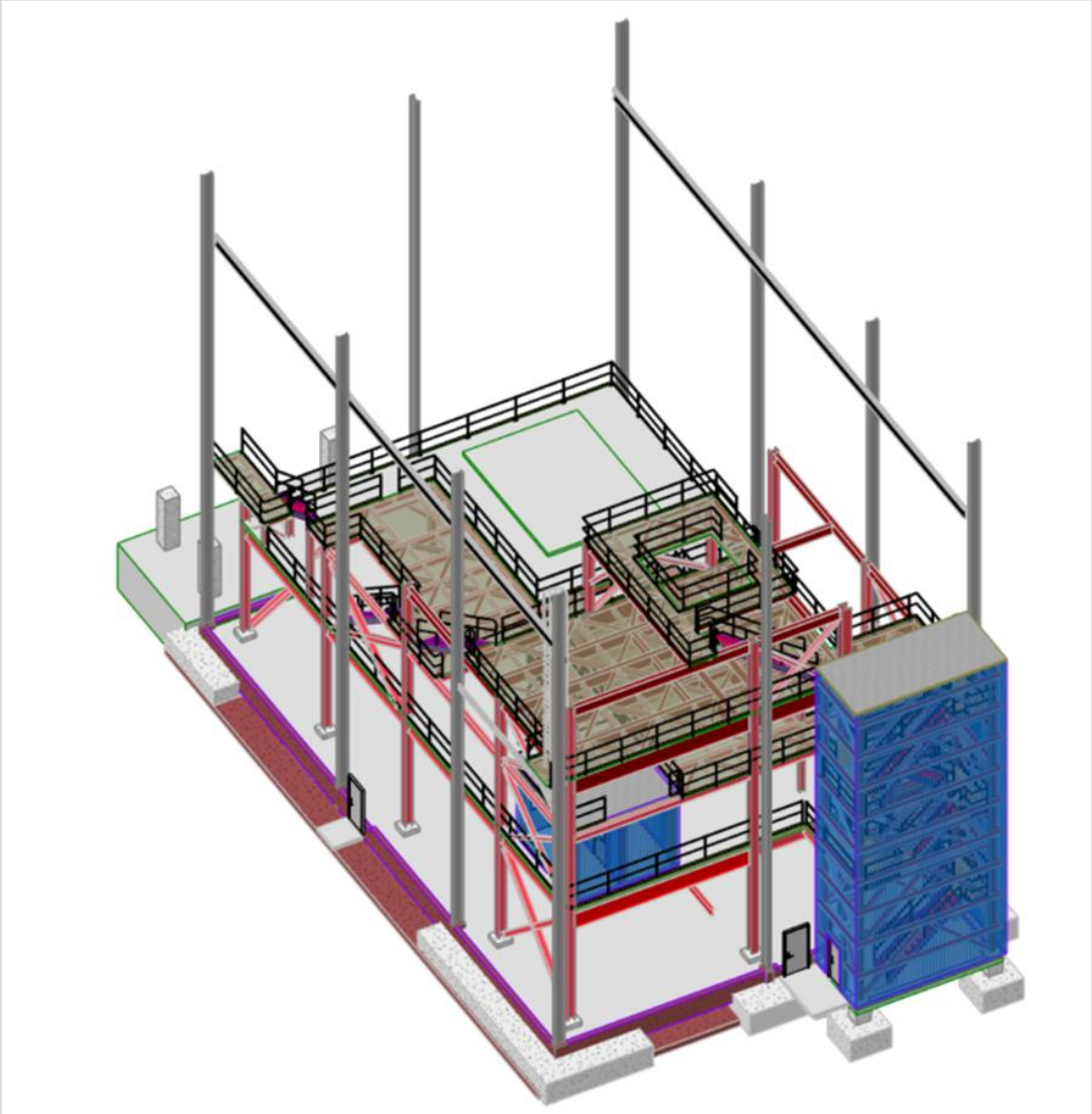
Source: WSP, 2025

Figure 18.17 – Filter Plant – View 2



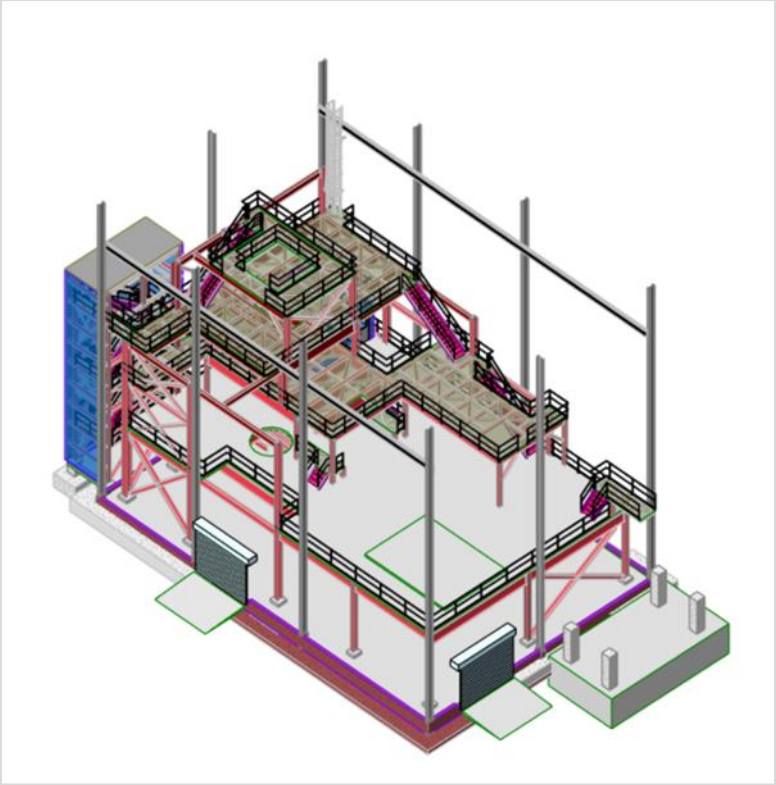
Source: WSP, 2025

Figure 18.18 – Paste Plant – View 1



Source: WSP, 2025

Figure 18.19 – Paste Plant – View 2



Source: WSP, 2025

18.7.3.3 Structural Material Take-Offs

Structural material estimates were compiled for the structural supports inside the Filter and Paste Plants, which are required to install, support, and access the process related equipment.

The quantities compiled in the Filter and Paste Plant structural material take-offs (MTOs) were used for the capital cost determination of the structural steel, concrete and architectural items inside the pre-engineered building envelope.

18.7.4 ELECTRICAL

18.7.4.1 Design Basis

The design of the electrical system for the Filter and Paste Plants was developed based on the DPM supplied documents and available Technical Specifications, ensuring all systems are compliant with safety and regulatory standards. The electrical scope of work included the preparation of:

- Cable Schedule.
- Cable Tray Routing.

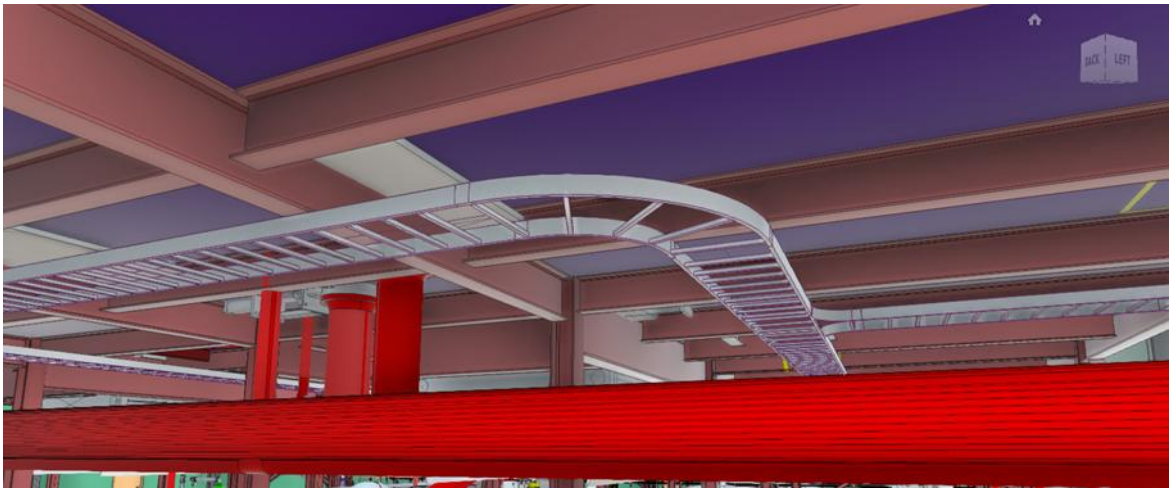
- Material Take-Off.
- Load List.
- Single Line Diagrams.

Neither the Filter or Paste Plants contain such things as power transformers, electrical rooms, or Motor Control Centres (MCCs), so specification related considerations were not examined.

18.7.4.2 Cable Routes

Power and Control cables will be distributed in the plant mainly via cable trays and when required, conduit will be installed to connect electrical equipment, especially motors. The cable trays will be run from the main electrical room, supplied by others outside the plants, and within the plants supported along the beams and racks in the building. An example of the cable tray routing is presented in Figure 18.20.

Figure 18.20 – Filter and Paste Plants – Example of Cable Tray Routing



Source: WSP, 2025

18.7.4.3 Load List

The majority of the Filter and Paste Plant's equipment utilizes a 400V supply, with the only exception being the Positive Displacement Paste Pumps' Hydraulic Power Packs which are supplied with 6,300V.

The estimated electrical Installed Power for all Filter and Paste Plant equipment combined is 2,851 kW. The Load List was based on the Mechanical Equipment List.

For the Filter Plant only, the equipment power is:

- Installed Load – 1,303 kW.

- Absorbed and Annual Consumed Power - for the Filter Plant equipment which largely operates 100% of the time the Mill is operating (excluding Standby equipment), there is 658 kW of absorbed power and 4,790 MW hours of annual power consumption based on 336 operating days per year.
- Connected Emergency Power – 491 kW.
- Operating Emergency Power – 276 kW.

The single largest connected loads are associated with the Pressure Filter Air Compressors at 200 kW each.

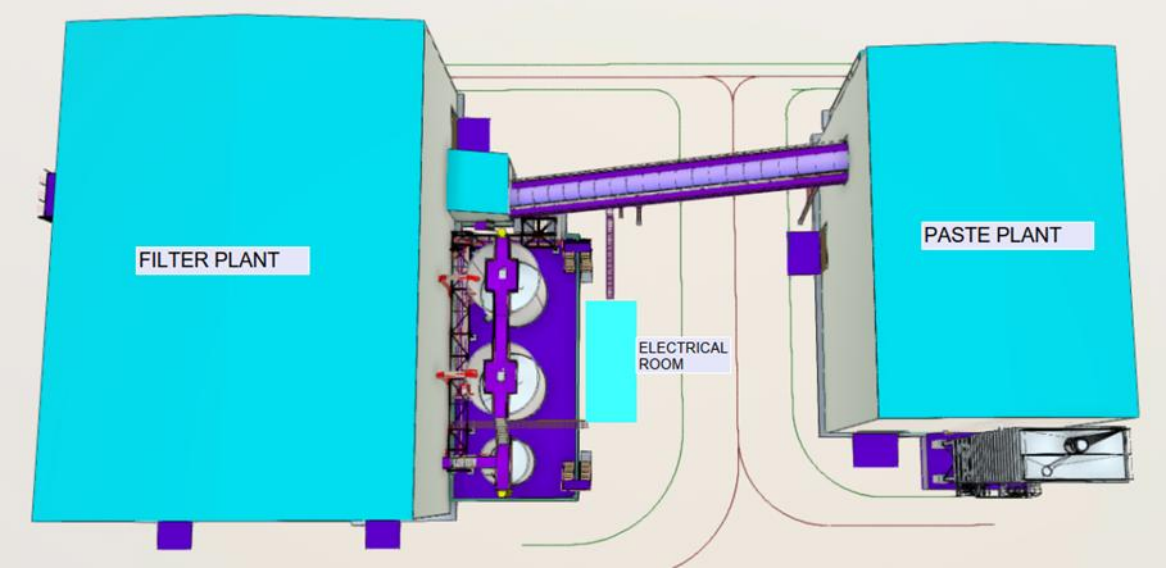
For the Paste Plant only, the equipment power is:

- Installed Load – 1,548 kW.
- Absorbed and Annual Consumed Power - for the Paste Plant equipment which operates approximately 48.7% of the time the Mill is operating, there is – 790 kW of absorbed power and 3,021 MW hours of absorbed power for 52% of 336 operating days.
- Connected Emergency Power – 1,120 kW.
- Operating Emergency Power – 560 kW.

The single largest connected loads are associated with the Positive Displacement Pump Power Packs at 560 kW each.

There is no power transformer station or electrical room in either the Filter or Paste Plants. The power supply infrastructure to service the Filter and Paste Plants is provided via a standalone modular power distribution system (Electrical Room) located between the Filter and Paste Plants, as illustrated in Figure 18.21.

Figure 18.21 – Filter and Paste Plants Electrical Room (by Others)



Source: WSP, 2025

The main Electrical Room design was not part of WSP's Scope.

18.7.4.4 Single Line Diagrams

The Single Line Diagrams for the Filter Plant and Paste Plant were developed on the basis of the Mechanical Equipment List, Electrical Load List, and Cable Schedule.

18.7.4.5 Filter and Paste Plant Electrical Material Take-Offs

The electrical materials estimates are for the materials required inside the Filter and Paste Plants to provide power to the process equipment, lighting, heating, and maintenance related activities.

The quantities listed in the Filter and Paste Plant electrical MTOs were used for the capital cost determination of the electrical items inside the Filter and Paste Plants.

18.7.5 INSTRUMENTATION

18.7.5.1 Instrumentation and Control Design

The Instrumentation and Control (I&C) design detailed in the Feasibility level P&IDs and I&C deliverables outlines the minimum specifications needed for the operation of the Filter and Paste Plants, see Table 18.3.

The I&C design considered best practices sourced from similar applications, the specifications documented within the Project's Instrumentation & Control Design Criteria, and the topics discussed during the Feasibility HAZOP.

Table 18.3 – I&C Features Represented in the P&ID

Controls implemented on the Process Control System (PCS)
Relative location of instruments on vessels and pipelines
Control loops and major interlocks and critical alarms
Valve failure positions
Graphics symbols and tags for field instrumentation and final control devices

The Filter and Paste Plant Feasibility Level I/O and Instrument Lists documents were developed to facilitate the completion of preliminary material take-offs, list preliminary process details, and device signal types.

Similar I/O and Instrument Lists were requested from the third-party mechanical vendors. This was done to establish the preliminary material take-offs and confirm the tie-in design needed to connect the skid PLC with the plant PLC. A list of the third-party mechanical vendors is presented in Table 18.4.

Table 18.4 – Filter and Paste Plants – Third-Party Mechanical Vendor Packages

Package Name	Plant
Pressure Filter package	Filter Plant
Conveyor package	Filter Plant & Paste Plant
Binder Silo package	Paste Plant
Paste Mixer package	Paste Plant
Positive Displacement Paste Pump package	Paste Plant
Live Bottom Feeder package	Paste Plant

The MTOs were incorporated into the Capex estimate for the Filter and Paste Plant.

18.7.5.2 Field Instrumentation Cable Management

WSP has implemented the following features for the proposed home-run instrumentation cable routed between the respective PLC Panels located in the electrical room and the instrumentation junction boxes installed in the field:

- Separation between the Paste Plant Network Architecture and the Filter Plant Network Architecture.
- Sufficient clearance between instrumentation and electrical cables.

18.7.5.3 Network Architecture

The network architecture for Filter and Paste Plants has been primarily designed to seamlessly integrate with the preliminary plant-wide automation scheme, consisting of a surface ring topology. This approach will ensure operations are uninterrupted, support high-speed data exchange, and preserve the integrity of process control functions with smart devices. The system architecture also includes a dedicated HMI in the Filter Plant for both the Filter and Paste Plants. This design will allow the effective commissioning of both Plant and allow operators to monitor, but not control, the function of the Plant's equipment.

18.7.6 PROCUREMENT MAJOR EQUIPMENT AND CONSUMABLES

WSP has compiled specifications in the form of Vendor Bid Packages, and received bids from DPM sourced and approved Vendors to determine the expenditure for major mechanical equipment:

- Pressure filters.
- Paste mixer.
- Positive displacement, paste pumps and pipeline PIG system.
- Cement binder system.
- Live bottom feeder.

For each of these Vendor packages, WSP completed a technical bid evaluation, including an economic evaluation, on all Vendor bids received from DPM; and provided recommendations on technical suitability in combination with costs.

WSP compiled equipment data sheets (DAT) and pricing and payment schedules (PPS) for process equipment for both the Filter and Paste Plants. These DATs and PPSs were submitted for inclusion as addendums to the integrated Vendor packages that were issued to potential suppliers. The process equipment included the following items:

- Conveyors.
- Slurry pumps.
- Water pumps.
- Sump pumps.
- Overhead cranes.
- Tanks and hoppers (platework).

For the integrated packages where DRA completed a technical bid evaluation and provided a recommended Vendor to DPM, WSP also, assessed that Vendor's bid as it specifically applied to the equipment in the Filter and Paste Plants and provided their recommendation.

For general vendor packages, not Filter Plant or Paste Plant specific equipment, issued to the vendors by DPM, WSP reviewed the recommended Vendor bids, as well as any other received bid, and interpreted or derived unit costs to satisfy the Filter and Paste Plant equipment requested, in some instances applying the “0.6 Rule” if practical.

The final equipment capital costs were used in WSP’s Capex cost tabulation.

The MTO(s) generated by the various disciplines listed above were also supplied and used in WSP’s Capex / SUSEX cost tabulation.

WSP supplied the quantities of consumable materials to DRA for inclusion in their operating cost compilation of the Project, inclusive of power consumption and cement binder for paste backfill.

18.8 Shotcrete and CRF Slurry Batch Plant

The Feasibility-level design of the plant is preliminary in that it is based on WSP’s design experience on similar plants and two (2) Vendor quotes.

The Batch Plant will be modular in construction with separate, but integrated modules serving to produce shotcrete and CRF slurry in an “on demand” basis to suit the mine’s needs; delivered into the mine via transmixers, through both the mine upper and lower declines.

The purpose of the plant is two-fold, to provide:

- Shotcrete for mine geotechnical support during preproduction and production development, and
- CRF binder slurry to be mixed with mine development waste rock and used as backfill during mine production

The Batch Plant will be located on the Mine Truck Shop (lower) terrace, which offers a reasonable compromise between travel distance to both the upper and lower declines, and accessibility for the delivery of binder and shotcrete aggregate to the site.

18.8.1 PROCESS

The process flow diagram (PFD) and materials mass balance (MB) of Batch Plant process were developed based on the developed design criteria

The raw material supplied to the processes are:

- Clarified water supplied from the mill’s clarified water tank.
- Sand and cement binder supplied from off-site sources by others.

Any water generated in the process (i.e., equipment cleaning), will be collected and pumped as discharge water to the mill tailings thickener.

18.8.2 STRUCTURAL

All earthworks required for and in the area where the Batch Plant will be located will be done by others and was not considered and was not considered in the structural evaluation.

Since the Batch Plant will be of modular construction, all the necessary structural components will be supplied by a Vendor.

The only additional structural elements required would be:

- The concrete footings for structural steel support such as the binder silos aggregate bins and conveyors.
- A concrete slab for containerised modules.
- A concrete bunker to receive and store the aggregate required for the production of shotcrete.

Structural drawings were not produced but a concrete material take-off (MTO) was prepared, based on Vendor supplied information.

18.8.3 MECHANICAL

A preliminary Major Equipment List (MEL) was developed, based on the Vendor quote supplied information as well any ancillary equipment that was required but considered outside the scope of the Vendor.

18.8.4 ELECTRICAL

A preliminary electrical load list was developed from the MEL.

The electrical power supply required for the Batch Plant will be supplied by others to the electrical room(s) in the modular plant.

For the Batch Plant the equipment power is:

- Installed Load – 210 kW.
- Absorbed Power - 156 kW
- Annual Consumed Power - 965 MW hours
- Connected Emergency Power – 0 kW.
- Operating Emergency Power – 0 kW.

Based on the load list, preliminary single line diagrams were developed.

An electrical MTO was not developed since as stated earlier, the Batch Plant will be a modular system supplied by a Vendor and a functional package as requested, would include all necessary electrical equipment.

18.8.5 INSTRUMENTATION

Instrumentation documents such as instrument lists and MTO were not developed since, as stated earlier, the Batch Plant will be a modular system supplied by a Vendor; and all the necessary documents and components will be supplied by the Vendor in a functional package.

18.8.6 PROCUREMENT MAJOR EQUIPMENT AND CONSUMABLES

WSP has compiled specifications in the form of a Vendor Bid Package and received bids from DPM sourced and approved Vendors to determine the expenditure for a modular integrated Shotcrete CRF Slurry Batch Plant.

WSP assessed the Vendor offerings and found them to be technically acceptable. The price quoted by the selected Vendor was used in the Capex estimate for the Project.

18.9 Mine Waste Management

The FS LOMP and Mining strategy indicate that 59% of mine waste will be stored on the surface, with the remaining waste stored underground via a paste backfill system. Mine waste at the Čoka Rakita Project will be managed via two (2) processes:

- 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.
- Filtered tailings will be transported via conveyor to a loading area, then trucked to the deposition point, progressively covering the PAG waste rock.

Detailed design, transportation and construction information for the DTSF is present in Section 18.10.

18.9.1 PRELIMINARY ORE STOCKPILE

The optimisation, design and stability analysis of the Preliminary Ore Stockpile was completed by WSP. The key design criteria are presented in Table 18.5 with the design summary details presented in Table 18.6.

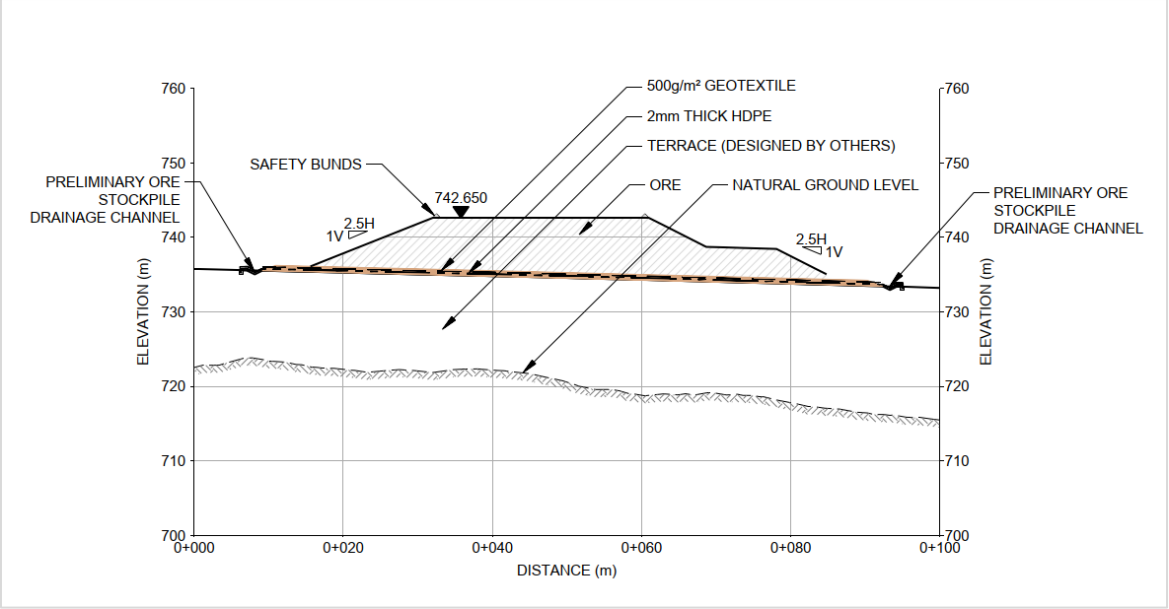
Table 18.5 – Preliminary Ore Stockpile Design Requirement

Description	Quantity (t)	Quantity (m^3)	Comment
Preliminary Ore Stockpile Capacity	100,000	50,000	FS Design.

Table 18.6 – Preliminary Ore Stockpile Design Summary

Element	Value
Ore Storage Volume Required (m³)	50,000
Total Stockpile Height (m)	11.5
Stockpile Footprint (m²)	12,800
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 18.22 – Preliminary Ore Stockpile Design



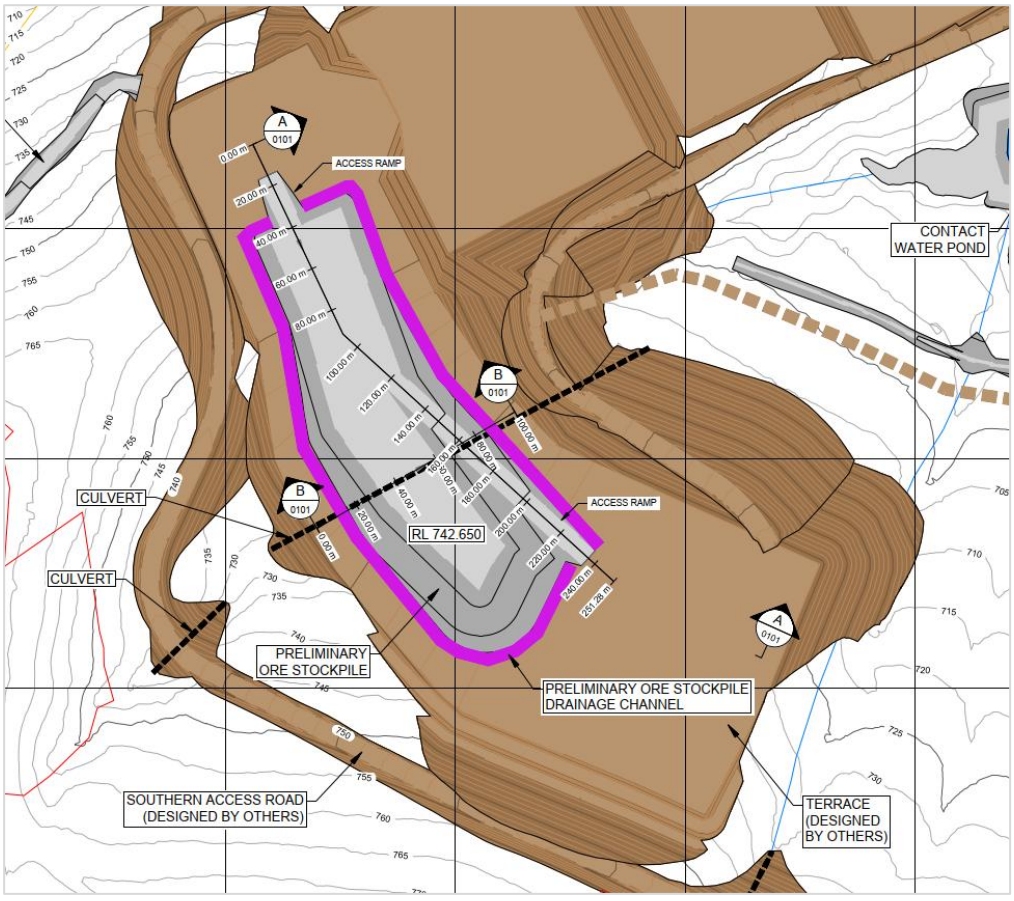
Source: WSP2025

18.9.1.1 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.

Figure 18.23 – Preliminary Ore Stockpile Design



Source: WSP, 2025

18.9.1.2 Liner

The preliminary ore stockpile will have a 2 mm HDPE liner covering 12,800 m² to prevent potentially acidic leachate from seeping into the ground. The liner will be protected with a 500 g/m² geotextile and 200 mm granular fill protection layer. Perimeter drainage channels will capture runoff and filter hazardous materials into the terrace drainage system.

18.9.1.3 Stability Analysis

Slope stability models have been developed using GeoStudio SLOPE/W limit equilibrium software, using the Morgenstern-Price method of analysis with optimised surfaces. The stability of the slopes within the scheme has been checked in accordance with BS EN 1997-1, Eurocode 7 (EC7), Rules on Technical Requirements for the Surface Exploration of Mineral Resources Deposits (SEM) and a Seismic Pseudo-static scenario.

The stockpile terrace will be built from overburden material, with stability analysis parameters inferred from the Ground Investigation Interpretive Report (WSP, 2025). Reinforcement of the slopes through a series of 100 kN/m tensile strength geogrids was identified to be required. The results are presented below in Table 18.7. Note, no geogrid was required for Section 2.

Table 18.7 – Preliminary Ore Stockpile Slope Stability Summary

Description	Scenario	Minimum Required FoS	Calculated FoS (MorganStern-Price) [1]
Section 1	DA1-C1	1.0	1.4
	DA1-C2	1.0	1.1
	Seismic	1.0	1.0
	SEM/SLS	1.3	1.4
Section 2	DA1-C1	1.0	1.4
	DA1-C2	1.0	1.2
	Seismic	1.0	1.1
	SEM/SLS	1.3	1.6
Section 3	DA1-C1	1.0	1.2
	DA1-C2	1.0	1.1
	Seismic	1.0	1.0
	SEM/SLS	1.3	1.3
Section 4	DA1-C1	1.0	1.1
	DA1-C2	1.0	1.0
	Seismic	1.0	1.0
	SEM/SLS	1.3	1.4

1. 100 kN/m strength geogrids used. Length and area vary per section. For full analysis see the updated stability analysis memo (WSP, 2025).

Additional testing will be recommended between the FS and the detailed design to optimise the design of the slopes analysed.

18.9.2 TOPSOIL AND OVERBURDEN STOCKPILES

18.9.2.1 Design Inputs

Topsoil stockpiles are required to provide total storage for all excess site-won topsoil excavated during the LOM. Table 18.8 presents the general assumptions regarding topsoil generation as well as the basic design criteria for stockpile design as specified by DPM.

Table 18.8 – Topsoil Stockpile Design Criteria

Criteria	Value	Unit	Reference/Comment
Topsoil thickness in situ	0.3	m	From previous project experience and initial FS 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

18.9.2.2 Optimisation

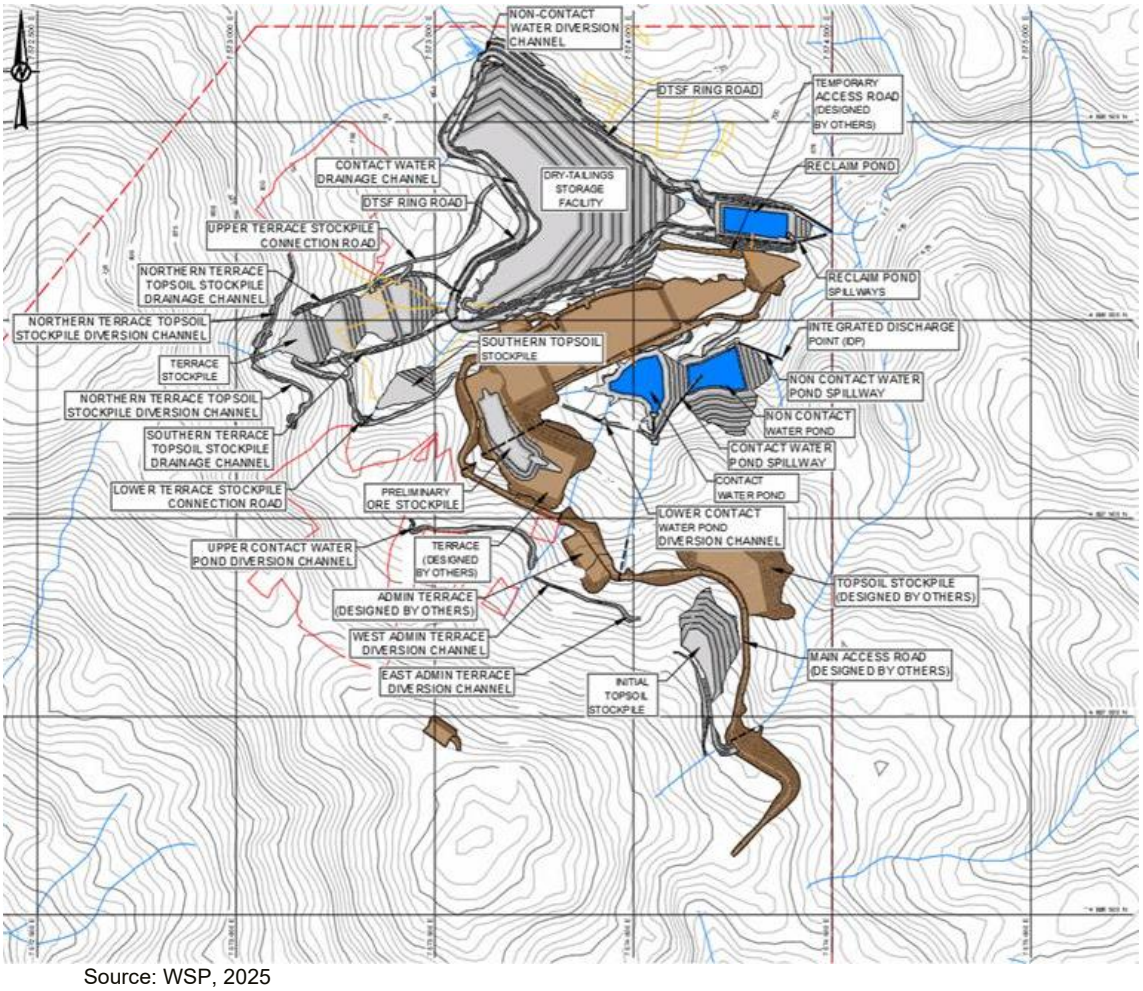
Table 18.9 presents a summary of the finalised FS topsoil and overburden stockpile designs.

Table 18.9 – Čoka Rakita Topsoil Stockpile Quantities

Stockpile ID	Volume (m³)	Material Storage
Terrace Stockpile	554,200	Combination (Topsoil and Overburden)
Initial Topsoil Stockpile	103,000	Topsoil
Southern Stockpile	73,900	Optional
Additional Stockpile	470,400	Overburden

The locations of these stockpiles are presented in Figure 18.24. The Southern and Additional stockpiles (blue) are shown as optional storage.

Figure 18.24 – Optimised Topsoil Stockpile Locations



Source: WSP, 2025

18.9.2.3 Stability Analysis

The Serbian Standard SRPS U.C5.020, 1980) was used to assess the stability of topsoil stockpiles. The stockpiles were also 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. Reinforcement of the slopes through a series of 100 kN/m tensile strength geogrids was identified to be required.

The findings of the stability assessment are presented in Table 18.10.

Table 18.10 – Topsoil Stockpiles Slope Stability Summary

Description	Scenario	Minimum Required FoS	Calculated FoS (MorganStern-Price)
Terrace Stockpile – Global Failure[1]	CDA Static	1.5	1.7
	CDA Seismic	1.0	1.1
	SEM/SLS	1.3	1.7
Initial Topsoil Stockpile	CDA Static	1.5	1.5
	CDA Seismic	1.0	1.0
	SEM/SLS	1.3	1.5

1. The results presented are for global failure when topsoil is stockpiled. Further analyses were run on the individual slopes, with minimum 100 kN/m tensile strength geogrid included in the design. For the full results please refer to the updated stability analysis memo (WSP, 2025).

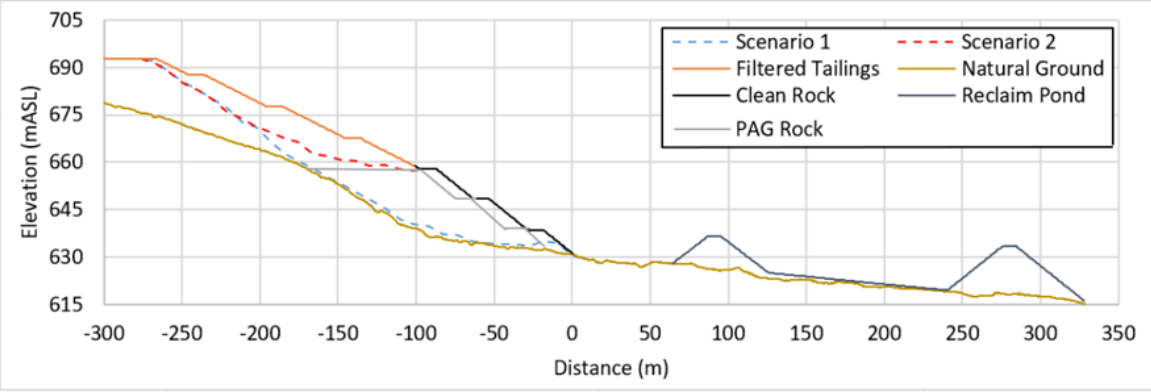
Further refinement of the design will be undertaken in the next phase of the Project.

18.10 Dry Tailings Storage Facility

18.10.1 Dam Break Analysis Summary

A Breach Analysis for the DTSF during the operational phase has been conducted. As there is no dam confining the tailings, the assessment does not include a breach process that involves loss of containment through flow conditions, rather a slumping of compacted filtered tailings. Two (2) credible failure Scenarios were adopted for the analysis and shown in Figure 18.25.

Figure 18.25 – Failure Scenarios



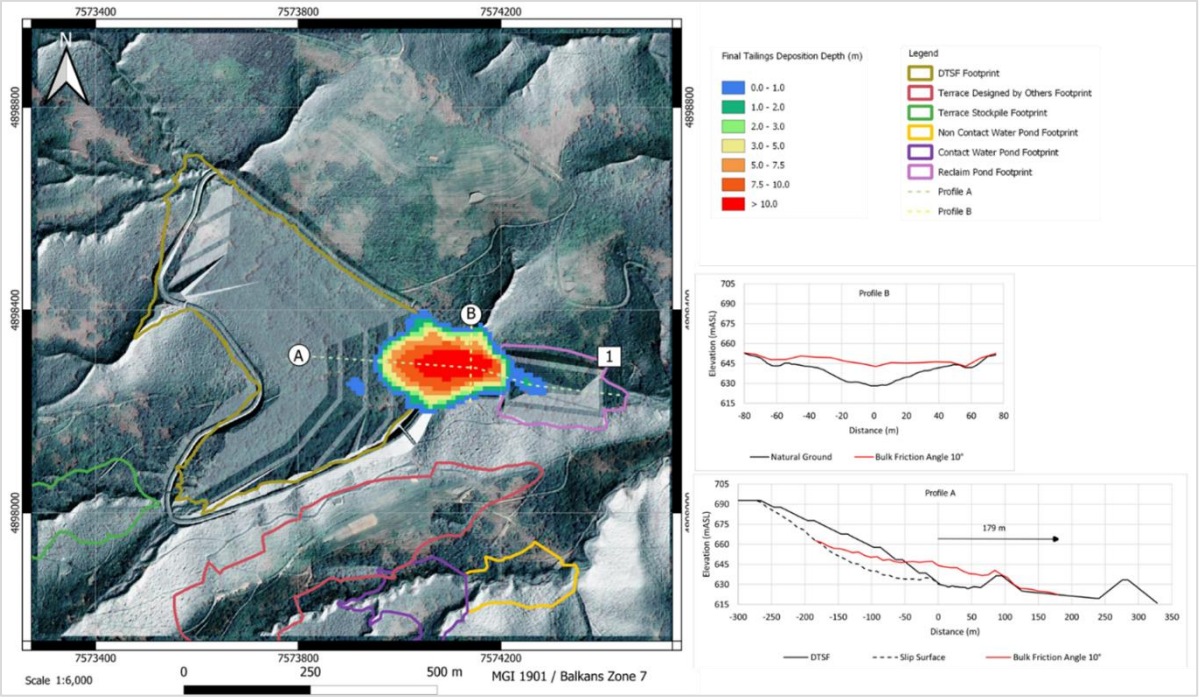
Source: WSP, 2025

The maximum runout distance for Scenario 1 at high mobility was estimated to be 166 m, with a final material depth deposition inside the Reclaim Pond of 2.0 m, which is the only case where the outflow material reaches the bottom of this facility, see Figure 18.26. For Scenario 2 at high mobility,

the maximum runout distance is 73 m, which does not enter the Reclaim Pond, while for normal mobility, the runout distance is negligible.

Therefore, it is unlikely that the tailings runout will inundate any of the locations of interest downstream of the DTSF, except for the Reclaim Pond, which is only reached by the worst case evaluated in this study, considering a volume of 230,000 m³ at high mobility (Scenario 1 with bulk friction angle of 10°).

Figure 18.26 – Final Deposition Depth for Scenario 1 High Mobility



Source: WSP, 2025

Runout profiles for sensitivity scenarios show a run-out distance about 160 m, this considers a Voellmy rheological model. This model accounts for velocity-dependent resistance terms by adding an empirically turbulent term to the frictional resistance incorporating the influence of air resistance. Initially, this model was used for snow avalanches, but in this runout context, it accounts for possible sources of velocity-dependent resistance.

18.10.2 DAM CLASSIFICATION

Following the consequence category classification using GISTM and CDA, the Čoka Rakita DTSF has been determined as “Low”. This classification is a change from the PFS due to the availability of the 3D model enabling a better understanding of the site and surroundings. he runout footprint is contained within the project area and only reaches the Reclaim Pond for Scenario 1 considering High Mobility and Sensitivity, which are not considered representative cases.

Mine personnel that could be present within the inundation area is unknown, but conservatively, it cannot be ruled out. The area between the DTSF and the Reclaim Pond will not be used for regular operations and will be only accessed by Operations, Maintenance, and Surveillance (OMS) personnel under exceptional conditions, such as inspections or emergency maintenance. Therefore, the probability of exposure to personnel is very low, and access is fully manageable. Itinerant population and exposure were considered. PAR of zero is estimated for Scenario 1 and 2, and therefore Potential Loss of Life (PLL) is lower than 1 and it can be considered as negligible. The standards and guidelines are presented in Table 18.11.

