NI 43-101 TECHNICAL REPORT ON A PRELIMINARY ECONOMIC ASSESSMENT OF THE RAJAPALOT GOLD-COBALT PROJECT, FINLAND

Effective Date: 15 October 2022

Prepared For

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1 EXECUTIVE SUMMARY

1.1 Introduction

This Technical Report has been prepared for Mawson Gold Limited (Mawson) by SRK Consulting (Finland) Oy (SRK) with contributions from AFRY Finland Oy (AFRY), Resources Engineering & Management Pty Ltd (RE&M), Paterson & Cooke Nordic AB (P&C), Gosselin Mining and Sweco Oy (Sweco) to disclose the results of a maiden Preliminary Economic Assessment (PEA), completed in October 2022, in accordance with (NI) 43-101 on the Rajapalot Gold-Cobalt project in northern Finland.

This Technical Report is a PEA on the Rajapalot gold cobalt project which has been prepared by a team of SRK consultants for, and on behalf of Mawson, a publicly-listed company in Canada. This report also incorporates a restatement of the Inferred Mineral Resource estimate for the Rajapalot project as at 26 August 2021.

AFRY was commissioned by Mawson to report on the results of a Mineral Resource estimate (MRE) on the Rajapalot Property in Lapland, Finland where gold and cobalt are the primary elements of economic interest. Environmental Impact Assessment (EIA) and land use planning processes are supported by Vahanen (acquired by AFRY during 2021/22) and Sweco, respectively.

SRK was commissioned by Mawson to prepare the mine planning including mine geotechnical, waste, tailings, backfill (supported by P&C) and water management. SRK also prepared the overall economic assessment.

Mawson, AFRY, RE&M, Gosselin Mining and Sweco have cooperated in the drafting of this document to ensure factual content and conformity with the preparation of the Technical Report and the requirements of reporting under the National Instrument NI 43-101.

The effective date of this Technical Report is 15 October 2022 with reliance on the Mineral Resource estimate reported in accordance with the NI 43-101 guidelines and the 2014 Canadian Institute of Mining and Metallurgy definition standards for reporting Mineral Resources and Mineral Reserves (the "2014 CIM Definition Standards") with an effective date of 26 August 2021.

Unless otherwise stated, information, data, and illustrations contained in this Technical Report or used in its preparation have been prepared by the Qualified Persons (QP) for the purpose of this Technical Report. The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized.

1.2 Reliance on Other Experts

The qualified persons have relied on information and opinions forming the basis for parts of this technical report in the following areas:

- Online data on permits from Finnish Government authorities (TUKES). These data are current as at 16 November 2022 and have been reviewed by the AFRY QP.
- Detailed technical geological work up to August 2021 of Mawson's Finnish geological team, supervised by their Chief Geologist, Dr Nick Cook (FAusIMM). These data have been independently verified by the AFRY QP during field visits in 2021.
- Reports of the GTK, BATCircle 1.0 and SGS were reviewed by the Mineral Processing QP.

1.3 Property

The property is located in the northern Finland region known as Lapland, close to the Arctic Circle (25.0°E and 66.6°N) as shown in Figure 1-1. In local Finnish grid coordinates (KKJ(3), EPSG:2393), the Rajapalot project is centred on 3408700mE and 7373200mN.

Currently, the Rompas-Rajapalot property consists of 5 granted exploration permits for 7,210 hectares and 9 exploration permit applications for a combined total of 18,685 hectares and is held 100% in the name of Mawson Oy, Mawson's 100% owned Finnish subsidiary. Exploration permits are granted for up to 15 years with standard two or three yearly renewals. The Rajapalot resource reported here occurs within two granted tenements (Kairamaat 2/3 and Hirvimaa).

Certain areas of the Rompas-Rajapalot areas (namely claim areas Kairamaat 2/3, Uusi Rumavuoma and Rompas) are defined as European Union (EU) Natura 2000 designated areas.

Figure 1-1: Location of the Rajapalot project area

1.4 Setting and Local Resources

The Property is located approximately 35 km west-southwest of the city of Rovaniemi in southern Lapland, Finland. Access to site is by standard vehicle on tar sealed roads and well-maintained gravel roads.

The topography is gently rolling to almost flat, heavily glaciated and inundated with numerous post-glacial lakes, till, eskers, lacustrine and fluvial deposits with a mean elevation of approximately 170 metres. Rajapalot has a typical Laplandic subarctic climate with cold, snowy winters and mild summers. Its closeness of the sea Bothnian Bay leads to milder winters, at least compared to rest of the Lapland. There is no permafrost in the region surrounding the project.

The project is well serviced by local infrastructure. The city of Rovaniemi is a regional logistics hub, hosts an international airport and a population of over 65,000 people. Finland has an established industrial economy and mature mining sector with over 40 operating mines.

1.5 History

On 30 April 2010, Mawson entered into an agreement with AREVA Finland (AREVA) whereby the Company acquired 100% of AREVA's mineral properties and exploration database in exchange for EUR 1 M and 10% equity in Mawson. Mawson continues to own 100% of the property. There are no underlying royalties (except a statutory Finnish mining royalty of 0.15% of the value of the exploited mineral / metal payable to the landowner), back-in rights or other underlying agreements or encumbrances over the property.

The entire property had seen minimal surface exploration prior to Mawson's ownership and Rajapalot was a grass roots discovery by Mawson geologists in 2012. A small outcrop a few metres across is the only surface exposure of any of the resources discussed within this

Technical Report, which were predominately drilled out between 2018 and 2021.

1.6 Geological and Mineralization

The host sequence comprises a polydeformed, isoclinally folded package of amphibolite facies metamorphosed Paleoproterozoic supracrustal rocks of the Peräpohja belt. The Paleoproterozoic of northern Finland are highly prospective for gold and cobalt, and include the Europe’s largest gold mine, Kittilä, operated by Agnico Eagle Finland Oy, and the recent Ikkari discovery by Rupert Resources.

At the project scale, Mawson recognizes two host rock packages; firstly, a siliciclastic, dolomitic carbonate and albite-altered metasedimentary sequence interpreted as forming in a platformal to continental margin setting (Kivalo Group) ; and he secondly, metasedimentary sequence comprising pelitic turbidites, arkosic sands, carbonates, impure and pure quartzitic sandstones and sulphidic bituminous rocks corresponds to the Paakkola Group. Mafic volcanics and intrusives and post-tectonic granitoids are locally abundant.

Stratabound gold-cobalt mineralization occurs near the boundary of the Kivalo and Paakkola groups with two contrasting host rocks, either iron-magnesium or potassic-iron types. Multi-stage development of the mineralization is evident, with early-formed cobalt and a post-tectonic hydrothermal gold event.

1.7 Exploration

Drill core recoveries are excellent, averaging over 99.9 % across the Rajapalot Resource estimate. All core is photographed with sampling details evident prior to cutting at the GTK core facility in Rovaniemi. Core orientation occurs on all core of NQ size and above (PAL series of drillholes; 96% of diamond drilling). Core orientation lines are marked on base of hole and the orientation line is kept in the core tray for verification purposes which also ensures the same half of the drill core is always used for assay. Fabric determinations are conducted using standard alpha/beta measurements or an oriented core holder.

1.8 Sampling, Analysis and Verification

The bulk of gold assays are conducted using the certified PAL1000 technique through CRS Laboratories in Kempele, Finland. Coarse crush samples, generally of 1 kg, are loaded into steel pots with abrasive media in the presence of cyanide. They are rolled for a standard period and then the gold in solution is determined by flame AAS. Lowest detection limits of this method with 1 kg of sample is 0.01 g/t Au. Fire assay methods using standard procedures to lower detection limits have been used as required.

Inter-laboratory testing of the PAL1000 technique using fire assaying at ALS laboratories has validated the technique.

On-site verification and on-line inspection of the assay data by the QP has found no internal or external laboratory issues of concern and finds that the methods employed by Mawson make the assay database suitable for estimation and reporting of the Mineral Resource estimates.

1.9 Metallurgical Summary

Testwork conducted over multiple stages has shown that the Rajapalot deposit is amenable to conventional processing techniques to liberate gold and cobalt into saleable products.

Geometallurgical analysis had identified two distinct feed types designated “Raja-South Palokas” (R-SP) and Palokas (Pal). Separate composite samples of each type were selected, prepared, and tested separately. Although roughly aligned with designated mining domains, this is not exclusively so, as the metallurgical-type classification is based on defined mineralogical characteristics and ratio of sulphur to cobalt in the feed.

Mineralogical examination of the feed type samples showed that the materials were distinguished by differences in silicate mineral occurrence. The R-SP sample had significant cobalt as Cobaltite, whereas the Pal sample had very minor cobaltite. Both types had cobalt intimately associated with iron-sulphide minerals – pyrrhotite and pyrite. For the Pal sample, the latter was the dominant occurrence of cobalt.

Comminution results classified samples as “slightly abrasive”, and “medium” with respect to coarse and fine grindability.

Gravity recoverable gold was identified for both types, with typical recoveries ranging from 10% to 18%. Gravity recovery of gold was shown to be beneficial in consistently achieving high gold leach recovery. Combined gravity plus leach gold recoveries of the order of 95% was demonstrated for both feed types, at a P80 grind of 75 μm and leach residence time of ~30 hours. Reagent consumptions were in the normal range of expectations.

Flotation testing of both mineral types demonstrated that high recoveries of cobalt can be achieved with appropriate pulp chemistry and collector additions. For the R-SP sample a simple rougher-cleaner flotation (non-optimised) achieved 88% cobalt recovery to a concentrate grading >0.6% Co. Actual (and modelled) concentrate grades will be a function of the relative grades of cobalt and sulphur in the proportion of the feed which will be directed to the flotation circuit. Anticipated concentrate sulphur grade is 35%, resulting in a concentrate cobalt grade ranging between 0.6% to 1.0%.

For the Pal sample, the equivalent was 78% cobalt recovery to 0.3% Co grade, reflecting the different cobaltite occurrence for the two samples. Where cobalt head grades are low, the response indicates that this mineralogical type will not meet current criteria to justify flotation processing after gold recovery. Further investigation is warranted.

1.10 Mineral Resource Estimate

Mineral Resource estimates under CIM Definition Standards 2014 are presented for the underground-only base case scenario discussed in Section 14.7 which represents the most reasonable prospect for eventual economic extraction (RPEEE).

The previous inferred mineral resource estimate outlined in the technical report titled “Mineral Resource Estimate NI 43-101 Technical Report – Rajapalot Property” dated 26 August 2021 (“Previous MRE”), is the basis for the updated mineral resource statement and all estimations remains the same, only calculated gold equivalent has been updated. Updated gold equivalent ($AuEq^2$=Au x 95% + Co x 87.6% / 911) is based on updated assumed commodity prices of Co USD27.22/lb and Au USD1,700/oz and includes recovery factors for Au (95%) and Co (87.6%). The updated $AuEq^2$ results in a total underground inferred resource of 958 $AuEq^2$ koz.

The Inferred MRE for the Rajapalot Project, with an effective date of 26 August 2021, is summarized in Table 14-9 based on the underground-only option.