Table 18.11 – 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	Low	Low	Not Expected. Itinerant personnel potentially affected is less than 1.
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.
Overall DTSF Consequence Classification	Low	Low	Overall consequence category is based on the highest consequence rating given during the assessment
Design Seismic Event	Between 1/100	1/200-year event	
Flood Event (IDF)	Between 1/100 and 1/1,000 Note	1/1,000-year event	
Peak Ground Acceleration (Active)	0.099	0.134	Seismic Assessment (2021) PGA determined from Design classification and WSP evaluation (WSP 2024b), (WSP, 2024). For stability assessment, Significant classification was used in FS, a classification of Low can be used for the Detailed Design.
Peak Ground Acceleration (Closure)	0.7	0.7	Seismic Assessment (2021), GISTM, PGA determined from Design classification and WSP evaluation. (WSP 2024b), (WSP, 2024). Initial values adopted from PFS and seismic assessment (WSP, 2024). Following completion of the FS, these values will be re-evaluated and refined in Detailed Engineering.

Note: Selected on basis of incremental flood analysis, exposure, and consequences of failure

18.10.3 DESIGN CRITERIA

The design of the DTSF during the FS was initially assessed against the tailings through-put outlined within the PFS; these figures have since been refined by WSP over the course of the FS. The newly established throughput values have been implemented as part of the current feasibility design and are presented in Table 18.12.

Table 18.12 – Čoka Rakita Mine Throughput

Description	Quantity	Units	Remarks/Assumptions
Life of Mine	9	Years	As per LOMP June 2025.
Annual Throughput	850,000	Tonnes Per Year (Average)	RFI 2024 Full annual tailings throughput presented in Basis of Design.
Total Tailings	7.00	Mt	FS Data (Received from DPM June 2025) (PFS value of 6.31 used for majority of FS project)
Tailings (Surface Disposal) Design Value	3.93	Mt	Was 5.9 Mt through PFS and was revised to 3.93 Mt on 1st October 2024.
Tailings (surface Disposal) FS Mine Plan Value	4.13 2.23 (assuming densities of 90% of MDD)	Mt Mm³	Initial finding of the Paste backfill study from WSP indicates greater storage requirements on surface compared with PFS. Approximately 59% to be stored on surface.
Tailings (Paste Backfill) FS Value	2.875	Mt	Approximately 41 % 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 June 2025.
Surface Mine Rock Storage FS Value	0.269	Mm³	Revised May 2025, generated in initial 2 years of construction (No bulking factor applied to this).
Surface Mine Rock Storage FS Value	0.368	Mm³	Placed rock including bulking factor (36.5%).

18.10.4 TAILINGS CHARACTERISATION

A summary of the geotechnical parameters derived from this suite of laboratory testing on the filtered tailings sample is presented in Table 18.13.

Table 18.13 – Summary of Geotechnical Parameters

Type	Parameter
Classification	Non-Plastic SILT
Specific Gravity	3.14 Mg/m³
Compaction	SPMDD = 2.04 Mg/m³ OMC = 12.6% MPMDD = 2.13 Mg/m³ OMC = 11.2 %
Shear Strength	Effective Friction Angle = 34° Effective Cohesion = 0 kPa
Hydraulic Conductivity	4.3 x 10^{-7} m/s

The index properties indicate the material is a non-plastic slightly clayey, sandy SILT.

Laboratory testing was used to derive geotechnical parameters for the tailings mass; these are presented in Table 18.14.

Table 18.14 – Geotechnical Design Parameters - Tailings

Material	Unit Weight (t/m³)	Effective Friction Angle (°)	Effective Cohesion (kPa)	Undrained Strength Ratio (Peak)	Undrained Shear Strength (kPa)	Reference
Tailings (Fine – Silt)	1.85	34	0	0.45	55	Tailings Characterisation Report (WSP, 2025)

Further geotechnical testing has been proposed on future tailings samples for use in the Detailed Design. The next batch of tailing samples is anticipated to be produced at the end of 2025.

18.10.5 GROUND CHARACTERISATION

Ground characterisation of the Čoka Rakita project site has been defined from the 2024-2025 PFS/FS Ground Investigation. Some elements of the final investigation will be incorporated into updates to the ground model in Detailed Design. This characterisation is based on a comprehensive 3D geological model developed using Leapfrog Works (WSP, 2025), alongside the geotechnical interpretive report (WSP, 2025) and the geotechnical interpretive report – surface infrastructure design (UoB. University of Belgrade, 2024) and geophysics report (University of Belgrade, 2024) provided by the University of Belgrade.

The geotechnical and hydraulic parameters are presented in Tables 18.15 and 18.16.

Table 18.15 – Čoka Rakita Ground Characterisation Model

Material	Weathering	Unit Weight ɣ (kN/m³)	Mohr-Coulomb		Hoek-Brown	
			Effective Cohesion c' (kPa)	Effective Friction Angle ϕ' (°)	Unconfined Compressive Strength UCS (MPa)	Geological Strength Index (GSI)
Fill (Cohesive)	-	21	5	28	N/A	N/A
Fill (Granular)	-	21	0	34	N/A	N/A
Overburden (In-situ)	-	18	5	28	N/A	N/A
SFD	Heavily weathered	23	16	32	2	14
	Moderately weathered	24	N/A	N/A	24	30
	Fresh	25	N/A	N/A	>50	40
GDI/CFP	Heavily weathered	23	22	38	3	20
	Moderately weathered	24	N/A	N/A	25	25
	Fresh	25	N/A	N/A	30	35
XMO	Heavily weathered	23	25	40	2.7	25
	Moderately weathered	24	N/A	N/A	8.6	35
	Fresh	25	N/A	N/A	20	50

Table 18.16 – Hydraulic Conductivity Lithological Units

Lithological Unit	Hydraulic Conductivity		
	Average (m/s)	Minimum (m/s)	Maximum (m/s)
Overburden	2.7E-07	1.4E-08	1.3E-06
SFD	2.5E-07	1.7E-09	1.5E-06
GDI	5.1E-07	2.2E-08	3.6E-06
XMO	6.7E-07	5.6E-08	1.8E-06

18.10.6 PROPOSED GEOTECHNICAL INVESTIGATION

It is proposed that a further limited investigation will be undertaken for confirmatory purposes of the ground conditions.

18.10.7 Dry Tailings Storage Facility Design

18.10.7.1 Tailings Conveyance Method Summary

A semi-quantitative SWOT assessment completed at PFS stage ranked each tailings technology with weighted sub-category from 1 to 5. "Filtered Tailings" was identified as the most advantageous due to easier permitting and higher technical scores.

18.10.7.2 Optimisation Summary

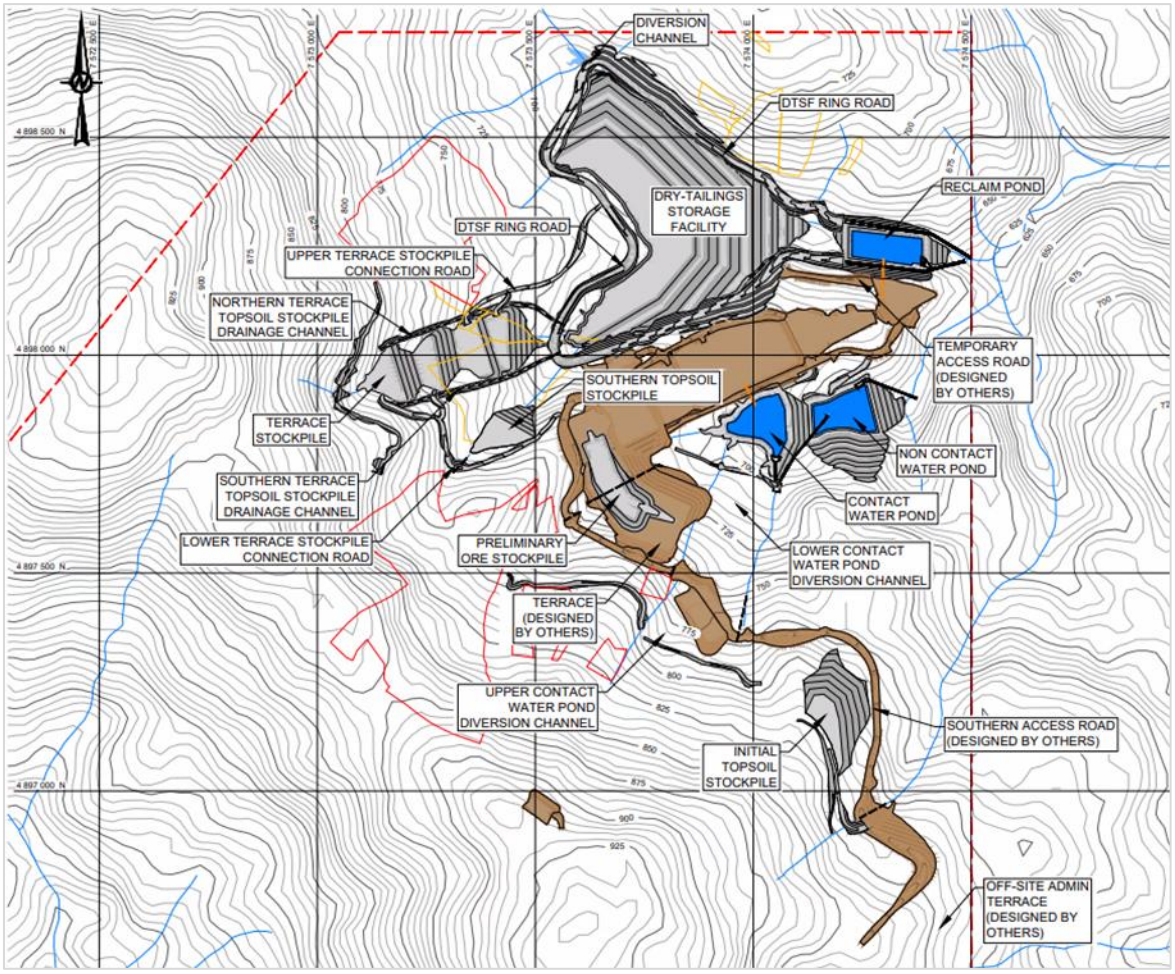
The DTSF was designed to PFS storage requirements. Storage requirement increased at the end of the FS through LOMP updates. Throughput changes are to be incorporated into the Detailed Design.

The optimised DTSF storage capacity and overall design basis are summarised in Table 18.17. The DTSF plan is presented in Figure 18.27.

Table 18.17 – Čoka Rakita DTSF Optimisation Summary

Locations	Units	Site 1 Volume	Notes
Required DTSF Capacity	m³ t	2,143,000 3,537,000	Initial basis from PFS storage requirements with values increased at the end of the FS through LOM plan updates. Throughput changes to be incorporated into the Detailed Design
Total DTSF Height	m	64.75	
Modelled Capacity	m³	2,163,843	
Capacity (Net Tailings)	m³	1,917,843	Capacity of only tailings with assumed 1.5 m FS design capping layer.
Modelled Capacity	t	3,528,831	Assuming 90% SPMDD Provides sufficient capacity against the PFS LOM values, requires additional approximately 330k tonnes capacity for final FS LOM values, sufficient space is available to increase final elevation of facility to manage this additional capacity.

Figure 18.27 – DTSF Location Plan



Source: WSP, 2025

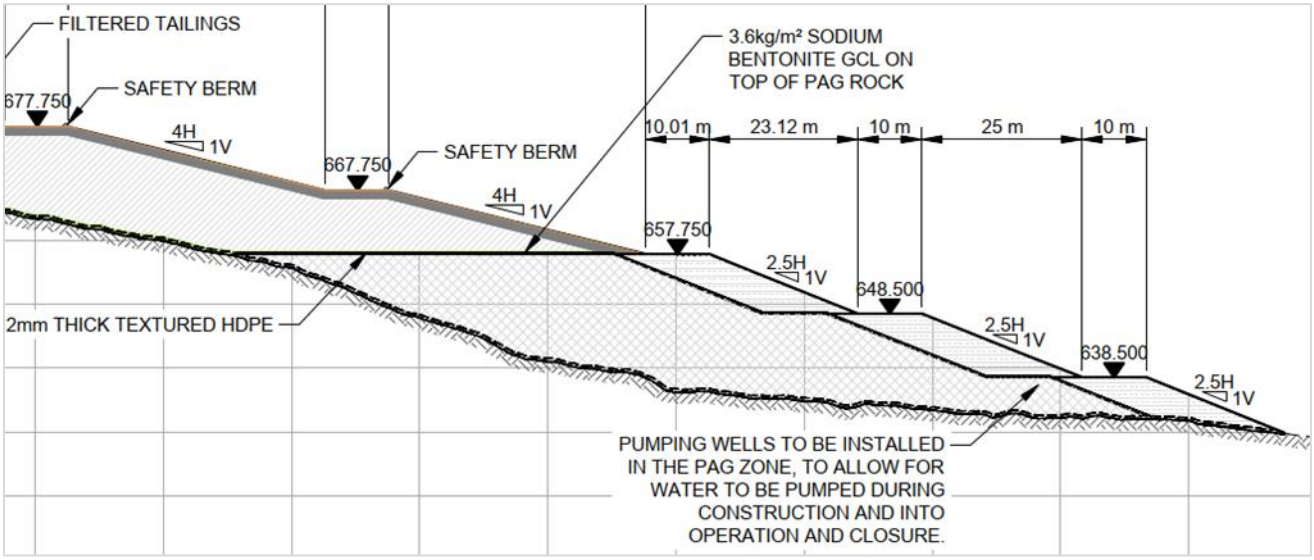
18.10.7.3 Potentially-Acid Generating Zone Design

During the initial years of mining, excavation will produce Mine Waste Rock, considered Potential Acid Generating (PAG) for this FS. Since backfilling underground stopes is not feasible 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. The PAG will be stored within the base of the DTSF. Containment is provided through the installation of a lining system across the base of the valley. Once all the PAG Mine Waste Rock has been placed within the initial two years of mine development and following the first year of operations, the top of the PAG rock will be composite lined and then the placement of the filtered tailings can commence above the PAG Mine Waste Rock.

Figure 18.28 presents the cross-section FS design of the DTSF PAG Foundation.

Figure 18.28 – DTSF PAG Foundation Design



Source: WSP, 2025

If geochemical testing confirms the mine rock is PAG, it could produce 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 fully encapsulated to prevent ARD.

A seepage assessment was completed to rationalise the volume of water anticipated to seep through the DTSF, the results show that during the operational life, the PAG zone will remain unsaturated. The long-term closure case indicates that seepage from ground water may begin to infiltrate the PAG zone from the base, this works on the assumption that in the long term, there will be some clogging of the overall underdrain from fines infiltration. The FS design includes a composite liner on top of the PAG rock to prevent seepage. This design will be reassessed at detailed design.

Table 18.18 presents the required and designed PAG storage capacity as per the May 2025 LOM plan.

Table 18.18 – PAG Zone Design

Description	Quantity		Units	Remarks / Assumptions
Required Surface Mine Rock Storage	0.269		Mm³	Revised May 2025, generated in initial 2 years of construction (No bulking factor applied to this).
UG Waste Rock Bulking Factor	36.5		%	Assumption from in-situ density of 2.73 t/m³ from mining team and 2.0 t/m³ for placed construction rockfill.
Surface Mine Rock Storage	0.368		Mm³	Placed rock including bulking factor (36.5%).
Annual Surface Mine Rock Disposal	Year -2	127,975	m³	Reduction in anticipated PAG material to be delivered to the DTSF footprint from update mine plan, contingency for additional material to be storage in the PAG footprint. As per Mine plan from FS (May 2025).
	Year -1	141,975		
PAG Foundation Design				
Bench Height	10		m	
Bench Width	10		m	
Downstream Slope:	1:2.5		V:H	Defined in FS design, considering stability, closure requirements, construction practicalities, sustainability of vegetation etc.

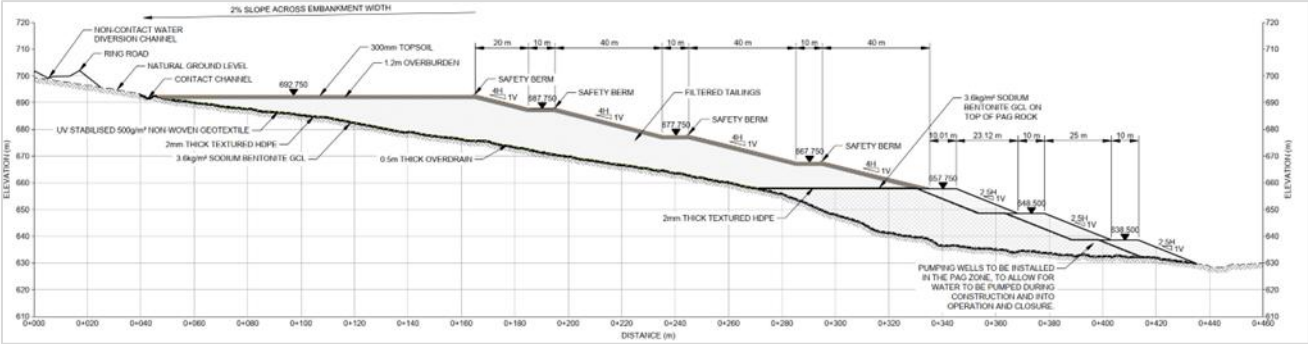
18.10.7.4 Filtered Tailings Design

Filtered tailings will be placed across the top of the PAG foundation to create four (4) benches, as detailed in Section 18.10.7.13. Table 18.19 provides information on the overall filtered tailings design and annual surface stacking rate.

Since the delivery of this design, DPM has updated the LOMP resulting in an increase in the total quantity of tailings required to be stored from that presented in the PFS. It is recommended that the design is re-assessed using the values as per the mine plan received in May 2025 in the Detailed Design prior to the final sizing and location of infrastructure.

A cross section of the FS filtered tailings design is presented in Figure 18.29.

Figure 18.29 – Filtered Tailings Design - Cross Section



Source: WSP, 2025

Table 18.19 – Filtered Tailings Design

Criteria	Input	Units	Notes and References
Required DTSF Capacity	2,143,000	m³	Initially designed to PFS storage requirements with values increased at the end of the FS through LOMP updates. Throughput changes to be incorporated into the Detailed Design.
	3,537,000	t	
DTSF Construction			
Operating Hours Per Day (24 Hours)	8	h	DPM
Daily Rate of Tailings Placement	820	m³ per day	FS Design
Crest Elevation	692.75	mRL	FS Design
Total Height	64.75	m	FS Design
Tailings Height	20	m	FS Design
Rockfill Height	30	m	FS Design
Bench Height	10	m	FS Design
Bench Width	10	m	FS Design
Bench Slope Length	40	m	FS Design
Downstream Slope: Overall Inter slope	 1:5 1:4	 V:H V:H	Defined in FS Design, considering stability, closure requirements, construction practicalities, sustainability of vegetation etc.

Year	Tailings for Surface Disposal (FS Design Values) (t)	Tailings for Surface Disposal (FS Final Values) (t)	Comment
Year -2	-	-	PFS throughput values used for FS Design. Updated throughput values at the end of the FS. New values will be adopted for Detailed Design.
Year -1	-	27,244	
Year 1	439,731	584,155	
Year 2	340,626	581,565	
Year 3	384,924	461,408	
Year 4	409,704	490,495	
Year 5	398,549	452,620	
Year 6	361,701	463,348	
Year 7	270,234	393,322	
Year 8	240,046	399,802	
Year 9	459,890	275,717	
Year 10	231,908	-	
Total	**3,537,313**	**4,129,676**	

18.10.7.5 Liner Design

The liner system identified from this study is summarised in Table 18.20.

Table 18.20 – Čoka Rakita DTSF Liner Selection

Description	Quantity	Surface Preparation
Entire DTSF Footprint	Composite – GCL and 2 mm HDPE GCL Needle Punched GCL (Bentonite weight 3.6 kg/m²) Further overlain by 500 g/m² carbon rich protector geotextile	Unsuitable material will be removed to required elevation and gradient; GI results indicate that some places may require 2 m+ of OVB removal. Strength testing to be completed of founding material to determine suitability. 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.

18.10.7.6 Liner System Characterisation

Table 18.21 presents a summary of the recommended laboratory testing to be conducted.

Table 18.21 – 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

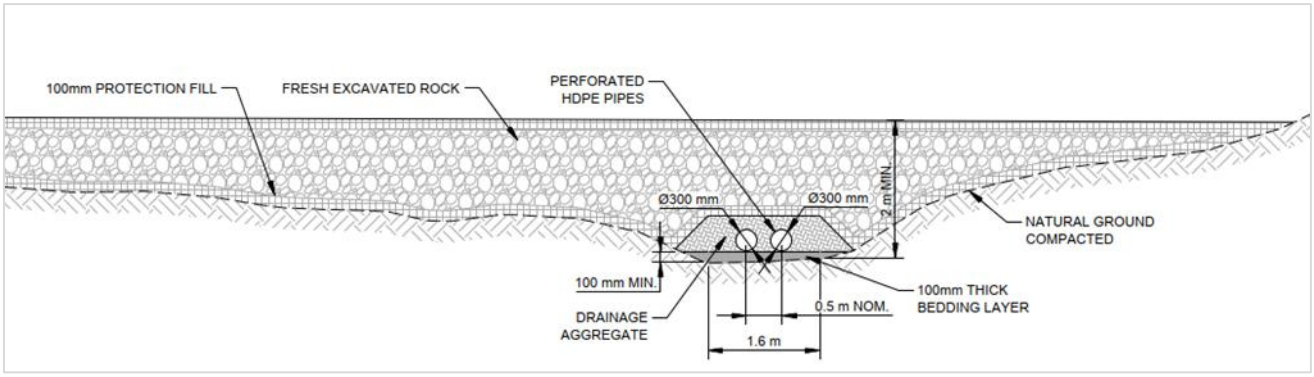
18.10.7.7 Underdrain

An underdrain will be installed at the base of the northern & southern valleys in which the DTSF will be constructed. The underdrain will run underneath the composite liner of the DTSF, conveying the flow of groundwater beneath the facility. The drain will continue beneath the Reclaim Pond, discharging into the original stream bed of the valley. Access points will be constructed at the downstream toe of the DTSF to enable water to be monitored and sampled for environmental testing, assessing if any leaks have occurred in the DTSF basal lining system. In the instance that contamination is detected, the water within the basal drain can immediately be diverted away from the natural environmental to the Reclaim Pond for treatment and/or re-use. The total quantity of drainage material required to construct the basal drain is approximately 58,000 m³.

The underdrain design will comprise of infilling the valley base with approximately 25 m² per metre run of locally won rockfill (>50 mm aggregates) with <5% fines content. Two (2) 300 mm OD HDPE pipes will be placed above the rockfill and surrounded by a finer drainage aggregate material (<20 mm) encasing the HDPE pipes by 200 mm. One (1) pipe will be perforated to collect any inflows into the drain, the second will be solid, this will be used for water management during the construction of the facility. A non-woven geotextile will be wrapped around the finer aggregate and perforated HDPE pipe, with the non-woven geotextile overlapping by at minimum of 300 mm, before being overlaid and buried by gravel filter material.

Figure 18.30 illustrates the cross section of the typical underdrainage design. Figure 18.31 presents a plan view of the underdrainage alignment beneath the DTSF facility.

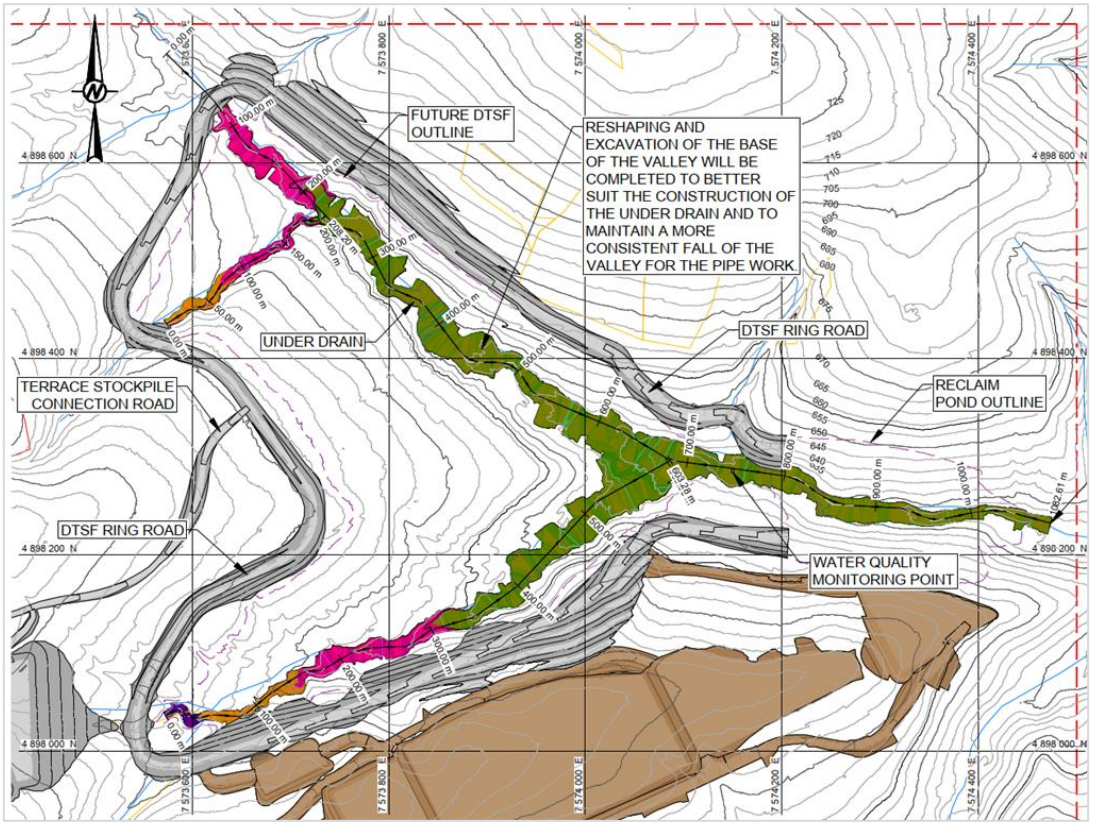
Figure 18.30 – Typical Underdrain Cross Section



Source: WSP, 2025

The top surface of the basal drain will be refined at Detailed Design to optimise the quantities of crushed rock required to construct the entirety of the underdrain. A filter compatibility assessment is recommended at Detailed Design to determine the gravel grading requirements and if a finer sand filter layer is necessary. Further hydrogeological studies will also assess the required HDPE pipe dimensions and perforations.

Figure 18.31 – DTSF Underdrain – Plan View



Source: WSP, 2025

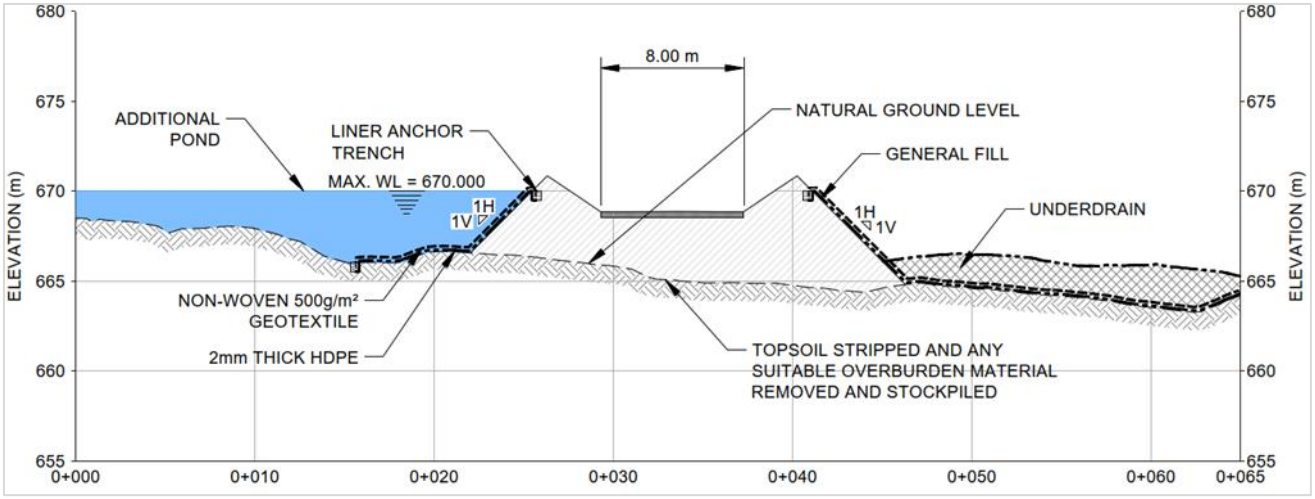
18.10.7.8 Northern Embankment

The Northern Embankment will be constructed at the northern extent of the Dumitru Valley; its purpose is to connect the access roads as part of the DTSF perimeter road and to allow for the DTSF footprint to encroach higher onto the northern side of the north valley. It will be constructed from locally won overburden and will be lined on both sides to prevent seepage into the DTSF. The underdrain will abut the southern side of the earthwork.

The upstream face will collect catchment water from the valley and will result in ponding of water upstream of the facility. Any water that is initially ponded will drain by gravity through the weir into the non-contact channel preventing it from breaching the project permit boundary to the north.

A section design of the northern embankment is presented in Figure 18.32. The geometry will be reviewed in detailed design.

Figure 18.32 – Northern Embankment Design



Source: WSP, 2025

Design details of the North Embankment are presented in Table 18.22.

Table 18.22 – North Embankment Design Summary

Description	Value
Fill Volume (m³)	1,419
Dam Height (downstream) (m)	8
Slope Gradient (V:H)	1V:1H
Crest Width (m)	8
Crest Height (mRL)	669
Freeboard (m)	1.5

Description	Value
Pond Height (mRL)	670
Pond Capacity (m³)	1,682

18.10.7.9 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. Access and works sequencing for the temporary lay-down area will be confirmed at detailed design stage.

The design details are as follows:

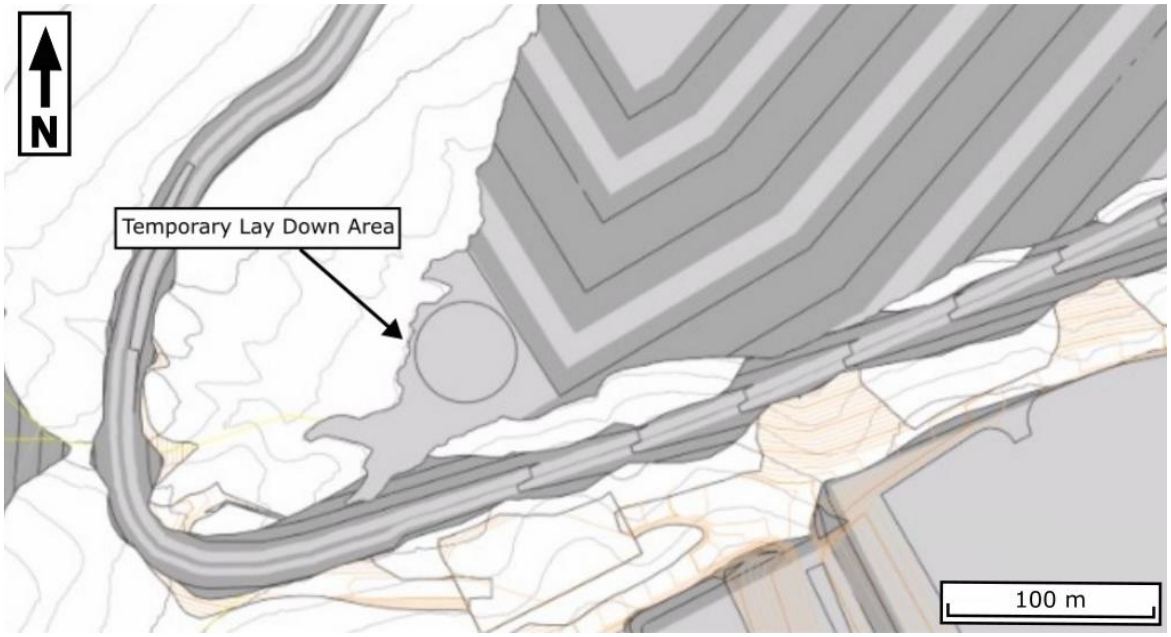
- Storage Capacity: 5 days, 1,400 kg/m³ density, approximately 9,000 m³.
- Design: Covered to prevent rain wetting, using a stacker to create an 8 - 10 m high conical stockpile with a 1,200 m² footprint.
- Parking Area: Adjacent to storage, 900 m² for surface mobile machinery
- Total Area: 2,200 m².

Table 18-20 summarises the construction equipment identified for the temporary lay-down area. Figure 18.33 illustrates a plan view and location of the temporary lay-down area.

Table 18.23 – 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 CS20	2

Figure 18.33 – Temporary Lay-Down Area - Plan View



Source: WSP, 2024

18.10.7.10 DTSF Stability Analysis

Material parameters were derived from the PFS/FS ground investigation (WSP, 2025) and laboratory testing data (WSP, 2025).

Table 18.24 presents the assigned material parameters.

Table 18.24 – Stability Analysis Material Parameters

Material	Weathering Grade	Unit Weight (kN/m³)	Material Strength
Filtered Tailings		20.6	Drained: c' = 0 kPa, φ' = 34 ° Undrained: τ/σ = 0.45, cu, min = 55 kPa
Clean Rock (Embankment)		20	c' = 0 kPa, φ' = 35 °
Mine Waste Rock (PAG)		20	c' = 0 kPa, φ' = 40 °
Geosynthetic Liner		20	c' = 0 kPa, φ' = 23 °
OVB (Capping)		17	c' = 3 kPa, φ' = 24 °
OVB		17	Drained: c' = 3 kPa, φ' = 24 ° Undrained: cu = 40 kPa *
SFD	Highly Weathered	19	c' = 25 kPa, φ' = 27 °
SFD	Moderately Weathered	20	UCS = 5,400 kPa, mi = 14, GSI = 35, D = 0, σ3, max = 362 kPa

Material	Weathering Grade	Unit Weight (kN/m³)	Material Strength
SFD	Slightly Weathered / Fresh	24	UCS = 16,600 kPa, mi = 14, GSI = 47, D = 0, σ3, max = 502 kPa c' = 36 kPa, φ' = 47 °
GDI	Highly Weathered	19	
GDI	Moderately Weathered	20	UCS = 120,000 kPa, mi = 26, GSI = 45, D = 0, σ3, max = 509 kPa
GDI	Slightly Weathered / Fresh	25	UCS = 138,200 kPa, mi = 26, GSI = 50, D = 0, σ3, max = 652 kPa

Four (4) 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 all of the cross sections (as seen in Table 18.25) exceed minimum required Factor of Safety (FoS) values when considering the drained and undrained strength parameters.

Table 18.25 – DTSF Slope Stability Summary

Drainage Condition	Loading Scenario	Required FoS	Section 1 FoS	Section 2 FoS	Section 3 FoS	Section 4 FoS
Drained	SEM	1.3	1.8	2.9	2.9	2.1
Drained	CDA (Static)	1.5	1.8	2.9	2.9	2.1
Drained	CDA (Pseudo-Static)	1.0	1.1	1.9	1.7	1.1
Undrained	SEM	1.3	1.8	2.3	2.3	2.4
Undrained	CDA (Static)	1.5	1.8	2.3	2.3	2.4
Undrained	CDA (Pseudo-Static)	1.0	1.0	1.0	1.0	1.0

18.10.7.11 Trade-Off between Truck and Conveyor

A trade-off study (ToS) between trucks and conveyors for tailings transportation was conducted during the PFS with trucks selected from this study. 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.

18.10.7.12 DTSF Ring Road

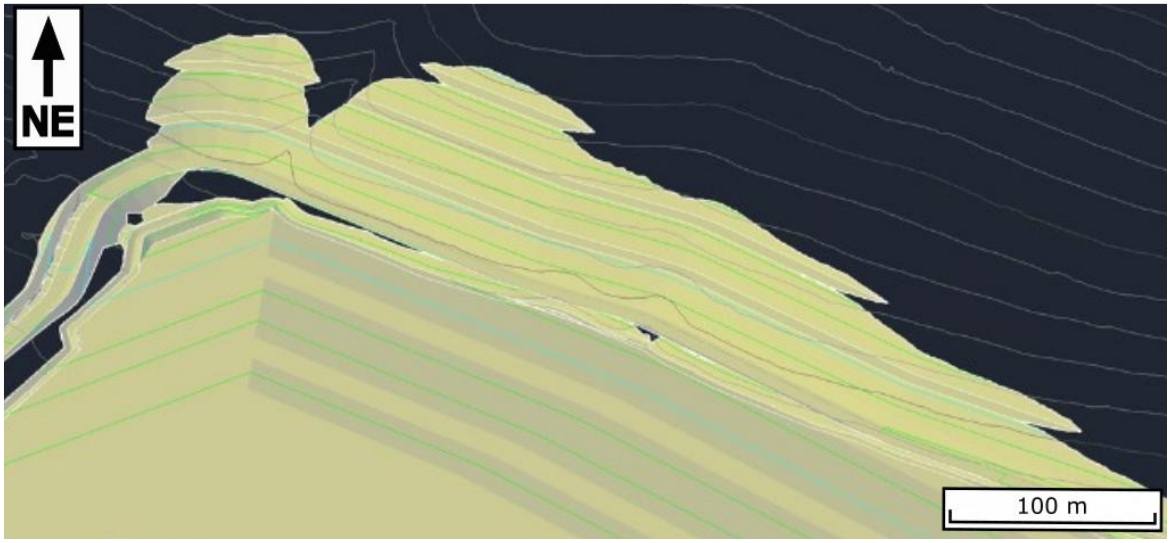
The DTSF Ring-Road has been designed as two (2) separate roads that surround the DTSF. The entire ring-road has a total design distance of 2,300 m, and varies in gradient between 1 % to 11 %. Where necessary, cut slopes were designed with a 1H:1V slope during the PFS. Following partial completion of the PFS/FS ground investigation, the design of some portions of the cut slopes has been altered to provide updated solutions.

The results of the PFS/FS ground investigation identified that the overburden material may in places have weaker horizons. From the limited investigation results, it was necessary to assume conservative material parameters to design suitable slopes. These designs may be amended through Detailed Design and construction if the ground conditions are found to be more favourable.

The general design for the cut slopes consists of 1H in 1V slopes with 5 m wide benches every 10 m of slope height. In addition, some slopes of highly weathered rock will require soil nailing at approximately 1.5 m centres and in some areas, excavation and replacement of overburden with the inclusion of geogrids at 1 m spacing will be necessary.

Figure 18.31 presents the north slope cut DTSF ring road design.

Figure 18.34 – DTSF Northern Embankment and Ring Road



18.10.7.13 Stage Storage Deposition

The sizing of the DTSF was competed at FS using the most recent LOMP (May 2025) and when compared with the updated PFS quantities, it has increased capacity. Refinement of the DTSF sizing will be completed at the Detailed Design stage.

The stage storage plan for the Čoka Rakita DTSF is presented in Table 18.26. The table presents the PAG and tailings deposition volumes with associated dam height elevation (mRL).

Table 18.26 – Stage Storage Deposition Summary

Year (Operation)	PAG Material (FS Mine Plan) * (t)	Tailings Surface Disposal (FS Mine Plan) * (t)	Elevation (mRL)	Rate of Rise
Year -2	174,686	-	-	-
Year -1	193,796	27,244	658.1	0.7
Year 1	-	584,155	664.4	6.25
Year 2	-	581,565	669.3	4.92
Year 3	-	461,408	673.0	3.68
Year 4	-	490,495	676.7	3.78
Year 5	-	452,620	680.4	3.7
Year 6	-	463,348	684.3	3.89
Year 7	-	393,322	687.7	3.33
Year 8	-	399,802	691.6	3.95
Year 9	-	275,717	692.75	0.7
Year 10	-	-	692.75	-
Total	**368 482**	**4,129.676**		

18.10.7.14 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 18.27 presents an overview of on the step-by-step process required for constructing the DTSF.

Table 18.27 – 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.

Subgrade Category	Occurrence	Treatment
	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.
Composite – GCL and HDPE	Entire DTSF Footprint	Unsuitable 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

18.10.7.15 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 CW channels as DTSF elevation increases.

18.10.7.16 Operations

The Čoka Rakita DTSF will operate 8 hours a day, 336 days a year. Surface mobile equipment details are presented in Table 18.28. Filtered tailings, produced in cycles via a filter press, will be transported to a storage terrace by a conveyor system. During site operational hours, the tailings material will be further loaded into articulated trucks & hauled into DTSF footprint to the designated deposition point. During non-operational hours and in poor weather, dry tailings will be moved to a covered temporary lay-down area within the DTSF to reduce saturation. A wheeled front loader will load articulated dump trucks with tailings from the storage area. Temporary access roads may be needed and will be designed at Detailed Design, considering truck weight and number. Reduced tire pressure is recommended for vehicles on the DTSF.

Articulated Dump Trucks will reverse into position with a banksman support, and tailings will be end-tipped into the deposition zone. A tracker dozer will spread tailings, and a soil compactor will

compact them in 300 mm lifts to MDD and OMC requirements. Additional equipment may be needed for moisture conditioning. The final layer thickness will be determined in Detailed Design studies. Excavator will maintain drainage ditches and create temporary channels.

Excavators will be utilized locally to aid in maintaining drainage ditches and producing temporary drainage channels necessary along the top of the filtered tailings surface.

Contact water (CW) channels will be raised progressively with the filtered tailings deposition and the liner system will be installed progressively with lining in the DTSF installed in four stages.

Table 18.28 – 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

18.10.8 RECLAIM POND

The Reclaim Pond receives runoff and seepage from the DTSF via the CW channels and has a total volume of 51,900 m³. The water collected is considered to be contaminated from contact with the tailings, therefore the lining system of the pond will comprise a composite lining system.