Table 1-1: Rajapalot Inferred Mineral Resources Effective 26 August 2021

Zone	Cut-off (AuEq¹)	Tonnes (kt)	Au (g/t)	Co (ppm)	AuEq² (g/t)	Au (koz)	Co (tonnes)	AuEq² (koz)
Palokas	1.1	5,612	2.8	475	3.1	501	2,664	562
Raja	1.1	2,702	3.1	385	3.3	271	1,040	288
East Joki	1.1	299	4.5	363	4.6	43	109	44
Hut	1.1	831	1.3	428	1.6	36	355	44
Rumajärvi	1.1	336	1.4	424	1.7	15	142	19
Total Inferred Resources		**9,780**	**2.8**	**441**	**3.0**	**867**	**4,311**	**958**

- The independent geologist and Qualified Person as defined in NI 43-101 for the mineral resource estimates is Mr. Ove Klavér (EurGeol). The effective date of the MRE remains unchanged to the Previous MRE (August 26, 2021, available on SEDAR), and will be restated in the PEA technical report when it is filed.
- The mineral estimate is reported for a potential underground only scenario. Inferred resources were reported at a cut-off grade of 1.1 g/t (AuEq[1] Au g/t + Co ppm /1005) with a depth of 20 meters below the base of solid rock regarded as the near-surface limit of potential mining. Refer to the Previous MRE for details on the cut-off grade calculation used in calculating the Inferred Mineral Resource.
- Resource gold equivalent grades (AuEq[2]) and ounces stated here are based on the updated PEA metal prices of USD1,700/oz Au and USD60,000/t Co and recovery assumptions of 95% Au and 87.6% Co. (AuEq2 = Au g/t x 95% + Co ppm x 87.6% / 911).
- Wireframe models were generated using gold and cobalt shells separately. Forty-eight separate gold and cobalt wireframes were constructed in Leapfrog Geo and grade distributions independently estimated using Ordinary Kriging in Leapfrog Edge. A gold top cut of 50 g/t Au was used for the gold domains. A cobalt top cut was not applied.
- A parent block size of 12 m x 12 m x 4 m (>20% of the drillhole spacing) was determined as suitable. Sub-blocking down to 4 m x 4 m x 0.5 m was used for geologic control on volumes, thinner and moderately dipping wireframes
- Rounding of grades and tonnes may introduce apparent errors in averages and contained metals.
- Drilling results to 20 June 2021.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

1.11 Mineral Reserve Estimates

There are no Mineral Reserve estimates for the Rajapalot property.

1.12 Mining Methods

The Rajapalot gold and cobalt project comprises five orebodies (Palokas, Raja, Joki, The Hut and Rumaj) within an area of approximately 3 km from west to east and 2 km from south to north, which commence from outcrops to 100 m below the surface, to a maximum depth of around 600 m.

The PEA mine plan considers a greenfield underground operation targeting a run-of-mine (RoM) production rate of 1.2 Mtpa through combined mining of three deposits at any one time to meet the target annual production. Each of the near surface deposits are planned to be individually accessed through decline box cuts with truck haulage to the RoM stockpile located at the process facility. RoM material is assessed against an economic cut-off for cobalt extraction, to be separately stockpiled, and campaign processed. All feed will be processed for gold recovery but only a proportion, on a feed campaign basis, for cobalt recovery.

The primary mining method selected for the Project is retreat longhole open stoping (LHOS) with 20 m level spacing and applied to the Palokas, Raja, The Hut and Rumaj deposits. Paste backfill is used to maximize mining extraction and reduce the tailings storage requirements on surface. The mining method selected for the Joki deposit is overhand Cut and Fill (C&F) due to its shallower dip angle with Cemented Rock Fill (CRF).

Figure 1-2 and Figure 1-3 provide respective plan and oblique views of the five mines for the Rajapalot Project showing the position and mining method. Individual ventilation designs have been developed for each deposit with vent raises and escape ways integrated within the mine development schedule.

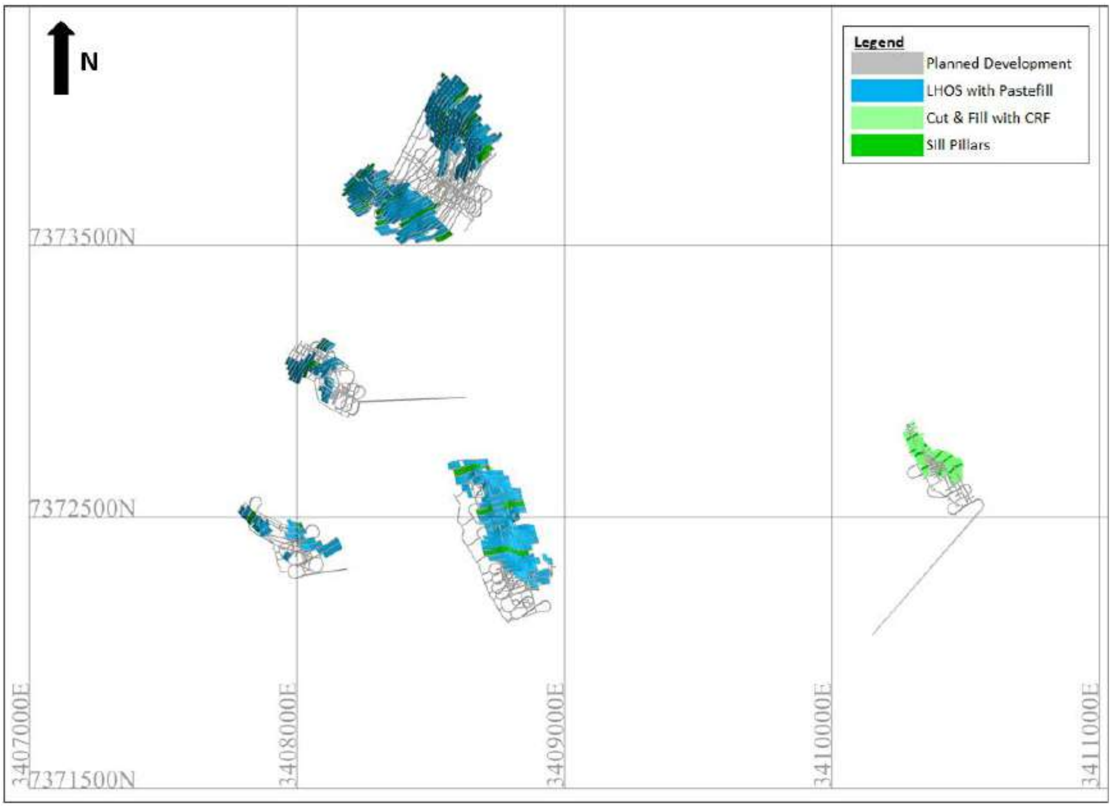

Figure 1-2: Plan view of the five mines for the Rajapalot by mining method

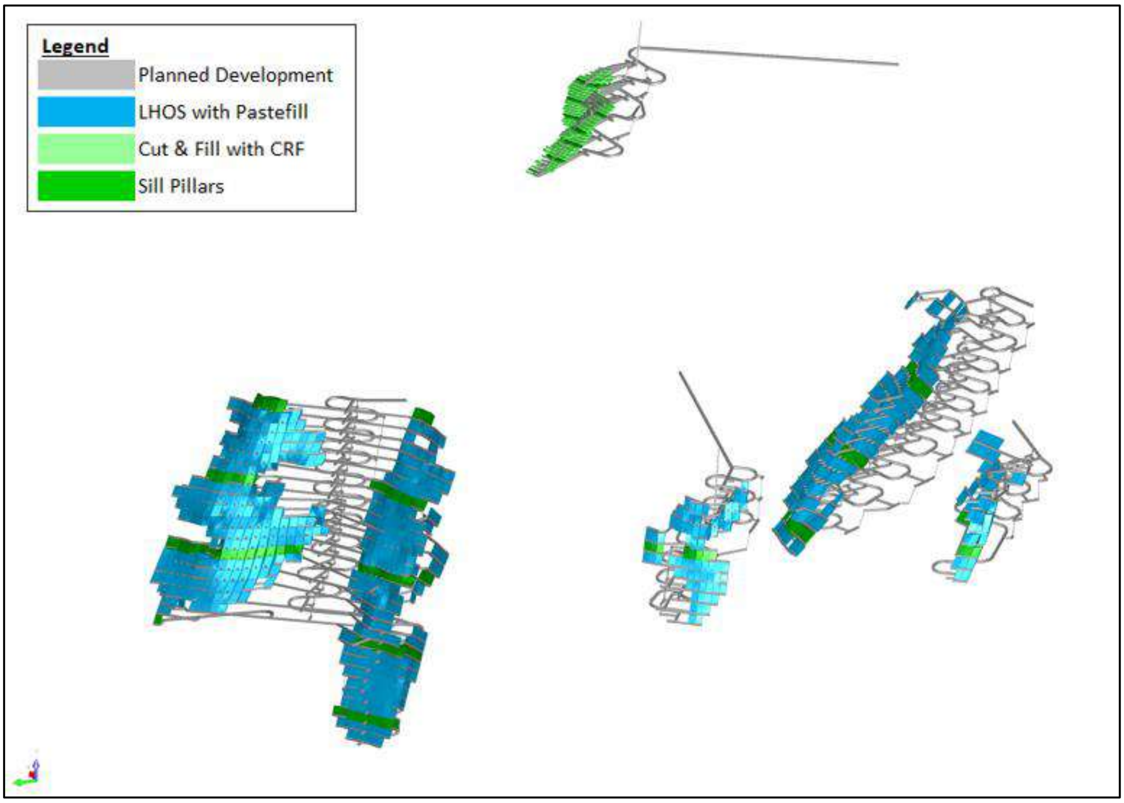

Figure 1-3: Oblique view of the five mines for the Rajapalot by mining method

A Net Smelter Return (NSR) Cut-off Value (CoV) of approximately USD52 per mined tonne was applied for the Rajapalot stope optimization, based on initial operating cost estimates for mining, processing and general and administrative (G&A). The Deswik Stope Optimizer module was used to generate mineable shapes with applied modifying factors (mine dilution and losses) to quantify the RoM inventory used as a basis for the LoM schedule.

Table 1-2: PEA RoM Tonnage

Deposit	RoM (Mt)	Au (g/t)	Co (ppm)	Au (koz)	Co (t)
Palokas	6.1	2.24	379	438	2,303
Raja	2.8	2.58	305	231	846
Joki East	0.4	2.87	225	37	90
The Hut	0.6	1.19	267	22	152
Rumajärvi	0.3	0.98	388	10	118
Total RoM	**10.1**	**2.26**	**347**	**736**	**3,509**
Cobalt Feed	6.1		529		3,203

The proposed Rajapalot mine targets a RoM production rate of 1.2 Mtpa through combined mining of three deposits at any one time to meet the target annual tonnage (Figure 1-4). The annual production schedule, which makes allowance for ramp up, is used to derive an equipment fleet schedule including commissioning and replacement periods over the duration of the operation. Fixed and variable labour is estimated for each annual period based on the development, production and equipment schedule. The mine operating cost estimate assumes an owner-operator approach, as is typical in Finland, with mine equipment purchased via a lease-to-own strategy on typical industry terms.

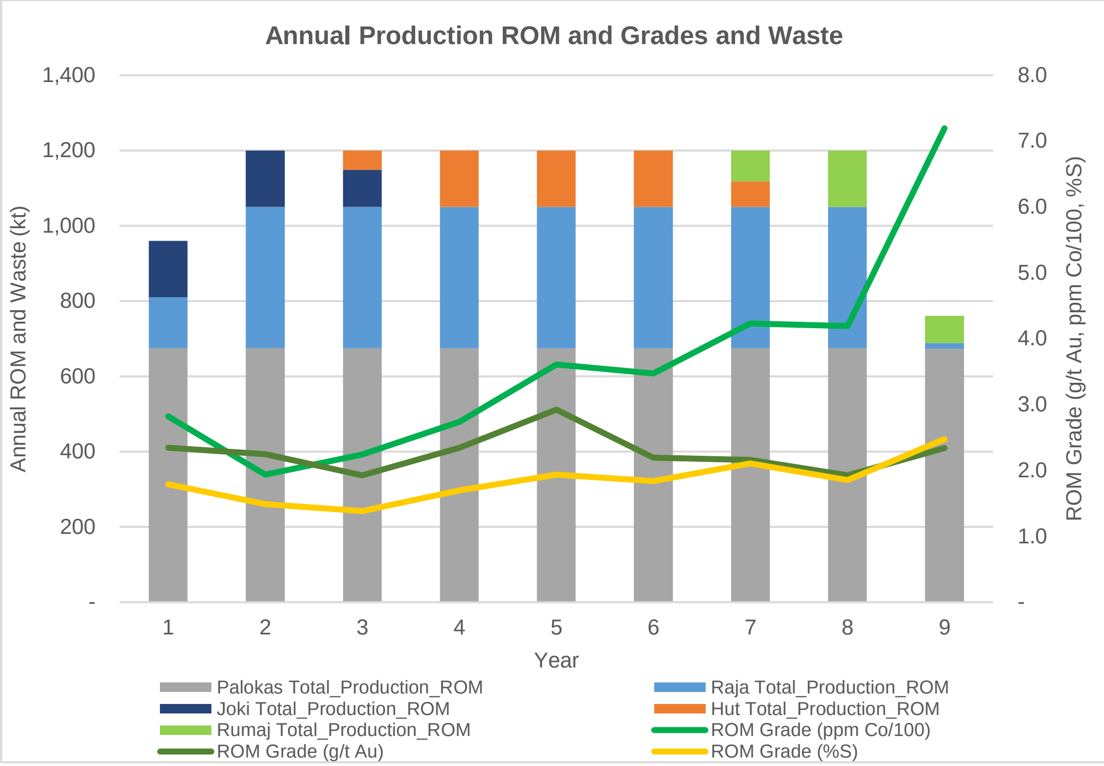

Figure 1-4: Annual Development and Production RoM and Grade for Rajapalot

There has been no hydrogeological assessment of the deposit bedrock; however, a preliminary mine dewatering model was developed and calibrated using hydrological parameters from other regional projects and similar geological settings. The base case inflow estimate for the mine complex ranges between 27 L/s and 32 L/s. Mine and surface water infrastructure has been provided to cater for these as a nominal flow rate.

1.13 Recovery Methods

The process flowsheet was designed based on metallurgical test-work carried out over multiple stages. A processing plant capacity was selected as 1.2 million RoM tonnes per annum, or approximately 3,600 RoM tonnes per day, which will produce on average 82 koz Au in doré and 306 t Co in concentrate per year using industry standard processes.

All feed will be processed for gold recovery but only a proportion (approximately 60%) for cobalt recovery. RoM material will be assessed against an economic cut-off for cobalt extraction, be separately stockpiled, and campaign processed. The process design, shown pictorially in Figure 1-5, consists of:

- Three-stage crushing followed by a single stage ball mill with integrated cyclone classification, with integral gravity gold recovery and intensive leach circuit and independent electrowinning. The target P80 grind size is 75 um.
- Gold leaching via cyanide in leach (CIL) process with a total leaching time of 30 hours. The carbon circulates through an elution and regeneration circuit. Gold reports to electrowinning before ultimately being smelted on site.
- Cyanide destruction using the INCO process is used to reduce free cyanide following CIL.

- Flotation via rougher and two cleaner stages to extract a marketable cobalt concentrate from the gold-cobalt feed. Flotation is bypassed when running a gold-only campaign. Concentrate is thickened and filtered for dispatch via highway truck.
- Backfill plant to split the material based on backfill feed requirements and pumps the balance to the wet residues facility where water can be returned to the process.

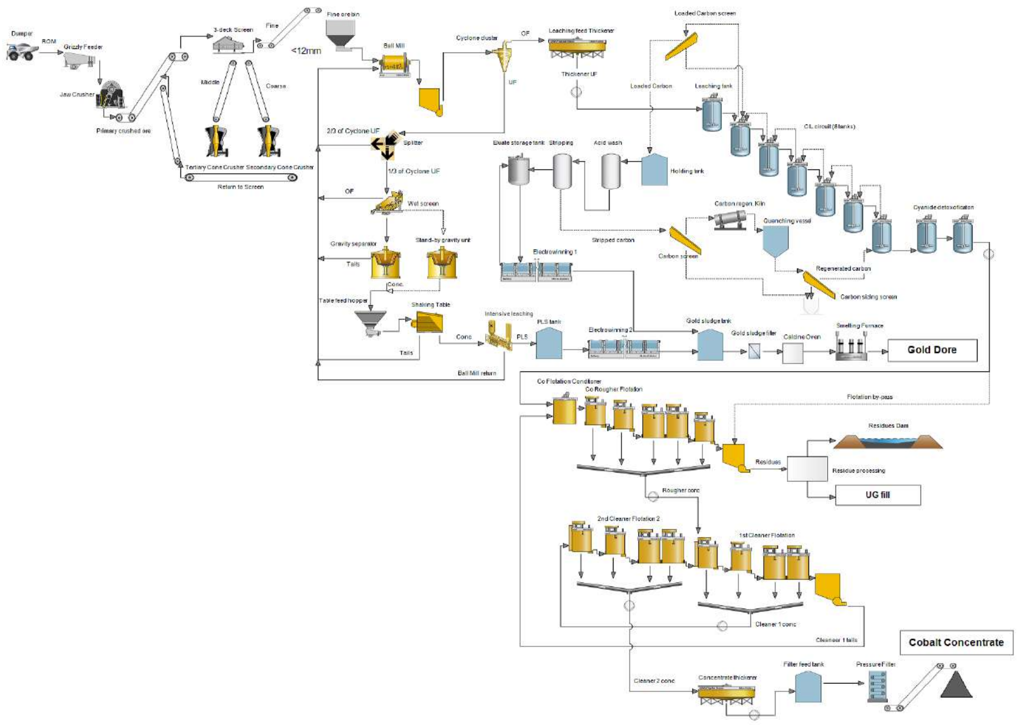

Figure 1-5: Proposed process flowsheet

1.14 Project Infrastructure

The Project is well supported by existing local infrastructure, being located 32 km from the capital of Lapland, Rovaniemi in northern Finland. Access to the Project is along an existing 3 km unsealed public road, which connects to a paved national highway (930). Figure 1-6 shows the conceptual site layout where the main surface infrastructure is located outside of the Natura 2000 area. Within the site, only typical infrastructure is necessary:

- Plant Residues (tailings) management facility for storage of 6 Mt of plant residues (net of backfill requirements). A ring dyke fully lined ring-dyke facility, raised in stages via downstream method, is envisaged for sub-aqueous deposition. Closure costs are also allowed for.
- Water treatment facilities of any potential discharge of water which could not be recycled into the process. The objective of water treatment concept is to reduce contaminants to below regulatory and/or discharge location levels.
- A 10 MW surface facilities heat plant fed by renewable forestry byproducts sourced locally (a typical source of heat in Finland).
- The majority of process equipment is housed inside buildings/enclosures, with laboratory, administration and workshop also provided.

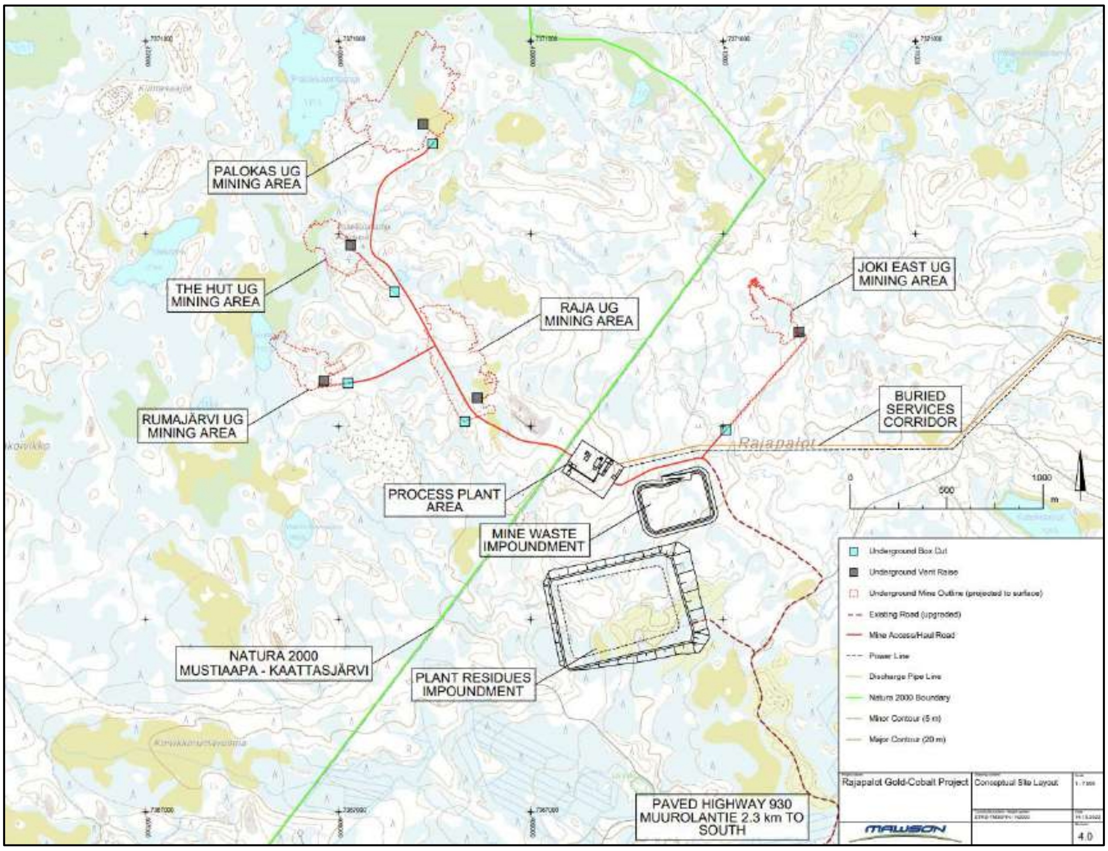

Figure 1-6: Plan view of the Rajapalot deposits and site layout

From an off-site infrastructure perspective, the Project benefits from good access and no significant logistics constraints or challenges exist for the proposed operation (Figure 1-7). New infrastructure connections to the project are limited to a 110 kV powerline 28 km to an existing substation of the Valajaskoski hydroelectric power station, and a 15 km water pipeline to take treated excess project water to a potentially suitable discharge point. It is envisaged the water line will share the power line easement.

Figure 1-7: Rajapalot Regional Plan

1.15 Market Studies and Contracts

The cobalt market is assessed to have a favourable outlook given its use in batteries and rise in global electrification. Cobalt from the project will be produced in concentrated form. Offtake discussions are in their very early stages.

There are no material contracts that pertain to the property.