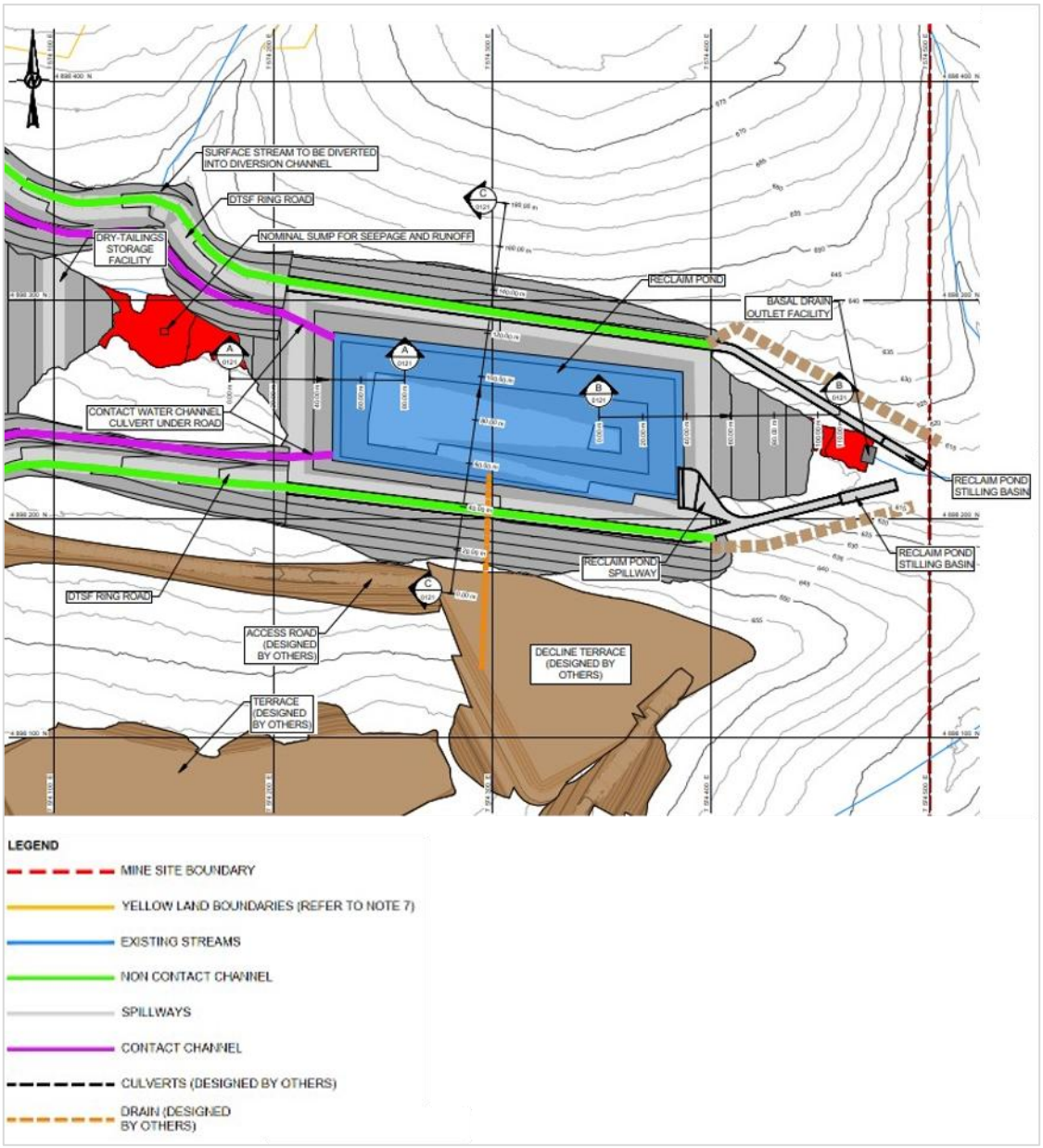
The Reclaim Pond has been positioned to provide sufficient space for the modelled dam breach for the DTSF therefore not causing structural damage to the upstream retaining embankment of the Reclaim Pond or cause a release of CW or tailings to the environment.

The valley slopes above the Reclaim Pond will require excavation and reprofiling to be constructed as per the design. The valley slopes above the Reclaim Pond access road will be cut at 1V:1H gradients. Due to the weathering profile anticipated from the ground investigation and the requirement for steep cut slopes due to proximity of other structures there will be a need to apply reinforcement to the slopes. The FS design includes an allowance for soil nails across the slope, these will be re-evaluated at Detailed Design with additional ground investigation and laboratory test work anticipated.

The basal drain that runs the length of the DTSF basin and DTSF foundation will extend beneath the Reclaim Pond. The purpose of the basal drain is to convey ground water within the foundations beneath the DTSF and Reclaim Pond to flow unobstructed and discharge into the environment downstream of the facility.

Figure 18.35 presents the plan view of the Reclaim Pond.

Figure 18.35 – Reclaim Pond – Plan View



Source: WSP. 2025

18.10.8.1 Reclaim Pond Design

The Reclaim Ponds design summary details are presented in Table 18.29.

Table 18.29 – 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) (1)	2 m (Operating level freeboard) 1 m (Spillway freeboard)	
Pond Capacity (m³)	51,900	

Note:

(1) The operating level freeboard is measured from the dam crest to the spillway invert level, i.e. the maximum water level before the pond starts overflowing via the spillway. The operating level freeboard meets the understanding of the Serbian regulation requirements of 2 m for dams higher than 15 m (SRPS U.C5.020, 1980). The spillway freeboard is measured from the dam embankment crest to the water level in the spillway when conveying the IDF. The freeboard is to be reviewed at detailed design stage.

18.10.8.2 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 18.30 presents a summary of the liner system selected for the DTSF Reclaim Pond.

Table 18.30 – Reclaim Pond Liner System

Feature	Lining System
Reclaim Pond	Composite – GCL and HDPE HDPE 2 mm Smooth CGL Bentonite weight (3.6 kg/m² sodium bentonite)

18.10.8.3 Reclaim Pond Stability Assessment

Slope stability models have been developed using GeoStudio SLOPE/W limit equilibrium software, using the Morgenstern-Price method of analysis with optimised surfaces. As part of the FS, stability analysis was completed following both global and local guidance.

Seepage analysis was completed for each of the embankments to determine the phreatic surface within the slopes. Given the design with the inclusion of upstream and basal liner systems the phreatic surface in the modelling does not develop within any of the slopes.

Based upon assumptions from the PFS, the structures were assigned a classification of "Significant", and the seismic coefficient used for the pseudo static state reflected this.

The results of the analysis are presented in Table 18.31 and indicate a compliant design. Reinforcement through the use of self-drilling anchors is required locally within rock cuttings to provide sufficient resistance for stability.

Table 18.31 – Reclaim Pond Stability Analysis Results

Section	Seepage Analysis	Loading Scenario	Minimum FoS Required	Location	Computed Overdesign Factor (FoS)
Section 1 - East embankment	Steady-State	SEM	1.3	Downstream	1.6
	Steady-State	CDA Static	1.5	Downstream	1.6
	Steady-State	CDA Pseudo-Static	1.0	Downstream	1.2
	Rapid-Drawdown	SEM	1.3	Upstream	1.4
	Rapid-Drawdown	CDA Static	1.3	Upstream	1.4
	Rapid-Drawdown	CDA Pseudo-Static	1.0	Upstream	1.0
Section 2 – West embankment	Steady-State	SEM	1.3	Downstream	1.4
	Steady-State	CDA Static	1.5	Downstream	1.6
	Steady-State	CDA Pseudo-Static	1.0	Downstream	1.0
	Rapid-Drawdown	SEM	1.3	Upstream	1.5
	Rapid-Drawdown	CDA Static	1.3	Upstream	1.5
	Rapid-Drawdown	CDA Pseudo-Static	1.0	Upstream	1.1
Section 3 - cut slopes of the access roads across the Reclaim Pond	-	SEM	1.3	Left cut slope	1.6
		CDA Static	1.5	Left cut slope	1.6
		CDA Pseudo-Static	1.0	Left cut slope	1.1
		SEM	1.3	Right cut slope	1.5

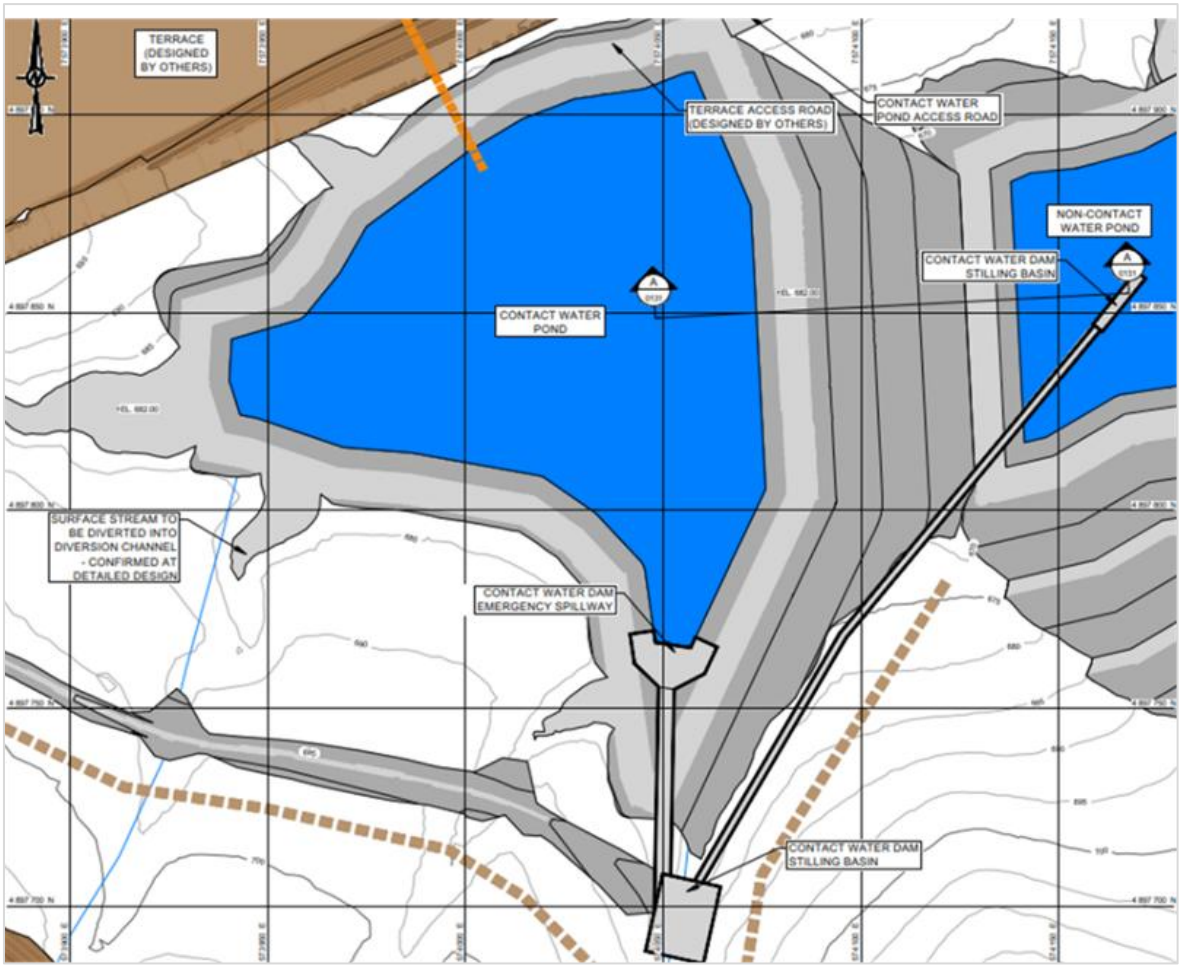
Section	Seepage Analysis	Loading Scenario	Minimum FoS Required	Location	Computed Overdesign Factor (FoS)
		CDA Static	1.5	Right cut slope	1.5
		CDA Pseudo-Static	1.0	Right cut slope	1.2

18.10.9 CONTACT WATER POND

The CW Pond primarily receives runoff reporting from the upper terraces of the ROM Pad and Terrace area. In case of a shutdown of the Clarifier or Water Treatment Plant, the CW Pond may receive additional pumped inflows from the Reclaim Pond, the underground mine dewatering and the water recirculated from the Processing Plant. The total pond volume is 62,000 m³.

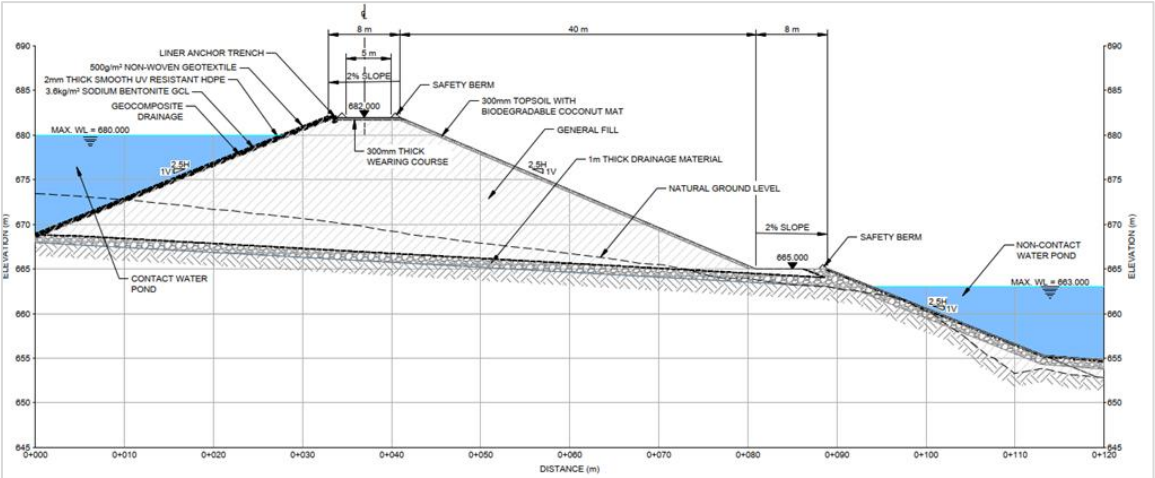
Figure 18.36 illustrates the plan view of the CW Pond and Figure 18.37 depicts the section view.

Figure 18.36 – Contact Water Pond – Plan View



Source: WSP, 2025

Figure 18.37 – Design Section for Contact Water Pond



Source: WSP, 2025

An access road will be constructed around the pond to facilitate monitoring and inspection of the dam during operations. The road will connect and lead to and from the process plant terrace. The pumping and decant systems to be installed within the CW Pond will be designed by others who will confirm the details and location at Detailed Design. The location of the equipment will need to not impair the operation of the facility spillway.

18.10.9.1 Contact Water Pond Design

The CW Pond design summary details are presented in Table 18.32.

Table 18.32 – Contact Water Pond Design Summary

Description	Value
Fill Volume (m³)	103,300
Slope Gradient	1V:2.5H
Crest Elevation (mRL)	682.0
Crest Width (m)	8
Freeboard (m) (1)	2 m (Operating level freeboard) 1 m (Spillway freeboard)
Pond Capacity (m³)	62,100
Pond Maximum Level (mRL)	680

Note:
(1) The operating level freeboard is measured from the dam crest to the spillway invert level, i.e. the maximum water level before the pond starts overflowing via the spillway. The operating level freeboard meets the understanding of the Serbian regulation requirements of 2 m for dams higher than 15 m (SRPS U.C5.020, 1980). The spillway freeboard is measured from the dam embankment crest to the water level in the spillway when conveying the IDF. The freeboard is to be reviewed at detailed design stage.

18.10.9.2 Contact Water Pond Liner Design

The CW Pond will receive contact water from the underground workings and process terrace, requiring a composite lining system similar to the DTSF and Reclaim Pond.

Table 18.33 presents a summary of the liner system selected for the CW Pond.

Table 18.33 – Mine Dewatering Pond Liner System

Feature	Lining System
Mine Dewatering Pond	Composite – GCL and HDPE HDPE 2 mm Smooth CGL Bentonite weight (3.6 kg/m² sodium bentonite)

18.10.9.3 Contact Water Pond Stability Assessment

During the course of the FS, the stability of the slopes within the scheme has been checked in accordance with Canadian Dam Association (CDA, 2014) Rules on Technical Requirements for the Surface Exploration of Mineral Resources Deposits (SEM) and a Seismic Pseudo-static scenario. Slope stability models have been developed using GeoStudio SLOPE/W limit equilibrium software, using the Morgenstern-Price method of analysis with optimised surfaces.

Seepage analysis was completed for the embankments to determine the phreatic surface within the slopes. Based on the modelling completed, a phreatic surface is not expected to develop within the slope; this is not unexpected given that the design includes an underdrain running underneath the pond and the embankment.

The structures were assigned a classification of “Significant”, and the seismic coefficient used for the pseudo-static state reflected this.

The results of the analysis are presented in Table 18.34.

Table 18.34 – Contact Water Pond Stability Analysis Results

Section	Seepage Analysis	Loading scenario	Minimum FoS Required	Location	Computed Overdesign Factor (FoS)
Section 1 - CWP	Steady-State	SEM	1.3	Downstream (CWP embankment)	1.4
	Steady-State	CDA Static	1.5	Downstream (CWP embankment)	1.6
	Steady-State	CDA Pseudo-Static	1.0	Downstream (CWP embankment)	1.1
	Rapid-Drawdown	SEM	1.3	Upstream (CWP embankment)	1.4
	Rapid-Drawdown	CDA Static	1.3	Upstream (CWP embankment)	1.4
	Rapid-Drawdown	CDA Pseudo-Static	1.0	Upstream (pond embankment)	1.0

18.10.10 NON-CONTACT WATER POND

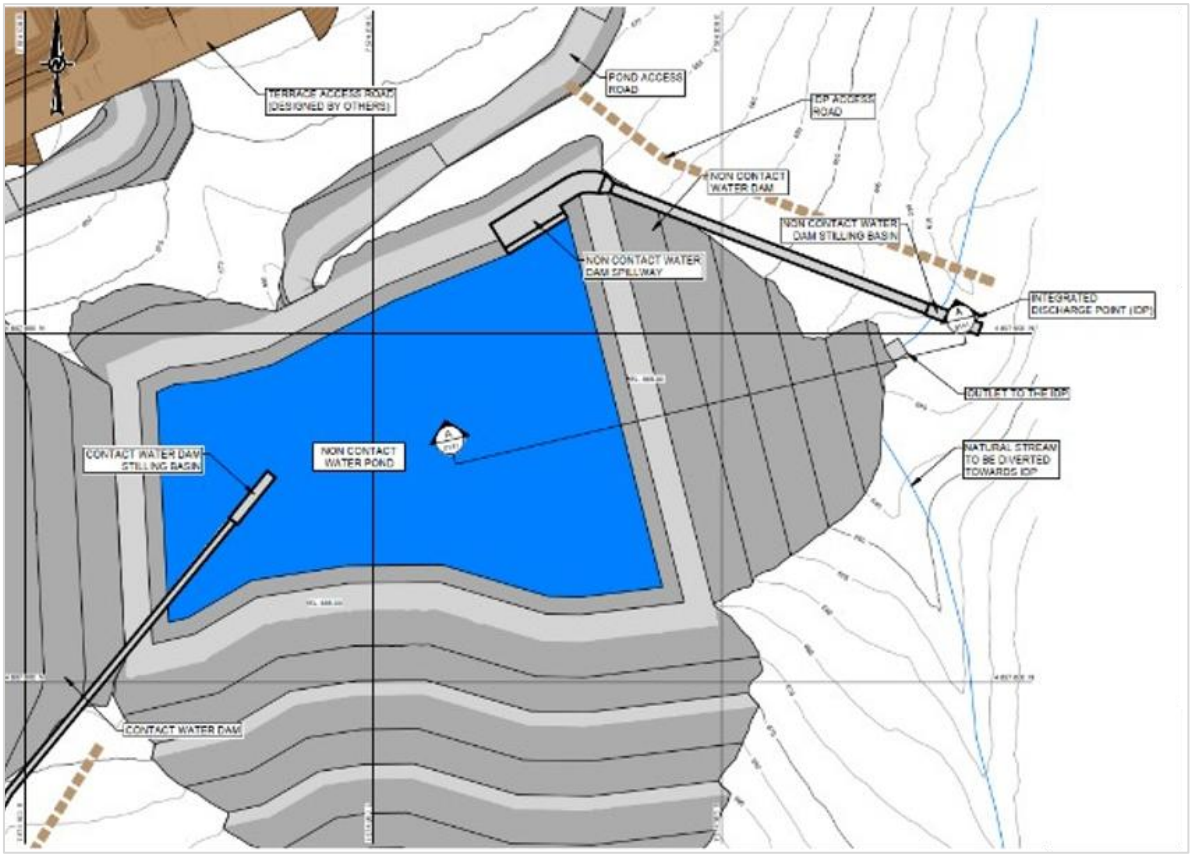
The Non-Contact Water (NCW) Pond has a maximum capacity of 58,500 m³ for the FS assessment.

The NCW Pond captures non-contact runoff originating from the catchment located south-east of the ROM Terraces including the catchment diverted by the Upper and Lower CW Pond diversion channels and the East and West Admin Terrace diversion channels. Once the maximum capacity of the pond is reached, it is designed to freely overflow into the pond spillway to the Integrated Discharge Facility.

An access road will be constructed around the pond to facilitate monitoring and inspection of the facility during operations. The road will connect to the process terrace. The pumping and decant systems to be installed within the NCW Pond will be designed by others who will confirm the details and location at a later stage in the design process.

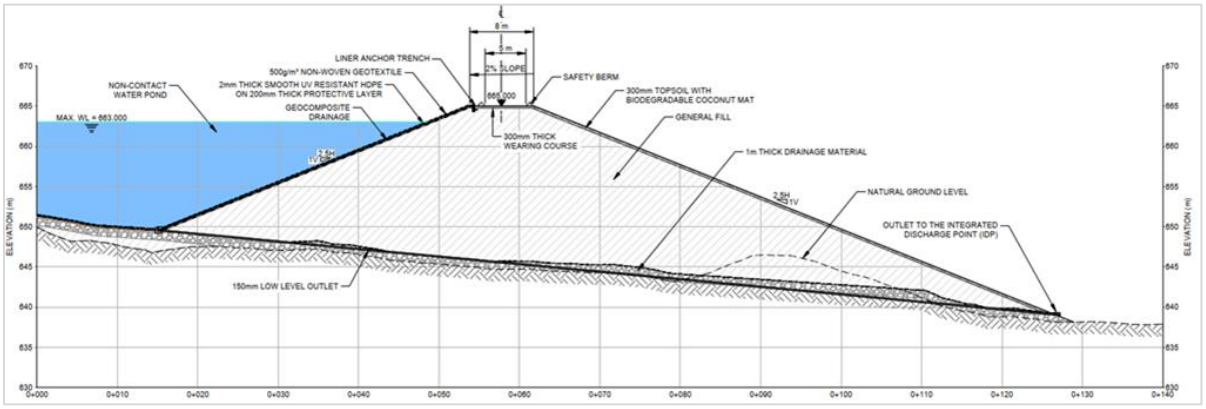
Figure 18.38 presents the plan view of the NCW Pond while Figure 18.39 presents a cross-section.

Figure 18.38 – Non-Contact Water – Plan View



Source: WSP, 2025

Figure 18.39 – Non-Contact Pond – Cross Section



Source: WSP, 2025

18.10.10.1 Non-Contact Water Pond Liner Design

The NCW Pond collects "clean" water, with no expected contamination from seepage through filtered tailings. Therefore, a single HDPE liner is initially considered for the lining system. A liner bedding layer will be constructed across the upstream face to protect the liner from damage during installation and operation.

Table 18.35 presents a summary of the liner system selected for the NCW Pond.

Table 18.35 – Non-Contact Water Pond Liner System

Feature	Lining System
Non-Contact Water Pond	HDPE 2 mm Smooth

18.10.10.2 Non-Contact Water Pond Design

The NCW Pond design summary details are presented in Table 18.36.

Table 18.36 – Non-Contact Water Pond Design Summary

Description	Value
Fill Volume (m³)	73,300
Slope Gradient	1V:2.5H
Crest Elevation (mRL)	665
Crest Width (m)	8
Freeboard (m) (1)	2 m (Operating level freeboard) 1 m (Spillway freeboard)
Pond Capacity (m³)	58,500
Pond Maximum Level (mRL)	663

Note:
(1) The operating level freeboard is measured from the dam crest to the spillway invert level, i.e. the maximum water level before the pond starts overflowing via the spillway. The operating level freeboard meets the understanding of the Serbian regulation requirements of 2 m for dams higher than 15 m (SRPS U.C5.020, 1980). The spillway freeboard is measured from the dam embankment crest to the water level in the spillway when conveying the IDF. The freeboard is to be reviewed at detailed design stage.

18.10.10.3 Non-Contact Water Pond Stability Assessment

During the course of the FS, the stability of the slopes within the scheme has been checked in accordance with Canadian Dam Association (CDA, 2014) Rules on Technical Requirements for the Surface Exploration of Mineral Resources Deposits (SEM) and a Seismic Pseudo-static scenario. Slope stability models have been developed using GeoStudio SLOPE/W limit equilibrium software, using the Morgenstern-Price method of analysis with optimised surfaces.

Seepage analysis was completed for to determine the phreatic surface within the slopes. Based on the modelling completed, a phreatic surface is not expected to develop within the slope; this is not unexpected given that the design includes an underdrain running underneath the pond and the embankment.

Based upon assumptions from the PFS, the structures were assigned a classification of “Significant”, and the seismic coefficient used for the pseudo static state reflected this.

The results of the analysis are presented in Table 18.37.

Table 18.37 – Non-Contact Water Pond Stability Analysis Results

Section	Seepage Analysis	Loading scenario	Minimum FoS Required	Location	Computed Overdesign Factor (FoS)
Section 2 - NCWP	Steady-State	SEM	1.3	Downstream (NCWP embankment)	1.3
	Steady-State	CDA Static	1.5	Downstream (NCWP embankment)	1.5
	Steady-State	CDA Pseudo-Static	1.0	Downstream (NCWP embankment)	1.1
	Rapid-Drawdown	SEM	1.3	Upstream (NCWP embankment)	1.4
	Rapid-Drawdown	CDA Static	1.3	Upstream (NCWP embankment)	1.4
	Rapid-Drawdown	CDA Pseudo-Static	1.0	Upstream (NCWP embankment)	1.0

18.10.11 CONSTRUCTION MATERIALS

This section provides a comprehensive overview of the material quantities produced and needed throughout the construction of the Čoka Rakita surface infrastructure project.

18.10.11.1 Material Balance

The material balance for the Čoka Rakita Project is divided into four (4) key construction phases or stages:

- During Stage 1, there will be significant clearing and de-vegetation activities. This phase will generate the largest volume of topsoil and overburden cut material. A significant portion of this overburden will be repurposed as fill material for constructing terraces and pond embankments.

- In Stages 2 and 3, the quantities of cut and fill will be relatively minor, primarily associated with the preparation of the DTSF as the development progresses to higher elevations. De-vegetation will be limited to the upper regions of the natural valley. Overburden and topsoil will be systematically placed on the outer slopes as the DTSF is gradually closed.
- By the end of Stage 4, additional topsoil will be generated during the final layers of DTSF preparation at higher elevations. This material will be utilized as a closure capping layer. A substantial volume of overburden (approximately 180,000 m³) and topsoil (approximately 48,000 m³) will be required to serve as the final closure capping layer for the DTSF.

The summary presented in Table 18.38 includes the major DTSF and ponds earthwork items. The Earthworks balance will be updated as part of the Detailed Design.

Table 18.38 – Summary of DTSF and Ponds Earthworks Items

Structure	Cut (m³)				Fill (m³)		
	Topsoil	OVB	Poor Quality Rock	Good Quality Rock	OVB	Granular Fill	Topsoil
DTSF	65,921	312,710	-	-	192,979	498,529	48,854
Reclaim Pond	9,842	29,892	18,031	21,575	40,507	2,500	1,045
NCW Pond	12,068	39,832	12,738	21,198	76,060	7,180	1,778
CW Pond	9,391	11,981	40	-	100,244	1,700	2,528
Ring Road (North)	12,618	77,408	22,032	179	52,129	10,657	12,618
Ring Road (South)	11,038	52,476	21,821	6,800	72,022	9,471	11,038
Total	**120,878**	**524,299**	**74,662**	**49,752**	**533,941**	**530,037**	**77,861**

18.10.11.2 Borrow Material

The mine site construction requires locally sourced borrow material. Initial DTSF development needs 36,322 m³ of clean rock for PAG Mine Waste Rock containment and 254,167 m³ of overburden fill and wearing course material from the DTSF ring-road. The surface infrastructure has a positive material balance, minimising the need for borrow areas, except for clean rock for road wearing course.

Initially, "clean" non-acid generating (NAG) rock was expected to be sources from the DTSF 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.

18.10.12 DTSF CLOSURE

The FS closure design of the DTSF is presented in detail in "Conceptual Closure Design for Čoka Rakita DTSF and Topsoil Stockpiles" (WSP, 2025i).

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 replacement 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 including:

- Groundwater and surface water quality and flow monitoring, to evaluate potential impacts of the DTSF on the surrounding environment.
- Vegetation growth monitoring, to assess the success of revegetation efforts.
- Landform integrity monitoring, envisaged to involve annual site inspections and data collection associated with the DTSF instrumentation plan (yet to be developed in full) to ensure structural stability and performance.
- Dust monitoring.
- Continued meteorological data collection at the established weather station for the site.

Further recommendation for additional studies with application to closure design is presented in the FS Closure Plan. The main areas of study are summarised below.

- Geochemistry.
- Geotechnical Stability.
- Hydrogeology.
- Material Balance.
- Vegetation.
- Legal Requirements.

18.11 Water Management

18.11.1 Surface Water Management System

18.11.1.1 Introduction and Methodology

As part of the Feasibility Study of the Čoka Rakita Project, the surface water management design was developed for the following areas:

- DTSF.
- Terraces and stockpiles.
- Contact Water Pond (CW Pond).
- Non-contact Water Pond (NCW Pond).

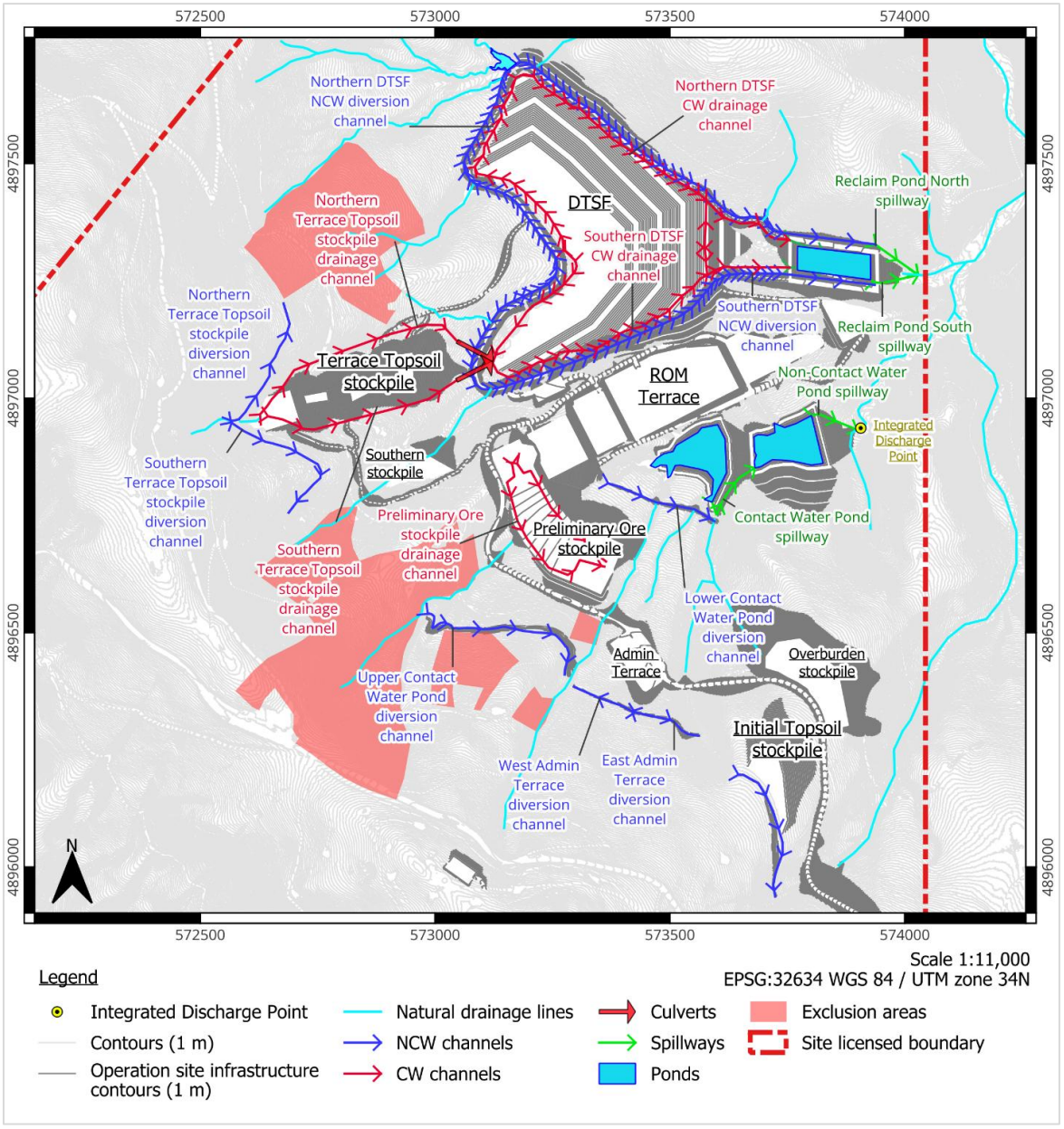
The climatic and hydrological basis adopted in the current report are presented in:

- Čoka Rakita Mine Surface Water Climate and Climate Change Studies (WSP, 2025a).
- Čoka Rakita Mine Surface Water Hydrology Report (WSP, 2025b).

This section describes the sizing of water management ponds, namely the Reclaim Pond, CW Pond and NCW Pond, with their corresponding spillways. The section also includes the sizing and design of the contact water and non-contact water drainage channels.

An overview of the infrastructure is presented in Figure 18.40.

Figure 18.40 – Čoka Rakita Water Management Infrastructure Overview



Source: WSP, 2025

18.11.1.2 *Design Criteria*

For the pond sizing, the 1:100 AEP spill frequency was adopted as the design criteria. A daily time step, rainfall runoff model with soil water budgeting, was used to generate a daily time series of pond inflows. This was combined with operating rules for the pump-out rates to simulate a daily pond water balance.

For the channel sizing, three main events have been adopted for the design criteria in agreement with DPM. These events have been determined from the historic climate timeseries and are as follows:

- 1:10 AEP storm event, for the sizing of the roadside channels around the DTSF.
- 1:100 AEP storm event, for the sizing of the DTSF CW channels, stockpile water channels and CW Pond drainage channels.
- 1:200 AEP storm event, for the sizing of the DTSF NCW diversion channels.
- 1:1,000 AEP storm event, based on the historical climate timeseries, for the sizing of the DTSF and ponds spillways.

The water conveyance structures, such as drainage channels and spillways, were sized based on peak runoff rates computed based on the hydrological and drainage model from the developed conditions, which incorporates FS catchments and drainage alignments.

18.11.1.3 Dry Tailings Storage Facility Water Management

The DTSF water management is presented in Figure 18.40. The Reclaim Pond is designed to store storm runoff and seepage from the DTSF, conveyed by the CW channels. Non-contact water channels intercepts and discharge downstream of the site non-contact runoff from the upstream catchments.

To reduce the cut required for the construction of the NCW channels in sections with a mild slope, a roadside channel has been designed to convey the 1:10 AEP storm, with the 1:200 AEP being conveyed on the road footprint. It is proposed that roadside channels will be lined with CC8 (8 mm thick) concrete canvas. These mild sections of the NCW channels include N1, N3 and S1.

The steep sections of the NCW channels, comprising of N2, N4, N5, and S2, are proposed as rectangular, stepped channels and were designed to convey the 1:200 AEP storm. They will be lined with gabion baskets and Reno mattress, with a reinforced concrete slab protecting the top and base of each step. The stepped design will ensure that the energy is dissipated along the entire channel length.

To effectively manage the runoff from the road and to facilitate the conveyance of the flows into the stepped NCW channels, the road will be sloped towards the non-contact channel. Additional structures like cross drainage will be evenly spaced along the road improving the diversion of flows from the road to the non-contact channels, reducing the erosion risk on the road.

The two (2) channel sections next to the Reclaim Pond, N6 and S3, have been designed as rectangular channels lined with reinforced concrete to reduce the required channel footprint and improve robustness in this critical location.

The final sections of the NCW channels will discharge the NCW downstream of the site. To keep within the site boundary a reinforced concrete chute discharging into a stilling basin has been adopted as a design for both the northern and southern spillways, with the southern chute also collecting the emergency spillway from the Reclaim Pond.

Forming part of the NCW management system around the DTSF, a nominal capacity pond is required to capture natural runoff from the external catchment on the northern side of the DTSF. This pond will have a nominal capacity of approximately 1,000 m³. The pond will receive inflows from the NCW channel N2 and will discharge via a weir to the NCW channel N3.

The contact water (CW) channels capture and convey the runoff generated within the DTSF area and discharge it to the Reclaim Pond. The CW channels will be built in stages to align with the progressive construction of the DTSF. During each DTSF stage, there will be a section of the CW channels with the definitive design and a section which will be temporary. The temporary channels will be covered during the subsequent DTSF raise and will be replaced with new temporary channels, constructed on top the of the latest DTSF stage. This will occur until the DTSF reaches its final design.

All the DTSF CW channels are designed to convey the 1:100 AEP flow event. The DTSF spillways (CWC_N8 and CWC _S7) are designed for the 1:1,000 AEP event and convey the DTSF CW channel flows into to the Reclaim Pond.

For sections with mild slopes, namely sections CW_N1, CW_N3, CW_N4, CW_N7, CW_S1 and CW_S6, the channels have been designed with a trapezoidal shape and riprap lining. To reduce velocities and improve the sediment capture in the steeper section of the contact channels, 0.5 m rock check dams are proposed to be evenly distributed along the channel to ensure velocities are maintained below 3.5 m/s. Check dam spacing between 4 and 15 m has been adopted for the channels CW_N2, CW_S2, CW_S3 and CW_S4.

The steeper downstream sections of the CW channels, which are also referred as DTSF spillways, have been designed for the 1:1,000 AEP event, comprising CW_N5, CW_N6, CW_N8, CW_S5 and CW_S7. Similarly to the NCW stepped channels, these stepped CW channels are designed as rectangular stepped channels lined with gabion baskets and Reno mattress, with reinforced concrete slab protecting the top and base of each step.

The Reclaim Pond receives runoff and seepage from the DTSF via the CW channels and has a total volume of 51,900 m³. Approximately 70% of the available storage volume is used as surge capacity to limit spills from the pond to less than a 1:100 AEP. The required pump out rate of 15 L/s is driven by the optimum drawdown time of the storm surge volume, being approximately 30 days.

The pond is equipped with an emergency spillway designed to convey the 1:1,000 AEP peak flow from the DTSF catchment area. The pond spillway sizing should be revised in the next phase of

studies taking into consideration of the updated design event adopted for the sizing of the NCW diversion channels. The feasibility of adopting a single Integrated Discharge Facility downstream of the Reclaim Pond should also be evaluated in the next phase, combining the two Reclaim Pond stilling basins and the DTSF underdrainage outlet.

The Reclaim Pond is to be lined with a composite liner system of GCL and HDPE. The pond emergency spillway will comprise of a 19 m wide weir on the southern side of the pond containment embankment. Flow from this weir will merge with the southern NCW chute and then discharge to the environment.

18.11.1.4 Terrace and Stockpiles Water Management

Three stockpiles for managing ore, overburden and topsoil require water management infrastructure designed to convey the 1:100 AEP flow event as shown in Figure 18.40. The stockpiles considered, include the following:

- Terrace Topsoil stockpile.
- Preliminary Ore stockpile.
- Initial Topsoil stockpile.

The water management of the Terrace Topsoil stockpile comprises of two (2) NCW diversion channels and two (2) stepped, CW drainage channels. The NCW diversion channels have been designed as trapezoidal channels, with a base width of 2 m, and are lined with CC8 concrete canvas. The stepped CW drainage channels will capture the runoff from the stockpile and ultimately reporting to the DTSF contact water channel system. These CW drainage channels are designed as rectangular, stepped channels. They will be lined with gabion baskets and Reno mattress, with a reinforced concrete slab protecting the top and base of each step.

The Preliminary Ore stockpile water management comprises of a triangular perimeter channel, with a top width of 1.5 m. The channel will have a similar lining system to that adopted for the stockpile, comprising of a HDPE liner with a 200 mm thick granular protection layer. The perimeter channels will feed into the ROM Pad and Terrace area water management channels and ultimately report to the CW Pond.

The Initial Topsoil stockpile water management comprises of an upstream trapezoidal diversion channel. The channel has a top width of 3.4 m and will be lined with CC8 concrete canvas.

18.11.1.5 Contact Water Pond Water Management

The water management around the CW Pond consists of the pond itself, its spillway and NCW diversion channels used to divert water around the CW Pond (see Figure 18.40). The CW Pond primarily receives runoff reporting from the upper terraces of the ROM Pad and Terrace area. In case of a shutdown of the clarifier or water treatment plant, the CW Pond may receive additional

pumped inflows form the Reclaim Pond, the underground mine dewatering and the water recirculated from the Processing Plant.

The total pond volume is 62,000 m³, where only approximately 50% of the available storage volume is used as surge capacity to limit spills from the pond to less than a 1:100 AEP. The required pump out rate of 15 L/s is driven by the optimum drawdown time of the storm surge volume, being approximately 30 days. It should be noted that at the time of the initial pond sizing, all the runoff from the ROM Pad and Terrace area was being managed in the CW Pond.

The CW Pond is to be lined with a composite liner system of GCL and HDPE. The pond emergency spillway will comprise of a 15 m wide weir on the southern side of the pond dam embankment. Flow from the weir discharges via a reinforced concrete chute into a stilling basin within the downstream NCW Pond. The stilling basin is required to protect the toe of the CW Pond containment embankment and the NCW Pond liner from scour in the event that the NCW Pond is empty.

NCW diversion channels intercept and diverts NCW away from the CW Pond to reduce the volume of water captured for treatment. These are identified as the Upper and Lower CW Pond diversion channels and are designed to convey the 1:100 AEP flow event. They are trapezoidal channels with a width of between 3.4 m and 3.9 m and are lined with CC8 concrete canvas.

To direct flow around the Admin Terrace two additional channels were included in the design: the East and West Admin Terrace diversion channels. The East Admin Terrace diversion channel routes water towards the east, where it flows down a natural drainage line and into the CW Pond stilling basin. The West Admin Terrace diversion channel directs water towards the west, where it is eventually collected in the Lower CW Pond diversion channel.