1.16 Environmental and Social Permitting and Management

The Rajapalot project is a greenfield site. The project area partly overlaps with Natura 2000 area protection area. Owing to exploration permitting conditions, surface baselining of the project area is extensive and to date no red flags has been identified. Further data and studies are needed to proceed to actual planning of the mitigation measures (to be undertaken during the Pre-Feasibility Study (PFS) stage).

Key environmental and social issues are related to permitting and Natura 2000 area. Environmental impact assessment (EIA) takes place prior to the environmental permitting and has been commenced by Mawson. EIA permitting requires geochemical and hydrogeological assessments (in addition to the technical planning to be carried out at PFS stage).

The project area has no Sami or community resettlement issues. Community support of the project is good, with positivity about the opportunities a mine will bring to the local area. Stakeholders' concerns are primarily related to the water quality and competing land uses such as recreation and reindeer herding.

1.17 Capital and Operating Costs

Life of mine capital costs for Rajapalot are estimated as USD291M, comprising USD191M initial capital and USD100M sustaining capital (Table 1-3).

The LoM operating costs average USD55.9/t RoM (Table 1-4). The PEA cost estimates in this section have been completed by SRK, P&C, Sweco and Mawson.

Table 21-3 provides a summary of responsibilities of each contributor to the cost estimates.

Table 1-3: Summary capital cost estimate

Capital Expenditure	Units	Project	Sustaining	Total
Underground Mine	USD M	3.9	57.2	61.1
Capitalized Mine Operating Costs	USD M	10.5	-	10.5
Process Facilities	USD M	125.5	13.7	139.3
Backfill Plant	USD M	10.8	10.8	21.7
Residue / Tailings Management	USD M	11.1	7.4	18.5
Contingency	USD M	29.5	2.2	31.7
Closure	USD M		8.4	8.4
Total Capital Expenditure	**USD M**	**191.4**	**99.7**	**291.1**

Table 1-4: LoM Project Unit Operating Costs (including royalty)

Total Operating Cost	LoM Total (USD M)	Unit rate (USD/t mill feed)	Contribution (%)
Mining (including backfill)	353.0	34.9	62%
Processing (including TSF)	170.6	16.9	30%
G&A	40.5	4.0	7%
Royalties	1.9	0.2	0%
Total	**566.0**	**55.9**	

1.18 Economic Analysis Summary

A technical economic model has been developed on an annual basis to assess the economic potential of the Rajapalot project. The current project schedule assumes a two-year construction period, followed by 9 years of production. The PEA mine production schedule as the main driver for the economic analysis, producing two products:

- doré with gold recovery of 95%, and the doré consisting of 75% gold (for shipment purposes); and
- cobalt concentrate: with Co recovery of 87.6%, S recovery of 88.0%, and a fixed S grade in Co con of 35%.

The following commodity prices have been applied in the PEA:

- Gold: USD1,700/oz; and
- Cobalt: USD60,000/t.

The following general assumptions have been made:

- All costs and revenues are presented in USD and are in real 2022 money terms.
- A 2-year pre-production period for construction, development and commissioning activities.
- Cash flows have been discounted to the start of construction using an end-year approach. Any cash flows, including cost of further studies, prior to the start of construction have been excluded from the analysis; however, a tax loss opening balance is currently allowed for.
- A discount rate of 5% has been applied for NPV calculations.
- A closure cost of USD8.4M has been included at the end of life.
- The cash flow model is presented post-tax and pre-finance.

1.18.1 Economic Evaluation Results

Based on the PEA economic analysis, the project has positive operating margins, a 26.5% post-tax IRR and 2.9 year payback period. The Project operating life of 9 years results in an estimated net cash flow of USD341M and NPV_5 of USD211M. LoM All-in Sustaining Cost (AISC) is calculated at USD824/oz Au, which places the project in the first quartile on the cost curve (as published by the World Gold Council as of 30 June 2022). These financial metrics indicate that the Rajapalot Project has good economic potential and warrants further studies. A summary of pre- and post-tax economic metrics presented in Table 1-5 with annual free cash flow (FCF) shown in Figure 1-8.

Table 1-5: PEA Pre- vs Post-Tax Economic Metrics Summary

	Units	Pre-Tax	Post-Tax
NPV (5%)	USD M	271	211
IRR	(%)	30.2%	26.5%
Undiscounted Payback	(year)	2.8	2.9

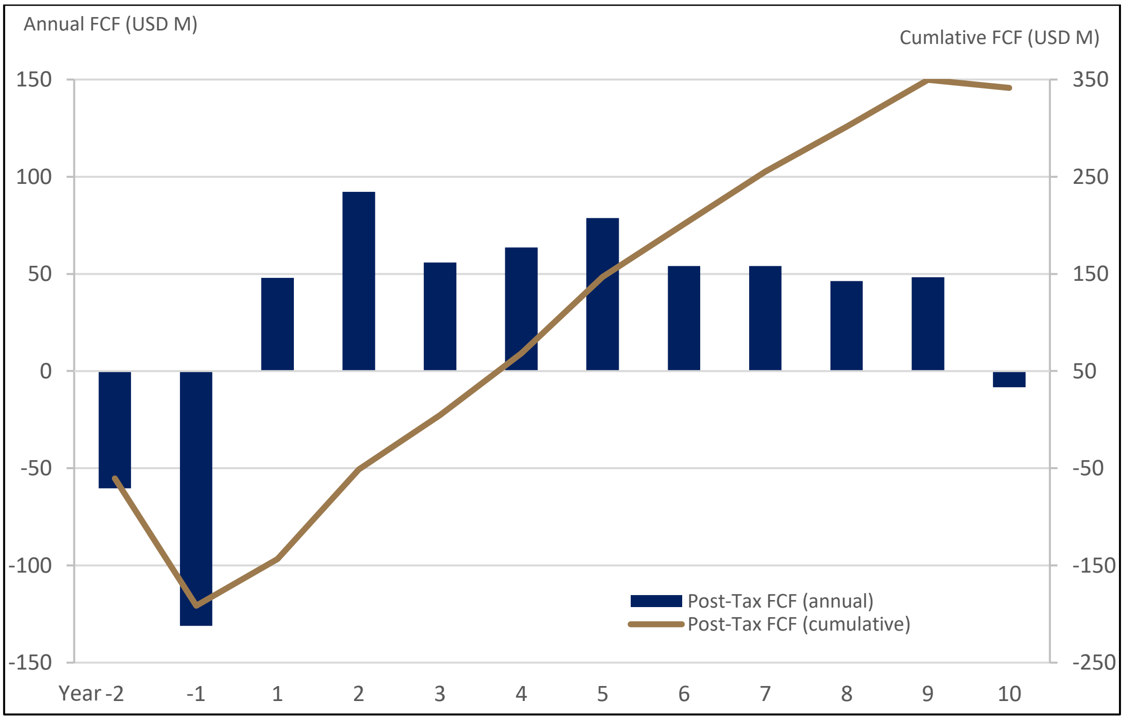

Figure 1-8: Rajapalot free cash flow (post-tax)

The Project is most sensitive to metal prices followed by capital expenditure with select multivariate analysis is shown in Table 1-6.

Table 1-6: Key sensitivities to gold price

Gold Price (USD/oz)	Post-tax NPV_5 (USD M)					Post-tax IRR	Y1-5 FCF (USD M)
	Base Case	CAPEX - 10%	CAPEX +10%	OPEX -10%	OPEX +10%		
1,400	89	112	66	106	72	15%	234
1,550	150	173	128	167	133	21%	286
1,700	211	234	189	228	195	27%	338
1,850	272	295	250	289	255	32%	390
2,000	333	356	310	350	316	37%	442

1.19 Interpretations and Conclusions

The PEA economic analysis indicates that the Rajapalot Project has good economic potential and warrants continued development. The following interpretations and conclusions are summarized for the Project:

- Mineral Resource and Exploration Potential
  - Significant potential exists to expand the MRE, locally as well as in the regional Project area. The defined resource bodies are all open down dip.
  - The wider property is relatively underexplored but has a significant number of anomalous gold occurrences that warrant follow up and present good potential for further discovery.
- Mining
  - The primary mining method of retreat LHOS with paste backfill selected for Palokas, Raja, The Hut and Rumajärvi deposits is appropriate and has the advantage of maximizing mining extraction and reducing tailings storage requirements on surface. The mining method selected for the Joki deposit is overhand C&F and considered appropriate due to its shallower dip angle with CRF for backfill support.
  - The overall production target of 1.2 Mtpa was based on an assessment of the RoM Inventory could be sustained over a 9-year period through mining three of the deposits at any time. The production strategy considers continuous mining of the larger two deposits (Palokas and Raja) over the LoM and mining the smaller three deposits (Joki, The Hut and Rumajärvi) sequentially with the order determined by higher gold grades.
  - All feed will be processed for gold recovery but only a proportion, on a feed campaign basis, for cobalt recovery.
- Geotechnical
  - The primary requirement to limit geotechnical uncertainty is for Mawson to conduct collection of the geotechnical parameters for calculation of the rock classification systems, Q and Modified Rock Mass Rating (MRMR). Currently this is not fully completed and is considered as standard for more detailed technical level studies.
  - Major geological structures have not yet been modelled at the deposit scale. If present these will influence the underground mine design, access placement and also the extraction sequence.
- Hydrogeology and Water Management

  - Based on the assumptions made in this study regarding the surface and groundwater regime at Rajapalot, it is likely that there will be no need for any active intervention pre-mining to advance dewater the rock mass around the underground operations.
  - Whilst the low K and S properties of the formations that host the deposits still need to be confirmed through in situ testing, if they exist as expected, then conventional sump and pump arrangements should be adequate to dewater the underground mines.

- Mineral Processing and Metallurgical Testwork
  - For the purpose of the current PEA, the process design is based on maximizing recovery of gold and cobalt by gravity recoverable gold recovery and WoO leach for gold recovery as bullion, followed by bulk sulphide flotation from the leach tailings to produce a cobalt concentrate.
  - Cobalt recovery and product grade are dependent on feed source and target concentrate product. Flotation testing of both mineral types demonstrated that high recoveries of cobalt can be achieved with appropriate pulp chemistry and collector additions.
  - As demonstrated by the test results, however, the mineralogical type represented by the Pal sample, with low cobalt head grade and only minor cobaltite content, will only produce a low Co grade concentrate, such that flotation processing after gold recovery may not always meet current economic criteria.

- Tailings residue management
  - The following details need to be investigated further in future phases of study including ground conditions beneath the footprint of the TSF; geotechnical testing of representative tailings samples; impact of cold climate on tailings properties and storage capacity; geochemical and settlement properties and sources of materials for TSF construction.

- Environmental Studies, Permitting, and Social or Community Impact
  - Mawson has completed a significant number of environmental studies and has been conducting baselining assessments across the relevant parts of its tenement package as well as surrounding areas, in support of its exploration activities and the evaluation of the impact of a future mining project. The studies have thus far confirmed that there are no such plant or animal species which would be unique to the project area or the larger vegetation zone area.
  - Mawson has had an active ESG program operating for many years, which is being constantly adjusted as its projects grow and develop.
  - Mawson considers stakeholder engagement and collaboration to be a critical part of the potential development of the Rajapalot project, and social aspects will be a key part of the EIA preparation process.
  - Closure costs for the TSF and plant areas have been included in the project estimates. At this stage closure requirement considerations are only preliminary assumptions. The EIA and various permits may set additional requirements to the closure

measures. Full assessment of closure costs will be completed when the needs are studied in future stages.

- Key objects remaining on the site after closure are the extractive waste facilities. Mawson aims to minimize extractive waste rock areas by utilizing waste rock in mine backfill and also in other infrastructure related projects.

1.20 Recommendations

The main recommendations arising from the PEA study relate to collecting of more empirical data, particularly geotechnical and hydrogeological, and completion of more detailed engineering studies to increase cost estimate accuracy. Further gold and cobalt metallurgical test work is necessary and will be used to refine recoveries and operating assumptions, and alongside cobalt marketing studies, optimize the cobalt NSR. Upgrading the resource classification to indicated through drilling would be required to consider future Mineral Reserve assessments. Environmental baseline studies should continue in support of the in-progress EIA, including assessing opportunities to reduce reliance on fossil fuels and the carbon footprint of the Project.

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NI 43-101 TECHNICAL REPORT ON A PRELIMINARY ECONOMIC ASSESSMENT OF THE RAJAPALOT GOLD-COBALT PROJECT, FINLAND

2 INTRODUCTION

2.1 Terms of Reference and Purpose of the Report

This Technical Report has been prepared for Mawson Gold Limited (Mawson) by SRK Consulting (Finland) Oy (SRK) with contributions from AFRY Finland Oy (AFRY), Resources Engineering & Management Pty Ltd (RE&M), Paterson & Cooke Nordic AB (P&C), Gosselin Mining and Sweco Oy (Sweco) to disclose the results of a maiden Preliminary Economic Assessment (PEA), completed in October 2022, in accordance with (NI) 43-101 on the Rajapalot Gold-Cobalt project in northern Finland.

The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in SRK's services, based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Mawson subject to the terms and conditions of its contract with SRK and relevant securities legislation. The contract permits Mawson to file this report as a Technical Report with Canadian securities regulatory authorities pursuant to NI 43-101, Standards of Disclosure for Mineral Projects. Except for the purposes legislated under Canadian securities law, any other uses of this Technical Report by any third party is at that party's sole risk. The responsibility for this disclosure remains with Mawson. The user of this document should ensure that this is the most recent Technical Report for the property as it is no longer valid if a new Technical Report has been issued.

Unless otherwise stated, information, data, and illustrations contained in this Technical Report or used in its preparation have been prepared by the Qualified Persons (QP) for the purpose of this Technical Report.

The PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized.

2.2 Qualified Persons and Details of Site Inspection

This Technical Report is a PEA on the Rajapalot gold-cobalt project which has been prepared by a team of SRK consultants for, and on behalf of Mawson, a publicly-listed company in Canada. This report also incorporates a restatement of the Inferred Mineral Resource estimate for the Rajapalot project as at 26 August 2021.

AFRY was commissioned by Mawson to report on the results of a Mineral Resource estimate (MRE) on the Rajapalot Property in Lapland, Finland where gold and cobalt are the primary elements of economic interest.

Environmental Impact Assessment (EIA) and land use planning processes are supported by Vahanen (acquired by AFRY during 2021/22) and Sweco, respectively.

SRK was commissioned by Mawson to prepare the mine planning including mine geotechnical, waste, tailings, backfill (supported by P&C) and water management. SRK also prepared the overall economic assessment.

Mawson, AFRY, RE&M, Gosselin Mining and Sweco have cooperated in the drafting of this document to ensure factual content and conformity with the preparation of the Technical Report and the requirements of reporting under the National Instrument NI 43-101.

These consultants have extensive experience in the mining and metals sector and are members in good standing of appropriate professional institutions. The consultants comprise specialists in the fields of geology and resource estimation; mining engineering and mineral reserves; mining geotechnical engineering; hydrogeology/hydrology; mineral processing; waste and dry filtered tailings engineering; geochemistry; water management; environmental and social; and financial evaluation (hereinafter the “Technical Disciplines”).

The individuals listed in Table 2-1, by virtue of their education, experience and professional association, are considered QPs as defined in the NI 43-101 standard, for this Technical Report, and are members in good standing of appropriate professional institutions. Table 2-1 provides a summary of the designated Qualified Persons responsible for the disclosure in this Technical Report and Table 2-2 provides details of the personal inspections undertaken. SRK was given full access to the relevant data requested and conducted discussions with Mawson technical staff and management as well as other consulting groups who contributed to the Technical Studies. QP certificates of authors are provided in Appendix A.

SRK will receive a fee for the preparation of this Technical Report in accordance with normal professional consulting practices. This fee is not dependent on the findings of this Technical Report and SRK will receive no other benefit for the preparation of this Technical Report. SRK does not have any pecuniary or other interests that could reasonably be regarded as capable of affecting its ability to provide an unbiased opinion in relation to the PEA, technical economic parameters (TEP), the Life of Mine (LoM) plan for the project and the projections and assumptions included in the various technical studies completed by Mawson, opined upon by SRK and reported herein.

Consequently, SRK, the Qualified Persons and the Directors of SRK consider themselves to be independent of Mawson, their directors and senior management.

Table 2-1: Qualified Persons and Contributors to this Technical Report

Qualified Persons Responsible for the Preparation of this Technical report						
Qualified Person	**Position**	**Employer**	**Independent of Mawson**	**Date of Last Site Visit**	**Professional Designation**	**Sections of the Report**
Christopher Bray	Principal Consultant (Mining)	SRK	Yes	none	BEng (Mining), MAusIMM(CP)	Sections 1 to 6, 15, 16, 18.1, 18.5 to 18.8, 19, and 21 to 27 and overall preparation of Technical Report
Ove Klavér	Principal Geologist	AFRY	Yes	29 June to 2 July, 2021	MSc (Geology), Eur.Geol., FAMMP	Sections 7 to 12, and 14
Eemeli Rantala	Senior Geologist	AFRY	Yes	29 June to 2 July, 2021	MSc (Geology), P.Geo.	Sections 4.3, 4.4 and 20
Craig Brown	Principal Consultant	RE&M	Yes	none	B.E.(Chem), GradDipGeosci, FAusIMM	Sections 13 and 17
Mathieu Gosselin	Industry Expert - Mining	Gosselin Mining	Yes	none	P. Eng.	Sections 17, 18.2 to 18.4, and 21
Other Experts who assisted the Qualified Persons						
Expert	**Position**	**Employer**	**Independent of Mawson**	**Date of Last Site Visit**	**Professional Designation**	**Sections of the Report**
Eemeli Rantala	Senior Geologist	AFRY	Yes	29 June to 2 July, 2021	MSc (Geology), P.Geo.	Sections 7 to 12
Maarit Korhonen	Senior Consultant (Environment)	AFRY	Yes	none	MSc (Geology), BBA	Sections 4.3, 4.4 and 20
Michael Di Giovinazzo	Principal Consultant (Geotechnical)	SRK	Yes	12 April 2022	BSc (Geology), GCertEng, MAusIMM	Section 16.4
William Harding	Principal Consultant (Water Management)	SRK	Yes	12 April 2022	MSc, FGS, CGeol	Sections 16.5, 16.14, 16.15
Hannah Wickham	Consultant (Tailings and Geotechnical Engineering)	SRK	Yes	none	MSc (Hons) Environmental Geology, MIMMM	Section 18.6
Brian Prosser	Principal Consultant (Ventilation)	SRK	Yes	none	B.S. (Mining), P.Eng	Section 16.13
Inge Moors	Principal Consultant (Mineral Economist)	SRK	Yes	none	MSc (Mining), MIMMM	Section 19, 21 and 22
Andy Beveridge	Senior Engineer	P&C	Yes	none	BSc (Hons) Geology, ACSM, MAusIMM(CP)	Section 16.10
Juha Vehviläinen	Leading Specialist	Sweco	Yes	none	D.Sc. (Technology)	Section 21
Timo Pekkala	Process design engineer	Sweco	Yes	none	MSc (Process Engineering)	Section 17

Table 2-2: Details of Site Inspection by the Qualified Persons and Other Experts

Qualified Person	**Company**	**Independent of Mawson**	**Expertise**	**Date(s) of Visit**	**Details of Inspection**
Ove Klavér	AFRY	Yes	Geology and Mineral Resources	29 June to 2 July, 2021	Visit to Mawson office and core storage facility in Rovaniemi. Checked assay results, confirmed field locations of drillholes and prospects. Verified QAQC for sampling of drill core and reviewed database system. Inspected mineralized drill core from the each of the prospects that are included in the Inferred Mineral resource estimate.
Eemeli Rantala	AFRY	Yes	Geology, Environmental and Social	29 June to 2 July, 2021	
Other Experts	**Company**	**Independent of Mawson**	**Expertise**	**Date(s) of Visit**	**Details of Inspection**
Michael Di Giovinazzo	SRK	Yes	Geotechnical	12 April 2022	Visit to Mawson office and core storage facility in Rovaniemi.
William Harding	SRK	Yes	Water Management	12 April 2022	

2.3 Sources of Information

SRK's opinion contained herein is based on its review of the technical and scientific information provided to SRK by Mawson throughout the course of its investigations. SRK has relied upon the work of other consultants in the project areas in support of this Technical Report with major authoring contributions from AFRY, RE&M, P&C, Gosselin Mining and Sweco.

This report is based on technical data, documents, reports and information supplied by Mawson, including copies of Concession application and award documents; historical reports on exploration and drilling; and internal reports by Mawson staff and consultants/contractors. The specific reports referred to are listed in Section 27 of this report.

This report contains data from drilling carried out by Mawson from mid-2011 to June 2021. Preliminary metallurgical testwork commenced in 2014 (SGS) and the most recent testwork was undertaken by Wardell Armstrong International (WAI) and the Geological Survey of Finland (GTK) Circular Economy Solutions laboratory during 2022.

Ongoing work, such as baseline environmental studies, community engagement and initial permitting activities are also described in this report.

2.4 Effective Date

The effective date of this Technical Report is 15 October 2022 (the "Effective Date") with reliance on:

- The Mineral Resource estimate reported in accordance with the NI 43-101 guidelines and the 2014 Canadian Institute of Mining and Metallurgy definition standards for reporting Mineral Resources and Mineral Reserves (the "2014 CIM Definition Standards") with an effective date of 26 August 2021.

SRK's opinion contained herein as of the Effective Date, is based on information collected and completed by SRK throughout the course of the PEA, which in turn reflects various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

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

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

2.5 Units of Measure

Currency is expressed in United States dollars (USD) unless stated otherwise; units presented are typically metric units, such as metric tonnes, unless otherwise noted.

3 RELIANCE ON OTHER EXPERTS

The qualified persons have relied on information and opinions forming the basis for parts of this technical report in the following areas:

- Online data on permits from Finnish Government authorities (TUKES). These data are current as at 16 November 2022 and have been reviewed by the AFRY QP. The portion of the report where this disclaimer applies is Section 4.
- Detailed technical geological work up to August 2021 of Mawson's Finnish geological team, supervised by their Chief Geologist, Dr Nick Cook (FAusIMM). These data have been independently verified by the AFRY QP during field visits in 2021. The portions of the report where this disclaimer applies are Sections 7 to 9 and 14.
- Reports of the GTK, BATCircle 1.0 and SGS were reviewed by the Mineral Processing QP (Craig Brown). The QP relied upon this information in forming an opinion on the mineral processing and metallurgical testing and this disclaimer therefore applies to Section 13.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Project Location

The property is located in the northern Finland region known as Lapland, close to the Arctic Circle (25.0°E and 66.6°N) as shown in Figure 4-1. In local Finnish grid coordinates (KKJ(3), EPSG:2393), the Rajapalot project is centred on 3408700mE and 7373200mN.