18.11.1.6 Non-Contact Water Pond Water Management

The NCW Pond captures NCW runoff originating from the catchment located south-east of the ROM Pad and Terrace area (see Figure 18.40). This includes the catchment diverted by the Upper and Lower CW Pond diversion channel. The pond currently has a maximum capacity of 58,700 m³. Once the maximum capacity of the pond is reached, it is designed to freely overflow via a 20 m wide weir into a reinforced concrete lined spillway chute and stilling basin to the Integrated Discharge Facility. Water from the pond is used for domestic water supply and can be used for emergency water supply for the site. The pond will be lined with a single layer of smooth 2 mm HDPE liner.

The site has a minimum environmental flow requirement, that under normal operating conditions would be satisfied by the regular discharge from the water treatment plant into the Integrated Discharge Facility. In the case where the water treatment plant is under maintenance and out of operation, it is proposed that minimum environmental flows will be supplemented with water stored in the NCW Pond via a low-level outlet. The outlet design comprises of a 150 mm HDPE pipe that is regulated via a gate valve.

18.11.1.7 *Sediment Management*

A general erosion and sediment management philosophy of preventing erosion and removing sediment at source, has been adopted. In locations where this is not possible, sediment management check dams will be placed in channels to limit the movement of sediment downstream. Despite these measures it is expected that some sediment will settle out within the water management ponds, which may need removal.

Sediment originating from the DTSF is expected to settle out within the DTSF CW channels due to low velocity conditions resulting from the installation of permeable check dams. Finer particles not captured in the CW channels will ultimately be captured in the Reclaim Pond. When the sediment deposited in the Reclaim Pond starts to affect its storage capacity the sediment will need to be removed. High sediment loads are expected to be deposited within the pond following high flow events.

To reduce the sediment load reporting to the channels, localised solutions can be implemented. One potential solution is to reduce the erosion potential of the DTSF by adopting polymer sprays. Another potential solution is to install compost-filled filter socks that will capture sediment from runoff before it enters the CW channels.

Stockpiles are considered areas of high erosion potential. The Terrace Topsoil stockpile sediment will be managed through the DTSF CW channels and Reclaim Pond. For the Preliminary Ore stockpile, it is expected that majority of the sediment will settle into the perimeter channel due to the mild channel slope. Sediment in the channel should then be cleared so that the channel capacity is not significantly reduced.

Additional sediment management is expected to occur within the terraces water management, with excess sediment reporting to the CW Pond. Due to the limited space and temporary use, the Initial Topsoil stockpile sediment management is proposed to occur via either portable sedimentation tanks or geotextile filter bags.

18.11.1.8 *Closure Water Management*

The water management for the closure design is based on the following concepts:

- The DTSF will be progressively rehabilitated during operations.
- The Reclaim Pond, CW Pond and NCW Pond and their associated spillways will be removed as part of closure. All the stockpiles will be removed as well as the associated water management structures.
- All water management structures sized for closure will be designed to the 1:10,000 AEP storm event with climate change. This is to align with the Global Industry Standards on Tailings Management (GISTM) (ICMM, 2020).

As part of the closure plan two different DTSF designs have been considered, a base case terraced design, and a geomorphic influenced design. For both DTSF designs, the same NCW conveyance structures around the DTSF would apply. The existing DTSF NCW channels from the operation stage will be expanded to convey the additional closure peak flows. The area between the DTSF and the NCW channels will require backfilling to convey the closed DTSF runoff into the closure channels. The closure NCW channel top widths would range between 8 and 17 m.

Riprap was initially proposed as the channel lining for the closure NCW channels. However, due to the potential for scour, the riprap sizes required are excessive and not practical. Two alternative solutions have been evaluated: block stone channel lining and mass concrete interlocking blocks to stabilize the channel bed.

The block stone channel lining has been taken forward as preferred option, with mass concrete blocks to be used if sufficient quantities and quality of block stone cannot be sourced.

Depending on the design chosen for closure, long-term maintenance may be required for the closure channels.

18.11.2 Waste Water Treatment Plant

The waste water treatment Plant is described in Section 18.6.5.13 of the Report.

18.11.3 Site-Wide Water Balance

The Čoka Rakita site-wide Water Balance Model (WBM) and Water Quality Model (WQM) assessed flows and transfers at the Čoka Rakita Project. The model was built using GoldSim™ software and encompasses the entire LOM, from pre-operations through active closure, building upon and refining previous work from the PFS with updated inputs and logic.

18.11.3.1 Purpose and Methodology

The model was designed to appraise internal water transfer pump rates, mining process water consumption, storage pond sizing, and environmental discharge rates. It incorporates all on-site facilities, including the DTSF, and integrates a contaminant transport component to assess Site water qualities. The model was run stochastically for 1,000 realisations to evaluate water management under varying climatic conditions such as droughts and extreme rainfall. Although the current LOM is 9 years, at the time of developing the WBM it was taken to be 8 years. As a result, the three (3) defined LOM phases simulated by the WBM are as follows:

- Pre-operations and start of decline construction (Year -2 to Year -1).
- Operations (Year 1 to Year 8).
- Active closure (Year 9 to Year 11, also described as Closure Year 0 to Closure Year 2).

18.11.3.2 Key Inputs and Scenarios

Model parameters included process plant demands, pond geometries and pump rates to maintain a spill frequency for the Reclaim Pond and Contact Water Pond with less than a 1:100 Annual Exceedance Probability (AEP). Catchment areas were delineated from the agreed site layout. Source terms were derived from available geochemical test work and monitoring data as inputs into the WQM. The stochastic climate model parameters reflected both historical climate data and climate change projections to stress-test future resilience. A critical inflow to the model is underground dewatering predictions from a 3D numerical groundwater model, providing high, normal, and low inflow rate scenarios based on varying hydraulic conductivities.

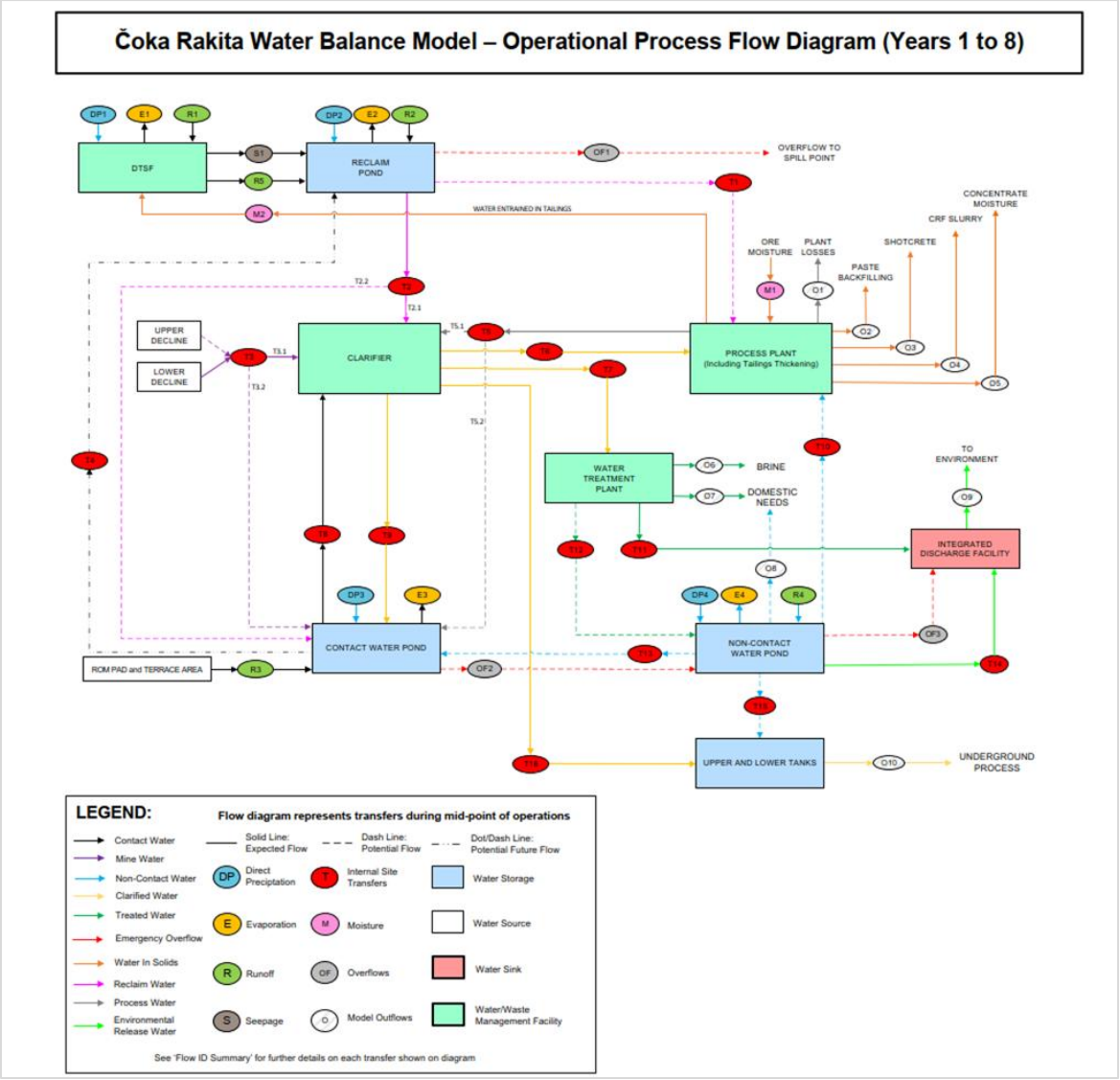
The groundwater inflow rates used within the WBM were an earlier iteration of the groundwater model comparing to what has been presented in (WSP, 2025c). The most recent modelled groundwater inflows show a reduction in average groundwater inflows across the LOM of approximately 14%, and a reduction in maximum annual inflows of approximately 8% for all three groundwater inflows scenarios.

The study evaluated a Base Case and three additional scenarios:

- Base Case: Climate model using historic climate data parameters, with high groundwater inflows, with water quality modelling.
- Climate Change: Climate model using climate change data parameters, with high groundwater inflows, no water quality modelling.
- Low Groundwater Inflow: Climate model using historic climate data parameters, with low groundwater inflows, no water quality modelling.
- Normal Groundwater Inflow: Climate model using historic climate data parameters, with normal groundwater inflows, no water quality modelling.

The WBM includes all phases of the LOM, including pre-operations, operations and active closure. A flow diagram representing the model logic during the mine operations stage is provided in Figure 18.41.

Figure 18.41– Site Water – Process Flow Diagram



Source: WSP, 2025

18.11.3.3 Key Findings

- Operational Demands: The Site's operational water demands are met across all simulated scenarios throughout the LOM. For the Base Case and Climate Change scenarios, this is primarily due to underground dewatering consistently exceeding process requirements. In low and normal groundwater inflow scenarios mine dewatering alone may not suffice, however additional inflows to the Clarifier (from Reclaim Pond, CW Pond, and Process Plant return water) ensure demands are met.

- Non-Contact Water Pond Sizing: While the Base Case and Climate Change scenarios indicate the mine can largely function without significant NCW supply, the Low Groundwater Inflow scenario highlights a potential need for a significant supply of NCW. In this scenario NCW resources were drawn down to zero in pre-operations phase, with an increasing need in later operational stages. Therefore, a further review of the NCW Pond sizing is recommended for future planning.
- Spill Frequencies: The model confirms that the Reclaim Pond and CW Pond spill frequencies remain below the target 1:100 (1%) AEP, achieving 1:385 AEP and 1:333 AEP respectively, indicating there is scope to reduce the pump rates of water from the ponds in order to provide a more consistent water supply to the Clarifier and WWTP.
- Environmental Flows: Minimum environmental flows are met for the Base Case, Climate Change and Normal Groundwater Inflow scenarios throughout the LOM. However, the Low Groundwater Inflow scenario indicates potential shortfalls during pre-operations, attributed to the lower groundwater inflows.
- Treatment Facility Rates: The Clarifier and WWTP flow rates adjust according to water availability, with higher rates in scenarios with greater water supply and vice-versa. Depending on the scenario, average Clarifier flow rates vary between 4,194 m³/d and 5,737 m³/d during operations. Similarly, WWTP flow rates vary from 1,596 m³/d to 3,070 m³/d.
- Contaminant Transport: The WQM results were compared with derived Class I water quality limits. The average results for the NCW Pond, which discharges to the environment via the integrated discharge facility, showed potential exceedances of chromium, copper, iron, and mercury. This is due to the natural runoff water quality, the main source term to the NCW Pond, exceeding Class 1 limits for these parameters. Moreover, evaporation of the NCW Pond further increases the concentration beyond the limits. The Reclaim Pond, which overflows to the environment with less than a 1:100 (1%) AEP via the emergency spillway, showed average results that exceeded water quality limits for arsenic, chromium, copper, mercury, lead, sulphate, and zinc. Therefore, emergency overflows to the environment must be prevented where possible to negate the exceedance of Class 1 water quality in the emergency discharge.

18.12 Power Supply and Distribution

18.12.1 Krst Substation and Local Voltage Regulation

The Krst 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.

The 35 kV power supply will feed a pair of transformers, each with a capacity of 16 MVA, 35/10.5 kV located at Čoka Rakita. 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.

Voltage regulation at the plant will be supported by an appropriately sized capacitor bank and harmonic filter installed on the 10.5 kV bus. The capacitors have been sized to provide a power factor of 0.95 at the projected Peak Load for the site. This equipment will improve site voltage regulation and reduce demand from the utility.

Two (2) standby 1.4 MW diesel generators will be provided for emergency backup power to the mine and critical processes within the plant.

18.12.2 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 18.39 presents the nominal voltage levels used on-site.

Table 18.39 – 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

18.12.3 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.

To supply power to each of these electrical rooms, 10 kV cables will be run in cable tray from main substation to each individual E Room. Where feasible, Ring Main Units (RMUs) will be employed

to provide a redundant power supply to each E Room, meeting DPM's requirements for redundancy within the system.

18.12.4 Refurbishment of Ada Tepe Equipment

To minimise equipment costs across the site, DPM has requested the reuse of the existing Ada Tepe electrical equipment, wherever feasible. For the FS, the equipment manufacturer has been contacted and provided with a detailed 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.
- Review structural integrity of buildings and perform repairs, where applicable.

18.12.5 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.

18.12.6 Site Load

The total power demand is estimated at 14.4 MVA with 7.0 MVA for the surface plants, 6.4 MVA for the underground mine and ventilation facility, and 1.0 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.

18.13 Automation

The overall control system philosophy decided on with DPM for the FS was as follows:

- A central control room located in the process plant equipped with a SCADA control system will include all servers, HMI, camera monitors, and control stations for both the process plant and the underground mine.
- One (1) small local control room located in the Paste Plant including one Operators station to control the Paste Plant and Filter building.
- The preliminary estimates for this system will include the following:
  - Four (4) SCADA servers (2 for surface, 2 for underground).
  - One (1) PI Historian (virtual server).
  - Four (4) Operator Consoles located in the central control room (2 for surface, 2 for underground).
  - Operator Screens.
  - One (1) CCTV Server (virtual server).
  - Two (2) Application workstations (virtual machines).
  - One (1) Business Application workstation (virtual machines).
- Each process area should have its own PLC cabinet including processor and IO modules. All programming for each process area shall be in this processor.
- In general, the use of Remote IO cabinets should be minimised. IO should be located in the PLC cabinets located in the E-Room.
- Where possible, the Čoka Rakita site will attempt to re-use existing PLC cabinets installed in the existing E-Rooms. IO Cards, Processors, and other components will be replaced in the factory based on design and engineering assessment.
- The existing Ada Tepe site utilises the Schneider Electric platform for all SCADA. DPM prefers to utilise an equivalent system for Čoka Rakita.
- Redundancy in the system will be achieved by way of a ring topology for all SCADA communications between the process PLCs and the central control room.
- It is assumed that licences for SCADA will be purchased on a 3-year contract from the manufacturer.

18.14 Communication

The Čoka Rakita Gold mining 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

Čoka Rakita will produce two (2) different concentrates, specifically a gold pyrite flotation concentrate and a high-grade gravity gold concentrate, and a gold doré. The potential markets and expected terms for these products are outlined in this section.

The gold in the flotation concentrate represents just over 50% of the LOM gold production at Čoka Rakita. The gold grade in flotation concentrate is expected to be approximately 100 g/dmt for the first 3-4 years of production and gradually dropping off to approximately 40 g/dmt by Year 10. Copper and silver are expected to be below payable thresholds. The flotation concentrate is clean with respect to impurity levels and therefore no penalty charges are expected to be levied. As a result of the high-grade gold content and the lack of impurities in the flotation concentrate, there will be many options for sales to smelters globally, mainly in Asia, Europe and North America, as well as to trading companies. DPM has extensive experience in marketing gold flotation concentrates and the terms provided in this study are based on its market knowledge and discussions with potential buyers of these concentrates. While market terms fluctuate over time, the terms provided represent DPM's vies of the long-term market for this quality of concentrate

The gold in the gravity concentrate represents approximately 20% of the gold production over the mine life. The gold grade in the gravity concentrate starts at over 2,000 g/dmt in the initial years, falling to just under 1,000 g/dmt in the final years. While copper smelters are potential buyers for these concentrates, DPM has identified gold refineries that can receive these types of materials and pay for a higher percentage of the gold. While the market is smaller, DPM has received indicative terms on which it is basing the terms provided in this study.

The gold in Čoka Rakita's doré production represents approximately 30% of LOM gold production. The doré grade is expected to be approximately 95% gold and without impurities. There are several gold refiners in Europe that will be possible buyers for this material. European refineries will provide the simplest logistics, which is a crucial factor in selecting a customer. Doré terms are relatively standard in the industry, so logistics and security are important considerations. The commercial terms for doré provided herein are based on the typical market terms for standard doré.

The gold market is mature globally with numerous reputable smelters, refiners and traders located throughout the world with whom DPM has extensive relationships. As such, a market study for gold products or for the expected price of gold was not undertaken.

19.2 Treatment Charges and Other Selling Costs

The terms contained within any future sales contracts are expected to be typical and consistent with standard industry practice and contracts the sales of gold concentrates and doré.

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 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	100	-
Refining Cost (Gold)	$/oz	6.0	6.0	0.4[2]
Penalty	$/dmt	-	-	-
Gold Payable	%	99.8	97.5[1]	99.9
Concentrate Freight	$/wmt	100	100[4]	0
Doré Freight				
Shipping Charge	$/oz			0.75
Handling Charge[3]	$/lot			280
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 refining charge is $500/lot
3. Lot size is expected to be 50-100 kg, US$280.00 per lot up to 100 kg, additional US$80.00 per additional 100 kg or fraction thereof
4. This will either be inland freight + freight cost to Asia, or inland freight to local smelter + freight capture that smelter will add to the TC

19.3 Commodity Price

The QP, Daniel Gagnon, P. Eng. has adopted the following price projection for the FS financial model base case as presented in Table 19.2. The selected gold price is conservative compared to three-year trailing average of US$ 2,387/oz or Q3 2025 average spot price of US$3,454/oz.

Table 19.2 – Commodity Pricing

Element	Unit	Financial Model
Au	US$/oz	1,900

19.4 Contracts and Off-Takes

No sales contracts have been established to-date for either of the Čoka Rakita concentrate qualities or the doré. However, DPM expects, based on its experience in selling Chelopech pyrite and gold-copper complex concentrates, that the full production of both concentrates will be sold in line with the above terms.

Contract terms and treatment charges will be negotiated with international smelting, refining and trading 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 by the London Bullion Market Association (LBMA).

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 2026. This Section addresses the current Project footprint of the mine infrastructure and landforms described in this FS.

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 four 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 FS 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 Lu Gjori 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 will be produced and transported via conveyor to the DTSF area and a paste backfill plant which will utilise dry filter cake tails in the backfill paste production.

Contact water collected from mine dewatering and surface runoff from process areas, roads, stockpiles and the DTSF and will be routed to two contact water collection ponds (a DTSF reclaim pond and the CW pond). All contact water collected will be sent to the WWTP 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 NCW pond is proposed to provide water supply contingency for the process plant and domestic use. There will be a soil storage facility (stockpiles) located to the west of the DTSF and 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 18.11) has been developed for the FS 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 discharged to the Reclaim Pond before pumping to the WWTP for treatment prior to discharge to the environment. Non-contact water diversions will be constructed to divert water away from the DTSF and will report to spillways prior to discharging downstream of the Reclaim Pond into the environment.

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 lower 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, contact water pond or further pumped to the WWTP 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 pumped to the clarifier and the WWTP 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 NCW away from infrastructure). Freshwater make-up for the process plant will be supplied in the first instance from the clarifier, and NCW pond (when required). Treated water will be pumped from the WWTP to the Integrated Discharge Facility from which it will be discharged to the Ogasu Lu Gjori stream which drains into the Lipa River. Figure 18.40 illustrates the surface water management approach.

Two (2) contact water ponds CW and the 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 CW pond emergency overflow spillway would drain into the NCW pond.

The NCW pond will have a single liner system and will supply domestic and emergency water. Once the NCW pond is full, water will be directed via a spillway to the Integrated Discharge Point. A spillway has also been designed beneath the NCW pond, this flows underneath the pond and discharges to the Integrated Discharge Point. Water from the Integrated Discharge Point will be discharged to the environment into the Ogašu Lu Gjori Stream.

On over drain system will be constructed above the DTSF liner at the base of the tailings to prevent re-saturation of the DTSF.

Process water balance calculations and diagrams have been developed in the site-wide water balance (Section 18.11.3). 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 Preliminary Economic Assessment and Pre-Feasibility studies. 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.

20.5 Permitting

The Project has prepared a Permitting Strategy (DPM, 2025) 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 five (5) exploration licenses (Potaj Čuka, Pešter Jug, Čoka Rakita, Tulare and Umka). The Potaj Čuka license hosts the Timok Gold Project, which was advanced to a Prefeasibility Study (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 (Čukaru Peki copper-gold mine operated by Zijin Mining is the most recently permitted project and has been in operation for four (4) 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).
- 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 third year of exploration and will apply for a renewal of the exploration licence when the initial

three (3) years are completed in late 2025. The MME has approved the Certificate of Resources and Reserves of Mineral Resources.

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:

- A Certificate of Mineral Resources and Reserves.
- Confirmation of compliance with the Spatial Plan.
- 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 the shared permitting jurisdiction between the 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. DPM initiated the SPSPA process on April 2, 2024, with the MCTI. The MCTI approved the development of the SPSPA process on the November 14, 2025. 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 waiting on a decision by the Serbian government to initiate the development of the SPSPA, DPM's approach has been to have 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 the Institute of Architecture and Urban and Spatial Planning Serbia, who is then appointed, and the overall process is owned by the MCTI.

DPM's role will be limited to a stakeholder; however, project information and data will be provided by DPM.

The Spatial Plan (SP) will require a Strategic Environmental Assessment (SEA). The SP 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 which the Project is currently gathering will be shared with the plan developer to ensure that the SEA that will accompany the SP will be compatible with the Project-specific EIA. Close engagement with the plan developer during the SEA development process is highly recommended to ensure proper and efficient guidance of the process. At the time of the SEA Report development, the land acquisition approach for the entire project area should also be finalised.

The process of development of a plan takes approximately 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.

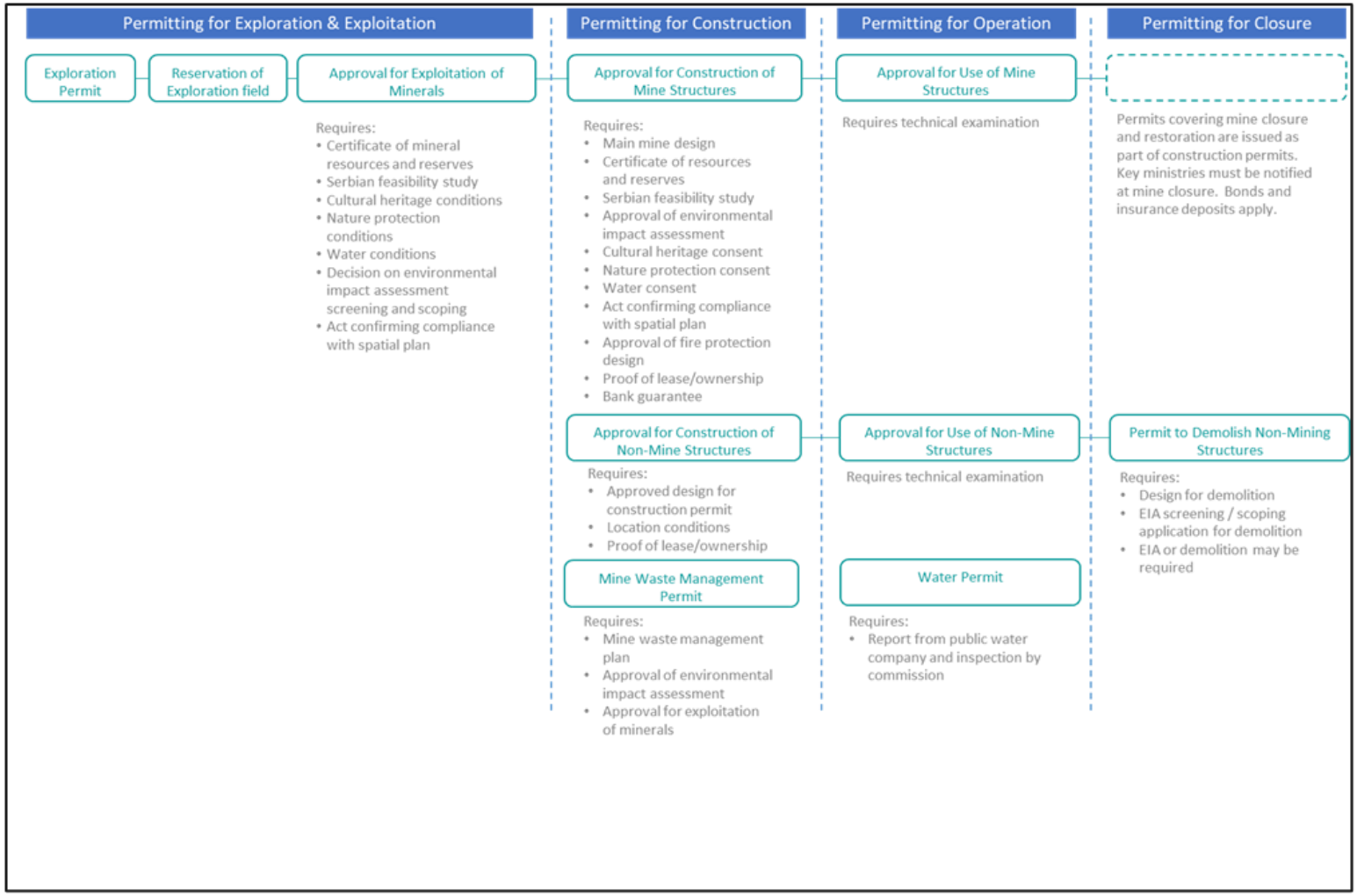
The Serbian Feasibility Study will build on the studies developed as part of the FS.

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 EIA 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.

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.

Figure 20.1 – Serbian Permitting Requirements



Source: ERM, 2024

The delay in starting the SP process in Serbia 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 Žagubica, on the mountain range between Bor to the southeast and Žagubica and Laznica to the west. Majdanpek 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 at 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.
- Traffic and transport study.
- Further cultural heritage baseline data collection (buried archaeology, historical farms, intangible heritage, particularly for certain minority ethnic groups).
- Identifying local users of water.

- 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.

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 June 2025 and periodically reviewed throughout the FS. 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; small areas of habitats that may host protected species; adverse changes in laws and delays to the permitting process; increase in local community opposition to the project; 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 contact water pond (Figure 18.40).

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, Bigar stream, Valja Saka stream, Ogašu Lu Gjori Stream, Lipa River, Vrkaluca stream, Ogašu Krloši stream and Valja Srteži stream. The flow of surface water is monitored continuously, and data is collected monthly, while the quality of surface water is tested monthly. Groundwater wells have been constructed and data collection for the Project area has commenced. Besides monitoring of the groundwater quality, a comprehensive hydrogeological monitoring has been introduced with piezometers equipped with divers or vibrating wire devices. A detailed hydrological study has been completed, and a 3D numerical groundwater flow model was developed for the project area.	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. Sixteen (16) surface water locations will be monitored (13 on a monthly basis, three (3) on a quarterly basis). thirty-nine (39) groundwater monitoring wells will be monitored for groundwater quality, twenty-four (24) on a monthly basis and fifteen (15) quarterly. In addition, six (6) groundwater monitoring wells in the adjacent Timok Project area will continue to be monitored on a quarterly basis. Twenty (20) springs will be monitored on a monthly basis and four (4) on quarterly basis. Hydrogeological monitoring includes 37 vibrating wire piezometers, and on Sixty-one (61) groundwater level is measured.
Terrestrial ecology - biodiversity	Habitat and species surveys were undertaken in 2013 and 2024 in the Project area and a wider research area. Surveys included habitats and flora, aquatic microbiology, mammals, birds, amphibians, invertebrates and reptiles. Biodiversity surveys concluded in 2024, and a report of the study produced in Q2 2025. 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.	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 2013 and 2024 in the Project area and a wider research area. Surveys included phytobenthos, microbiology, 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 (3) locations, monitoring for total suspended solids and gases (NO_x and SO_x) is planned for five (5) locations.
Noise and vibration	Summer monitoring campaign undertaken in 2022. Winter monitoring campaign undertaken in March 2025 at 20 locations. Additional monitoring locations were included to the northeast. Campaigns are two (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.	No further work planned.

Environment	Work Completed	Work Planned
Social baseline	Desktop review of secondary publicly available data completed. A socioeconomic baseline study was undertaken in Q4 2024, this characterised a Social Area of Influence and included field work to gather primary data through a combination of key informant interviews and focus group discussions. Based on the results of the socioeconomic baseline study a socioeconomic impact assessment was undertaken.	No further work planned at present.
Land acquisition	Land Ownership has been surveyed and a Land Acquisition Strategy prepared based on official cadaster data and information collected from landowners. Meeting with landowners was undertaken to inform them of the method of the property assessment.	Land acquisition is on-going.
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 on-going.	Soil descriptions and sampling for land capacity (agricultural) and chemical quality (pollution) required by Law on Soils Protection.
Geochemistry	Sampling, static testing and analytical programme to characterise ARD/ML (ARD/ML) capacity of waste rock, tailings and overburden material including soil to be disturbed has been conducted in 2023/2024. Humidity Cell testwork (HCT) started in 2025 to determine the long-term ARD/ML capacity of rocks.	Ongoing HCT studies for ARD/ML capacity to be conducted during 2025. Solidified tailings test according to Eurocode NEN 7375 is on-going.

Environment	Work Completed	Work Planned
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 / geophysical survey.

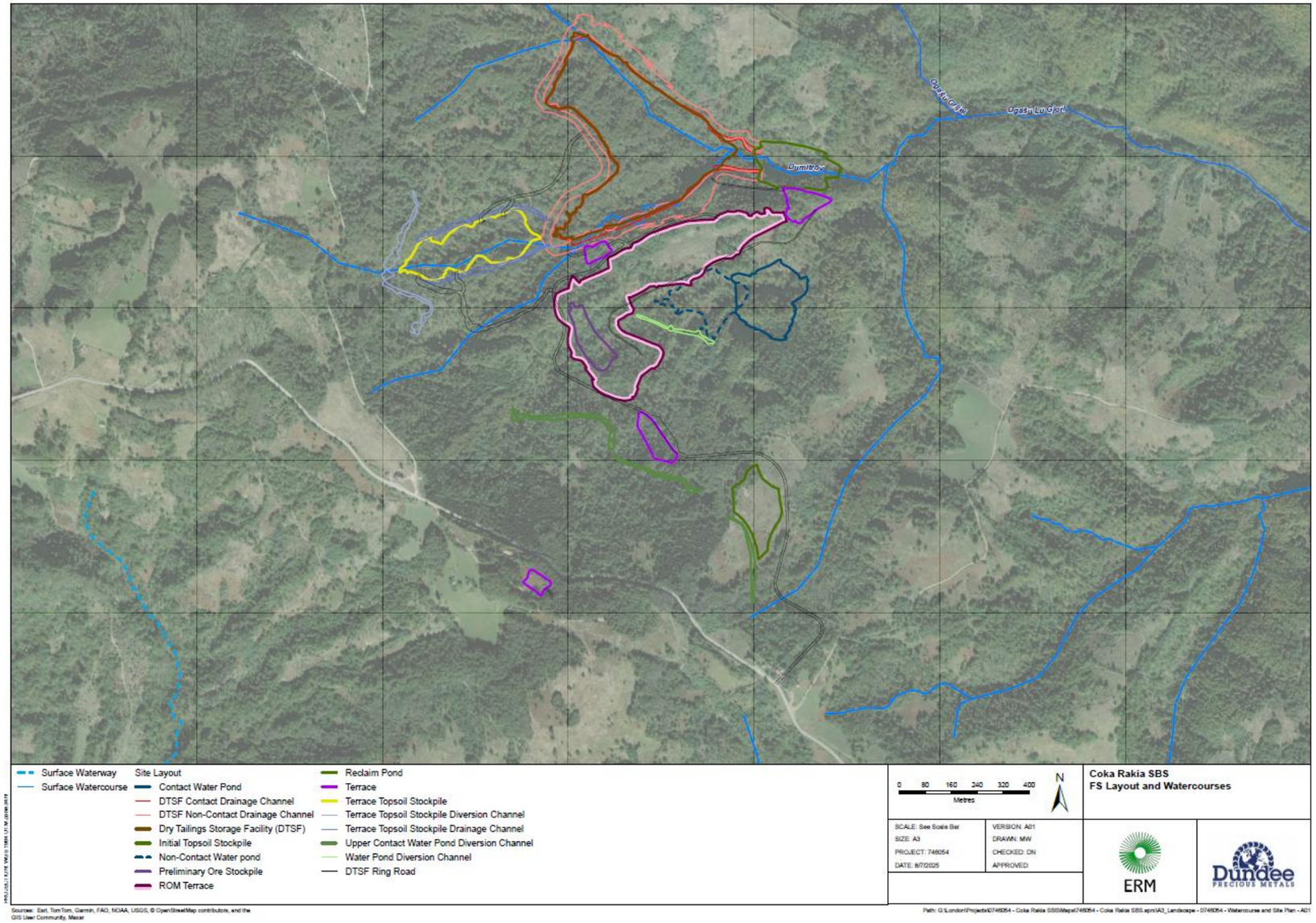
Source: ERM, 2025

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 FS Area



Source: ERM, 2025

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 storm-related flood events. Control of these risks is a design objective of the mine water management plan (Section 18.11.3).

An underdrainage system has been designed beneath the DSTF liner allowing NCW to drain directly to the environment, discharge will be to the Ogašu Lu Gjori Stream.

All CW 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 be used in underground processes, dust suppression, domestic maintenance water, and for maintaining green areas and gardens on site, any excess water will be discharged to the environment.

No part of the Project site lies within existing water supply protection zones. Dewatering required for underground mining may affect water availability for several seasonally active households and weekend houses in vicinity, as well as possible impacts to springs, wells and streams, some used seasonally for agriculture, in a small part of the large Crna Reka catchment to the south. Groundwater models and a water balance are required for the wider catchment area 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. Part of the wider research area was surveyed in 2013 as part of the Timok Gold Project, further 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 Figure 20.3) 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. 26 of these plants are listed on national legislation as protected, 34 species are designated as Least Concern on the global ICUN 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.

Aquatic surveys in the project area identified a total of 173 taxa of macroinvertebrates from 22 taxonomic groups in the study area, a significant diversity of species. The species *Epeorus yougoslavicus* (a species of mayfly) 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 stream, in the Ogašu lu Gjori, Ogašu Krloši and Vrkaluca streams and in Bigar, which include nationally protected species and those listed in Annexes II and IV of the EU habitat directive.

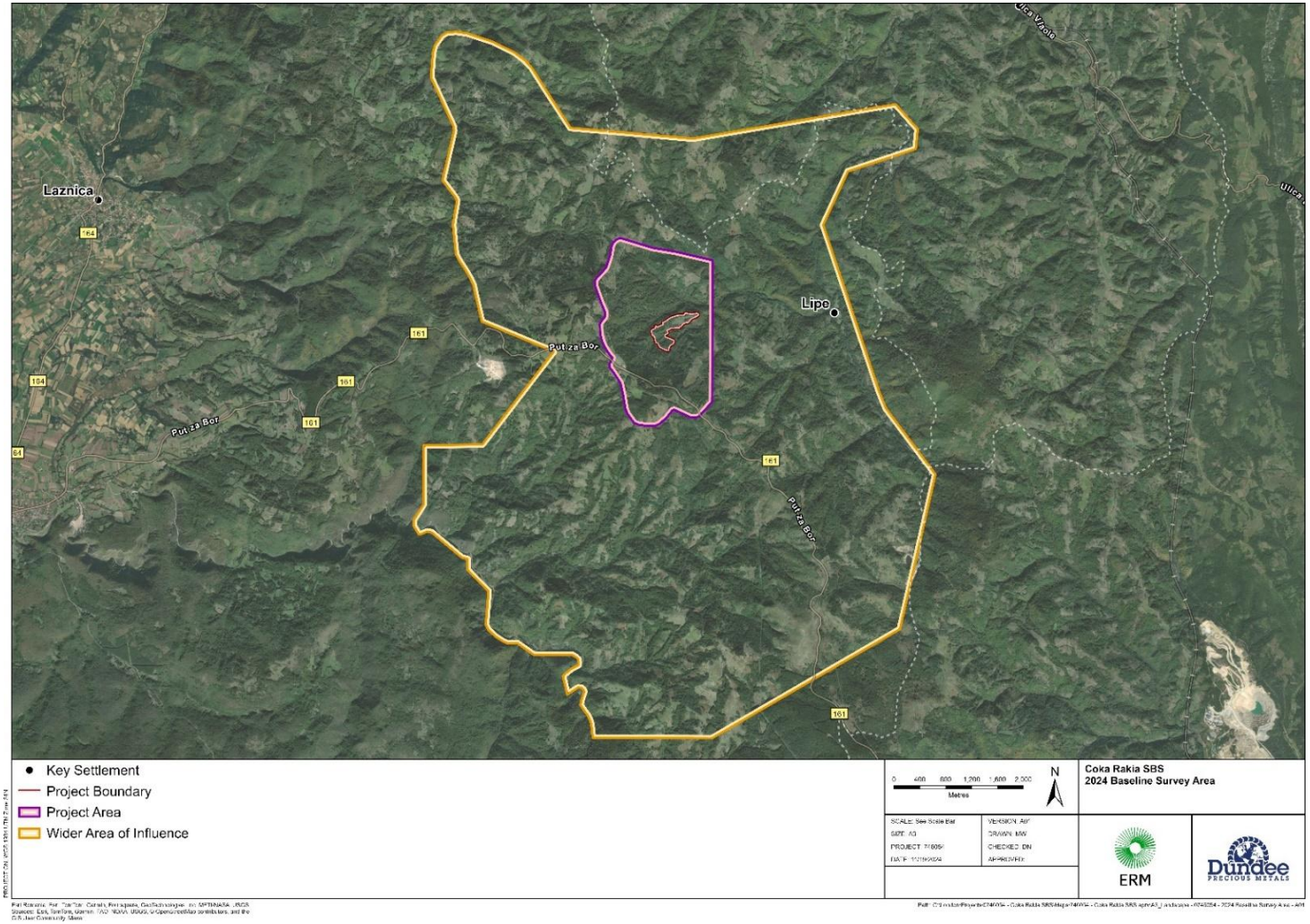
Aquatic surveys recorded a low abundance of fish species with only three (3) species being recorded. Two (2) of these species, bullhead (*Cottus gobio*) and the brook barbel (*Barbus balcanicus*), are protected under national legislation. During these surveys it was noted that the Lipa River water was 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 in the Dumitrov stream, Ogašu lu Gjori, Ogašu Krloši and Vrkaluca streams. This species is 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, eight 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 II and IV of the Habitats Directive or Near Threatened on the IUCN Red List.

Figure 20.3 – 2024 Biodiversity Baseline Survey Area of Influence



Source: ERM, 2024

Bird surveys were undertaken in spring 2024. The surveys recorded a total of 64 species and of these 51 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 wider area of the projects influence. 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 18 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 ICUN Red List.

Field surveys identified 19 species of mammals (excluding bats) including four (4) species listed in Annex IV of the Habitats Directive: the Hazel dormouse (*Muscardinus avellanarius*), Grey wolf (*Canis lupus*), European wildcat (*Felis silvestris*) 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, parts of which are located within the Crna Reka catchment. The Nature Park is currently in the process of being upgraded to National Park status. This area is a forested mountainous region which hosts a number of endangered 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 and 2025, 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. 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. The winter campaign was completed in early 2025. Additional monitoring locations to the northeast of the Project area were identified and surveyed. All three (3) campaigns were two (2) weeks long with attended and unattended measurements at identified receptor locations over and around the Project footprint.

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. Air quality and noise models have been developed however 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.

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. Limit values were exceeded in some sample locations for heavy metals including antimony, arsenic, copper, barium, chromium, zinc, cadmium, cobalt, nickel, lead, vanadium and mercury, all likely naturally occurring. Consistent with the FS planned layout, where topsoils are to be disturbed, the principle will be to conserve these in stockpiles for ultimate reuse as a rehabilitation medium. The disturbance and re-use or long-term storage of soil and subsoils from these areas will need to account for potential contamination, the potential for Acid Rock Drainage / Metal Leaching (ARD/ML) has been assessed in a comprehensive screening program using industry-standard static and kinetic testing. The results of the geochemical assessment are discussed further in Section 20.8.1.

It is noted that many of the Project infrastructure locations, especially access roads and the DTSF, are on very steep terrain. Geotechnical studies are on-going and further detailed investigations to address landslide and ground instability risks are planned to inform comprehensive risk assessments and mitigation measures.