The local Finnish coordinate system is being modified to a European standard and is partially implemented by the authorities, as such, agencies such as TUKES require reporting in the newer ETRS89/TM35FIN (EPSG 3067) coordinate format. All reporting in this document remains in the KKJ system.

Figure 4-1: Location of the Rajapalot project area

4.2 Property Ownership

Currently, the Rompas-Rajapalot property consists of 5 granted exploration permits for 7,210 hectares and 9 exploration permit applications for a combined total of 18,685 hectares and is held 100% in the name of Mawson Oy, Mawson's 100% owned Finnish subsidiary.

4.3 Permits and Compensation Arrangements

4.3.1 Exploration Permit

According to the Finnish Mining Act, prospecting and advanced exploration are subject to an exploration permit. An exploration permit on a site entitles its holder to the following rights:

- To conduct exploration on the permit holder's own land and that owned by another landowner, or exploration area, in the area referred to in the permit.
- To explore the structures and composition of geological formations.
- To conduct other exploration in order to prepare for mining activity.
- To conduct other exploration in order to locate a deposit and to investigate its quality, extent, and degree of exploitation, as provided for in more detail in the exploration permit.
- To build, or transfer to the exploration area, temporary constructions, and equipment necessary for exploration activity, as specified in more detail in the exploration permit.

An exploration permit gives its holder first refusal to apply for a mining permit to extract any minerals found within the site. According to the Finnish Mining Act an exploration permit can be granted for a maximum period of 4 years, with an option to extend the permit by three years at a time, up to a maximum of 15 years in total. Reservations are valid for two years.

The permit holder is liable to pay annual compensation to any landowners affected by the permit (known as a 'prospecting fee'). The prospecting fees payable to landowners are as follows:

- EUR20 per hectare per year for the first four years of the exploration permit.
- EUR30 per hectare per year for the fifth, sixth, and seventh year of the exploration permit.
- EUR40 per hectare per year for the eighth, ninth, and tenth year of the exploration permit.
- EUR50 per hectare per year for the eleventh year of the exploration permit and for any subsequent years.

The permit holder is also liable to compensate any inconvenience and damage caused in the area by exploration activities based on the Finnish Mining Act.

The Rajapalot Project refers to an approximate 3 x 3 km area surrounding the reported mineral resource, which occurs within two granted tenements (Kairamaat 2/3 and Hirvimaa). According to the Finnish Mining Act, after the first permit period of up to four years, all exploration permits in Finland can be renewed in 3-year maximum intervals, for a combined total of 15 years. Reservations are valid for two years. According to the Finnish Mining Act, exploration work cannot take place until the renewal has been accepted and completed.

The 1,462 hectare Kairamaat 2/3 and 1,717 hectare Kultamaat exploration permits are granted but appealed, hence not in legal force. However, Mawson is permitted to explore in Kairamaat 2/3 according to an enforcement order granted by TUKES (the Finnish Mining Authority) as soon as the administrative court agrees the ruling set by the mining authority.

The evaluation of the enforcement order is currently in progress in the administrative court. Figure 4-2 shows the locations of the local claims described above and summarized in Table 4-1.

Details of exploration permits were obtained from TUKES and verified on their website (last online update 16 November 2022; https://tukes.fi/karttatiedostot-rss-atomfeedina).

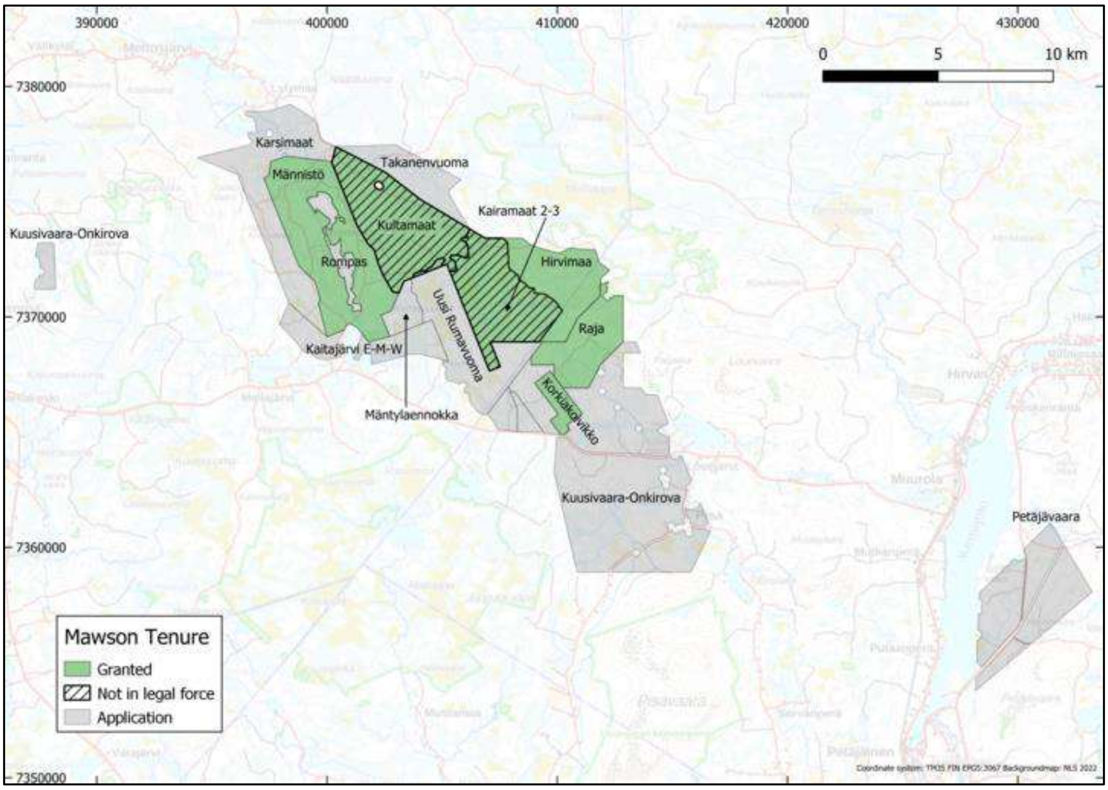


Figure 4-2: Mawson exploration permit status as at 9 November, 2022 (Source: TUKES)

Table 4-1: Status and details of Mawson exploration permits in Rompas-Rajapalot property

Permit type	Name	Mining registry number	Area (ha)
Exploration Permit	Raja	ML2014:0061	883
Exploration Permit	Männistö	ML2016:0046	2,141
Exploration Permit	Kultamaat[1]	ML2015:0005	1717
Exploration Permit	Kairamaat 2/3[2]	ML2013:0041	1,462
Exploration Permit	Hirvimaa	ML2014:0033	1,007
Total			**7,210**
Exploration Permit Application	Rompas	ML2014:0060	265
Exploration Permit Application	Korkiakoivikko[3]	ML2012:0168	232
Exploration Permit Application	Karsimaat	Ml2014:0075	1,618
Exploration Permit Application	Takanenvuoma	ML2022:0015	660
Exploration Permit Application	Uusi Rumavuoma	ML2015:0042	1,283
Exploration Permit Application	Kaitajärvi E-M-W	Ml2014:0100	809
Exploration Permit Application	Mäntylaenokka N -S	ML2015:0054	398
Exploration Permit Application	Kuusivaara	ML2014:0077	4,565
Exploration Permit Application	Petäjävaara	ML2014:0074	1,645
Total granted plus application permits			**18,685**

1. Granted but not in legal force, pending hearing of a permit approval appeal

2. Granted but not in legal force, pending hearing of a permit approval appeal or reinstatement of previously issued enforcement order

3. Extension permit application filed

4.3.2 Compensation Payable under the Environmental Permit

In addition to the fees payable according to the mining permit, obligations relating to compensation and securities may also be imposed in environmental permits. Typical examples of such obligations include compensation for effects on fishing and waste management securities to ensure that the rehabilitation phase will be completed satisfactorily. Decisions relating to the environmental permit and any compensation payable under the permit rest with the Regional State Administrative Agency for Northern Finland.

4.3.3 Factors Affecting Work on the Property

SRK is not aware of any other significant factors or risks that would prevent the right or ability to work on the property.

4.4 Preservation Areas

Parts of the Rajapalot project area, namely claim areas Kairamaat 2/3, Uusi Rumavuoma and Rompas are defined as European Union Natura 2000 designated areas. Natura 2000 sites cover approximately 17% of Mawson's current exploration permit area at Rompas-Rajapalot (Figure 4-3).

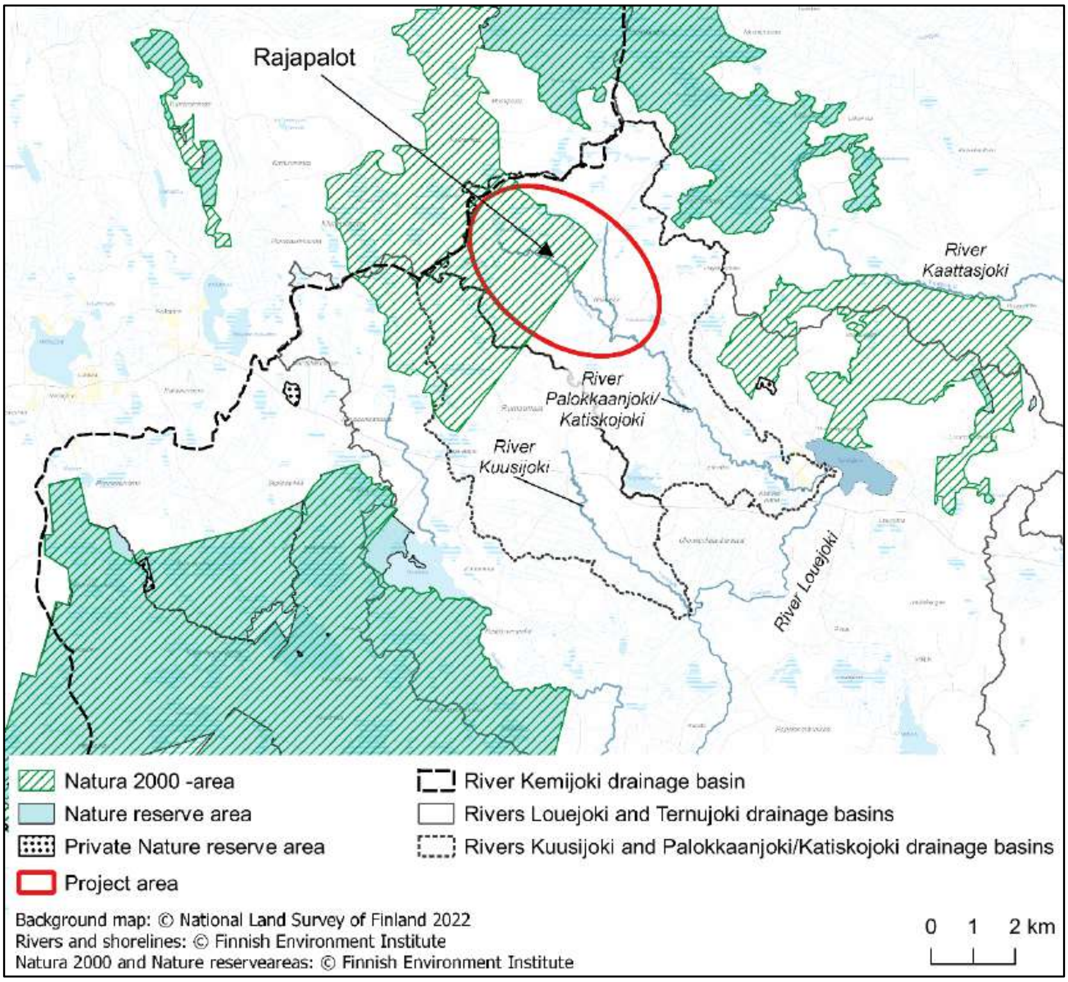


Figure 4-3: Rajapalot project area and adjacent Natura 2000 area

The Natura 2000 network is in place to conserve important biotopes and species throughout Europe. The purpose of the Natura 2000 program is to preserve nature's diversity. Natura 2000 sites cover approximately 13% of Finland and over 30% of Northern Finland. Natura 2000 is the centrepiece of EU nature and biodiversity policy. Mineral exploration is an allowed activity within Natura 2000 areas.

At Rajapalot, the Natura 2000 area boundary is approximately a kilometre to the northwest of the planned tailings facility area. The laws that govern Natura 2000 do not demand any buffer zones to be set between the conservation area and the land surrounding it. Impacts of projects or plans in or near Natura 2000 sites will be assessed unless it is certain that they will not undermine conservation objectives. The combined effects of different projects are also assessed. Projects can only be approved if the assessment has ensured in advance that they will not have a significant detrimental effect on the conservation objectives of the Natura 2000 sites. The Government may grant a permit for a project that impairs the natural values of a Natura 2000 site if it has to be implemented for an overriding reason in the public interest and there is no alternative solution. In that case, compensatory measures must be taken to maintain the coherence of the network.

A private conservation area (ERA206740) is located approximately 1.5 km to the south of the planned process plant site. The area is outside of the Rajapalot project area.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Project Setting and Accessibility

The Property is located approximately 35 km west-southwest of the city of Rovaniemi in southern Lapland, Finland. Access by road from Rovaniemi is via highway E75 south-westerly for 24 km to the junction of highway 930, just past the town of Muurola. Heading westerly on highway 930 (Aavasaksantie) for about 18 km, the Property is accessed via a secondary / tertiary gravel road that heads northerly. This is roughly the south-eastern boundary of the property which extends for several kilometres to the west and northwest; the project lies about 10 km south of the Arctic Circle.

The project lies across the boundary of the local municipalities Ylitornio and Rovaniemi. The area has good road connections from the south, east and west from the Swedish border. Mawson's all-year core logging, storage and office facilities are located at Ahjotie 7, Rovaniemi. Mawson also has an office in Ylitornio which is used as a community relations hub.

The project area is accessible from the main roads via existing forest roads in good condition all year round. The distance from the main road to the project area is 4 km; however, the deposits are located in the project area in such a way that there are no existing roads in the immediate vicinity.

5.2 Climate

Finland is one of the most Northern countries, and with its long north-south geography the climate conditions vary across the country. According to the Finnish Meteorological Institute's (FMI) data from 1991 to 2020, the annual mean temperature can vary from -2 to over +5°C, and the annual average precipitation from 400 to over 750 mm. Note that owing to the presence of the Gulf Stream along the Norwegian Coast (450 km to the northwest), the winter season is less harsh than equivalent latitudes in Canada.

The Rajapalot project area is in Lapland, the Northern part of Finland, about 10 km south from the Arctic circle. Rajapalot has a typical Laplandic subarctic climate with cold, snowy winters and mild summers. Rajapalot is also part of an area called Lapin kolmio (Lapland triangle), which is climatically affected by the closeness of the sea Bothnian Bay that leads to milder winters, at least compared to rest of the Lapland. There is no permafrost in the region surrounding the project.

According to the Finnish Meteorological Institute's climate statistics from 1991 to 2020, the project area's annual mean temperature is between 1 to 2°C and the annual rainfall averages 535 mm with snow on the ground averaging 183 days per year. Rajapalot lies near a watershed boundary and is topographically almost 100 m higher than the neighbouring areas. In general, the topography relief of the Rajapalot area is rather flat despite alternating moraine hummocks and peatland areas.

5.3 Local Resources and Infrastructure

Rovaniemi is the largest city and provincial capital of Lapland with a population of 63,000 (July 2020). Rovaniemi has an international airport (IATA: RVN) and is the third busiest airport in Finland in terms of passenger numbers and is located about 10 km north of Rovaniemi city centre. Daily flights from the all-weather sealed airport link with the Finnish capital Helsinki. Regular trains also service Rovaniemi.

For international overseas shipments, the ports of Kemi and Oulu are suitable for mine operations. Rajapalot property is connected to Kemi and Oulu via routes 930, 929 and highway E75 to Kemi (110 km) and via routes 930, 929 and highway E75 and E8 to Oulu (230 km).

Highly educated skilled labour is readily available in Rovaniemi and surrounding communities. There is adequate raw material (water, gravel, timber) and forestry roads inundate the entire area. The smaller communities along highway 930 on the southern margins of Mawson's tenement package are serviced with electricity. As mining is an established and recognized industry in Finland, there would appear to be no hindrances to surface rights. The terrain is suitable for a mine/processing plant, dumps, tailings and storage facilities.

Powerlines, water, and telecommunications are all available within short distance from any of the prospects.

5.4 Physiography

Most of Lapland is characterized by lowland topography with the maximum elevations of lowland being 200 m elevation. The topography is gently rolling to almost flat, heavily glaciated and inundated with numerous post-glacial lakes, till, eskers, lacustrine and fluvial deposits. The mean elevation on the property is approximately 170 m above sea level (masl), ranging between 150 and 200 masl. At Rajapalot, swamps and small creeks drain east and south into the Kemijoki River which in turn flows into the northern Gulf of Bothnia, while waters in the western part of the project drain to the Tornio River also flowing into the northern Gulf of Bothnia.

5.5 Seismicity

Seismic activity in Finland is low, due to Finland's location in the middle of the Eurasian plate. Small earthquakes are mainly caused by the release of stresses in the bedrock. Seismic Institute monitors earthquakes in Finland and tens of Finnish earthquakes are registered each year. These earthquakes are relatively weak, with magnitudes within 100 km of the Project generally less than 1, and a maximum of 2.2 in the last 10 years.

6 HISTORY

6.1 Prior Ownership

On 30 April 2010, Mawson entered into an agreement with AREVA whereby Mawson acquired 100% of AREVA's mineral properties and exploration database in exchange for EUR 1 M and 10% equity in Mawson. At that time, the Rompas project had 30 grab samples collected while the Rajapalot area had not yet been discovered nor was part of the Areva exploration area. The Rajapalot project was a grassroots discovery by Mawson in 2012 and is located 8 km east of Rompas.

6.2 Exploration History

Commencing in the 1950s, the GTK conducted molybdenum and uranium exploration to the south and west of the Rajapalot area before moving north in the 1970s in a search for copper and tungsten. Rautaruukki undertook uranium exploration approximately 15 km south of Rajapalot and discovered the Mustamaa prospect (Vanhanen, 2010).

Airborne radiometric surveys were flown by GTK and interpreted anomalies were followed up on the ground by geologists with prior GTK exploration experience. GTK drilled a fence of diamond drillholes along strike from what is now the Uusi Rumavuoma exploration permit application. Radioactivity was first discovered within Uusi Rumavuoma in the early 2000s.

Rompas, a gold-uranium discovery made by AREVA in 2007 was the first gold occurrence described in the region where there is no evidence for prior exploration, development or production.

AREVA began reconnaissance exploration in June 2007, consisting of mostly ground radiometric surveys. Further work was completed in 2008 with some follow-up work done in 2009. More than 150 new, separate occurrences of high uranium and extremely high gold content were located in bedrock with 30 grab samples taken at the time on the Rompas project. At that time, however, AREVA decided to reduce activities in Finland and started negotiations with Mawson.

Located 8 km east of Rompas, Rajapalot is a grassroots discovery made by Mawson geologists in 2012. A small outcrop a few metres across is the only surface exposure of any of the resources discussed within this Technical Report.

There are no visible signs of prior exploration activity at Rajapalot.

6.3 Mineral Resources and Production

No historical Mineral Resource or Mineral Reserve estimates or production have occurred on the property before Mawson commenced exploration activities.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology Setting

7.1.1 Craton, Regional and Peräpohja belt

The bedrock of Finland is defined by an Archean basement (3.5–2.5 Ga), its Paleoproterozoic sedimentary-volcanic cover (2.5–1.9 Ga) and the Svecofennian orogenic domain (1.93–1.8 Ga; Hanski E., 2015; see Figure 7-1(a)). Archean crustal segments are attributed to the Kola and Karelian cratons and are separated by the Lapland Granulite Belt and the Belomorian terrain (Lahtinen R. et al, 2005). Throughout the Paleoproterozoic tectonic evolution (2.5–1.9 Ga), the Archean cratons underwent several stages of intracontinental extension and rifting, as well as continental margin rifting, which resulted in the formation of deep-scale structures and shallow water-basins (Lahtinen R. et al, 2005).