Detailed geological mapping by DPM and others has been available for the FS, 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 and archaeo-geophysical 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

A Social Impact Assessment (SIA) was conducted for the Project in 2024 (ERM, 2025). Key socio-economic characteristics of the Project area are summarised as follows:

- Significant out migration of youth for employment and education opportunities elsewhere in the country or abroad, leading to an aging population.
- Key land uses in the area include agriculture, livestock rearing, and forestry.
- The Project area has higher rates of youth unemployment compared to national averages, with key employers being the Mining and Quarrying sector and Wholesale Trade sector. The aging population and prevailing economic conditions have led to a locally depressed economy in Žagubica Municipality.
- Many households in the Project area rely on agriculture, livestock rearing and forestry for livelihoods and subsistence.
- 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. However, it should be noted that the area where the surface infrastructure is located, which accounts for the majority of the land acquired, is not used in this way.
- Incomes are lower than national averages. Households, especially the elderly in the Project area, rely on remittances from family members living abroad to meet their basic needs.
- Education at the elementary level is good in the Project area. However, there are plans to potentially close an elementary school due to the lack of students in the area. High school and tertiary education opportunities are limited. Consequently, educational attainment is lower in the Project area than the national average.
- Prominent health issues in the Project area are characteristic of an aging population and the demands for healthcare are high due to this. However, the healthcare services available in the Project area are limited and lack funding. Many people have to travel significant distances to receive specialised medical care. The costs associated with this can be detrimental to low income (vulnerable) households.
- Infrastructure and service delivery in the wider Project area of influence is mixed, with good electricity connections, water supply and good telecommunications within the main settlements of Žagubica and Laznica. The areas surrounding the settlements, including isolated homes and agricultural holdings which do not have access to grid electricity or a piped water supply. These households rely on natural water sources such as springs that may run dry during periods of drought. Some isolated areas are not covered by the telecommunications network, including the mobile network. There is a no public sewerage network and the transport network is not well connected to other parts of the country with certain roads requiring substantial upgrades.
- Anti-mining protests have also been taking place in Žagubica Municipality. Protests in the municipality in 2024 focused on the exploration activities of DPM and the use of drinking

water in the exploration drilling process. Activists from elsewhere in Serbia have travelled to Žagubica to protest against DPM.

- The proposed Project is expected to generate positive impacts at local, regional and national levels through the generation of direct and indirect jobs - to different extents - during the different phases of the Project, creation of long-term benefits associated with capacity enhancement of local Serbian labour force through on-the-job and formal training, increased spending capacity, a growing market to cover supply chain needs, and tax and royalty payments.
- At a much smaller scale, the Project is likely to result in economic displacement caused by land acquisition for the Project and access restrictions to ecosystem services used for subsistence and livelihoods. However, economic displacement was taken into consideration in the land acquisition strategy and in the determination of the unit value of the properties to compensate landowners for it. At this stage, no physical displacement or resettlement is envisaged.
- The presence of the Project could affect the health, safety and security of the communities in the Project area as a result of increased road accidents and road trespassing and environmental health impacts (e.g. noise, air quality and water quality). It was also found that stakeholders engaged with during the primary data collection noted road safety and environmental health as key concerns relating to the Project. The decommissioning phase of the Project will be associated with a marked decrease in employment associated with the mine and wider supply chains. Given the limited alternative employment opportunities in the area, it can be expected that communities in the Project area will have become highly reliant on the mine as a source of income during the construction and operational phases. Likewise, Žagubica Municipality may have become dependent on contributions to the municipal budget as a result of the presence of the mine and the associated economic activity. As such, they are highly vulnerable to the mine's closure and the resultant loss of direct and indirect employment unless alternative economic activities are developed throughout the life of the mine. There is also a potential for an economic downturn in the region following mine closure.

Management plans including a Stakeholder Engagement Plan, Livelihood Restoration Plan, Community Health and Safety Management Plan and Conceptual Closure Plan will form part of an overarching Social Management Framework. The Framework is a system to address and manage the socio-economic impacts of activities during the Project lifecycle for Čoka Rakita. Other recommended management plans that will form part of the framework include project level Code of Conduct, Human Resources Policy, Health, Safety, Security and Environment Policy, Human Rights Policy, Contractor and Supplier Management Plan, Retrenchment Plan, Recruitment and Procurement Procedure, Workers Grievance Mechanism, Occupational Health and Safety (OHS) Management Plan, Emergency Preparedness and Response Plan (EPRP), Traffic Management Plan, and Security Management Plan.

20.6.8.1 Land Acquisition

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, and 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.

DPM developed a Land Acquisition Strategy (DPM, 2024) 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 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. The Project maintains an office in the town of Žagubica that employs local

people to be an interface between the Project and the community. Community members are able to visit the office to ask questions, raise concerns, and access information about the Project.

During the SIA process, as part of the primary data collection, stakeholders ranging from local authorities, Non-governmental Organisation (NGO) representatives and local community members, were asked about their perception of the Project, including their concerns and views as to whether there may be any benefits.

It was found that stakeholders interviewed generally expressed a positive sentiment towards the Project and that the mining sector in the region is seen as a key source of employment. Stakeholders expressed that they have already seen benefits from the mining exploration activities for the Project and others in the area, particularly in the hospitality sector. The Project is seen as positive in that it will provide local employment, especially for youth, and may lead to a reduction in their emigration in search of education and employment opportunities elsewhere.

- Most stakeholders interviewed stated that they were not well informed about the Project. This has created growing concerns and confusion around potential impacts. The information vacuum that has been created due to the lack of engagement has led many local residents to take heed of the warnings from environmental activists that have become more prominent in the area. Many concerns about the Project have stemmed from lack of information and confusion between various proposed mining projects in the area since 2004.

Engagement activities include:

- Direct engagement with local community members on an individual basis.
- Engagement with community groups (schools, hunting groups, local institutions, and organizations).
- 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.

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.

Baselines surveys have identified the presence of IUCN Red List Vulnerable species and species listed in Annex II and Annex IV of the Habitat Directive, which meets 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 EIA 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

To evaluate the potential for metal leaching (ML) and acid rock drainage (ARD) associated with mine development at Čoka Rakita, ERM has worked with DPM to implement a phased geochemical characterisation program. This program includes both static and kinetic testing of representative rock and tailings materials, designed to inform environmental management strategies for mine waste including material segregation and storage and predictive water quality modelling. The testing program is being conducted in alignment with Serbian environmental regulations and European Union and draws on international best practices such as those outlined in the Global Acid Rock Drainage (GARD) and Mine Environment Neutral Drainage (MEND) guidelines.

20.8.1.1 Static Testing

The static testing program comprises a total of 101 samples, including 29 composite samples from the initial Phase 1 campaign and 72 variability samples selected for Phase 2. These samples were chosen to represent the lithological diversity and spatial distribution of ore and waste rock units expected to be disturbed during underground development and construction activities.

Phase 2 sample selection was guided by a lithogeochemical classification (LithClass), which was developed using multivariate analysis of drill-core assay data to identify compositionally consistent material types. Sample numbers were determined based on MEND guidelines and the relative proportions of each lithology within the planned development envelope.

Static testing included:

- Acid-base accounting (ABA) using the modified Sobek method.
- Sulphur speciation (total S, sulphide-S, sulphate-S).
- Neutralization potential (NP) and acid potential (AP).
- Mineralogical characterization via semi-quantitative X-ray diffraction (XRD).
- Multi-element analysis (ICP-MS) via four-acid digestion.
- Shake flask extraction (SFE) for leachability assessment.

Results from the static testing indicate that several lithologies, particularly mineralized skarn units (S2S1), volcanogenic epiclastics (SFD), and early mineral porphyry (EMP), exhibit characteristics consistent with potentially acid-generating (PAG) behaviour. These units typically show low neutralization potential ratios (NPR < 1) and elevated sulphur contents. In contrast, unmineralized skarn and sedimentary units generally contain higher carbonate content and exhibit higher NPR values, classifying them as non-PAG (NPR > 3) or uncertain (NPR 1 to 3).

The total carbon content was found to be a more reliable proxy for neutralization potential than calcium content, due to the presence of non-carbonate calcium minerals (e.g., garnet, wollastonite, pyroxene) and limitations in assay detection thresholds. As such, it is recommended to consider routine analysis of total carbon in future drill-core assays to improve spatial prediction of ARD potential.

In addition to underground development rock, three (3) composite samples of flotation tailings have undergone static testing and mineralogical analysis. The samples were classified as non-PAG, attributed to its high calcite content and absence of detectable sulphide minerals. Cemented tailings that will be used for underground backfill are expected to have increased neutralization potential due to the addition of cement.

Materials to be excavated at surface for the construction of terraces and infrastructure (e.g., access roads, building and plant sites) also underwent geochemical characterisation. This includes static testing of seven (7) overburden samples to assess their ARD/ML potential and inform construction material management and sourcing.

20.8.2 KINETIC TESTING

To complement the static testing results and assess the long-term leaching behaviour of waste rock under simulated atmospheric conditions, a kinetic testing program was initiated in March 2025. Six (6) composite samples representing the dominant waste rock lithologies were prepared for humidity cell tests (HCTs), with one duplicate included for quality control. These composites were generated by blending residual material from the Phase 2 variability samples and reflect the major lithogeochemical units identified at Čoka Rakita:

- SFD_upper: Intermediate volcanics and volcaniclastics near the declines.
- SFD_lower: Deeper stratigraphic levels of the SFD unit, including mafic subunits and mixed sediment-volcanic rocks.
- S2S1_high_Cu: Mineralized skarn with elevated copper and gold.
- S2S1_low_Cu: Mineralized skarn with lower copper content.
- S2S1_low_Cu_low_Au: Unmineralized skarn with low sulphide and carbonate content.
- EMP: Early Mineral Porphyry intrusive unit.

The HCTs are being conducted in accordance with ASTM D5744 and are planned to run for a minimum of 20 to 52 weeks. Weekly leachate sampling was performed during the initial 20 weeks, followed by bi-weekly frequent sampling intervals. Analytical parameters include pH, electrical conductivity, acidity, alkalinity, sulphate, nitrate, and a suite of dissolved metals. Pre- and post-kinetic ABA are also included.

In addition to waste-rock and ore samples, one HCT of two composited tailings samples as well as a NEN 7375 (monolithic tank test) leaching test of solidified tailings with 5% cement are currently underway. Furthermore, HCTs of two (2) overburden composite samples were initiated in June 2025. The tailings and overburden HCTs are carried out using the same methodology as described above.

The kinetic testing program is ongoing and regularly evaluated. The kinetic program's purpose is to characterize the long-term geochemical behaviour of waste rock materials under oxidizing conditions, assess the potential for acid generation and metal release, and support the development of a robust ARD/ML management plan. These results will also be integrated with multi-element drill-hole assay data to attempt spatial modelling of ARD/ML characteristics across the deposit.

20.8.3 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

The closure of the Čoka Rakita Gold Project has been carefully planned to ensure that the site is rehabilitated in a manner that is environmentally sound, socially responsible, and economically viable. The closure strategy has been developed in accordance with Serbian legislation and international best practices, particularly those outlined by the International Council on Mining and Metals. From the outset, the project has integrated closure considerations into its design and operational planning, allowing for a more seamless transition from active mining to post-closure land use.

The overarching aim of the closure plan is to return the site to a condition that is physically and chemically stable, while also supporting beneficial future land uses. This includes restoring the landscape as closely as possible to its pre-mining state, and exploring opportunities for tourism, renewable energy, agriculture, forestry, or a combination of these. The closure vision emphasizes the importance of minimizing long-term environmental impacts, engaging stakeholders throughout the process, and leaving a positive legacy for the surrounding communities.

Closure activities are scheduled to occur progressively throughout the life of the mine, beginning in year five and continuing through to year fifteen. This phased approach allows for early rehabilitation of certain areas, particularly the dry tailings storage facility, which is designed to accommodate progressive closure. The underground mine will be decommissioned by removing hazardous materials and salvageable equipment, followed by backfilling with paste tailings. Mine portals will be sealed using engineered plugs or concrete caps, and groundwater recovery will be monitored to ensure that discharge meets environmental standards.

The DTSF will be capped with a multi-layer system that includes a drainage layer, frost protection, and topsoil. Native vegetation will be established to stabilize the surface and promote biodiversity. Seepage and runoff will be treated as necessary, with the goal of eliminating the need for long-term active water treatment. Processing facilities will be dismantled and removed unless retained for future use, and the terraces on which they are built will be regraded and revegetated. All demolition and decontamination activities will be conducted in accordance with national regulations.

Water management infrastructure will be adapted to post-closure conditions, with ponds and ditches redesigned to handle extreme weather events. Some ponds may be converted into passive treatment wetlands, and all liners will be removed and disposed of appropriately. Miscellaneous infrastructure, including buildings and roads, will be either removed or repurposed depending on the

final land use plan. Disturbed areas will be covered with topsoil and revegetated to restore ecological function.

Following the completion of physical closure activities, it is recommended that a ten-year monitoring program be implemented to ensure the long-term stability of the site. This will include inspections of landforms, water quality sampling, and assessments of vegetation health. Monitoring frequency will be adjusted based on performance, and if water quality consistently meets discharge criteria, active treatment may be discontinued.

The estimated cost of closure is approximately US$38.9 million, based on a Class 4 estimate prepared in accordance with the standards of the Association for the Advancement of Cost Engineering. This figure includes direct costs for earthworks, revegetation, water treatment, demolition, and monitoring, as well as allowances for project management, studies, and contingency. The indirect costs and contingency are not taken forward to the calculations in Sections 21 and 22, where a total of US$27.3 M direct closure costs is used. Serbian law requires a financial guarantee of at least 30% of the closure cost, which must be maintained throughout the life of the mine and beyond.

While the closure plan is comprehensive, it acknowledges several uncertainties that may affect its implementation. These include potential changes in regulatory requirements, evolving understanding of the geochemical characteristics of waste materials, and the impacts of climate change. Opportunities for cost savings and enhanced outcomes have also been identified, such as reusing infrastructure, optimizing topsoil use, and integrating progressive closure into operational planning. As the project advances, the closure plan will be regularly updated to reflect new information and stakeholder input, ensuring that it remains robust and responsive to changing conditions.

Closure planning is a continuously evolving process involving regular review and planning during every stage of the Project life cycle. Future updates will include:

- ESIA – Conceptual Closure Plan.
- Operational Closure Plan Updates – Every two (2) years.
- Detailed Closure Plan – Five (5) years prior to / from closure.
- Closure Execution Plan – Two (2) years prior to / from closure.
- 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 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.
- An Emergency Response and Contingency Plan.
- A Social management Framework.

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

A Health and Safety Execution Plan (HSEP) has been drafted (DPM, 2025) detailing how health and safety responsibilities will be organised for the Project.

The HSEP will form the basis of a Health and Safety Plan to be developed for future work activities. The HSEP includes a review of applicable legislation and is designed in line with the Serbian, International best practice (e.g., IFC Environmental, Health and Safety Guidelines and ISO45001) and DPM Internal Standards.

The HSEP 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 HSEP 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 Risk Factors for the Environment, 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 programme of further baseline collection planned.

Environmental and social factors have been considered in the assessment of the various alternatives, trade-off and options studies in the PFS and in this FS 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 had been delayed 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 have been developed for the Project, and this will help to identify potential impacts and inform the final 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, reduction in baseflow to springs and streams 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 18.11) 18.11that 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 non-contact or treated water to the environment.
- The Project may result in potential impact to habitats, 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 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.
- 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 hunting in the Project area will also be restricted during operations and possible cultural heritage features may 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, however, there have been public protests against the project and others in the last year. The stakeholder engagement plan, going forward, is a key tool which may need agility to ensure relationships are effectively managed. In addition, any EIA submitted for consideration to the Serbian authority requires engineer sign off, and the current public unrest is causing uncertainty on the ability to fulfil this requirement, by local subject matter specialists, across the industry. The key risks and mitigations for these different social aspects 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

21.1 Capital Cost Estimate

21.1.1 SUMMARY CAPITAL COST ESTIMATE

The Capex was developed from material take-off quantities and equipment lists with unit rates, labour-hours and labour costs applied. The Capex was incorporated by DRA using all formal MTOs, lists, and Bills of Materials prepared and issued by DRA and other specialists as detailed in this FS Report. DRA completed the Capex incorporating:

- DRA quantities and costs.
- Quantities and costs from WSP for the mining operation and other areas as detailed in Section 21.1.2.
- DPM's costs for the Owners costs, operational readiness and refurbishment and transfer of ownership costs for the Ada Tepe relocated equipment and materials.

The compiled Capex is summarized in Table 21.1 and is presented in United States Dollars (US$). All quoted prices are input in their denominated currency and converted to US$ based on the currency exchange factors provided by DPM and shown in Table 21.2.

The Capex date is based on 2nd Quarter 2025 rates. No provision for escalation has been provided.

Table 21.1 presents a summary of the Initial Capex, and Sustaining Capital (Susex) distributed over the LOM indicated separately. Owner's costs, contingencies and risk amounts are included in this Capex.

The Capex consists of direct and indirect capital costs as well as contingency. Estimated costs have been calculated for sustaining capital (Susex) which mainly include for mining and dry tailings storage expansion. These amounts are shown separately, by Area. Amounts for the mine closure and rehabilitation of the site have been estimated as well.

Table 21.1 – Capital Cost Summary

Area	Description	Initial Capex (US$ M)	Sustaining Capex (US$ M)	Total (US$ M)
2000	Underground Mine	129.14	22.86	152.00
3000	Ore Handling	18.96	-	18.96
4000	Processing Plant	62.95	-	62.95
5000	Filtered Tailings / Water Treatment Facilities	52.38	6.07	58.45
6000	On-Site Infrastructure, Site-Wide General	65.30	-	65.30
7000	Incoming Power	2.41	-	2.41
8000	Operational Readiness	26.39	-	26.39
9000	Indirect Costs	32.96	2.89	35.85
9100	Owner's Costs	14.18	-	14.18
9900	Project Contingency	43.53	-	43.53
9900	Project Closure and Rehabilitation[1]	-	29.90	29.90
	Total Major Area Capex	**448.18**	**61.72**	**509.92**

- Closure costs include the non-recoverable VAT of approximately $2.6 M.
- All capital costs, including closure costs are assumed to be net of recoverable VAT.
- Numbers may not sum precisely due to rounding.

21.1.1.1 Sustaining Capital Costs (SUSEX)

The Sustaining Capex (SUSEX) commences upon production of concentrate and continues throughout the mine life. The Sustaining costs are included in the overall Project Capex.

These costs include the following:

- Ongoing Mine Development and Underground Installations.
- Mobile Equipment Replacements and / or rebuilds.
- Process Plant Sustaining Capital.
- Dry Tailings Storage Facility Sustaining Capital.
- Paste Backfill Sustaining Capital.

The sustaining capital cost was developed, by production year, using quantities prepared on MTOs sheets and equipment additions when required.

These sustaining capital costs cover several areas, including mining and the DTSF as itemised in Table 21.1.

The mining sustaining capital consists of all activities from Years 1 to 8 inclusive in the LOMP, such as:

- All underground development capital costs after process plant startup.
- Mobile equipment rebuilds and replacements.
- Ventilation installations.
- Dewatering installations.
- Electrical installations.
- Mine safety installations.
- Backfill reticulation, and
- Service water installations.

21.1.1.2 Closure and Rehabilitation Costs

At the end of the Project life, it is required that all disturbed areas are rehabilitated, and equipment and buildings are disposed of. Closure costs have been included in the sustaining capital. This includes the non-recoverable VAT of approximately $2.6 M.

The Čoka Rakita Closure costs include the following:

- Mining closure costs.
- Paste Backfill closure costs.
- Process plant closure costs.
- Filtered tailings storage facility closure costs.
- Ponds and general site area.

The closure cost was based on the ERM closure plan and costs were developed by ERM with input from DPM.

Sustaining and closure costs are included in the total Project Capex of $509.92 M. For the purpose of the financial model, these costs are scheduled in the years they occur post-production.

21.1.2 QUALIFICATIONS AND EXCLUSIONS

21.1.2.1 Qualifications

The following qualifications form the basis of the Capex:

- The Capex assumes typical terms and conditions for contractor bidding and execution. Quotes from Vendors for equipment and materials to be valid for budget purposes.

- The Capex assumes no limitation with respect to local content in terms of labour, materials, equipment and economic impact.
- The Capex assumes no underground obstruction for all excavation activities to be performed during the construction.
- Suitable backfill material is available from the Project site excavations or from nearby sources within 45 km.
- Soil conditions are adequate for foundation bearing pressures.
- The Capex assumes no restriction to site at any time during execution of the Project.
- The Capex is based on no delays in execution from time of Project start as a result of either of the following:
  - Owner's financing changes.
  - Owner's permitting delays.
  - Labour disputes.
  - Relevant permits obtained in a timely manner to meet the Project schedule as outlined in Section 20.5.
- 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.

21.1.2.2 Major Exclusions

The following items have not been included in the Capex.

- Consultant fees associated with the preparation of a feasibility study and basic engineering.
- Earthworks and General Site:
  - Remediation of contaminated soil.
  - Removal and disposal of hazardous materials.
  - Provisions for adverse soil conditions beyond those based on available site data or assumed.
- General
  - Local community public relations caused delays and/or design changes.

- Cost impact resulting from “force majeure” and/or inclement weather.
- Allowance for schedule acceleration or deceleration.
- Minor adjustments to the plant following start-up and commissioning.
- Scope or design changes.
- Re-stocking or cancellation charges caused by changes to material specifications after award.
- Change to the construction methodology following completion of the Capex.

- Financial
  - Provision for inflation, escalation, currency fluctuations and interests incurred during construction.
  - Project financing costs.
  - Fluctuations to nominated currency exchange rates.
  - All sunk costs.
- Extraordinary climatic events.
- Labour disputes.
- Camp and catering will not be required as construction contractor pricing will be based on an all-inclusive rate, including accommodation, food, incidentals and transportation to and from the camp and the construction sites. It is further assumed that accommodation is available in the surrounding areas for the EPCM personnel and vendor supervisors.

21.1.3 CURRENCY

The Capex, Opex and Financial evaluation was prepared in United States dollars (US$). The 2nd Quarter 2025 exchange rates used for preliminary evaluation of all work are shown in Table 21.2. The final exchange rates reflect a three-month period ending June 30th, 2025, and then applied to the Capex. All future escalation is excluded from this Capex but will be covered in the Financial Evaluation. DPM provided the currency exchange rates that were used in the development of the final Capex, Opex and Financial Evaluation.

Table 21.2 – Currency Exchange Rates (June 30, 2025)

Currency Description	Currency Code	Exchange Rate	US Dollar
United States Dollars	USD	1.0000	1.0000
Canadian Dollar	CAD	0.7226	1.3839
Euro	EUR	1.135	0.8814
British Pound	GBP	1.336	0.7488

Currency Description	Currency Code	Exchange Rate	US Dollar
South African Rand	ZAR	0.056	18.2315
Serbian Dinar	RSD	0.0096	104.1687
Bulgarian Lev	BGN	0.5801	1.7240
Bosnia-Herzegovina	KM	0.6000	1.6667

21.1.4 ESTIMATE CRITERIA

21.1.4.1 Scope Of Activities

The scope of facilities includes the following areas and facilities. Unless otherwise indicated, all work will be by DRA:

DRA's direct scope of facilities is as follows:

- Refurbishment and transportation of existing equipment and buildings from Ada Tepe to the Čoka Rakita Project site. Retagging of equipment and steel will be part of the Ada Tepe closure costs.
- Process Plant.
- All on site surface infrastructure:
  - Access, haul roads and parking areas.
  - All Process and non-process buildings such as Plant and Mine Offices, Mechanical, and EC&I.
  - Workshop, Warehouse Building, Truck shop.
  - Process Auxiliaries as pipe racks, channels, and tunnels.
  - Mine auxiliaries (wash bay, tire shop and etc.).
  - Fuel station.
- Power Supply and Distribution:
  - Offsite power to the main sub-station.
  - On site power.
- Terraces, topsoil dumps and waste dumps (waste dumps based on mine production schedule).

WSP's direct scope of facilities include the following:

- Underground mine.
- Surface decline, vent shafts/raises and portal (by WSP).

- Paste Backfill Plant – by WSP, but DRA will be responsible for co-ordination as well as pricing and quantities for the building structure and foundation as well as the structure containing the paste backfill pipe from the paste backfill plant to the upper decline portal. WSP shall be responsible for the design and quantities for the interior foundations and equipment support structures. WSP shall be responsible for the pipeline and pipe supports for delivering the paste backfill from the paste backfill plant to the upper portal opening.
- Filter Plant (WSP) but DRA will be responsible for co-ordination as well as pricing and quantities for the building structure and foundation. WSP shall be responsible for the design and quantities for the interior foundations and equipment support structures.
- Dry Tailings Storage Facility – by WSP, but DRA will be responsible for co-ordination and pricing of quantities provided by WSP, quantified by Year.
- Shotcrete and CRF Slurry Batch Plant – by WSP, but DRA will be responsible for co-ordination. The Shotcrete and CRF Slurry Batch Plant will be a unit design including E-house electrics and foundations by WSP and priced based on quotes received by DPM. Foundations will be priced by DRA based on WSP MTO.
- Surface ventilation facilities for the underground mining operation – by WSP, but DRA will be responsible for co-ordination.
- Water management infrastructure – by WSP but DRA will only be responsible for co-ordination and pricing of quantities provided by WSP.

ERM was responsible for the geology and mineral resources. However, there are no facilities associated with this consultant and therefore, no applicable costs to be generated in the Project Capex. However, ERM was responsible for providing the closure costs with input from DPM which formed closure costs.

DPM will be responsible for development of the refurbishment cost of the relocated equipment, the transfer of ownership costs for such equipment, Operational Readiness costs as well as DPM's costs on the Project.

DRA incorporated all WBS codes for the Project into the overall Capex.

The Lead Estimator prepared the Capex and sustaining capital estimate and coordinated all estimating personnel to ensure details were sufficiently defined and properly estimated to achieve the required accuracy.

21.1.5 Estimate Accuracy

As agreed with DPM, the estimate was developed on the basis of an overall accuracy range of -10% to +20%.

Although some individual elements of the estimate may not achieve the target level of accuracy, the overall estimate should fall within the parameters of the intended accuracy.

21.1.6 Capex Cost Basis and Methodology – Direct Costs

Each company involved in the estimate development, upon completion, provided a detailed estimate in Excel spreadsheet format for full consolidation of the estimate by DRA.

21.1.6.1 Existing Facilities

Much of the existing facilities at Ada Tepe will be reused as a basis for the new facilities at Čoka Rakita. Therefore, work was required to provide an estimate of the cost to inspect the existing facilities, provide an examination and estimate of repair or refurbishment of the various elements, transport to Čoka Rakita and reinstalled.

The selected equipment, deemed suitable for reuse on the new Project, will be dismantled, re-tagged and repaired at the existing site, using the existing component available spares and/or with new spares and materials. The equipment will then be transported to the new Project site to await installation.

The dismantling cost for the selected equipment will be applied to the closure costs for Ada Tepe and was not included in the Project Capex. However, DPM will obtain refurbishment and transport costs to Čoka Rakita from local contractors who will provide such information on every piece of equipment. The dismantled equipment and steel will be stored at Ada Tepe and transported to the Project site just in time for installation.

The selected equipment was identified in the mechanical equipment list (MEL), the electrical equipment list (EEL), the instrument list and the mobile equipment list and indicate that the equipment is relocated from the existing plant. All equipment and materials not reused from the Ada Tepe plant will be new purchases.

21.1.7 Underground Mine Capex

The Project Capex for the underground mine is based on a mining contractor working in conjunction with owner crews on the development of the mine. The development contractor first establishes the two portals, including their box cut excavations and portal face ground support with culvert travelway, followed by development of the two decline headings, the spiral ramps, level accesses, and level infrastructure, excepting the garage facility excavation that will be done as part of the owner crew development.

The mining Capex includes ongoing owner crew development and owner crew construction that immediately follows completion of the contractor development activity on each level with preproduction level crosscuts, ore sills, truck loadout areas, backfill cuddies, and all underground construction performed as owner crew work activities.

21.1.7.1 Summary of Estimated Costs

Table 21.3 presents WSP's Summary Capex for the underground mine in US$. The estimate's accuracy level aligns with DPM's required accuracy as stated in the introduction of this Section 21.

Table 21.3 – Underground Mine Capex (US$ Millions)

Description	Year -3	Year - 2	Year -1	Total Cost
Development Contractors: Lateral + Vertical; Direct Costs	2.73	26.11	29.11	57.95
Contractor Indirects: Mob, Demob, Temporary Facilities, Engineering of Temporary Installations	0.00	15.13	1.35	16.48
Personnel: Owner Hired & Supervised	0.00	0.06	1.98	2.04
Owner's Equipment Operation	0.00	0.18	1.81	1.99
Owner Provided Materials and Services	0.00	1.29	12.36	13.65
Capital Acquisitions: Owner Equipment*	0.00	19.71	17.32	37.03
Total Capex	**2.73**	**62.48**	**63.92**	**129.14**

*Shotcrete / CRF Batch Plant included with surface equipment costs
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 was derived from the following sources:

- Quotations from suppliers for new equipment.
- Quotations from suppliers for major consumables
- Quotations from contractors,
  - for lateral development,
  - for vertical development,
  - for pre-grouting surface holes,
  - for earthworks,
  - for surface construction.
- Local labour cost survey provided by DPM.
- 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 Contractor Direct Costs, Contractor Indirect Costs, Owner Operational Personnel, Owner Equipment Operation, and Owner Provided Materials and Services.

The Underground Mine Capex also includes Capital Acquisition costs that comprise mobile and fixed equipment purchases, not including associated expenses such as shipping to the site and commissioning which are included with the Project's indirect costs provided by DRA.

Assumptions

The following assumptions have been made in the preparation of the cost estimate for the underground mine:

- The DTSF has been established to the point that the development rock from the portal and decline excavations can be deposited directly to that location without intermediate stockpiling requirement.
- The terraces have been established to the point that the development contractor can mobilize to site.
- Quotations obtained from a development contractor for the two portals with box cut excavations and portal face ground support and corrugated steel travelway, both declines, spiral ramp, level access and level infrastructure excavations are provisional and are used only for budgetary purposes. WSP has reviewed and has deemed reasonable for the FS Report.
- Personnel with the required experience and skills will be available starting from the development contractor completion 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.
- Except for activities related to contractor development and 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.
- 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.

- Quotations obtained from vendors for equipment and materials are provisional and are intended solely for budgetary purposes. WSP has reviewed and has deemed reasonable for the FS.
- DPM will purchase all equipment for the underground mine except for the raise bore machine.
- 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.

21.1.7.3 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, excepting contractor indirect costs specific to the twin decline and spiral ramp development bid.
- Operational Readiness.
- Any work outside the defined battery limits for the underground mine.

Surface facilities exclusively for the use of the mine such as the paste backfill plant and the Shotcrete / CRF backfill plant are included with the surface capital costs.

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 Piping and instrumentation Diagrams (P&IDs).
- 3D Model.
- Layout Drawings.
- 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.

- Based on available borehole data, a Leapfrog model was created, from which 3D surfaces were extracted for the individual lithological varieties. This model was considered 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 21.4.
- 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.
- Geotechnical studies.
- Other applicable information.

The pricing of the site preparation and bulk earthworks quantities was based on contractor rates from a number of 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.

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.

- For all areas, the 3D model and design sketches were used to develop concrete quantities.
- 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, provided by DPM, received from four (4) local 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.

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 was based on contractor rates from two (2) contractors, which also included the installation of the mechanical equipment and platework. 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.

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.
- Laboratory, and Control Room.

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 was 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 was 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.

21.1.8.8 Platework Fabrication and Liners

Certain platework items such as chutes, hoppers, pumpboxes, tanks and other platework required for the Project, were separately issued for pricing.

21.1.8.9 SMP Contract

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.

The budget quotations covered the supply and installation of new steelwork, installation of Ada Tepe steelwork (dismantled and delivered to site at no cost to the contractor), new mechanical equipment and platework, and installation of Ada Tepe equipment (dismantled and delivered to site at no cost to the contractor). The piping requirements were separated and priced by a local piping contractor.

21.1.8.10 Piping

The development of the piping quantities was based on the following methodology.

- The quantities were based on 3D models and P&ID drawings for the process plant, and piping and valve quantities by type and size were used to form the quantity of piping and valves.
- For piping lines that were not measured, estimation factors were used to develop the total length in meters.
- An allowance was included for building service piping 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 quotations received from two or three qualified equipment suppliers and fabricators. The following piping systems were issued to suppliers for bidding, and the returned quotations were evaluated accordingly:

- 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.

21.1.8.11 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 the Project were estimated by type of material and type of process.
- Bulk electrical quantities of raceway were developed from the plot plan, equipment layouts and single line diagrams.

- All power was distributed to the various e-houses, substations and pumphouses via cable trays within the building and on pipe racks.
- 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.
- Incoming power has been developed based on a 35 kV line from Žagubica. No modifications for the Žagubica sub-station is included in the Capex, nor any alternate power sources from other areas. Two 35 kV power lines (buried single core cables in trefoil formation) from the Substation "Krst" are planned. The total length of route is about 6 km.
- On-site, the main substation with fully redundant transformers steps down the 35 kV grid incoming voltage to 10 kV which is then 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 up to 75 kW to be locally controlled and de-energised with local disconnect switch.
- Permanent emergency gensets are 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 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 three (3). 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.

21.1.8.12 Instrumentation

Where possible, the existing control system was dismantled and relocated to the new plant. All associated instruments were reviewed and relocated with the existing equipment. New equipment and instruments were also incorporated into the Capex.

- The PFDs, PIDs 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 pricing of the instruments was based on the receipt of two to three quotations from qualified equipment suppliers and fabricators.

The pricing of the installation of the instruments and the supply and installation of the cables, cable trays, interconnections, etc., was based on contractor rates from three (3) 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.

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 was applied to quantities and included 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 21.4.

Table 21.4 – Growth Allowances per Discipline

Discipline Code	Discipline Description	Design Growth
1200	Earthwork – Site Preparation	1.10
1300	Earthwork – Terraces	1.10
1400	Earthwork – Roads	1.10
1500	Earthwork – Fencing	1.10
1600	Earthwork - Underground Piping	1.10
1700	Earthwork – Liners	1.10
1800	Earthwork - Soil Nails	1.10
2100	Sub-Foundation	1.10
2200	Concrete	1.05
2300	Structural Steel	1.05
1120	Dismantled Steel	1.10
2400	Architectural Structural	1.10
2500	Architectural Finishes	1.10
2700	Prefabricated Structures	1.10
2800	Building Mechanical Services	1.05
2900	Building Electrical Services	1.05
4000-4800	New Mechanical Equipment	1.00
1140	Dismantled Ada Tepe Equipment	1.10
4900-4940	Mobile Equipment	1.05
5000-5700	Piping (with quantities)	1.10
6100	Electrical Bulks	1.05
6200	Process Electrical - Conduit, Cable Tray and Bus Duct	1.05
6400	Process Electrical - Power Generating Equipment	1.05
6600	Process Electrical - Middle Voltage Power Equipment	1.05
6700	Power Equipment up to 1000 V	1.05
6800	Process Electrical - E-Rooms	1.05
6900	Process Electrical - Miscellaneous Electrical	1.10
7100	Instrumentation - Cables	1.05
7400	Instrumentation - Switches	1.05
7500	Instrumentation - Valves	1.05
7700	Instrumentation - Process Panels and Controls	1.05
7900	Instrumentation - Piping, Tubes and Fittings	1.05
2000	Underground Excavations	1.00

21.1.10 CAPEX PRICING BASIS

The input cost sources are outlined in Table 21.5.

Table 21.5 – Input Cost Source

Description	Firm Quotes	Budget Quotes	Database	Factored
Mining	0%	80%	15%	5%
Mechanical Equipment	19%	71%	10%	0%
Electrical Equipment	45%	40%	10%	5%
Steel Works	0%	90%	5%	5%
Prefabricated Buildings	0%	95%	0%	5%
Earthworks	0%	80%	10%	10%
SMP	0%	90%	0%	10%
Piping	0%	70%	20%	10%
Architectural	0%	60%	20%	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. Excepting the lateral and vertical development contractors, owner's crews perform the balance of the mine development and all mine construction with the costs of such based on local labour rates provided by DPM.

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.

- Development contractor

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 OPERATIONAL READINESS

The operational readiness during the pre-operational phase comprises of the Owner's cost to train and recruit the plant operating personal so the mine and plant personnel are ready for the smooth transition to the mining operation. The personnel are usually hired during the construction period, so they are familiar with the equipment and operations.

In addition, during this period, the computer systems are purchased and are set-up to ensure that the systems are up and running for effective control of the operations prior to the start of the mining operations and plant production. This area covers the following activities:

- Training and recruiting of plant operating personnel (Operational Readiness).
- Working capital (covered in the economic evaluation) (Operational Readiness).
- Computer IT hardware and software.

21.1.13 INDIRECT COSTS

21.1.13.1 Owner's Indirect Costs

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:

- 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).
- All production costs prior to commercial production.
- All costs associated with the permitting process.
- Community relations.

- Emergency response.
- Environmental / permitting / government relations.
- Finance and general administration.
- Human resources.
- Information technology.
- Insurances.
- Legal.

21.1.13.2 Contractor's indirect Costs

The contractor's indirect costs include for mobilization and demobilisation only. All other contractor indirect costs are included in the direct costs of the work.

The contractor's indirect costs which have been included in the direct costs consist of but not limited to the following:

- Construction mobile equipment, fuel and operating costs.
- Construction power.
- Cranage.
- Scaffolding.
- Concrete and aggregate batch plants.
- Room and board and transportation to and from site.
- All overheads and profits.

21.1.13.3 External Consultants

External consultants are those entities that are outside of DPM's internal personnel and services. The external consultants cover the following activities.

- Construction Support by a local design engineering firm.
- Geotechnical services.
- Fire protection services.
- Environment consultants.
- Drilling services such as drilling assays, third party supervision.
- Miscellaneous consultants.

21.1.13.4 Vendor Representatives & Commissioning

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. It is assumed that 17 representatives are required for the Project with a portion of the representatives assisting in the dismantling of the Ada Tepe equipment and reinstalment of such equipment at the Project site.

Commissioning

Commissioning of the facilities includes the cost for a commissioning team required for preparing the facilities for the commissioning and ramping up of the plant operation. It is assumed that this is a four (4) month activity involving:

- 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.

The 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.

21.1.13.5 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.

21.1.13.6 Engineering, CM, PM (Owners)

DPM provided estimated costs for its on-site staff to manage the construction management requirements, as well as costs for surveyors, QA/QC specialists and induction.

21.1.13.7 Temporary Services, Camp and Catering

DPM provided estimated costs for temporary services required to support its staff as well as the construction personnel on site. It should be noted that DPM is not providing any camp nor camp catering costs. Each contractor is responsible for its on-site staff and construction person power.

21.1.13.8 Capital Spares and Inventory

Capital Spares and Inventory comprise a complement of spares to be used during the commissioning process, the establishment of an inventory of spares for the initial period of the operations, and operational spares.

In addition, a complement of grinding balls, reagents, fuel and lubricants is required for the commissioning phase and start of operations. These “first fills” are also identified and included in the Capex.

It is assumed that the existing inventory at Ada Tepe will be utilised for the refurbishment of the mechanical equipment. Therefore, a full complement of spares would be required for the Project.

- Capital Spares.
- Operational Spares.
- Commissioning Spares.
- Initial Fills.

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.

A Monte Carlo simulation was performed based on DPM’s established parameters and the final Capex included a P50 value of $43.53 M as the assigned contingency value.

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.

The operating cost expenditure (Opex) for this Report covers the costs to operate all the facilities required for the Project.

Costs are expressed in second quarter 2025 US$ unless otherwise stated. Cost elements are input in their denominated currency and converted to US$ based on the currency exchange factors provided by DPM and shown in Table 21.2.

The Opex cost basis is prepared in 2025 constant money terms and results shown herein are in such terms, unless otherwise indicated.

For valuation purposes, constant money terms are escalated to nominal terms based on official inflation indices and foreign exchange forecasts.

Further Opex details are presented in Section 21.2 of this FS Report.

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 (7,344,769 tonnes milled provide the unit costs), is summarised in Table 21.6.

Table 21.6 – Opex by Major Area

Area	**LOM Total Opex** (US$ M)	**Average Opex** (US$/t milled)
Mining	267.15	36.37
Processing and Tailings	204.84	27.89
G&A[1]	109.71	14.94
Total Opex	**581.70**	**79.20**

1 The operating costs presented above exclude property taxes of $5.8M., which are included in the economic analysis
2 Numbers may not add due to rounding.

21.2.2 Summary of Personnel Requirements

Table 21.7 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 local area.

Table 21.7 – 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 total mine Opex was estimated based on Owner furnished equipment, materials and mine operators and based on the annual mine plans that totalled 7,261,167 tonnes mined after mill startup. Table 21.8 presents the unit rates that were derived by applying the 7,344,769 ore tonnes milled to these expenditures to then arrive at the total cost per tonne milled for the mine operations.

Table 21.8 – Summary of Mining LOM Average Opex

Description	**LOM Total Opex** (US$ M)	**Average Opex** (US$/t milled)
Labour	91.5	12.46
Equipment Operation	53.8	7.32
Materials and Services	30.5	4.15
Utilities	91.4	12.44
Total	**267.2**	**36.37**

Numbers may not add due to rounding.

The mining Opex estimate generally excludes:

- The capital expenditures related to the pre-production tonnage of 83,612 ore tonnes placed to stockpile for future milling are not part of the total operating costs as shown in Table 21.8.
- 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 21.9 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 21.9 – Summary of Processing LOM Average Opex

Description	**LOM Total Opex** (US$ M)	**Average Opex** (US$/t milled)
Labour	42.35	5.77
Power	54.91	7.48
Reagents and Consumables	81.69	11.12
Laboratory	2.16	0.29
Contract Maintenance	2.41	0.33
Mobile Equipment Maintenance	2.12	0.29
Spares and Miscellaneous	19.20	2.61
Total	**204.85**	**27.89**

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 44,571,545 kWh which results in a unit energy consumption rate of 52.44 kWh per tonne or $7.50 per tonne milled.

21.2.4.2 Reagent Consumption Costs

Reagents consist of flotation reagents (e.g., 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 $6,655,37 or $3.32 per tonne milled.

Cement is required for the paste backfill operation. Total cement requirement is 17,493 tpa based on a 5% addition rate and represents $2,798,859 annually or $3.29 per tonne milled.

The total annual cost of above reagents is 9,454,235 or $11.12 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 $109.71 M or $14.94 per tonne milled.

22 ECONOMIC ANALYSIS

22.1 General

The economic analysis presented in this Section contains forward-looking information as defined by Canadian securities legislation. 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. 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, although DPM may decide in the future to finance a portion of the Project with debt. Current Sebian tax regulations were used to assess corporate tax liabilities. DPM retained KPMG d.o.o. Beograd to review tax calculations for the after-tax economic evaluation of the Project.

The cash flow model uses inputs from all elements of the Project to provide a detailed financial projection for the entire Project on an annual 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. A sensitivity analysis was carried out to assess the impact of variations in gold price, Capex, and Opex on the financial metrics. The sensitivity of the after-tax metrics was also assessed to a broader range of gold prices ranging from US$1,500/oz to US$3,500/oz.

The production schedule used in this analysis is based on the LOMP and the Concentrate Production Schedule outlined in Sections 16 and 17, respectively. The Capex and Opex are taken from estimates detailed in Section 21. The economic analysis is developed in terms of financial years with appropriate adjustments made to the production and cost schedules to convert the data from mine plan years.

All costs and pricing are in Q2 2025 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 by Canadian securities legislation. 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.
- Timing of the Project in terms of permitting approval, project development, construction, operation, and closure.
- General business and economic conditions.
- Ability to maintain social license to operate.
- Unrecognised environmental risks.

22.3 Assumptions

The following assumptions were made in the development of the economic analysis:

- The reference date for the analysis has been set to align with the commencement of Development Capital expenditure, as per DPM's planned project execution schedule.
- The Development Capex spending is spread over four financial years, namely 9% in Year -3, 40% in Year -2, 40% in Year -1, and 11% in Year 1, which aligns with DPM's expected spend profile based on the planned project execution schedule.
- Revenue from the Project is derived from the sale of gold concentrate and gold doré.
- All product is sold in the same year that it is produced.
- A flat Net Smelter Return (NSR) government royalty of 5% was applied.
- Corporate tax liabilities were assessed based on current Serbian tax regulations.
- The Project is eligible for a tax holiday under current Serbian income tax regulations.
- A conservative assumption has been made regarding the Closure Cost with the entirety of the estimated cost scheduled in the final year of production.

- No consideration has been made in this model for salvage value from the sale of mechanical equipment.
- Discount rate of 5%.
- Long-term gold price of US$ 1,900/oz selected.
- All costs and pricing are in Q2 2025 US dollars.
- No provision is made for the effects of inflation in this analysis.
- All environmental approvals, required permits, licenses and authorisations will be obtained from the relevant governments and other relevant stakeholders.
- The analysis is conducted on an unlevered, all-equity basis.

22.4 Economic Analysis Parameters

The technical parameters and cost inputs in the economic analysis are summarised in Table 22.1.

Table 22.1 – Economic Analysis Parameters

Description	Unit	Value
Macroeconomic Parameters		
Gold Price	US$ per oz	1,900
Project Parameters		
Discount Rate	%	5.0
Mine Life	years	10
Mineral Reserve (LOM)	Mt	7.3
Grade mined (LOM average)	g/t Au	6.44
Annual mill throughput	tpa	850,000
Gold recovery (LOM average)	%	87.9
Gold payability (LOM average)	%	98.5
Total gold produced (LOM)	Moz	1.34
Average annual gold production (LOM)[3]	oz	148,000
Average annual gold production (first five years)[4]	oz	189,000
Government royalty (NSR)	%	5.0
Capital Cost Estimate		
Initial Capital	US$ M	448
Sustaining Capital (LOM)	US$ M	32
Closure Costs[2]	US$ M	30

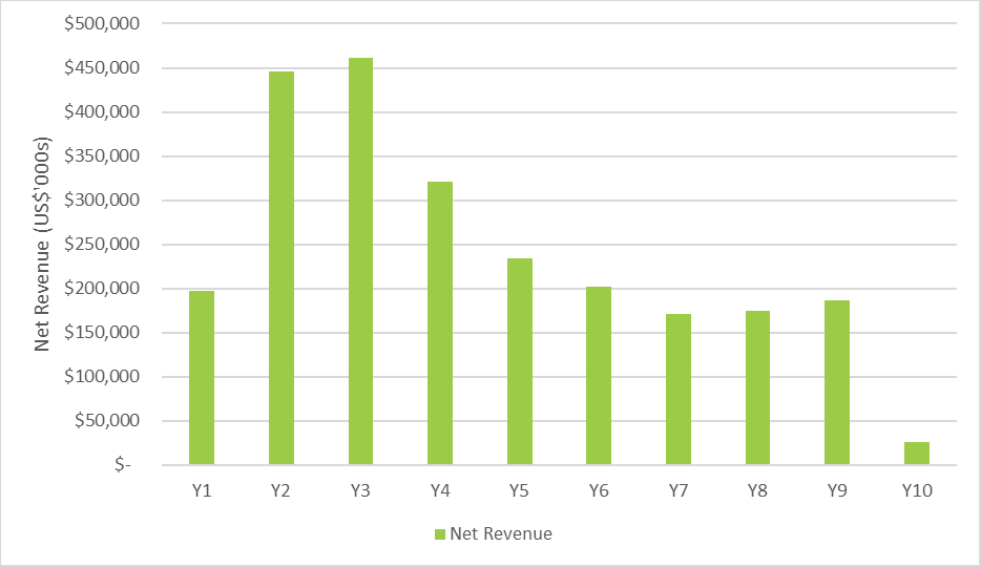
Description	Unit	Value
LOM Operating Unit Costs	**US$ M**	**US$/t ore processed**
Mining	267	36
Processing	205	28
General & Administrative[5]	110	15
Total Opex	***582***	***79***
Other Cash Costs	**US$ M**	**US$/t ore processed**
Property Taxes	6	1
Royalties	121	16
Cost Metrics (LOM Average)[6]		
Cash cost per Tonne of Ore Processed	US$/t ore processed	96
All-in Sustaining Cost per Ounce of Gold Sold	US$/oz payable gold	644

1. Numbers may not add due to rounding.
2. Closure costs include the non-recoverable VAT of approximately $2.6 M.
3. Average production from the 9 years with significant annual throughout (>50% of 850 ktpa).
4. Average production from first five full years of production.
5. The G&A operating costs presented above exclude property taxes of $5.8M, which are presented in a separate line.
6. Cash Cost; Cash Cost per Tonne of Ore Processed; and All-in Sustaining Cost per Ounce of Gold Sold are non-GAAP measures or ratios. 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.

22.5 Revenue

The Project generates US$2.5 billion in gross revenue over the LOM. After consideration of selling costs, the net revenue over the LOM is US$2.4 billion. Figure 22.1 presents the net revenue profile of the Project per operating year.

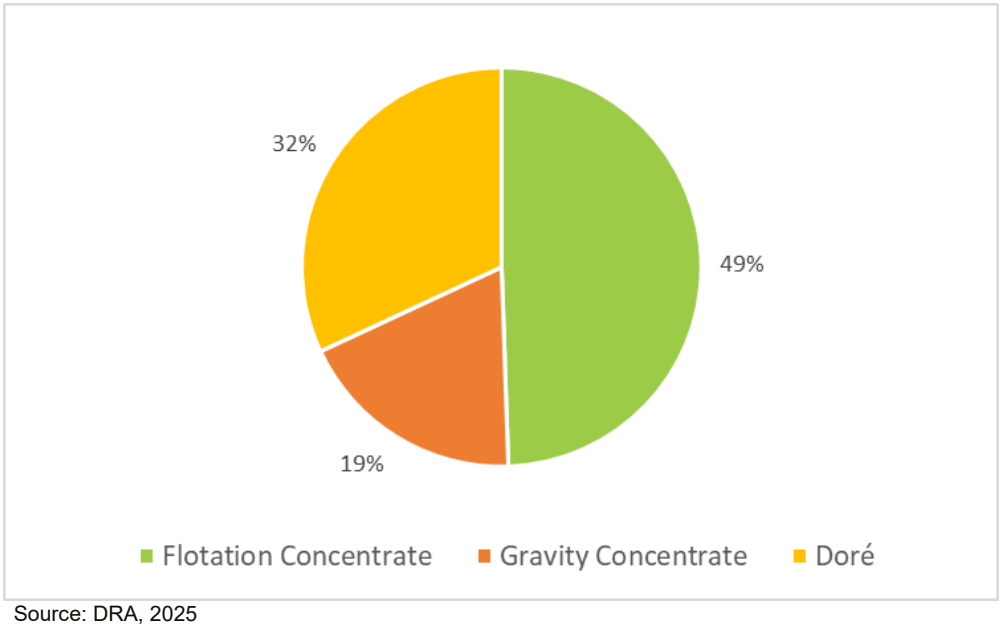
Figure 22.1 – Revenue Profile over the LOM



Source: DRA, 2025

Figure 22.2 shows that the flotation concentrate accounts for about half of the net revenue over the LOM of the Project. Of the remainder, doré accounts for about 32% and the gravity concentrate for about 19%.

Figure 22.2 – Proportion of Net Revenue Attributable to Each Product



Source: DRA, 2025

22.6 Taxes

The Project is subject to Income Tax and other non-income taxes such as Value Added Tax (VAT), property tax, and import duties. To support the after-tax economic evaluation of the Project, DPM retained KPMG d.o.o. Beograd to review the tax calculations included in the financial model.

22.6.1 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 rates and deductions 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. Operating losses cannot be carried back.

Current legislation allows for relief from income tax ('tax holiday') 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 approximately US$10 M (at an RSD:US$ exchange rate of 0.010) in qualified fixed assets for use in the taxpayer's registered business activities.
- Employment of at least 100 new employees for an indefinite period of time during the entire period of the tax relief.

DPM has advised that the Project is eligible for this tax holiday. 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. Based on the list of non-qualifying assets (second hand equipment previously used in Serbia, passenger cars, airplanes, furniture, mobile phones, air conditioners, video surveillance and other non-business-related assets), the percentage of the tax holiday was calculated to be around 99.5%. Therefore, the effective income tax rate applied in this analysis is close to 0% over the 10-year life of the Project.

22.6.2 Land Acquisition and Usage Taxes

It is assumed that the land acquisition tax will be paid prior to the reference date and, as such, is not included in this analysis.

Annual property taxes have been estimated using a 0.4% tax rate on an assumed land acquisition cost and the capital costs of land conversion, buildings, structures, and roads.

22.6.2.1 Value Added Tax

Value Added Tax (VAT) is levied at a standard rate of 20% on the supplies of goods/services in Serbia and on the importation of goods. A reduced VAT rate is applicable to certain items, while others are exempt from VAT based on the provisions of Serbian tax law. The sale of gold concentrate and doré is exempt from VAT with the right of deduction of the input VAT.

A detailed review of the Capex and Opex items has not been conducted at this stage to precisely determine the VAT payable for the Project. At the feasibility study stage, a flat 20% VAT has been applied to all Capex and Opex items except labour costs, which are considered exempt from VAT.

DPM will claim the input 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.

22.7 Base Case Cash Flow Analysis and Economic Results

Based on the assumptions outlined in this Section, the results of the analysis indicate a positive after-tax NPV of $782 M, at a discount rate of 5%, an after-tax IRR of 35.6% and a payback period of 1.8 years. Owing to the tax holiday applicable to the Project, the pre-tax metrics are similar to the after-tax metrics, as shown in Table 22.2.

Table 22.2 – Economic Results Summary

Description	Unit	Pre-Tax	After-Tax
NPV @ 5%	US$ M	783	782
IRR	%	35.7	35.6
Payback Period	Years	1.8	1.8

Numbers may not add due to rounding.

Figure 22.3 presents the annual and cumulative cash flows of the Project on an after-tax basis. A summary of the financial model is presented in Table 22.3.

Figure 22.3 – After-Tax Annual and Cumulative Cash Flows



Source: DRA, 2025

Table 22.3 – Summary of Production Schedule and Cash Flows

Period #			-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Period			FY-3	FY-2	FY-1	FY1	FY2	FY3	FY4	FY5	FY6	FY7	FY8	FY9	FY10	FY11
PRODUCTION SCHEDULE		*(Tot. / Avg.)*														
Ore Tonnes Milled	*(ktonnes)*	**7344.8**	-	-	-	400.2	828.8	850.0	850.0	850.0	850.0	850.0	850.0	850.0	165.8	-
Feed Grade, Gold	*(g/t)*	**6.44**	-	-	-	9.49	10.19	10.27	7.34	5.47	4.76	4.11	4.18	4.44	3.29	-
Contained Metal, Gold	*(koz)*	**1520**	-	-	-	122.1	271.5	280.8	200.6	149.5	130.2	112.2	114.3	121.4	17.5	-
Recovery Schedule																
Gold Recovery to Final Flotation Concentrate	*(%)*	**45.3**	-	-	-	41.5	40.4	40.3	45.3	48.6	49.9	51.2	51.0	50.5	52.8	-
Gold Recovery to Dore	*(%)*	**26.8**	-	-	-	29.3	30.8	30.9	26.9	24.2	23.2	22.2	22.3	22.7	20.9	-
Gold Recovery to Secondary gravity tails	*(%)*	**15.8**	-	-	-	17.2	18.1	18.1	15.8	14.2	13.6	13.0	13.1	13.3	12.3	-
Flotation Concentrate																
Grade, Gold	*(g/t)*	**63.8**	-	-	-	99.7	108.0	109.0	75.8	55.9	48.5	41.7	42.5	45.2	33.4	-
Contained Metal, Gold	*(koz)*	**688**	-	-	-	50.7	109.8	113.1	90.8	72.7	65.0	57.4	58.3	61.3	9.3	-
Tonnage Produced	*('000s dmt)*	**335.4**	-	-	-	15.8	31.6	32.3	37.3	40.5	41.7	42.8	42.7	42.2	8.6	-
Dore																
Contained Metal, Gold	*(koz)*	**407.9**	-	-	-	35.8	83.5	86.7	53.9	36.2	30.2	24.9	25.5	27.57	3.7	-
Gravity Concentrate																
Grade, Gold	*(g/t)*	**1583.8**	-	-	-	2,218.3	2,381.2	2,400.9	1,715.0	1,278.7	1,113.0	959.4	977.2	1,038.2	768.3	-
Contained Metal, Gold	*(koz)*	**240**	-	-	-	21.0	49.0	50.9	31.7	21.3	17.7	14.6	15.0	16.2	2.1	-
Tonnage Produced	*('000s dmt)*	**4.7**	-	-	-	0.30	0.64	0.66	0.57	0.52	0.50	0.47	0.48	0.49	0.1	-
Total Gold Production																
Contained Metal, Gold	*(koz)*	**1,336.0**	-	-	-	107.6	242.3	250.7	176.4	130.2	112.9	96.9	98.8	105.10	15.1	-
Payable Metal, Gold	*(koz)*	**1,316.0**	-	-	-	106.1	239.1	247.4	173.8	128.1	111.0	95.3	97.1	103.36	14.8	-
Period #			-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Period			FY-3	FY-2	FY-1	FY1	FY2	FY3	FY4	FY5	FY6	FY7	FY8	FY9	FY10	FY11
PRE-TAX CASH FLOW		*(Tot. / Avg.)*														
Net Smelter Revenue	*($'000s)*	**$2,422,510**	-	-	-	$197,713	$446,391	$461,915	$321,387	$234,193	$201,516	$171,464	$174,942	$186,858	$26,130	-
Less: Government Royalties	*($'000s)*	**($121,125)**	-	-	-	($9,886)	($22,320)	($23,096)	($16,069)	($11,710)	($10,076)	($8,573)	($8,747)	($9,343)	($1,307)	-
Less: Total Operating Costs	*($'000s)*	**($581,707)**	-	-	-	($50,978)	($73,550)	($68,548)	($67,488)	($67,100)	($64,345)	($63,072)	($61,300)	($56,631)	($8,695)	-
Less: VAT Payable after Credit Applied	*($'000s)*	**($2,620)**	-	-	-	-	-	-	-	-	-	-	-	-	($2,620)	-
Less: VAT Refund Account		**-**	($1,726)	($5,633)	$20	$3,012	$1,501	$229	$181	($29)	$189	($112)	$217	$136	$2,016	-
Less: Property Taxes	*($'000s)*	**($5,814)**	($97)	($97)	($97)	($552)	($552)	($552)	($552)	($552)	($552)	($552)	($552)	($552)	($552)	-
Operating Earnings	*($'000s)*	**$1,711,243**	($1,823)	($5,731)	($77)	$139,309	$351,470	$369,948	$237,458	$154,802	$126,732	$99,155	$104,559	$120,468	$14,972	-
Capital Expenditures																
Pre-Construction & Early Works Capital	*($'000s)*	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Development Capital exc. Capitalized Amounts	*($'000s)*	**$448,183**	$42,044	$179,273	$179,273	$47,592	-	-	-	-	-	-	-	-	-	-
Sustaining Capital	*($'000s)*	**$31,834**	-	-	-	$11,343	$4,519	$4,852	$1,278	$2,285	$1,180	$5,536	$428	$413	-	-
Closure Capital	*($'000s)*	**$27,300**	-	-	-	-	-	-	-	-	-	-	-	-	$27,300	-
Salvage Value	*($'000s)*	**-**	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Changes in Working Capital	*($'000s)*	**($0)**	-	-	-	($31,730)	($39,025)	($2,643)	$23,071	$14,025	$5,221	$4,810	($617)	($1,599)	$28,487	-
Pre-Tax Cash Flow	*($'000s)*	**$1,203,927**	**($43,868)**	**($185,004)**	**($179,350)**	**$48,643**	**$307,927**	**$362,453**	**$259,251**	**$166,542**	**$130,773**	**$98,429**	**$103,515**	**$118,456**	**$16,159**	**-**
Discounted Pre-Tax Cash Flow	*($'000s)*	**$782,855**	**($42,811)**	**($171,948)**	**($158,755)**	**$41,007**	**$247,227**	**$277,147**	**$188,795**	**$115,506**	**$86,379**	**$61,919**	**$62,017**	**$67,589**	**$8,781**	**-**
Pre-Tax IRR	*(%)*	**35.7%**														
Payback Period	*(years)*	**1.8**														
AFTER-TAX CASH FLOW		*(Tot. / Avg.)*														
Pre-Tax Cash Flow	*($'000s)*	**$1,203,927**	($43,868)	($185,004)	($179,350)	$48,643	$307,927	$362,453	$259,251	$166,542	$130,773	$98,429	$103,515	$118,456	$16,159	-
Less: Income Tax Paid	*($'000s)*	**($955)**	-	-	-	($1)	($236)	($250)	($151)	($89)	($68)	($46)	($51)	($63)	-	-
After-Tax Cash Flow	*($'000s)*	**$1,202,973**	**($43,868)**	**($185,004)**	**($179,350)**	**$48,643**	**$307,690**	**$362,203**	**$259,100**	**$166,453**	**$130,705**	**$98,383**	**$103,464**	**$118,393**	**$16,159**	**-**
Discounted After-Tax Cash Flow	*($'000s)*	**$782,162**	**($42,811)**	**($171,948)**	**($158,755)**	**$41,007**	**$247,037**	**$276,956**	**$188,685**	**$115,444**	**$86,335**	**$61,890**	**$61,987**	**$67,554**	**$8,781**	**-**
After-Tax IRR	*(%)*	**35.6%**														
Payback Period	*(years)*	**1.8**														

Source: DRA, 2025 Closure costs referenced in this table do not include the non-recoverable VAT of approximately $2.6 M, which is reflected in a separate line (VAT Payable after Credit Applied).

22.8 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 Feasibility Study level is typically -15% +20%. 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 and Figures 22.4 and 22.5. The NPV and IRR of the Project are most sensitive to variations in the gold price. The Project retains a positive NPV at the lower limit of the gold price interval tested.

Figure 22.4 indicates that variations in Capex and Opex have similar impacts on the after-tax NPV, while Figure 22.5 shows that the after-tax IRR is more sensitive to Capex than to Opex based on the steeper slope of the Capex line.

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	1,120	951	782	613	444
IRR	%	44.7	40.3	35.6	30.5	24.9
Payback	years	1.5	1.7	1.8	2.0	2.3

Capex	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	693	737	782	827	872
IRR	%	29.2	32.2	35.6	39.6	44.2
Payback	years	2.1	1.9	1.8	1.7	1.5

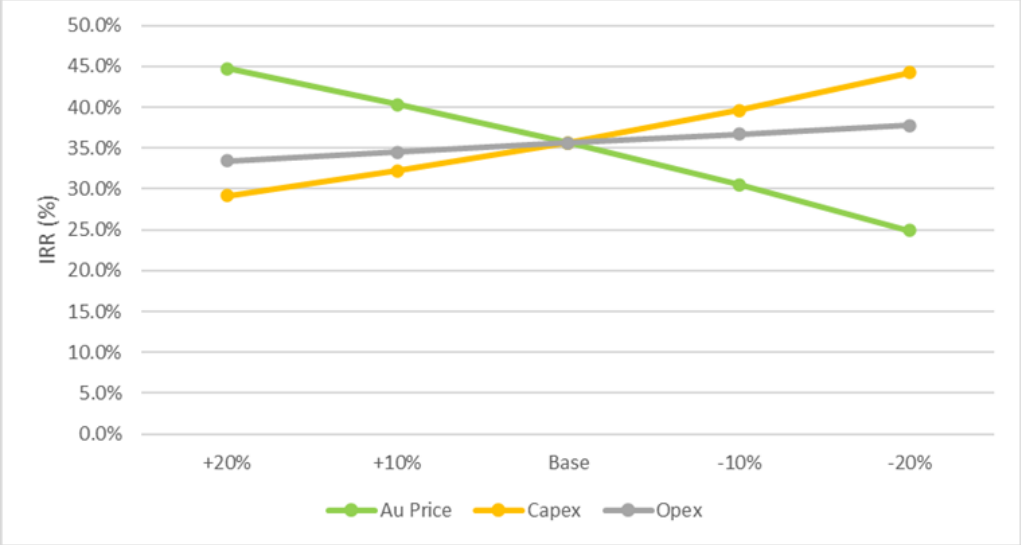
Opex	Units	+20%	+10%	Base	-10%	-20%
NPV @5.0%	US$ M	701	742	782	823	863
IRR	%	33.4	34.5	35.6	36.7	37.8
Payback	years	1.9	1.9	1.8	1.8	1.7

Figure 22.4 – After-Tax NPV 5%: Sensitivity to Capex, Opex and Gold Price



Source: DRA, 2025

Figure 22.5 – After-Tax IRR: Sensitivity to Capex, Opex and Gold Price



Source: DRA, 2025

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 22.5 with the base case highlighted.

Table 22.5 – After-Tax NPV and IRR Sensitivity to Varying Gold Price

Gold Price	Unit	US$ 1,500/oz	US$ 1,700/oz	US$ 1,900/oz	US$ 2,300/oz	US$ 2,500/oz	US$ 3,500/oz
NPV at 5%	US$ M	427	605	782	1,139	1,317	2,207
IRR	%	24.3	30.3	35.6	45.2	49.5	67.8
Payback Period	years	2.3	2.0	1.8	1.5	1.4	1.0

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 Prefeasibility Mining Study for the Timok Project in 2021 (De Weedt et al., 2021) which is available on SEDAR+ at www.sedarplus.ca. The MRE used as a basis for the study (effective date of 29 May 2020) includes 32.3 Mt of Indicated Mineral Resource with an average grade of 1.27 g/t Au and 1,319 koz of contained gold and 0.9 Mt of Inferred Mineral Resources with an average grade of 1.5 g/t Au and 45 koz of contained gold (DPM, 2021). Mineral Resources were estimated based on conceptual US$1,400/oz gold price pit shells to support RPEEE. Mineral Resources were reported in accordance with CIM definition standards (May 2014).

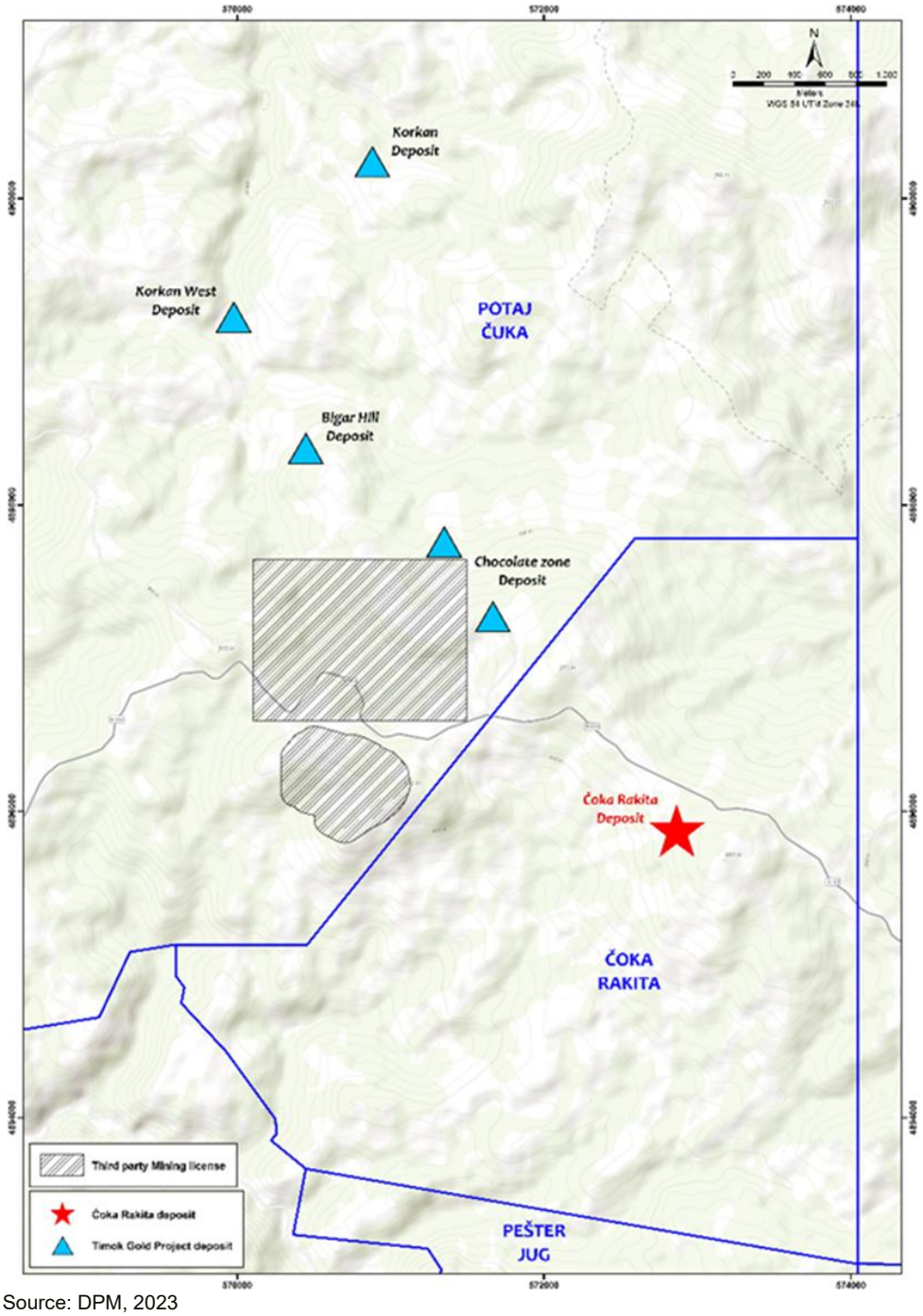
Probable Mineral Reserves of 19.2 Mt were reported, with an average grade of 1.07 g/t Au and 662 koz of contained gold (effective date of 29 May 2020). The reported Mineral Reserves assumed a conventional open-pit mining scenario (DPM, 2021) and were estimated at a gold price of US$1,250/oz and included modifying factors related to mining cost, and dilution and recovery, process recoveries and costs, G&A, royalties, and rehabilitation costs. A marginal cut off of 0.21 g/t Au was used for the Oxide material and 0.24 g/t for the Transitional material for all deposits. Mineral Reserves were also reported in accordance with CIM definition standards.

Given the high-grade gold potential of the Čoka Rakita Project, DPM has paused further FS work on the Timok Gold Project to focus on Čoka Rakita.

Please note that the QP, Malcolm Titley, MAIG, 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 the Timok Gold Project Deposits in Relation to the Čoka Rakita Project



Source: DPM, 2023

23.2 Dumitru Potok, Rakita North and Frasen Exploration Prospects (DPM)

On September 11th, 2024, DPM announced the discovery of high-grade copper gold mineralisation one kilometre to the North of Čoka Rakita[1] .

Subsequently on February 19, 2025, DPM reported the results from exploration drilling which further extended mineralisation[2]. DPM's 2025 exploration program includes approximatively 55,000 metres of drilling to follow-up on positive results to extend existing zones of mineralisation and to continue testing enhanced targeting models. A portion of this drilling will be dedicated to the Dumitru Potok and Frasen target delineation programs, which will aim to define the stratabound-hosted mineralisation, which is open in multiple directions, and to better understand the continuity of the mineralisation along strike and away from the causative intrusions.

Figure 23.2 shows a plan view highlighting the location of Dumitru Potok, Rakita North and Frasen prospects relative to Čoka Rakita and recent drilling on these prospects.

At the Dumitru Potok prospect, located approximately one kilometre northeast of Čoka Rakita, a target delineation drilling program has outlined a sub-vertical monzodiorite intrusive body, with proximal high-grade stratabound copper-gold-silver mineralisation on the conglomerate-marble unconformity and along the marble-intrusive contact zone.

Based on the drilling to date, stratabound conglomerate-marble unconformity hosted mineralisation extends outward from the eastern and western margins of the causative intrusion. Mineralisation can be traced for more than one kilometre of strike length and up to 500 hundred metres away from the causative intrusion and varies in thickness from five (5) metres to up to 40 m. Mineralisation on the marble-intrusive contact zone is developed for more than 300 m vertically, either as contact skarns or as stratabound lenses within reactive marbles, including on the lower marble-basement contact. The mineralisation is open in multiple directions; and its full footprint and grade continuity have yet to be defined. Figure 23.3 depicts are cross section through the Frasen and Dumitru Potok Targets, showing both the contact type skarns and the sub horizontal stratabound mineralisation.

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.

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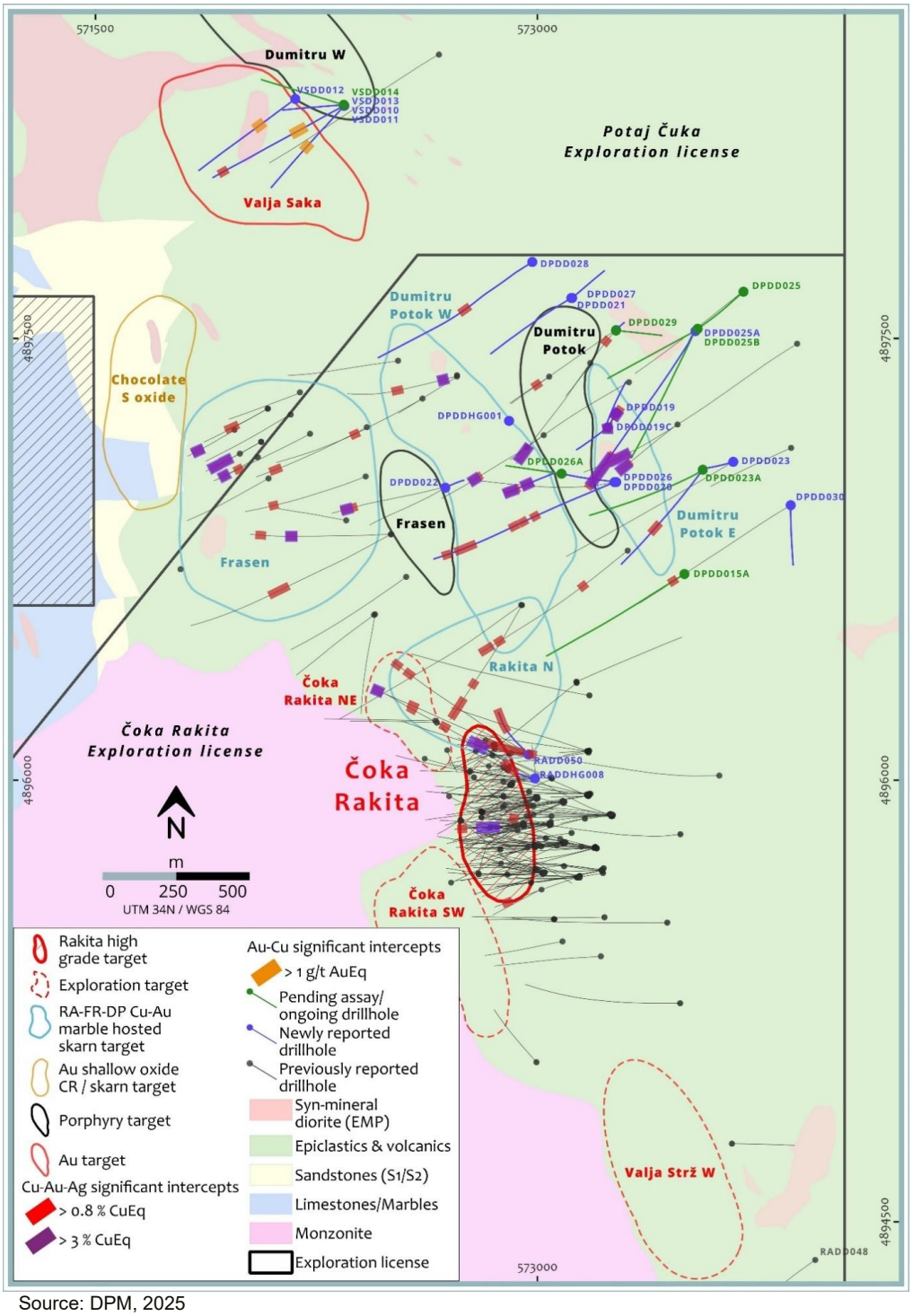
[1] Dundee Precious Metals Inc. | Dundee Precious Metals Reports High-Grade Copper-Gold Discoveries within One Kilometre of the Čoka Rakita Project; Results include 63 metres at 1.74% Cu, 2.18 g/t Au and 9.04 g/t Ag

[2] Dundee Precious Metals Inc. | Dundee Precious Metals Reports Wide High-Grade Intercepts from the Dumitru Potok Prospect; Results include 190 metres at 2.07% Cu, 1.23 g/t Au and 12.19 g/t Ag

At the Rakita North target, mineralisation is a combination of manto-like skarn mineralisation on the upper and lower contact, as well as more discrete replacement and stockwork veins over the entire marble lithological package. The extent of drilling shows significant copper-gold mineralization developed over an up to 300-metre vertical section, by more than 300 m by 150 m laterally, and remains open in multiple directions. Preliminary metallurgical testwork results demonstrated high recoveries are achievable, albeit with a significant mass pull, attributed to elevated levels of pyrite in certain samples, while others exhibit a higher presence of unspecified gangue. Future testing will focus on reducing mass pull by examining the effects of grind size on non-sulphide gangue and pyrite recovery, as well as exploring the advantages of incorporating a cleaning circuit for concentrate enhancement. Ongoing testwork includes a detailed mineralogical investigation to determine a range of key mineralogical criteria.

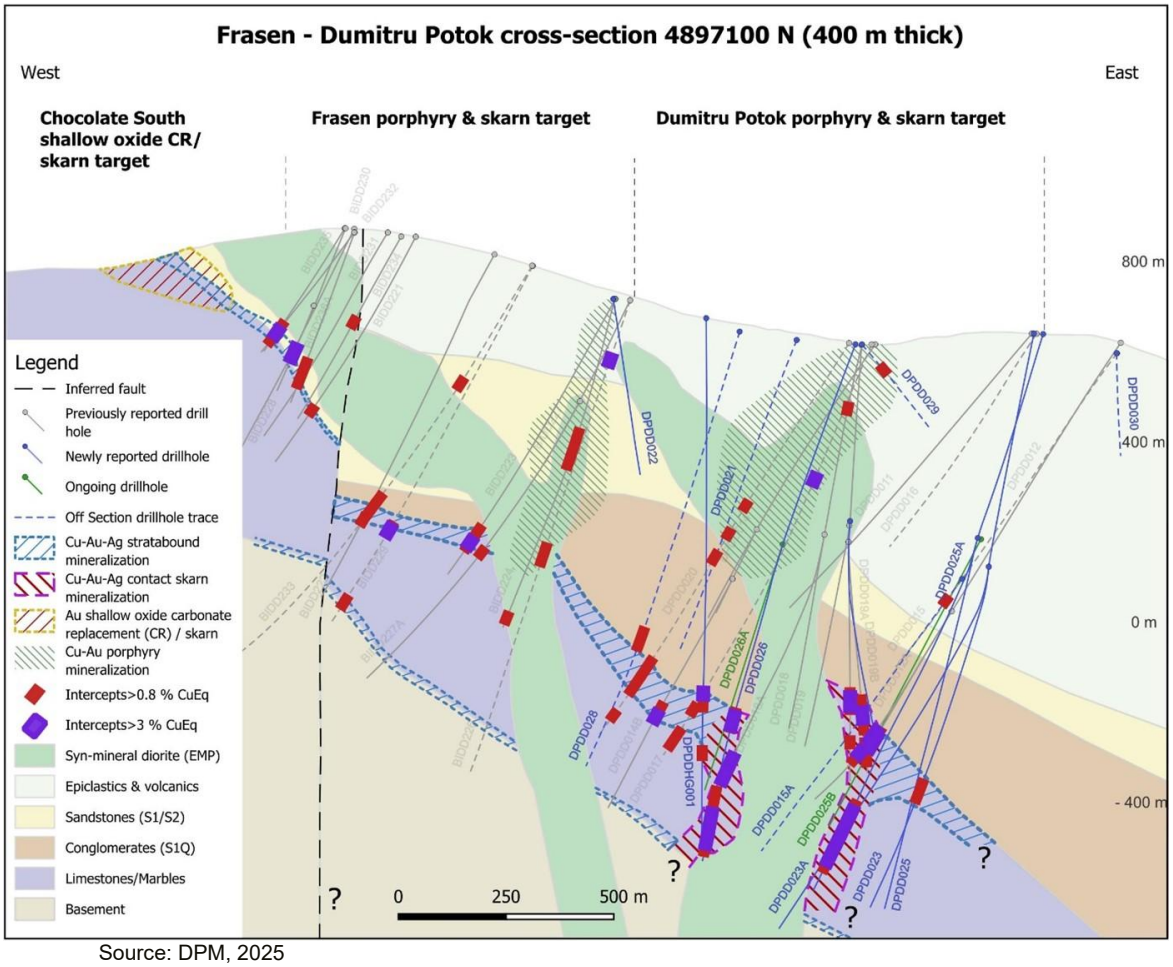
DPM's 2025 exploration program includes approximatively 55,000 metres of drilling to follow-up on positive results to extend existing zones of mineralisation and to continue testing enhanced targeting models. A portion of this drilling will be dedicated to the Dumitru Potok and Frasen target delineation programs, which will aim to define the stratabound-hosted mineralisation, which is open in multiple directions, and to better understand the continuity of the mineralisation along strike and away from the causative intrusions.

Figure 23.2 – Project Scale Map of Frasen, Rakita North and Dumitru Potok Targets and Results from the Scout Drilling Program



Source: DPM, 2025

Figure 23.3 – Cross section looking north at the Frasen and Dumitru Potok targets, located approximately one kilometre north of the Čoka Rakita deposit, showing scout drilling, the conceptual geology model and interpretation of target mineralization styles.



Source: DPM, 2025

24 OTHER RELEVANT DATA AND INFORMATION

This Section describes additional data relevant to the Project. The Project Implementation Schedule, Project Execution Plan, and Operational Readiness Development Plan will be discussed as part of this Section of the Report.

24.1 Project Implementation Schedule

24.1.1 INTRODUCTION

The Project Implementation Schedule includes the detailed engineering, procurement and construction activities as indicated. The information contained in this schedule is derived from information taken from supplier's firm bid quotes, budgetary quotes, in-house database(s); and contractor bids received by the DPM procurement team. The schedule presents the total duration of the Project considering Project Financing is available and environmental authorisations for construction are established.

The Project Implementation Schedule has been developed to include a ten (10) month Detailed Engineering 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.

24.1.2 EPCM PHASE

The Detailed 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 will be relocated, refurbished and installed, 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 Tepe Facility are planned to be executed by DPM's operation's team and relevant subject matter experts, with costs and durations established by DPM. The construction / execution portion of the schedule is predicated on the Ada Tepe sourced equipment being fully refurbished and delivered to the Čoka Rakita site just-in-time for installation. The construction periods at the Čoka Rakita site are based on DPM's expectation that unlimited resources (personnel and equipment) will be available at Čoka Rakita.

Most of the surface process plant areas were completed to a Feasibility Level and are advancing with development of Basic Engineering design and completion of purchase order activities to lock-in the pricing of the selected equipment packages and advance the acquisition of the vendor

preliminary data and vendor certified information. The Basic Engineering design phase is expected to be completed by the end of Q1 2026.

The Tailings Filtration Plant and the Paste Backfill plant were advanced to a Feasibility level of completion with budgetary pricing and associated design modelling.

DPM identified that a significant amount of the surface mobile equipment will be relocated from DPM's Ada Tepe facility.

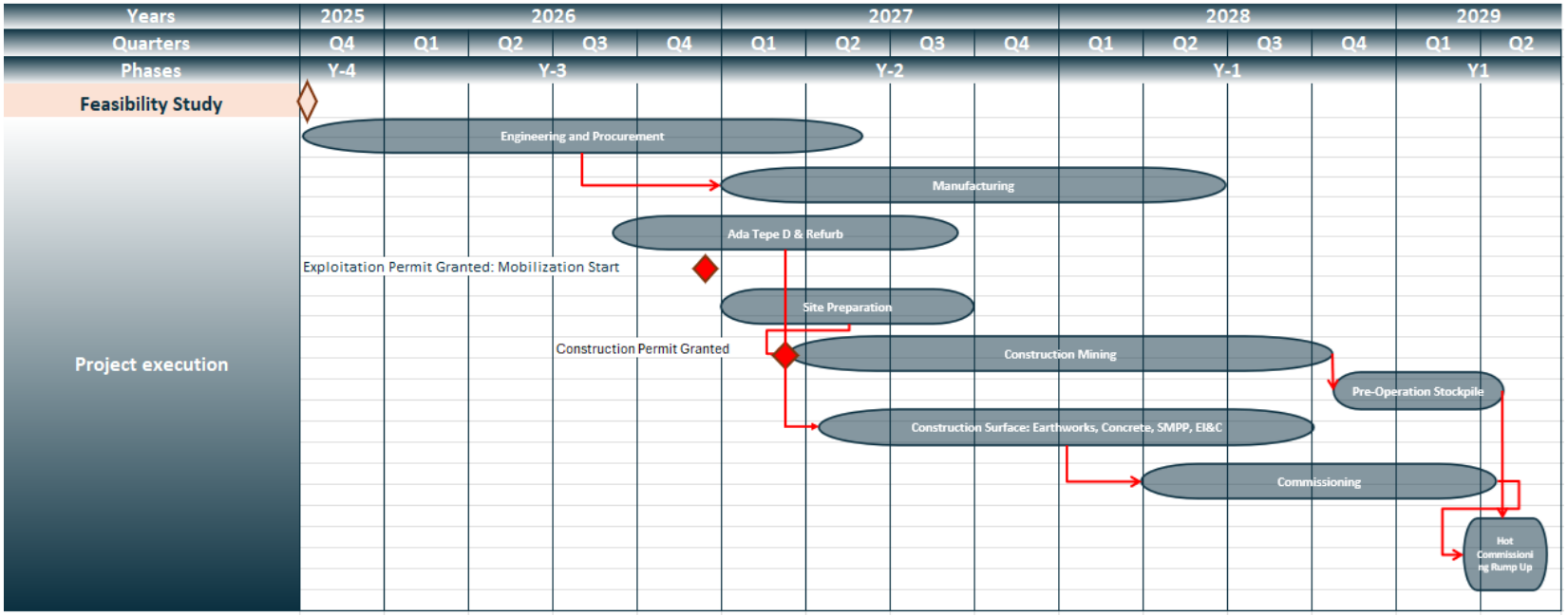
During the FS phase of the Project to the following items were conducted:

- Define the EP and CM phases, as two (2) distinct phases.
- Advanced firm bid pricing of most of the surface equipment, to enable advancement of equipment purchase order completion and acquisition of Vendor preliminary data and Vendor certified data.
- Developed the site engineering and design required to establish the infrastructure and site preparation to adequately prepare for the underground preparation and construction phases.
- Identified preliminary designs (including options) for temporary and permanent access to the Čoka Rakita site.
- Determined some logistics for equipment and materials deliveries, laydown areas and security.
- Identified procurement for the above including bid preparation and evaluation, organisation of site visits, contract preparation and contract administration.
- Started preparation of mobilising 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



Source: DPM, 2025

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 continue to be developed by the DPM project team during the Execution phase.

It is critical to ensure all stakeholders are aligned and that the project controls are in place. The following list outlines the content that is described in detail in the PEP.

- Project Goals and Objectives.
- Project Location.
- Scope of Facilities and Overview.
- Project Organisation.
- Health and Safety Execution Plan.
- Environmental Execution Plan.
- Risk Management.
- Engineering Execution Plan.
- Procurement and Contracting Execution Plan.
- Construction Execution Plan.
- Commissioning Execution Plan.
- Project Controls.
- Electronic Document Management Systems.
- Project Communications.
- 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 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.

- Ensure that the correct personnel are effectively trained and prepared to receive the handover of the plant at the correct time. Ensuring that these personnel are prepared to work with the equipment, systems and technology, adhering to the fundamental tenets of safety, environment, reliability and sustainability.
- Identify and address all significant operational issues, from the start of the operations readiness activities, through to Project handover.
- Ensure appropriate operational procedures, systems and personnel 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, commissioning to startup of operations and meeting of 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 can be 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.

24.3.2 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.

24.3.3 ORP Phases

The ORP commenced during the PFS stage with an initial study, by the DPM OR team. The plan and scope were matured during the FS phase and will be ready for rollout at the commencement of the execution phase. The ORP extends beyond the handover period into the ramp-up and steady state phases as an overarching plan linking the phases together.

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, Malcolm Titley, MAIG, conducted a personal inspection of the Project on between May 13 to 15, 2025 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 January 17, 2025. A total of 299 drillholes totalling 141,505 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 approximate 1 g/t Au cut-off value. Detailed lithology and structural models were developed and used to constrain domain extents, as well as to incorporate 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 (which is reported here), silver, 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 optimized mineable shapes using Deswik's Stope Optimiser (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

Testwork results support the selected flowsheet and is adequate to size equipment with confidence. Variability results shows that there is little variability within the mineralised material. The only exception is with the arsenic content in concentrates produced from distinct variability samples; this prompted the introduction of concentrate storage bunkers to allow blending of the final product prior to shipment.

Confirmatory testwork performed during the FS phase validates the design decisions made during the PFS and adds confidence to the design criteria selected for sizing equipment. Pilot testing of the Jameson cell confirms both the recoveries and mass pull numbers that were previously adopted based on bench scale testing.

25.3 Filter and Paste Plant Related Testing

The testing conducted by Jenike & Johanson (J&J) on the transportability of the 85 wt% solids Pressure Filter Cake, and its flowability concluded and confirmed the following elements of the current FS design:

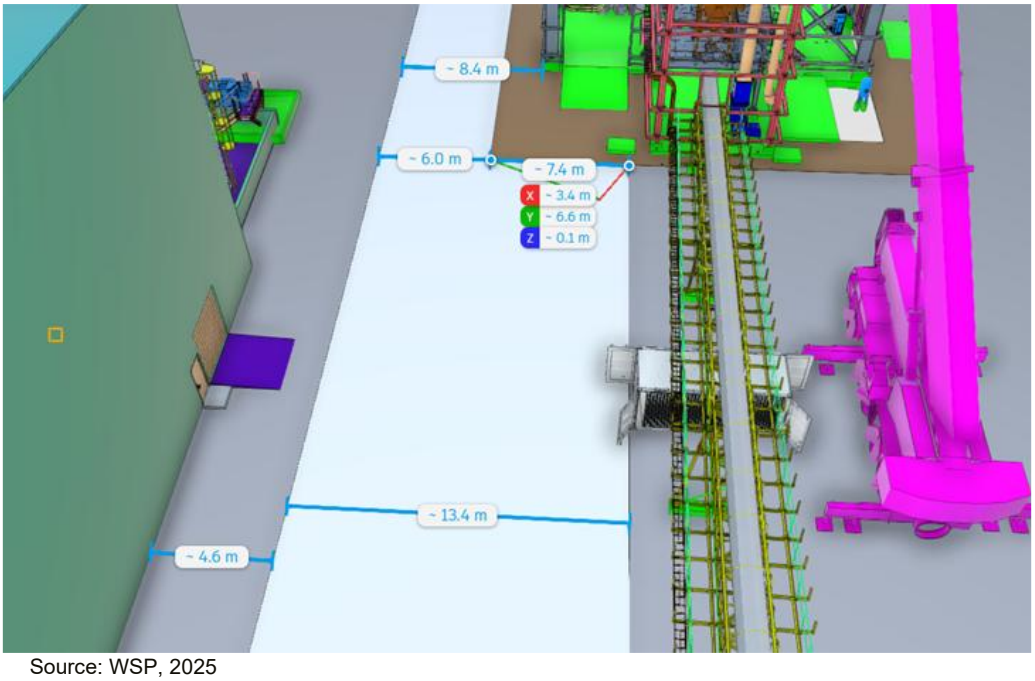
- The acceptability for the angle of the Paste Plant Incline Conveyor at +21.5 degrees (+39.4%) and having a conveyor belt of conventional construction.
- The acceptability of the 6 mm UHMW lined rectangular Live Bottom Feeder Hopper having two (2) straight sides and two (2) sides with a 71-degree taper at the outlet, to a nominally 8 m long by 6 m wide outlet, would allow mass flow of the Filter Cake onto the conveyor beneath it.

25.3.1 METSO DEWATERING TECHNOLOGY CENTER

The Metso Dewatering Technology Center (DTC) Pressure Filtration testing report concluded that a Metso Larox FFP (62/62) could dewater the 58 wt% solids tailings slurry to 86.2 wt% solids, which was below the testing's target Filter Cake solids content of 87 wt%. It should be noted however that there is insufficient information presented in the report to allow an analysis of the effect of varying such parameters as blowing duration or chamber depth (only a 50 mm deep chamber was tried) on the final cake moisture. Since no graphical data was provided in the Metso laboratory report, providing recommendations is difficult and it should be clearly stated that any future Vendor testing reports are to include graphical backup of the testing results.

The length of the Metso Larox FFP (62/62) Pressure Filter is larger than that currently shown in the FS 3D Filter Plant model. A longer Pressure Filter could be accommodated in the Filter Plant since there is adequate room available to expand the building to the north between the Primary Crusher building and the Filter Plant building as shown in Figure 25.1.

Figure 25.1 – Available Area Between the Filter Plant and Primary Crusher Buildings



Source: WSP, 2025

25.4 Mineral Reserve Estimate

The Mineral Reserve estimate for the Project has been prepared in accordance with industry best practices and is compliant with the 2019 CIM Definition Standards. Conversion of Mineral Resources to Mineral Reserves was based on a comprehensive mine plan and engineering analysis, incorporating key modifying factors such as dilution, minimum mining widths, ore and extraction

losses, appropriate mining methods, metallurgical recoveries, permitting considerations, and infrastructure requirements.

At the time of the reserves estimate, there are no Proven Mineral Reserves for the Čoka Rakita Deposit.

The Probable Mineral Reserve for the Čoka Rakita Deposit totals 7.344 Mt of diluted ore, grading 6.44 grams per tonne (g/t) of gold (Au), containing approximately 1.52 million ounces of gold. This represents an increase of 9.7% in tonnes and 10.5% in contained ounces compared to the Pre-Feasibility Study (PFS) reserve estimate as described in Section 15.6 of the Report.

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 cost assumptions.

25.5 Mining Methods

25.5.1 Underground Geotechnical

Rock mass classification assessments (RMR76 and Q' values) were completed based on the data available and divided into seven (7) geotechnical domains. The main rock masses are classified as exhibiting poor 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 metres in width (poor to very poor) that can have an impact on local stability and increase stope dilution. Where possible, the critical infrastructure (e.g., maintenance bay, surface ventilation raise, etc.) were moved to accommodate the fault zones. Ground support requirements for underground infrastructure that intersects fault zones will be heavier (type of support and density of support) compared to the standard support in the ramp, access, and development drifts.

Empirical and analytical stability assessments and 3D non-elastic and elastic numerical induced stress assessments were completed to review the stability of the mine plan and sequence and review standoff distances for the access drifts, spiral ramp, and surface ventilation raise. The pillars produced from stopes and drifts were assessed using 3D numerical methods and the sequence and spacing of stopes / drifts were adjusted to improve stability related to induced stress. Additional geotechnical assessments (empirical and numerical) were completed on the key infrastructure including surface vent raise, upper and lower declines to define ground support recommendations.

The available underground geotechnical data collected, and stability assessments completed confirmed the key assumptions for the FS mine design (e.g., stope size, stope mining general sequence, location of key infrastructure, ground support).

25.5.2 HYDROGEOLOGY

The objective of the update of the numerical groundwater flow model was to update estimates of potential groundwater inflows into the planned UG development. The current model would have to be further modified to evaluate potential environmental impacts of the Project, and to assist with closure studies

- The Hydro-stratigraphic unit zonation has been improved by including the updated Leapfrog geological model within the model domain. This improved zonation features a north-west draining high permeability limestone and marble basement.
- Main faults identified and tested during the field investigations, which are considered to potentially play a significant role in the groundwater flow regime within the model domain, have been implemented in the updated model.
- The calibration of the pre-mining FS model shows a notable improvement with a NRMSE of 4.9% compared to the previous 11% achieved by the pre-mining PFS model, with an updated correlation coefficient of 98.7% versus the earlier 92%.
- The base-case inflow simulation is accompanied by high and low inflow scenarios to provide high-level risk assessment.
- Mean annual inflows of up to 2,520 m³/d have been simulated for the Base Case scenario, and up to 3,271 m³/d for the High Inflow scenario.
- Highest mean annual inflows are expected in the second year of operation, with the minimum inflows occurring during the first year of operation.

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.

25.5.3 MINING

Multiple aspects of the Project have been progressed in the FS stage 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.
- Owner's lateral development rate was reviewed compared to the PFS which was 3.5 m/d to 5.4 m/d in the FS, due to revisions in ground support and equipment efficiencies.
- Stope production activities are more detailed than that of the PFS due to added stope preparation, slashes and fill preparation.

- Material handling has been progressed since the PFS with the addition of added remucks on level crosscuts while also incorporating truck turn arounds to increase cycle times.
- Sublevel long-hole open stoping longitudinal retreat is an appropriate method for mining the Čoka Rakita deposit.
- The number of units in the planned equipment fleet and their makes, and models are appropriate for the projected production rate, mining method, and development requirements at Čoka Rakita.
- LOM has been extended by an additional year when compared to the PFS to Year 9.
- Milling will commence at the beginning of Year 1 and all Year 1 waste will need to stay underground.

25.5.4 INFRASTRUCTURE

25.5.4.1 Electrical and Communications

The scope of the FS is limited to the underground power distribution system and above ground main exhaust fans E-House including 1 kV, 690 V, 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.

25.5.5 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 challenging at this time and are worthy of refinement in future phase of this Project.

25.5.6 MINE DEWATERING

Ground water inflow is a significant contributing component to the total water pumped from the mine.

The current design of the mine dewatering system utilises a good combination of gravity flow and pumping, while minimizing design complexities. The system has also been designed to account for flexibility on operating modes.

Sumps have been adequately sized to meet the minimum 8-hour surge capacity for water underground for any eventual upset or power failure. The Main dewatering pumps are arranged in a two (2) out of three (3) operating modes, with the ability to run one (1) through three (3) pumps, by using the redundant discharge pipe to surface. Two (2) pumps are sufficient to accommodate the high flow scenario with the third pump available to handle any unforeseen peaks. The main dewatering pumps are also on emergency power, to have the ability to operate under a plant wide power failure scenario.

25.5.7 MINE VENTILATION

The ventilation system has been designed to provide sufficient air for the planned production and development activities. Due to a combination of the relatively high airflow requirement dictated by the Serbian regulations and procurement of new and clean diesel fleet for the operation; it is expected that the contaminants levels emitted by the diesel equipment will always remain below exposure limits during a shift.

The ventilation automation system is deemed essential to this operation to maintain compliance with the operating diesel equipment and distribute fresh air efficiently and dynamically during a shift.

Where auxiliary duct is concerned, it is assumed that good maintenance practices and best-in-class products are expected to be deployed to minimize the risk of lack of airflow at the working face.

The greatest risk for the integrity of the ventilation system with the operation is open stope leakage through the ventilation curtains. It is therefore essential to provide enough resources for the maintenance and installation otherwise headings and other areas could lack of airflow which would put the health and safety of the workers at risk

25.6 Processing and Recovery Methods

The various risk and operability assessments revealed no major concerns. The design is robust and includes all reasonable measures to ensure a safe workplace.

A concern was raised regarding the potential for penalties applied to the flotation concentrate due to exceedance of the arsenic threshold when mining certain high arsenic pockets within the orebody. An analysis of the occurrence of arsenic showed that inadvertent blending within the surge capacities in the plant, combined with the 7-day concentrate stockpile post filtration, provides adequate blending to negate this concern.

25.7 Filter and Paste Backfill Plant

The Feasibility design of the Filter and Paste Plants is based on information from: the Pre-feasibility design, provided by DPM, and laboratory testing by Jenike and Johanson (J&J) as well as Metso Dewatering Technology Center (DTC) (Metso) conducted in this phase of the Project.

The design of the Filter Plant is focused on dewatering all tailings produced by the mill for Filter Cake disposal to either the DTSF or use in Paste Backfill production in the Paste Plant.

The consistencies of the Filter Cake applied in this phase of the study were 87 wt% solids to the DTSF and 85 wt% solids to the Paste Plant.

Geotechnical laboratory testing late in this phase of the Project determined that the optimum moisture content of the Filter Cake for trafficability in the DTSF was lower, being in the range of 89.0 to 89.2 wt% solids.

Metso's laboratory testing for Pressure Filter design indicates that 89 wt% solids (11 wt% moisture) is not achievable and that a filter with a larger footprint would be required in any event. Therefore, a solids and moisture content target must be established early in the next phase of the Project to advance the process and plant design because the selection of the Pressure Filter has a governing effect on all engineering work with potential impact to the footprint of the Filter Plant.

The J&J, Metso, and previous RMS testing were all conducted on tailings having a specific gravity (SG) around 3.2. It is understood that the SG of the tailings will vary throughout the LOM and to that end, the Design Criteria value of the tailings SG was changed to 3.05.

The reduction of the tailings SG prompted an empirical recalculation of the rheological characteristics of the paste backfill, and subsequent re-evaluation of the hydraulic flow model and design pressure of the Positive Displacement Paste Pump.

The positive displacement Paste Pumps' capital cost is the second largest component cost of the Filter and Paste Plant process equipment, surpassed only by that of the Pressure Filters.

The capital cost of a 150-bar design pressure Paste Pump has been included in the Capex estimate, while the revised empirically derived design pressure of 120-bar will result in a lower Capex which should be verified by Vendor quotes.

The J&J laboratory testing verified the acceptability of the angle of the Paste Plant Incline Conveyor at +21.5 degrees (+39.4%) with a conveyor belt of conventional construction. The testing of the Filter Cake at 85 wt% solids also confirmed the Filter Cake's flowability through the current design of the Live Bottom Feeder Hopper in the Paste Plant.

The routing and configuration of the surface Paste pipeline from the Paste Plant to the upper decline portal was finalised late in this phase of the Project. The incorporation of a second Paste pipeline and the need for a short trestle supported section of the pipeline will require a robust design of Paste Plant's discharge pipeline in the next phase of the Project.

25.8 Shotcrete/CRF Slurry Batch Plant

During mine development Shotcrete will be used as a component of the required geotechnical support of underground excavations.

Shotcrete will be formulated on site from offsite procured sand in a batch style modularised mixing plant located on surface. The Shotcrete will be produced and hauled underground by a fleet of transmixers in an "on demand" fashion.

Although Shotcrete will be required throughout the LOM, the greatest demand for it will be in the preproduction Year -2. The procurement of a modularised plant, with the majority of its components built and assembled off site, will facilitate the rapid commencement of Shotcrete production.

The other backfill methods complimenting the primary method of Paste backfill, is cemented and uncemented rockfill (CRF and RF) using development waste rock generated in the mine and eliminating the requirement to haul waste rock to surface for disposal beyond the initial pre-production years of the Project.

Backfilling with CRF will start in Year -1 and continue until Cemented Pastefill (CPF) is introduced in the second quarter of Year 1. After that time, both will be used until the available waste rock for CRF is depleted in Year 6.

The CRF Slurry will be formulated in a modular plant on surface, hauled underground by the fleet of transmixers in the same "on demand" fashion as Shotcrete, deposited on a measured amount of waste rock in a CRF Mixing Bay, then mixed by an LHD and deposited in the stope being backfilled.

The individual Modularised designs of a Shotcrete and a CRF Slurry Plant will be combined into a single, dual-purpose Shotcrete/CRF Slurry Batch Plan for this Project.

The design presented in this phase is preliminary and requires more development in the next phase of the Project in a multi-disciplinary approach, it should be noted however, the designs of individually purposed Shotcrete and CRF Slurry Plants as packages has been readily available on the market for many years. The Plant required for this Project will be an integrated combination of the two (2) Modularised Plants.

The integration of the two (2) Modularised plants should be done bearing in mind any significant redesign of the individual Modules will greatly impart delivery time and cost of the integrated Batch Plant.

25.9 Dry Tailings Storage Facility

The Project is designed to operate 24 hours a day, 336 days a year. Tailings material will be produced from the filter press and will be transported initially via a conveyor system to the designated tailings storage terrace. A proportion of the overall tailings stream will be delivered back to the underground.

A designated multi-purpose storage area has been designed to store tailings material during inclement weather conditions. There will be a requirement to have a roof over the storage area to prevent a re-saturation of the tailings.

During the initial years of mining operations, excavation of the decline portals and underground workings will produce Mine Waste Rock. The PAG will be stored within the base of the DTSF. Containment is provided through the installation of a lining system across the base of the valley. Once all the PAG Mine Waste Rock has been placed within the initial two (2) years of mine development and following the first year of operations, the top of the PAG rock will be composite lined and then the placement of the filtered tailings can commence above the PAG Mine Waste Rock. Following completion of the Underdrain and placement of the PAG zone in year -2 and -1, the facility will be raised 35.4 m over 9 years. The average rate of rise (RoR) for the facility is 3.8 m/y.

Trucking across the DTSF may require construction of a series of temporary access roads which may change frequently and require additional construction material.

Geotechnical stability analysis for the DTSF including the PAG formation design was performed and assessed against CDA guidelines, (CDA, 2013) and Serbian Standards (SEM) (SRPS U.C5.020, 1980). Stability analysis confirmed the minimum factor of safety for the different scenarios was achieved.

The FS closure design of the DTSF focusses on replicating natural patterns of drainage channels and slopes to integrate the final landform with the surrounding area.

Material management across the Čoka Rakita mine site will be a complex operation during both the early development years and also throughout the remaining LOM. Four stockpiles have been identified to store a combination of overburden and topsoil. The sizing of these additional structures will depend on the ground conditions identified during the construction of the mine site. Geotechnical stability analysis was performed and assessed against CDA guidelines, (CDA, 2013) and Serbian Standards (SEM) (SRPS U.C5.020, 1980). Stability analysis confirmed the minimum factor of safety for the different scenarios was achieved.

25.10 Surface Water Management

As part of the Feasibility Study of the Čoka Rakita Project, the surface water management design was developed for the site. This includes the sizing of the Reclaim Pond, CW Pond and NCW Pond, their corresponding spillways, and the CW and NCW channels.

The critical design decisions and conclusions are as follows:

- The pond sizes were maintained based on their PFS dam embankment geometries. The pond operations were optimised to minimise the pump out rate required and reducing flows to the Clarifier and WWTP. This also provided more consistent and manageable inflows to the Clarifier and WWTP.
- The Inflow Design Flood (IDF) for the pond operations has been defined in accordance with GISTM (ICMM, 2020) “Significant” consequence classification, resulting in the 1:1,000 Annual Exceedance Probability (AEP) event being adopted.
    - A preliminary Dam Breach Assessment (DBA) and Consequence Classification Assessment (CCA) was conducted on the cascading failure of the CW Pond and NCW Pond. This resulted in a “Significant” preliminary classification for the CW Pond and NCW Pond.
    - The “Significant” classification from the cascading failure was also applied to the Reclaim Pond. Given that the combined volume of the CW and NCW Pond is greater than that of the Reclaim Pond, this represents a conservative classification for the Reclaim Pond.
    - At the time of writing the ponds DBA has been updated confirming the “Significant” classification for the Reclaim Pond and resulting in a “High” classification for the NCW and CW Ponds. It is however noted that the Digital Elevation Model used to estimate the potential impacted area has a low resolution resulting in an overconservative estimate of the flooded area. This gap should be addressed in the next phase of the studies to review the classification.
- Runoff and seepage from the DTSF are conveyed via the perimeter CW channels into the Reclaim Pond. The Reclaim Pond has a total capacity of 51,900 m³, based on a 1:100 AEP spill frequency, and a pump out rate to the Clarifier of 15 L/s. Based on the sizing exercise only 67% of the total pond volume is used for storm surge capacity. The pond is equipped with an emergency spillway designed to convey the 1:1,000 AEP peak flow. The pond spillway sizing should be revised in the next phase of studies taking into consideration of the updated design event adopted for the sizing of the NCW diversion channels. The feasibility of adopting a single Integrated Discharge Facility downstream of the Reclaim Pond should also be evaluated in the next phase, combining the two (2) Reclaim Pond stilling basins and the DTSF underdrainage outlet.

- The feasibility of adopting a single Integrated Discharge Facility downstream of the Reclaim Pond should also be investigated in the next phase, combining the two (2) Reclaim Pond stilling basins and the DTSF underdrainage outlet.
- The freeboard for the pond containment embankments has been taken as 1 m from the water level at the spillway, when conveying the IDF, to the embankment crest.
- There is a 1 m freeboard from the pond water level, when the spillway is conveying the IDF, to the containment embankment crest. The spillway weirs have been sized to convey the IDF within a 1 m water depth from the spillway weir invert. This gives a total depth of 2 m from the spillway invert to the dam embankment crest.
- The CW Pond was sized to accommodate runoff from the ROM Pad and Terrace area and the volume of water treated via the Clarifier and WWTP (to account for scheduled and unscheduled maintenance of the WWTP), with a 1:100 AEP spill frequency. The pond has a total capacity of 62,000 m³ and a pump out rate to the Clarifier of 15 L/s. Based on the sizing exercise only 50% of the total pond volume is used for storm surge capacity. The pond is equipped with an emergency spillway designed to convey the 1:1,000 AEP peak flow.
- The NCW Pond will be used as an emergency water supply for the site. It has a storage capacity of 58,700 m³, based on the PFS design sizing. Once full, the pond will discharge excess water to the downstream environment via the spillway, which has been designed to convey a 1:1,000 AEP flow event. For managing minimum environmental flows or reducing the pond water level, water can be discharged through a low-level outlet which is controlled by a gate valve.
- Additional information and analysis related to the ponds sizing and performances in the overall site wide context under different climatic conditions and groundwater inflows scenarios has been presented in the Site Wide Water Balance and Water Quality Model Report (WSP, 2025g). The outcome of the water balance would ultimately support decision making around future optimisations of pond sizes and/or optimisation of pumping rates.
- NCW diversion channels will be constructed around the DTSF to divert NCW around the contact areas of the site. The channels will be designed to convey the 1:200 AEP peak flow.
- The Upper and Lower CW Pond diversion channels intercept and divert NCW to the NCW catchment, limiting water volumes requiring treatment. The channels will be constructed to convey the 1:100 AEP peak flows.
- To manage runoff from the three stockpiles areas perimeter and diversion channels will be constructed to convey the 1:100 AEP peak flows.
- Five (5) different channel designs have been applied to the site water management channels:

    1. Trapezoidal channels lined with CC8 concrete canvas:

These were adopted for mild slopes which do not require energy dissipation. To maintain a smaller channel footprint, concrete was chosen over a rougher channel lining (i.e., riprap) as it results in lower flow depths.

2. Trapezoidal channels lined with riprap:

   These were adopted where velocities needed to be reduced to limit scour. Where the riprap lining alone was not sufficient, rock check dams have been included to maintain velocities below 3.5 m/s.

3. Rectangular stepped channels lined with gabion baskets, Reno mattresses and concrete slabs:

   These channels are required where high flows are conveyed down steep slopes. The objective of the steps is to dissipate flow energy along the channel to maintain flow velocities below 5 m/s. Near vertical channel sides retained with gabion baskets reduces the channel footprint and consequently the cut requirements into the natural slopes.

4. Rectangular reinforced concrete channels.

   Adopted only next to the Reclaim Pond, to provide an enhanced robustness to scour in this critical location, as well as minimizing the channel footprint.

5. Rectangular reinforced concrete chutes

   These were used for the DTSF and pond spillways. In these locations there was a need to manage and control flows with high velocities. Energy dissipators in the form of stilling basins are provided at the downstream end of each chute.

- The erosion and sediment management design focuses on preventing erosion and removing sediment at its source. Where this is not feasible, sediment controls can be installed along water conveyance structures to capture transported sediment, with the water management ponds providing the backstop for sediment control.
- Several sediment control options have been considered for the site sediment management. These include polymer sprays, portable sedimentation tanks, geotextile filter bags, compost-filled filter socks and check dams.
- Closure will involve the removal of all the operational water ponds and ancillary infrastructures. The area between the DTSF and the ring road will also be backfilled to effectively manage runoff into the closure channels. These channels will mostly adopt the alignment of the operational DTSF NCW channels but with a larger footprint to accommodate the 1:10,000 AEP with climate change design flow. Depending on the design chosen for closure, long-term maintenance may be required for the closure channels.

25.11 Site-Wide Water Balance

The Water Balance Model (WBM) provides a robust assessment of Čoka Rakita's water management under various conditions. While operational demands are consistently met, the study identifies a potential shortfall to satisfy the minimum environmental flow during the pre-operational period in case of a low groundwater inflow scenario. Based on the outcomes of the water balance the NCW Pond sizing should be revised in the next phase of the study. Based on the source terms derived from available geochemical data, exceedances with respect to Class 1 water quality limits are noted in the water management ponds. It is important to note that these results reflect data available during model development, and future iterations should incorporate evolving project data and system updates.

25.12 Capital and Operating Cost Estimates

The capital and operating costs were estimated using methods described in this Report, which are typical of FS-level cost estimation exercises.

The costs as presented are suitable for use as "the budget" for the project implementation.

25.13 Economic Analysis

Based on the available information, the Project has an after-tax NPV of $782 M at a discount rate of 5%, an IRR of 35.6% and a payback period of 1.8 years from the start of operations. The sensitivity analysis indicates that the Project economics are most sensitive to the gold price. Significant sensitivity is also observed to changes in the estimates of Capex and Opex. Even with a gold price 20% below the base case of US$1,900/oz, the Project maintains a positive after-tax NPV.

25.14 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 this FS 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.
- 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.

25.15 Other Relevant Data and Information

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

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

DPM will finalise the ORP during the detailed engineering 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.

25.16 Opportunities

25.16.1 MINING

Exploring alternative vertical drilling equipment or technology. Rotary vertical drilling systems (RVDS) demonstrate great accuracy and could assist with the surface boreholes to the lower decline as well as the paste backfill bypass holes.

Infrastructure - Electrical and Communications

Evaluating the potential to route cables down internal ventilation raises can save on service borehole development.

25.16.2 Filter and Paste Backfill Plant

The footprint required for the Filter and Paste Plants is constrained by the angle of the inclined conveyor between the Filter and Paste Plants.

The inclined conveyor is somewhat steeper than ideal; however, it is still acceptable over the short distance between the plants. The conveyor's inclination has been verified as being acceptable in third party material transportation and flowability testing that was commissioned by DPM in this phase of the Project.

Hydraulic flow modelling indicates that system operating pressures can be limited to below 90 bar allowing for the use a 120-bar design pressure positive displacement paste pump instead of a 150-bar design pressure pump as outlined in the PFS. Careful consideration needs to be given to engage in hydraulic flow loop testing to confirm pressure loss estimates before the pump is ordered, so as to de-risk this specification change and ensure that the 120-bar design specification is adequate for the purpose.

25.16.3 Shotcrete/CRF Slurry Batch Plant

An integrated plant design offers the opportunity for the use of common components, such as the Binder Silos, which has been done in this study. Other possibilities such as the common use of a colloidal mixer to produce binder slurry for the production of Shotcrete could eliminate the need for a Binder Weigh Hopper.

25.16.4 Processing and Recovery Methods

The AG mill and Vertimill have some spare capacity with respect to the installed power. This provides an opportunity to increase throughput marginally or grind finer to improve gold recovery.

Filter and Paste Plants

With additional laboratory rheological testing and a flow loop, the pipeline friction losses can better quantify the applicability for the use of a 120 bar Design Pressure Positive Displacement Paste Pump and the inherent Capital cost saving.

25.17 Risks

The Project team (DPM, DRA, ERM, WSP) conducted a multi-day project risk review workshop during the week of June 16, 2025, 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 FS risk review workshop utilised the risk register established during the PFS as the basis for the FS workshop.

The total number of risks identified, in accordance with the DPM ERM Framework, was 41 risks, which comprised: 4 “High” risks, 21 “Moderate” risks, and 16 “Low” risks.

During the detailed engineering and execution phases, additional work will be conducted to eliminate the remaining High-risk items.

The QPs noted that 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 basic-detailed engineering phase.

Process

DPM provided sufficient Geotechnical information to DRA and WSP, during the FS, for the process plant terrace elevations and area to be confirmed and frozen. WSP has identified that they will continue additional detailed analysis of the slope stability, in the next phase of the Project, to ensure that the terraces and their associated embankments will be stabilised.

Process Facilities

During the basic-detailed engineering phase, the Geotech needs to be resolved/confirmed, so that the engineering designs can be finalised.

Filter Cake Production and Paste Backfill Systems

There is no doubt that the specific gravity (SG) tailings solids will vary over the LOM. The rheological properties of the tailings solids form the basis for the Filter and Paste Plant equipment specifications and sizing. A sample of tailings with a “worst case” SG should be tested and the results used for the final design criteria of the Plants to be used in the next phase of the Project.

A significant change in the size and design of the Pressure Filters required to produce optimum Filter Cake moisture content for DTSF disposal could cause a redesign of the Filter Plant and the overall footprint of the Filter and Paste Plant area.

Shotcrete/CRF Slurry Batch Plant

Individual Modularised Shotcrete and CRF Slurry Batch Plants have been and are available on the market that have proven track records for functionality. Their simplistic, portable, and low-cost designs are intended for use in operations such as Čoka Rakita which have a relatively short LOM. Significant redesigns of Vendor supplied modular plants are likely not only to result in price increases but longer delivery and implementation times, with a greater risk of effective and harmonious

functionality. Therefore, caution should be exercised in the rigid applications of some design standards other that those related to safety and established accreditations.

Dry Tailings Storage Facility

The top five (5) risks identified within the Čoka Rakita FS DTSF design are as follows:

- Dam Failure – mitigated by use of filtered tailings technology to reduce impact of DTSF failures.
- Potentially Acid Generating – mitigated by lining the PAG rock zone with composite lining systems and inclusion of the material within the DTSF footprint.
- Seepage to environment – mitigated by use of composite lining system accompanied by basal underdrainage channel with access / monitoring point downstream of the DTSF.
- Instability caused by inadequate construction material – mitigated by Construction Quality Assurance (CQA) and frequency of laboratory testing.
- 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 Detailed Design.

Environment

All environmental mitigations, identified during the FS, need to be addressed during the basic-detailed engineering phase, to ensure a robust design which eliminates the remaining High-risk items, as identified in the FS.

Economic Analysis

The Project economic performance is highly sensitive to the gold price, 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 base case gold price of US$1,900/oz is significantly below the prevailing spot price of gold.

The current geopolitical uncertainty around trade policies has the potential to have inflationary effects on global markets. Significant cost inflation or cost overruns driven by other factors would negatively impact the Project economics given the sensitivity observed to both Capex and Opex.

The economic analysis presented in this Report considers that the Project benefits from the tax holiday available to companies making significant investments in Serbia. Any changes to

government policy around the tax holiday provisions (such as the cancellation or reduction of the tax holiday) have the potential to significantly impact the economics of the Project.

26 RECOMMENDATIONS

26.1 Geology

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

26.1.1 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.

26.1.2 DRILLING

No further drilling is planned at Čoka Rakita as the Project is moving into its development phase.

Once the decline has reached a reasonable depth and underground drill cuddies are established further infill drilling will take place.

26.1.3 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.
- 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.

26.1.4 ASSAY QA/QC

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

- Approximately 91% of samples in mineralisation (>4,800 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 mineralized 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. Additional CRM’s at similar or higher Au grades would be beneficial.

- Continual vigilance is required considering the extremely high gold grade values that have been encountered while drilling.
- The QP finds that the QA/QC procedures are adequate and that there are no fatal flaws evident.

26.2 Mineral Processing and Metallurgical Testing

26.2.1 DRA Recommendations

All recommended testing has been completed. There are no further recommendations pertaining to future testwork.

26.2.2 WSP Recommendations

WSP recommends the following:

- The laboratories that conducted the testing in this phase of the Project all received tailings whose solids specific gravity (SG) differed, thereby making an “apples to apples” comparison of the results more difficult. It is recommended that future laboratory testing be conducted using tailings solids with an SG that is to be considered acceptable over the LOM.
- The Pressure Filter Vendor testing should be conducted by at least one other Vendor, preferably two, since different Pressure Filter designs and configurations are available other than those offered by Metso. For example: different filter plate sizes that could reduce the overall footprint of the Filter while still meeting the production criteria; and varying the chamber depths that could provide lower cake moistures with varying air blow times.
- The friction loss values for the paste backfill underground distribution system (UDS) flow model were empirically estimated based on a tailings solids specific gravity of 3.05. Actual laboratory scale flow loop testing is recommended whose output would generate friction loss versus velocity curves for 150 mm diameter:
  - Sch 120 CS pipe.
  - Sch 80 CS pipe.
  - Sch 40 CS pipe.
  - HDPE DR9.
- The RMS 2024 unconfined compressive strength (UCS) testing for paste backfill 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 alone. 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 further develop the design of the Shotcrete/CRF Slurry Batch Plant the Shotcrete recipe should be finalised and a testing program conducted to validate the recipe in the next phase of the Project.
- Similarly, since cemented rock fill (CRF) is also used as backfill, and no laboratory testing has been conducted to determine the UCS with associated binder addition rates, testing should be conducted 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.

26.3 Mineral Resource Estimate

It is recommended that the MRE be updated on the basis of new drilling and screen fire assays where possible. Infill drilling to derisk the portion of the mine plan that supports the first year of mining / the payback period of the project has been conducted, further studies to prepare for grade control drilling and geology support to mine production should be considered. Consideration of how to predict where higher and lower grades are present during mining should be made, investigating possible grade control scenarios and optimal drill spacing. This should be investigated by underground drilling to better investigate the close spaced grade and geological variability.

26.4 Mineral Reserve Estimate

DPM should prioritise underground infill drilling within the deposit areas defined in the FS, with the primary goal of converting Probable Reserves to Proven Reserves and upgrading Inferred Mineral Resources to the Indicated category. Simultaneously, an exploration drilling program should be implemented to expand the mineral resource base, thereby enhancing the overall value of the Project.

The success of the Project mining and processing sequence is highly dependent on strict adherence to the construction sequence during the pre-production period. As such, this phase requires particular focus and careful management, especially in ensuring the timely hiring of personnel, procurement and delivery of equipment, installation of surface infrastructure, and acquisition of all necessary permits.

26.5 Mining Methods

The following recommendations are suggested:

26.5.1 Geotechnical

- Some geotechnical drilling and laboratory intact strength tests were completed along the declines after the FS geotechnical assessments were completed. This data should be reviewed to confirm geotechnical design parameters used for the portal and decline.

- In-situ stress testing within the mining horizon using industry standard practices (mini-frac or overcoring methods) to confirm values estimated by laboratory methods. This should be completed development of the initial drifting at the depth of mining.
- Continue to update the fault model and character matrix.
- DPM to review risk and response to potential surface ventilation raise instability at specific depths. Possible solutions include consolidation grouting, bolting, or acceptance of the increase instability risk.
- For boreholes, it is recommended to evaluate a typical standoff distance for adjacent excavations to prevent breakthrough holes in unwanted areas.
- Geotechnically log pilot drillholes for each underground raise and complete a local detailed design for the stability and support requirements for each raise.
- A pilot hole for the Upper Portal to assess bedrock quality with depth for the initial decline is recommended to validate study inputs. Detailed mapping of each portal face is recommended to be completed prior to commencement of the decline to validate geomechanical inputs.
- Hydrogeological conditions and regime are not well understood at the Upper and Lower Portals. Additional studies are recommended to be completed. Study findings should be incorporated into the Portal and Initial Decline designs.

26.5.2 HYDROGEOLOGY

- To update the developed numerical groundwater flow model with FS mine design (due to the cutoff date for data input for the groundwater modelling effort, the Phase 3 PFS mine design was used for the FS numerical groundwater flow model update).
- To update the developed numerical groundwater flow model as new data is made available, and updated prediction of inflows is required.
- To use the numerical groundwater flow model as a decision-making tool if advanced dewatering is proposed, and/or quantification of contact and non-contact water is required.
- The developed numerical groundwater flow model could be further modified to evaluate potential environmental impacts, and to assist with closure studies.
- An adequate network of piezometers and monitoring wells should be maintained, and detailed records of groundwater inflows as the mine begins its operation.

26.5.3 MINING

- All ore sills are recommended to be graded appropriately and be redesigned to suit proper drainage.
- To get a better understanding of the traffic management and congestion, WSP recommends a detailed traffic study for the next phase. This would test potential traffic congestion and validate if passing bays into the spiral ramp would enhance operational efficiency.

- Design safety cuddies into the spiral ramp design to enhance operational safety.
- Update the CRP120-2000-1700-BOD-0001-E-C. Design Basis & Criteria Document for ventilation when the main surface fans supplier is selected.
- Consider implementing dedicated escapeway raises in the Northern internal sections of the lower levels below RL440 to ensure safe egress in case of an emergency.
- The pre-production period is very important for the project, and special attention must be placed on it, in terms of purchasing equipment, recruiting and training personnel, and scheduling surface construction work.
- Review intersections to ensure that clearances are sufficient and create representative section views of areas showing final service arrangement (i.e. allowance for additional shotcrete for initial 80 m of each decline).
- For vertical development is it recommended that additional contractor input be gathered (i.e. review of excavations for equipment setup, shotcrete methodology and rates) to support and validate current excavations the next phase of design.
- Further investigation should be completed to determine the exact location and equipment that will be used to drill the service boreholes from surface to designate the borehole station in the lower declined underground.

26.5.4 INFRASTRUCTURE

- Finalise type of main dewatering pumps. Current design shows progressive cavity pumps, however a multi-stage centrifugal pump may also be considered, if there are limitations on availability of type of pump selected. Key infrastructure changes relating to selection would include, but not limited to:
  - 1 kV motors instead of 690 V motors. The progressive cavity pumps require VSDs that are only available at 690 kV. Switching to multi-stage centrifugal pumps allows for elimination of 1kV-690 V transformers.
  - Addition of flow control valves and minimum recirculation line.
- Electrical infrastructure optimisation on number of transformers to drop voltage from 10 kV to 1 kV, 690 V and 230 V. Current philosophy is to provide transformers for localised voltage supply, based on needs. This allows for minimising power loss across the infrastructure. Further optimisation studies may be conducted to assess reduction in transformers against the added cost of larger cable sizes.

26.5.5 VENTILATION

- The main surface fans have been equipped with additional isolation dampers in order to achieve reverse flow in the event of a fire underground. The intent is to switch from exhaust mode to fresh air intake, per Serbia regulations, without having to change the fan direction, by

using the dampers to redirect flow through the vertical raise. During the equipment evaluation process some of the suppliers are able to provide bi-directional fans that may be more cost effective than installing additional dampers and ducting for achieving the requirements for reverse flow.

- To prevent the formation of dust, WSP recommends regularly spraying water at the tapping points and use a water truck to spray water in the ramps as required.
- 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.
- Continue coordination of ventilation planning between mine owner and contractor(s) to ensure seamless transitions from development to primary ventilation fresh air supply.
- A procedure to inspect, maintain and repair flexible ducting should be put in place to ensure adequate airflow at the working face.
- Auxiliary fans pressure should be measured when the duct is extended to detect stall zones.
- Main fans should be equipped with static pressure measurement to detect if the fan would enter in stall zone.
- Airflow regulators seal requirements when fully closed is essential to maintain adequate airflow in other levels.

26.5.6 MINE BACKFILL

- Pressure filtration testing was conducted by one Vendor (Metso) in this phase of the Project. The testing was done based on the moisture content target determined from preliminary laboratory testing earlier in this phase. The pressure filter Vendor testing was conducted prior to the receipt of the final results of the DTSF deposition testing; the target filter cake moisture content at the DTSF. The results from the DTSF testing indicated that a lower moisture content filter cake was required, compared to that achieved with the Metso Vendor testing. A reduction in the target moisture content will influence the pressure filter size and operating parameters such as the filter cycle time. The change in the target moisture content of the DTSF deposited filter cake will not affect the target moisture content of the filter cake used in the production of paste backfill since its target moisture content is much lower. However it may affect the physical design of the filter plant due to the size or number of pressure filters required and impact the paste backfill plant due to a reduction in its available footprint. Further pressure filtration testing with at least one, preferably two (2) additional Vendor is recommended in the next phase of the Project. Further discussions should be held with Metso to examine the impact of cycle time and pressure filter offering based on the new target filter cake moisture content. Although it is still planned, no further paste backfill laboratory testing was conducted in this phase of the Project. However, it remains a recommendation 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.

- 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.
- 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 Project and better define the hydraulic flow characteristics of the paste in the distribution system.
- 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.
- The CRF modularized system design has been developed to a feasibility level. The designs and concepts are suitable for continued development of a modularized plant design in subsequent phases of the Project.

26.6 Recovery Methods

Process Plant

No additional trade-off studies were performed during the FS. The flowsheet and main equipment remained unchanged from the PFS.

26.7 Project Infrastructure

At the next stage of the Project, the following areas are recommended to be investigated further:

- Additional exploration and testing to complete detailed designs in the site infrastructure areas where no direct exploration was done.
- 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.
- Confirmation of Geotechnical recommendations based on a thorough Geotechnical investigation, to validate the proposed Engineering fills for terraces and roads using the blending of Overburden and granular material.
- Detailed design of structures and foundations based on final geotechnical recommendations and certified equipment information from the vendors.
- Equipment from Ada Tepe to be re-evaluated by the original supplier/vendors for the change in seismic conditions. All equipment loads need to be re-certified by the supplier/vendor.

- A thorough analysis and selection of the Water Treatment Plant and Clarifier need to take place in the next phase, following updates to the Site Water Balance. This will include reviewing options for handling the brine - the concentrated liquid byproduct of the water treatment process.

26.7.1 Filter and Paste Backfill Plants

The following are recommendations to be considered for inclusion in the next phase of the Project and to be investigated further:

- Future refinements in the LOMP will ultimately determining the maximum annual Filter Cake and Paste Backfill production requirements and therefore the overall capacities of both plants. Both the Filter and Paste Plants' design criteria and equipment sizing should be re-examined, and a determination made if revisions are required based on the newest LOMP.
- Laboratory testing associated with the acceptable moisture content of the filtered tailings being deposited in the DTSF conducted late in this phase of the Project, indicated a reduction in the Filter Cake's moisture content below that which was consider would be required. This could affect either the size or the number of Pressure Filters required in the Filter Plant, and consequently the Plant's multidisciplinary design and its footprint. At least one (1) additional Vendor should be selected to conduct Pressure Filtration testing early in the next phase of the Project and propose a Pressure Filter design. To minimise the footprint of the Pressure Filters and the Filter Plant, it is recommended that the individual filter plates be sized as large as practically possible.
- There was no additional Paste Backfill unconfined compressive strength (UCS) testing conducted in this phase of the Project and the same required binder addition rate as in the PFS was applied, at 5 wt%. Testing of Serbian sourced 90/10 binder with tailings of the agreed to specific gravity (SG) should be conducted to establish the suitability of the design of the Binder Silo/Bin Package for overall storage capability, as well as to more closely determine the operating cost of Paste Backfill.
- Upon selection of the major equipment Vendors, their equipment dimensions should be incorporated into the design of the Filter and Paste Plants and based on the equipment, the plants' functionality refined, as required.
- In this phase of the Project, all pipelines were sized according to flow rate, and specifications were selected based on service and expected pumping pressures. Future work in the next phase of the Project should include a reassessment of the specification selections based on Vendor stated pump dead pressures and unexpected operating conditions. Dynamic effects and thermal growth for pipe stress may also need to be considered for higher risk lines.
- Fire suppression systems in the form of a sprinkler system were not considered in this phase of the Project. The regulatory requirements and compliant designs should be incorporated in the next phase of the Project.

- There are no E-rooms in either the Filter or the Paste Plants, rather an E-room to service the plants will be supplied by others and was initially located on the east side of the Filter Plant. To allow the maximization of the footprint available for the Filter and Paste Plants on the crusher terrace, the E-room was relocated to an area between the Filter and Paste Plants such that an adequate amount of space was still available to allow the area's use by binder delivery trucks to have easy turning access for their off-loading into the Paste Plant's binder silos. The configuration of the E-room, its grounding bed, and transformers must be closely examined to maintain a binder truck's ability to safely and easily use the area between the Plants to reverse into its off-loading position at the silos.
- Additional improvements to binder delivery traffic patterns should be considered for the next phase. The crusher terrace is quite congested, and delivery trucks will have limited room for manoeuvring during deliveries to the Paste Plant. Strategically locating bollards to protect buildings from impacts and designated foot traffic paths will need to be planned to maximize the safety of this operation.
- Structural design and material selection should be refined in the next phase of the Project based on the weights of the major mechanical equipment when final Vendor selection is completed.
- Local control panels should be considered for the Paste Plant. Tablet type control stations would allow operators to exercise local control of conveyor belt travel, valve actuation, and Paste Mixer functionality, and assist with plant clean up duties during planned shutdowns.
- Vendor equipment specifications should also specify skid mounted instruments be pre-wired to skid junction boxes.
- Instrumentation, pipe, and valve specification selections should be reviewed when pump dead head pressures become available after Vendor selection.
- Paste pump selection should be reviewed to determine if a 150-bar design pressure pump or a 120-bar pump is preferred for this operation. This final decision on pump selection should be made when hydraulic flow loop pressure loss testing is performed and DPM has full confidence in the friction loss estimates for the paste line. Currently WSP's estimate carries a safety factor based on historical experience and Čoka Rakita tailings rheology testing however a flow loop test is the best way to have full confidence in the equipment selection.
- Detailed engineering of the paste pipeline from the Paste Plant to the upper decline portal will need to include a full stress analysis with dynamic effects modelled. The paste line represents a significant risk to the operation should it fail and DPM will require a robust design for this system.
- This design of the discharge piping configuration from the paste plant has not been developed to consider the second standby paste pipeline to the upper decline portal and the short, if required, supporting trestle before the lines submerge below ground level outside the

paste plant. The design of discharge piping and trestle will benefit from the afore recommended stress analysis.

- Due to the simplicity and small scope for the electrical discipline in the Filter and Paste Plants' design, the recommendation for reliability or improvements cannot be foreseen at this time. However, a power system study should be conducted in a later stage of the Project to make sure electrical equipment was sized accordingly and distribution equipment design can warranty the reliability of the plant.

26.7.2 SHOTCRETE AND CRF SLURRY BATCH PLANT

The following are recommendations to be considered for inclusion in the next phase of the Project and to be investigated further:

- Since the Batch Plant's engineering was not considered in earlier stages of this Project, the design and configuration of modularised plant should be further investigated and refined in a collaborative effort with Vendors who have a proven track record of supplying modularised plants for the purpose of formulating both shotcrete and CRF slurry and not solely accustomed to supplying concrete Batch Plants. This could be accomplished by engaging a Vendor in an initial engineering design contract with potentially a portion of the cost being credited towards the purchase of the plant.
- There has not been any CRF unconfined compressive strength (UCS) testing for the Project to date. Since this is the primary backfilling method in the preproduction years, and a significant component of the LOMP backfilling needs, it is critical that testing be conducted to quantify the binder addition rates, and Batch Plant design criteria early in the next phase of the Project.

26.7.3 DRY TAILINGS STORAGE FACILITY

At the next stage of the Project, the follow areas are recommended to be investigated further:

- Additional Overburden Material testing and evaluation of results for design optimisation.
- Optimization of Ring road slope reinforcement.
- Overdrains refinement through seepage analysis.
- PAG Zone design optimisation through additional seepage analysis.
- Additional assessment of PGA and possible further seismic analysis to input into deformation analysis if necessary.
- Deformation analysis for DTSF Seismic case if required following update to PGA.
- Additional tailings testwork if samples are available.
- Evaluation of operational design with updated filtered tailing parameters from process plant testwork.

- Additional assessment of sizing for temporary lay down area followed by evaluation of off-spec material handling.
- Definition of material stockpiles following completion of staged earthwork balance for whole project site. This includes the requirement to have a roof over the storage area to prevent re-saturation of the tailings.

26.7.4 Surface Water Management

The current assessment has identified key gaps that should be addressed as part of the next phase of the study:

- It is recommended to conduct a more detailed regional analysis of the extreme rainfall probability focusing on the probability distribution growth curve.
- The 1:100 Annual Exceedance Probability (AEP) CW management pond spill frequency requirement will need to be verified and agreed with the permitting authorities, with consideration of the water quality associated with the spills.
- The water management pond containment embankment freeboard allowance will need to be verified with the local regulators.
- It is recommended that the pond sizing is reviewed once the new groundwater inflows have been estimated and when the Clarifier and WWTP shutdown logic is confirmed (as it has a strong impact on the CW Pond sizing).
- It will be necessary to update the pond sizing to reflect the change for the final mine layout.
- Adopt the outcome of the water balance to ultimately support decision making around future optimisations of pond sizes and/or optimisation of pumping rates.
- To refine the sediment management strategy, sediment modelling is recommended in future design stages.
- The Reclaim Pond spillway has been designed to convey the IDF of 1:1,000 AEP peak flow from the DTSF catchment area. The NCW channels have been sized to convey the 1:200 AEP peak flow and would not have a sufficient capacity to convey the IDF defined for the Reclaim Pond. During such an event the NCW channels are expected to overtop into the DTSF CW channels which flow into the Reclaim Pond. As a result, the catchment area for the Reclaim Pond would then include the larger surrounding natural catchment during the 1:1,000 AEP event, leading to a larger pond spillway requirement. It is recommended that in future studies the Reclaim Pond spillway is resized to account for this increase in catchment area.
- The feasibility of adopting a single Integrated Discharge Facility downstream of the Reclaim Pond should be evaluated in the next phase, combining the two (2) Reclaim Pond stilling basins and the DTSF underdrainage outlet.

- To address the limitations around the hydraulic modelling of the stepped channels, it is recommended that future design stages incorporate 3D Computational Fluid Dynamics (CFD) modelling. This will allow for more accurate representation of complex flow behaviours and better inform the final design of stepped structures.

26.7.5 SITE-WIDE WATER BALANCE

Based on the outcomes of the FS Water Balance Model (WBM), the following recommendations are proposed:

- Model calibration: Though the model's climate modules (PrecipGen, WGEN and SNOW-17) are set up to be representative of the available data, a calibration of the WBM itself has not been undertaken. It is recommended that once Site monitoring data (e.g., mine flows and pond levels) are available the model undergoes a calibration process to allow mine operation optimisation and verification during the operational phase.
- Groundwater inflow update: the groundwater inflow rates used within the WBM were an earlier iteration of the groundwater model comparing to what has been presented in (WSP, 2025c). The most recent modelled groundwater inflows show a reduction in average groundwater inflows across the LOM of approximately 14%, and a reduction in maximum annual inflows of approximately 8% for all three groundwater inflows scenarios. Due to the significant role played by groundwater inflows within the Site's water management systems, it is imperative that future modelling work incorporates the most up-to-date groundwater inflows available.
- Production throughput and parameter update: The production throughput parameters used within the WBM were taken from calculations conducted in April 2025 (WSP, 2025e). It is noted that further updates have been finalised since this date (WSP, 2025f). However, due to time constraints related to modelling and reporting, it was not possible to incorporate the new results within the final model. It is recommended that the finalised production throughput data is implemented in future model iterations.
- Process Plant modelling: Within the WBM, the Process Plant is modelled as a 'black box' meaning that internal interactions and recirculation are not modelled in detail. Future model updates should consider the inner workings of the Process Plant.
- Underground modelling: The WBM considers any water going underground or coming from underground to be a global outflow and global inflow, respectively. In that respect, it only focuses on the water balance at surface and does not model underground interactions. Further model iterations should account for water reticulation in the underground mine workings.
- Pond water transfer pumps: In the WBM it has been assumed that the pumps used to drain the ponds are variable speed pumps, allowing improved optimisation of water supply to the

Clarifier. If fixed speed pumps are specified, the assumptions on the WBM will need to be reviewed.

- Water treatment plant supply: In the Base Case model scenario the influent supply to the WWTP varies between 21 l/s and 50 l/s during operations. This variability results from the contribution of water pumped from the Reclaim and CW Ponds. Consideration needs to be given to a WWTP design than can manage such a variability in influent supply. With a simulated annual exceedance probability spill risk of 1:385 and 1:333 respectively for the ponds (as compared to a design AEP of 1:100) there is scope to reduce the pond water transfer pump rates from 15 l/s to give a more consistent input to the water balance. This would need to be undertaken in consultation with the permitting authorities and with consideration of the results from the additional model scenarios simulated, and the impact of a major unscheduled shutdown on the CW Pond spill risk.
- Review NCW Pond sizing: The NCW Pond requires specific attention and optimisation. The pond has a volume of 58,700 m³ however, the water balance modelling indicates that the mine can function adequately without a significant non-contact water supply under the Base Case Scenario. Nevertheless, additional model scenarios demonstrate the pond's value as an emergency water store in case of lower-than-expected ground water inflows to the underground mine. Further analysis is recommended to refine the pond's optimal size.
- Reclaim Pond operating rule optimisation: The Reclaim Pond typically pumps water to the Clarifier. Yet, during a Clarifier shutdown, water from the Reclaim Pond can be diverted to the CW Pond. This redirection is contingent on the volume of water stored in the CW Pond; if it's more than 50% full, the Reclaim Pond cannot transfer any additional water. In such cases, water must be held within the Reclaim Pond until the CW Pond's level drops below 50% or the Clarifier resumes operation. The 50% threshold for the CW Pond was established during the modelling process to minimise the risk of spills and may be optimised in future model iterations
- Review geochemical test work data: It is recommended that the chemical source terms and water quality modelling be revisited during detailed design incorporating the geochemical test work, including kinetic testing. Quality assurance / quality control assessment of sampling, and analysis should be undertaken on the geochemical test work data. A comprehensive analysis should be conducted including a full suite of water quality parameters, as outlined in WSP (2025h) to allow for geochemical modelling, including an ion charge balance check.
- Underground mine dewatering source term update: The flows from the underground dewatering are the biggest flow and therefore their chemistry is an important component of site wide water quality. The currently derived source term concentrations do not incorporate scaling based on the characteristics of the underground mine's exposed workings, and do not incorporate leaching from underground paste backfill or cemented rockfill due to lack of test work data. This needs to be resolved with further testing and analysis.

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.

26.8 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.
- Conduct a Value Engineering workshop before proceeding to detailed engineering, focusing on the highest-cost areas (Underground Mine, Process Plant).

Mining

- Revisitation of the single source bids and quotations as part of tendering the underground mining work, material supply, and equipment procurement has potential to establish lower costs and can lead to establishing better contractor performances that can enable advancement of the scheduled work.
- Revisitation of the decision registry to ensure all ramifications of the decisions as recorded are understood.

26.9 Economic Analysis

The following recommendations have been identified with regards to the Project:

- Conduct a detailed review of the Capex and Opex estimates to develop a detailed assessment of the import duties and VAT payable for the Project, including an assessment of the schedule of required payments.

- Maintain engagement with a tax consultant to ensure that the taxation calculations applied in the analysis are aligned with the latest regulations from the Government of Serbia.

26.10 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.

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 Project.

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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WSP. (2025h). Source Term Derivation Water Quality Modelling (CRP120-5000-1250-RPT-0006-O-B).

WSP. (2025i). Conceptual Closure Design for Čoka Rakita DTSF and Topsoil Stockpiles (CRP120-5000-1260-0019).

WSP. (2025j). Čoka Rakita Feasibility Study DTSF Geotechnical Stability Assessment (CRP120-5000-1260-RPT-0016-O-B).

WSP. (2025k). Čoka Rakita Feasibility Study Ponds Dam Breach Assessment (CRP120-5000-1250-RPT-0003-E-B).

28 ABBREVIATIONS

Abbreviation	Unit or Term
"	Inch
$ M	Million Dollar
$/dmt	Dollar per Dry Metric Ton
$/h	Dollar per Hour
$/kg	Dollar per Kilogram
$/kWh	Dollar per Kilowatt Hour
$/lot	Dollar per Lot
%	Percent
%w/w	Percentage by Weight
€	Euro
<	Less Than
>	Greater Than
±	Plus and Minus
°	Degree
°C	Degree Celsius
µm	Micron
1D	One-Dimensional
2D	Two-Dimensional
3D	Three-Dimensional
4D	Four-Dimensional
A	Ampere
AAS	Atomic Absorption Spectrometry
ABA	Acid-Base Accounting
AC	Air Condition
AEP	Annual Exceedance Probability
Ag	Silver
AG	Autogenous
Ai	Abrasion Index
Al	Aluminum
ALS_BO	ALS Bor
ALS_VA	ALS Vancouver
AP	Acid Potential

Abbreviation	Unit or Term
ARD	Acid Rock Drainage
As	Arsenic
Au	Gold
AUT	Automatic
Ba	Barium
BaseMet Labs	Base Metallurgical Laboratories
BAT	Best Available Techniques
BBWi	Bond Ball Mill Work Index
BD	Bulk Density
BFS	Blast Furnace Slag
BHP	Big Hornblende Porphyry
Bi	Bismuth
BM	Block Model
BOQ	Bill of Quantities
Ca	Calcium
$CaCO_3$/t	Calcium Carbonate per Tonne
CCA	Consequence Classification Assessment
CCLAS	Comlabs Computerised Laboratory Automation
CCTV	Closed-Circuit Television
Ce	Cerium
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CITES	Convention on International Trade in Endangered Species
cm	Centimetre
CM	Contained Metal
CN	Curve Number
CO	Carbon Monoxide
Co	Cobalt
COG	Cut-Off Grade
CPF	Cemented Paste Backfill
Cr	Chromium
CRF	Cemented Rock Fill
CRM	Certified Reference Materials

Abbreviation	Unit or Term
Cs	Caesium
CS	Carbon Steel
csv	Comma-Separated Values
Cu	Copper
$CuSO_4$	Copper (II) Sulphate
CV	Coefficient of Variation
CW	Contact Water
CWP	Construction Work Packages
d/y	Day per Year
DA	Dynamic Anisotropy
DAT	Data Sheets
DBA	Dam Break Analysis
DBE	Distribution Equipment
DC	Direct Current
DC	Design Criteria
DDF	Depth Duration Frequency
DEM	Digital Elevation Model
DFP	Coarse Feldspar Porphyry
DGPS	Differential Global Positioning System
DHSS	Drill Hole Spacing Study
dmtpa	Dry Metric Ton per Annum
DOL	Direct On-Line
DPM	DPM Metals Inc.
DRA	DRA America Inc.
DSO	Deswick Stope Optimiser
DTC	Dewatering Technology Center
DTM	Digital Terrain Model
DTSF	Dry Tailings Storage Facility
EBRD	European Bank for Reconstruction and Development
EDS	Elektro Distribusia Serbia
EGM	Engineering Geology Model
EGRG	Extended Gravity Recoverable Gold

Abbreviation	Unit or Term
EIA	Environmental Impact Assessment
ELOS	Equivalent Linear Overbreak Slough
EMP	Early Mineralised Porphyry
EPCM	Engineering, Procurement and Construction Management
EPRP	Emergency Preparedness and Response Plan
ERM	Enterprise Risk Management
E-room	Electrical Room
ESMP	Environmental and Social Management Plan
ETP	Effluent Treatment Plant
EU	European Union
E-W	East-West
EWS	Engineering Workstation
FA	Fire Assay
FAAS	Flame Atomic Absorption Spectrometry
FAG	Fault Gouge
FB	Fault Block
Fe	Iron
FOS	Factor of Safety
FPT	Full Plant Tailings
FRS	Fibre Reinforced Shotcrete
FS	Feasibility Study
FSE	Fundamental Sampling Error
FW	Footwall
g	Gram
G&A	General and Administrative
g/dmt	Gram per Dry Metric Ton
g/m³	Gram per Cubic Metre
g/oz	Gram per Ounce
g/t	Gram per Tonne
GARD	Global Acid Rock Drainage
GCL	Geosynthetic Clay Liner
GDI	Grano-Dioritic Porphyry

Abbreviation	Unit or Term
GEN_PE	Genalysis Perth
GEV	General Extreme Value
GGIBFS	Ground Granulated Blast Furnace Slag
GISTM	Global Industry Standards on Tailings Management
GPS	Global Positioning System
H&S	Health and Safety
H_2O_2	Hydrogen Peroxide
H_2SO_4/t	Sulphuric Acid per Tonne
HAZID	Hazard Identification
HAZOP	Hazard and Operability
HDPE	High Density Polyethylene
HEC	Hydraulic Engineering Circular
HMI	Human-Machine Interface
HQ3	HQ Triple Tube
HR	Hydraulic Radius
HSEP	Health and Safety Execution Plan
HSU	Hydrostratigraphic Unit
HW	Hanging Wall
I&C	Instrumentation and Control
ICP	Inductively Couple Plasma
ICP-AES	Inductively Coupled Plasma-Atomic Emission Spectrometry
ICP-MS	Inductively Coupled Plasma-Mass Spectrometry
IDF	Intensity Duration Frequency
IDF	Inflow Design Flood
IDP	Integrated Discharge Point
IDW2	Inverse Distance Weighting Squared
IO	Input/Output
IP	Induced Polarisation
IP	Ingress Protection
J&J	Jenike & Johanson
JLS	Jurassic Limestone

Abbreviation	Unit or Term
K	Potassium
kg	Kilogram
kg $CaCO_3$/t	Kilogram of $CaCO_3$ per Tonne
kg/kWh	Kilogram per Kilowatt-Hour
kg/m	Kilogram per Metre
kg/m²/h	Kilogram per Square Metre per Hour
kg/t	Kilogram per Tonne
KLS	Lower Cretaceous
km	Kilometre
km²	Square Kilometre
kN	Kilonewton
kN/m	Kilonewton per Metre
KNA	Kriging Neighbourhood Analysis
koz	Thousand Troy Ounces
kPa	Kilopascal
ktpa	Kilotonne per Annum
ktpm	Kilotonne per Month
kV	Kilovolt
kVA	Kilovolt Ampere
kVAR	Kilovolt-Ampere Reactive
kW	Kilowatt
kW/m³	Kilowatt per Cubic Metre
kWh/t	Kilowatt Hour per Tonne
L	Litre
L/s	Litre per Second
La	Lanthanum
LAN	Local Area Network
LCS	Local Control System
LCT	Locked Cycle Tests
LHD	Load Haul Dump
Li	Lithium
LithClass	Lithogeochemical Classification

Abbreviation	Unit or Term
LOM	Life of Mine
LOMP	Life of Mine Plan
LV	Low Voltage
m	Metre
m/h	Metre per Hour
m/m	Metre per Metre
m^3	Cubic Metre
m^3/d	Cubic Metre per Day
m^3/h	Cubic Metre per Hour
m^3/s	Cubic Metre per Second
$m^3/s/kW$	Cubic Metre per Second per Kilowatt
$m^3/t/h$	Cubic Metre per Tonne per Hour
m^3/y	Cubic Metre per Year
mA	Milliampere
Ma	Million Years
MAAT	Mean Annual Air Temperature
MAET	Mean Annual Evapotranspiration
MAFWM	Ministry of Agriculture, Forestry and Water Management
MAN	Manual
MAP	Mean Annual Precipitation
masl	Metre above Sea Level
MB	Mass Balance
MC	Master Composite
MCC	Motor Control Centre
MCTI	Ministry of Construction, Transport and Infrastructure
MDD	Maximum Dry Density
MEL	Mechanical Equipment List
MEND	Mine Environment Neutral Drainage
MEP	Ministry of Environmental Protection
Mg	Magnesium
Mg/m^3	Million Gram per Cubic Metre
MI	Ministry of Interior
MIBC	Methyl Isobutyl Carbinol

Abbreviation	Unit or Term
min	Minute
ML	Metal Leaching
mm	Millimetre
Mm³	Million Cubic Metre
MME	Ministry of Mining and Energy
Mn	Manganese
Mo	Molybdenum
Moz	Million Ounce
MPa	Megapascal
MRE	Mineral Resource Estimate
MSE	Mechanically Stabilized Earth
MSO	Minable Shape Optimizer
Mt	Million Tonne
MTO	Material Take-off
MV	Million Volt
MW	Million Watt
Na	Sodium
Nb	Niobium
NCW	Non-Contact Water
NGO	Non-Governmental Organisation
Ni	Nickel
NI 43-101	National Instrument 43-101
NNP	Net Neutralising Potential
NO_2	Nitrogen Dioxide
NP	Neutralising Potential
N-S	North-South
NSR	Net Smelter Return
OEM	Original Equipment Manufacturer
OF	Overflow
OHS	Occupational Health and Safety
OIS	Operator Interface Station
OMS	Operations, Maintenance, and Surveillance

Abbreviation	Unit or Term
OR	Operational Readiness
ORP	Operational Readiness Plan
OVB	Overburden
oz	Ounce
P	Phosphorus
P&ID	Piping and Instrumentation Diagram
P_{80}	Passing 80%
P_{85}	Passing 85%
P_{90}	Passing 90%
P_{95}	Passing 95%
Pa	Pascal
PaCMAP	Pairwise Controlled Manifold Approximation Projection
PAG	Potentially Acid Generating
PAR	Population at Risk
PAX	Potassium Amyl Xanthate
Pb	Lead
PCS	Process Control Station
PCS	Process Control System
PD	Positive Displacement
PDC	Process Design Criteria
PEA	Preliminary Economic Assessment
PEP	Project Execution Plan
PFD	Process Flow Diagram
PFS	Pre-Feasibility Study
PHA	Process Hazards Analysis
PLC	Programmable Logic Controller
PLL	Potential Loss of Life
PM	Particle Matter
PMP	Probable Maximum Precipitation
PoC	Push-to-Talk over Cellular
PoF	Probability of Failure
POS	Preliminary Ore Stockpile
ppm	Parts per Million

Abbreviation	Unit or Term
PPS	Pricing and Payment Schedules
PRV	Pressure Reducing Valve
psi	Pound per Square Inch
PSV	Pressure Safety Valve
PVC	Polyvinyl Chloride
PXP	Pyroxene Porphyry
Py	Pyrite
QA/QC	Quality Assurance and Quality Control
Qty	Quantity
RAR	Return Air Raise
Rb	Rubidium
RC	Reverse Circulation
RMU	Ring Main Unit
ROM	Run-of-Mine
RoR	Rate of Rise
RPEEE	Reasonable Prospects for Eventual Economic Extraction
RQD	Rock Quality Designation
RTD	Resistance Temperature Detector
RVDS	Rotary Vertical Drilling System
S	Sulphur
Sb	Antinomy
SCADA	Supervisory Control and Data Acquisition
SCS	Soil Conservation Service
SEA	Strategic Environmental Assessment
sec	Second
SEM-EDS	Scanning Electron Microscopy with Energy Dispersive X-ray Spectroscopy
SFA	Screen Fire Assay
SFD	Sequence Felsic Debris
SFE	Shake Flask Extraction
SG	Specific Gravity
SGS_BO	SGS Bor

Abbreviation	Unit or Term
SGS_BUR	SGS Burgas
SGS_CH	SGS Chelopech
SIA	Social Impact Assessment
SM	Screened Metallics
SO_2	Sulphur Dioxide
SP	Spatial Plan
SPSPA	Spatial Plan for Special Purpose Area
SQL	Simple Query Language
Sr	Strontium
t/d	Tonne per Day
t/m^2	Tonne per Square Metre
$t/m^2/h$	Tonne per Square Metre per Hour
t/m^3	Tonne per Cubic Metre
T_{80}	Transfer Size
Ta	Tantalum
Tb	Terbium
Th	Thorium
Ti	Titanium
TMC	Timok Magmatic Complex
TMI	Total Magnetic Intensity
ToS	Trade-off Study
tpa	Tonne per Annum
tph	Tonne per Hour
tph/m^2	Tonne per Hour per Square Metre
TSS	Total Suspended Solids
U	Uranium
UCS	Unconfined Compressive Strength
UDS	Underground Distribution System
UF	Underflow
UG	Underground
UofB	University of Belgrade
UPS	Uninterruptible Power Supply

Abbreviation	Unit or Term
US$	United States of America Dollar
US$/dmt	United States of America Dollar per Dry Metric Ton
US$/oz	United States of America Dollars per Ounce
US$/t	United States of America Dollars per Tonne
UTM	Universal Transverse Mercator
V	Vanadium
V	Volt
VDC	Volt Direct Current
VHG	Very High Grade
VOC	Volatile Organic Compounds
VOD	Ventilation-on-Demand
VSD	Variable Speed Drive
VTEM	Versatile Time Domain Electromagnetic
VWP	Vibrating Wire Piezometer
WB, WBM	Water Balance, Water Balance Model
WGS	World Geodetic System
WQM	Water Quality Model
wt%	Weight Percentage
WTP	Water Treatment Plant
WWM	Welded Wire Mesh
WWTP	Waste Water Treatment Plant
XMO	Monomictic Breccia
XRD	X-Ray Diffraction
Y	Yttrium
Zn	Zinc
Zr	Zirconium

29 CERTIFICATE OF QUALIFIED PERSON

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 –Feasibility Report – Čoka Rakita Project, Eastern Serbia*" dated January 09, 2026 with an effective date of November 26, 2025 (the "Technical Report"), prepared for DPM 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 the preparation of Sections 2, 3, 18.1 to 18.6 and 18.12 to 18.14, 21 (except 21.1.7 and 21.2.3), and 24; and portions of Sections 1, and 25 to 27 of the Technical Report.
8. I visited the properties (Ada Tepe and Čoka Rakita) that are the subject of the Technical Report on May 13 to 17, 2024; and December 4 to 5, 2024.

9. I have had prior involvement with the property that is the subject of the Technical Report:
    - Served as QP for "*Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia*" prepared by DRA Americas Inc., with an effective date of December 18, 2024 (issued Report Date January 30, 2025, 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 9th day of January 2026, Toronto, Ontario.

"Original Signed on file"

Ian Major, P. Eng.
Senior Project Engineer
DRA Americas Inc.

2nd Floor | Exchequer Court | 33 St Mary Axe | London EC3A 8AA

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*Technical Report – Feasibility Report – Čoka Rakita Project, Eastern Serbia*" dated January 9, 2026 with an effective date of November 26, 2025 (the "Technical Report"), prepared for DPM Metals Inc. (the "Company").

I, *Malcolm Titley, MAIG.* of *Horsham, United Kingdom*, do hereby certify:

1. I am an Associate Principal Consultant for Environmental Resources Management Limited (trading as CSA Global) located at Springfield House, Suite 2 First Floor, Horsham, West Sussex, RH12 2RG, United Kingdom.
2. I graduated from the University of Cape Town with a Bachelor of Science in geology and chemistry in 1979.
3. I am registered member of the Australian Institute of Geoscientists (AIG Membership # 2546).
4. My relevant experience includes over 44 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:
    - Over 30 years in mineral resource estimation including extensive gold and copper mine operational and mineral resource estimation experience.
    - Over 20 years of mine production operational and management experience.
    - Over 20 years as a consultant to the mining industry.
    - Participant and author for several NI 43-101 (as defined below) technical reports since 2007.
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 27 of the Technical Report.
8. I visited the Property that is the subject of the Technical Report on May 13 to 15, 2025.
9. I have not had prior involvement with the property that is the subject of the Technical Report.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with NI 43-101.
11. As at the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.

Dated this 9th day of January 2026, Horsham, UK.

"Original Signed on file"

Malcolm Titley, MAIG
Associate Principal Consultant
Environmental Resources Management Ltd.

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 –Feasibility Report – Čoka Rakita Project, Eastern Serbia*" dated January 9, 2026 with an effective date of November 26, 2025 (the "Technical Report"), prepared for DPM 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 33 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 33 years of experience, 6 years in research and development, 10 years in process plant operations and the remainder in process plant flowsheet design and engineering;
   - Polymetallic flotation testwork interpretation and flowsheet design for several studies and projects;
   - Participant and author for various NI 43-101 Technical Reports.
5. I have read the definition of "qualified person" set out in the NI 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfill the requirements to be a qualified person for the purposes of NI 43 101.
6. I am independent of the Company applying all the tests in Section 1.5 of NI 43-101.
7. I am responsible for the preparation of Sections 13 and 17, and portions of Sections 1 and 25 to 27 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 – "Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia"* prepared by DRA Americas Inc., with an effective date of December 18, 2024 (issued Report Date January 30, 2025, prepared for Dundee Precious Metals Inc.);
    - *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 Dundee Precious Metals Inc.);
    - *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 9th day of January 2026, Toronto, Ontario.

"Original Signed on file"

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 –Feasibility Report – Čoka Rakita Project, Eastern Serbia dated January 9, 2026 with an effective date of November 26, 2025 (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 "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 complete a personal inspection of the property (June 11 to 13, 2024) 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, DPM Metals Inc., as described in section 1.5 of NI 43-101.

(g) I have had prior involvement with the property that is the subject of the Technical Report: Served as QP for "Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia" prepared by DRA Americas Inc., with an effective date of December 18, 2024 (issued Report Date January 30, 2025, prepared for Dundee Precious Metals Inc.).

(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 9, 2026.

Signed by Khalid Mounhir

Khalid Mounhir; P.Eng.

CERTIFICATE OF QUALIFIED PERSON DARLENE NELSON

I, Darlene Nelson, state that:

(a) I am a Senior Geotechnical Engineer at:
WSP Canada Inc.
84 King St W
Brockville, Ontario, K6V 3P9

(b) This certificate applies to the technical report titled Technical Report –Feasibility Report – Čoka Rakita Project, Eastern Serbia dated January 9, 2026 with an effective date of November 26, 2025 (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 with a B.ASc. in Geological Engineering from 1997, I am a member in good standing of the Professional Engineers of Ontario (#90460965). My relevant experience after graduation, for the purpose of the Technical Report, includes over 22 years of Rock Mechanic experience in both operations and consulting in both underground and open pit mines and projects throughout North America, Europe, South America and Africa.

(d) I did complete a personal inspection of the property (December 5, 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, DPM Metals Inc., 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 Brockville, Ontario this January 9, 2026.

Original Signed and Sealed on file

Darlene Nelson, 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 –Feasibility Report – Čoka Rakita Project, Eastern Serbia dated January 9, 2026, with an effective date of November 26, 2025 (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 Item 16.3 and portions of Items 1, and 25 to 27 of the Technical Report.

(f) I am independent of the issuer, DPM Metals Inc., as described in section 1.5 of NI 43-101.

(g) I have had previous involvement with the property that is the subject of the Technical Report: Served as QP for "Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia" prepared by DRA Americas Inc., with an effective date of December 18, 2024 (issued Report Date January 30, 2025, prepared for Dundee Precious Metals Inc.).

(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 9, 2026.

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 –Feasibility Report – Čoka Rakita Project, Eastern Serbia dated January 9, 2026 with an effective date of November 26, 2025 (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 Items 16.12, 16.13, 16.15, 18.7 and 18.8 and portions of Items 1, 25, 26, 27 of the Technical Report.

(f) I am independent of the issuer, DPM Metals Inc., as described in section 1.5 of NI 43-101.

(g) I have had prior involvement with the property that is the subject of the Technical Report: Served as QP for "Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia" prepared by DRA Americas Inc., with an effective date of December 18, 2024 (issued Report Date January 30, 2025, prepared for Dundee Precious Metals Inc.).

(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 contains 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 9, 2026.

Original Signed and Sealed on file

Isaac Ahmed; P.Eng.

CERTIFICATE OF QUALIFIED PERSON PETER CORRIGAN

I, Peter Corrigan, state that:

(a) I am a Technical Director for Mine Waste at:
WSP Ireland Consulting Ltd.
Town Centre House
Dublin Road,
Naas, Co. Kildare
Ireland

(b) This certificate applies to the technical report titled Technical Report –Feasibility Report – Čoka Rakita Project, Eastern Serbia dated January 9, 2026 with an effective date of November 26, 2025 (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 Trinity College Dublin with a Bachelor Degree in Civil, Structural and Environmental Engineering. I am a Chartered Engineer member in good standing of Engineers Ireland (membership No. 040106) and a Professional Member (MIMMM) of the Institute of Materials, Minerals & Mining (IOM# number: 705809). My relevant experience after graduation, for the purpose of the Technical Report, includes over 26 years of experience in the design, construction and auditing of tailings storage facilities in numerous countries across Europe and in some countries in Africa. I have led numerous design teams for the design of tailings storage facilities and associated water management systems.

(d) I did complete a personal inspection of the property (from 17 to 19 June 2024 and from March 3 to 5, 2025) described in the Technical Report.

(e) I am responsible for Items 18.9 and 18.10 and portions of Items 1 and 25 to 27 of the Technical Report.

(f) I am independent of the issuer, DPM Metals Inc., 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 Wicklow, Ireland this January 9, 2026.

Signed by Peter Corrigan

Peter Corrigan; BA, BAI, C.Eng, MIEI, MIMMM

CERTIFICATE OF QUALIFIED PERSON RYAN SWEETMAN

I, Ryan Sweetman, state that:

(a) I am an Associate Director at:
WSP UK Ltd
3 Wellington Place
Leeds
LS1 4AP

(b) This certificate applies to the technical report titled Technical Report –Feasibility Report – Čoka Rakita Project, Eastern Serbia dated January 9, 2026 with an effective date of November 26, 2025 (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 Cardiff University with a BEng in Civil Engineering, and a postgraduate from Newcastle Upon Tyne University with a MSc in Sustainable Management of the Water Environment. I am a chartered member in good standing of the Institution of Civil Engineers (ICE number: 51169537) and a Professional Member (MIMMM) of Institute of Materials, Minerals & Mining (OIM3 number: 705766). My relevant experience after graduation, for the purpose of the Technical Report, includes over 24 years' experience in the design and planning of mine water management, flood risk and river engineering projects both in the United Kingdom and internationally.

(d) I did complete a personal inspection of the property (March 3, 2025) described in the Technical Report.

(e) I am responsible for Item 18.11 and portions of Items 1, 25 to 27 of the Technical Report.

(f) I am independent of the issuer, DPM Metals Inc., 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 Knaresborough, United Kingdom this January 9, 2026.

Original Signed and Sealed on file

Ryan Sweetman, Institution of Civil Engineers, CEng, MIMMM
IOM3 number: 705766

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 –Feasibility Report – Čoka Rakita Project, Eastern Serbia*" dated January 9, 2026 with an effective date of November 26, 2025 (the "Technical Report"), prepared for DPM 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 29 years continuously since my graduation.
5. I have worked on similar projects to the Čoka Rakita Project in North and South America; 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 (as defined below) 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 have participated in the preparation of this Technical Report and am responsible for Sections 19 and 22, and portions of Sections 1, 25 to 27 of the Technical Report.
9. I did not visit the property that is the subject of the Technical Report.

10. I have had prior involvement with the property that is the subject of the Technical Report:
    - Served as QP for "*Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia*" prepared by DRA Americas Inc., with an effective date of December 18, 2024 (issued Report Date January 30, 2025, prepared for Dundee Precious Metals Inc.).
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 9th day of January 2026, Montreal, Quebec

"Original Signed on file"
Daniel M. Gagnon, P. Eng.
Senior Vice President East Canada and Mining
DRA Americas Inc.

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 –Feasibility Report – Čoka Rakita Project, Eastern Serbia dated January 9, 2026 with an effective date of November 26, 2025 (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 visit the property that is the subject of 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, DPM Metals Inc., as described in section 1.5 of NI 43-101.

(g) I have had prior involvement with the property that is the subject of the Technical Report: Served as QP for "Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia" prepared by DRA Americas Inc., with an effective date of December 18, 2024 (issued Report Date January 30, 2025, prepared for Dundee Precious Metals Inc.).

(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(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Sudbury, ON, Canada this 9th day of January 2026.

Signed by William Richard McBride

William Richard McBride, P. Eng.

2nd Floor | Exchequer Court | 33 St Mary Axe | London EC3A 8AA

CERTIFICATE OF QUALIFIED PERSON

To accompany the Report entitled "*Technical Report –Feasibility Report – Čoka Rakita Project, Eastern Serbia*" dated January 9, 2026 with an effective date of November 26, 2025 (the "Technical Report"), prepared for DPM 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 Resource 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 (as defined below) 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 "*Technical Report – Pre-Feasibility Report – Čoka Rakita Project, Eastern Serbia*" prepared by DRA Americas Inc., with an effective date of December 18, 2024 (issued Report Date January 30, 2025, prepared for Dundee Precious Metals Inc.).

    - 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 9th day of January 2026, London, UK.

"Original Signed on file"

Kevin Leahy, B.Sc. (Hons), PhD, C.Geol., SiLC
Associate Partner
Environmental Resource Management Ltd