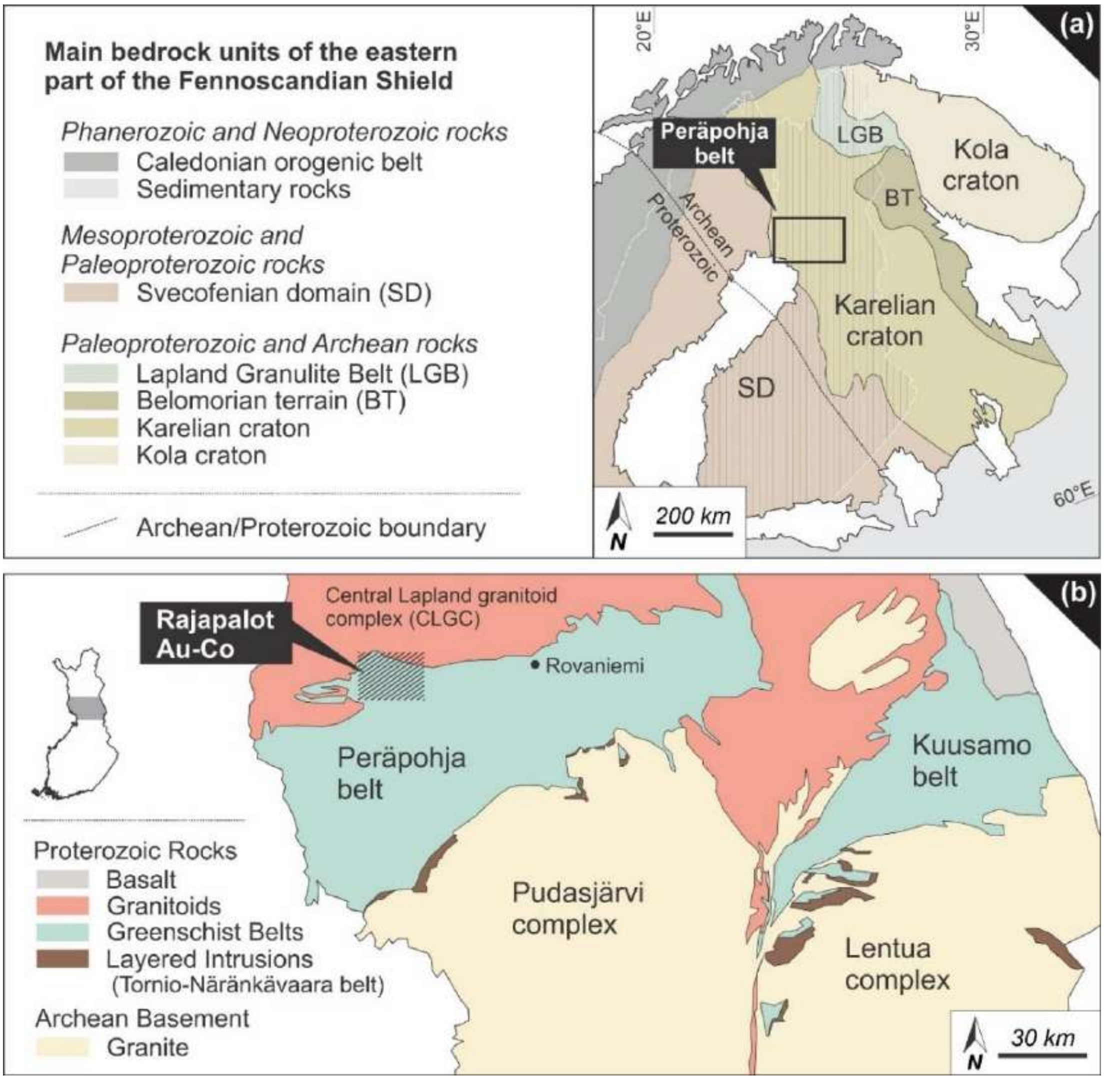


Figure 7-1: Fennoscandian Shield bedrock units (a) and enlarged area (b) of Archean and Paleoproterozoic rocks of the Peräpohja and Kuusamo belts (Source: Raič S. et al (in Prep), 2021)

The depositional history of these Karelian basins coincides with the Great Oxygenation Event, which created favorable conditions for a pre-concentration of metals (such as Co, Cu, Zn, Ni, Mo, and Au) in sulfidic sediments, as well as the deposition of carbonaceous material (Melezhik V.A. et al, 2013). Köykkä J. et al, 2019 proposed five basin evolution stages (see Figure 7-2) and summarized the volcano-sedimentary successions and intrusions with the following generalized lithostratigraphy (from bottom to top): (i) basaltic mafic volcanics and minor conglomerates (ii) clastic sedimentary rocks, (iii) subaerial mafic volcanics, komatiites and carbonate rocks; (iv) greywackes, carbonaceous material and sulfur-rich pelitic rocks and (v) phyllites and greywackes. All these units were then deformed and metamorphosed during the Svecofennian Orogeny (Lahtinen R. et al, 2005; Figure 7-2). Such a depositional and orogenic evolution of Archean to Paleoproterozoic settings is worldwide referred to as Paleoproterozoic greenstone belts. Within the northern part of the Karelian domain, the Svecofennian orogenic gold deposits were formed in the early stages of the accretion of microcontinents between 1.92 and 1.86 Ga, which resulted in the formation of the Fennoscandia Plate, and the far field effect of the collision of Fennoscandia and Sarmatia in the SE (Svecobaltic orogeny) and Amazonia in the west (Nordic orogeny) between 1.85 and 1.79 Ga (Weihed et al., 2005; Lahtinen R. et al, 2005; Molnár F. et al, 2017, 2018). The gold mineralization is mainly located in complexly folded thrust zones where contacts between the competent and less competent host-rocks have experienced several stages of deformation and alteration (Hanski E., 2015).

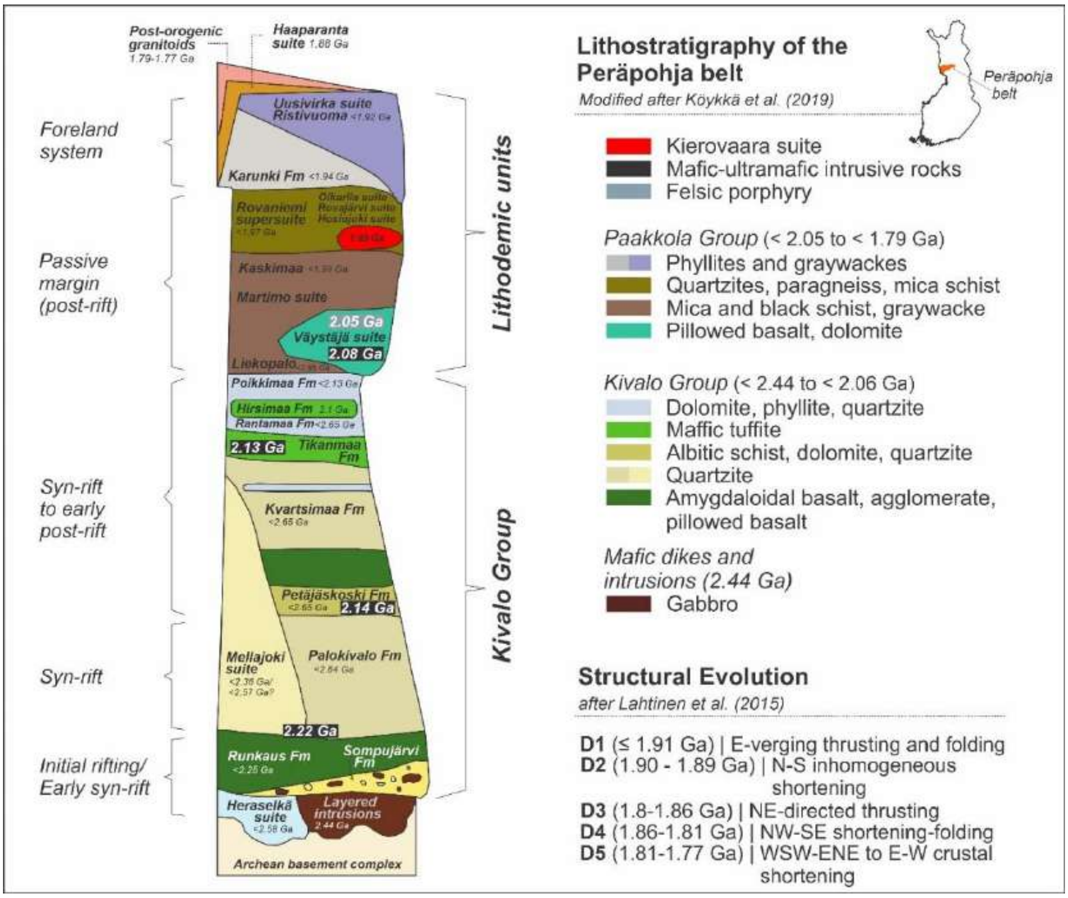


Figure 7-2: Paleoproterozoic lithostratigraphy of the Peräpohja belts (Source: Raič S. et al (in Prep), 2021)

One of these Paleoproterozoic rift-related basins is the Peräpohja belt (PB) in northern Finland, located between the Archean granitoid Pudasjärvi complex to the southeast and the Central Lapland granitoid complex (CLGC) to the north (see Figure 7-1(b); Vanhanen E. et al, 2015; Nironen M. (ed.), 2017). The maximum depositional age for the PB is defined by the NE trending 2.44 Ga layered intrusions of the Torino-Näränkävaara belt (Iljina and Hanski et al., 2005), scattered along the northern boundaries of the Pudasjärvi and Lentua complexes (Figure 7-1(b); Ranta, J.P. et al, 2015; Nironen M. (ed.), 2017). After emplacement, normal faulting of intrusions led to partial erosion of igneous layers, onto which the lowermost and oldest units of the PB were deposited (Sompujärvi conglomerates and Runkaus volcanic sequence, Figure 7-2; Hanski E. et al, 2005; Nironen M. (ed.), 2017). In the western part of the belt, the youngest supracrustal metasediments are cut by 1879 ± 3 Ma monzonite intrusions, which constrain the minimum age of the PB at 1.88 Ga (Lahtinen R. et al, 2005; Hanski E. et al, 2005; Ranta, J.P. et al, 2015; Nironen M. (ed.), 2017).

Following the classification of Perttunen et al. (1990), the Kivalo group and the Paakkola group are the two major lithostratigraphic units that characterize the supracrustal rocks of the PB (see Figure 7-2): (i) the base of the Kivalo group is defined by conglomerates (Sompujärvi Formation, 2.44 Ga), which are overlain by amygdaloidal basalts (Runkaus Formation, 2.25 Ga). Concordant mafic layered sills (2.22 Ga; Perttunen V. et al, 2001) cut the quartzites of the voluminous Palokivalo Formation, which is deposited on the Runkaus Formation (Ranta, J.P. et al, 2015). Mica-albite schist and dolomite of the Petäjäskoski Formation are overlain by the 2.1 Ga continental flood basalts of the Jouttiaapa Formation (Huhma H. et al 1990; Hölttä, P. et al, 2007; Kyläkoski M. et al, 2012). Sericitic quartzites and dolomites of the Kvartsima Formation define the upper part of the Kivalo group, while the Tikanmaa, Poikkimaa, Hirsimaa, Rantamaa and Lamulehto Formations characterize intervening mafic tuffite, dolomite and phyllite (Ranta, J.P. et al, 2015). (ii) Rocks of the Paakkola group comprise pillowed basalts (Väystäjä Formation, 2.05 Ga; Perttunen V. et al, 2001), mafic and felsic tuffs (Korkiavaara Formation, 1.97 Ga; Hanski E. et al, 2005), mica schists, black schists and metagraywackes (Martimo Formation; <1.92 Ga; Lahtinen R. et al, 2015).

The tectonic evolution of the PB is characterized by a polyphase deformation history (between approximately 1.9 and 1.8 Ga) and increasing metamorphic conditions towards the northern parts from lower-greenschist to upper amphibolite facies and local migmatization along the northeastern marginal zone (Figure 7-2; Hanski E. et al, 2005; Lahtinen R. et al, 2005; Laajoki, K., 2005; Hölttä, P. et al, 2017). A detailed description of the Svecofennian tectonic evolution of the PB and related emplacement periods of granitic intrusions is provided by Lahtinen R. et al, 2015 and Nironen M. (ed.), 2017, where the authors discuss the five deformation stages that affected the PB in great detail (D1–D5; see Figure 7-2).

Note that the section above (part of Section 7.1 Regional Geology) is based wholly on the work of Raič S. et al (in Prep), 2021. This is a cooperative paper in preparation co-authored by GTK and Mawson employees.

7.1.2 Regional Mineralization and Age Data

New research work of a GTK team led by Ferenc Molnár has shown a significant group of Au and Au-Co mineral deposits across the northern Finnish Paleoproterozoic rocks. It is clear from these data that early deposition of sulphides and redox sensitive metals commenced around 2.06 Ga and formation of new minerals with progressively younger closure ages continued through to approximately 1.78 Ga. The influx of Au, W, As, Bi, Te and S occur in two periods, that are approximately 1.92 Ga and 1.78 Ga. The latter event corresponds with the ages of Rajapalot and Rompas gold. Figure 7-3 and Figure 7-4 show abbreviated data on the Paleoproterozoic Au and Au-Co systems in northern Finland.

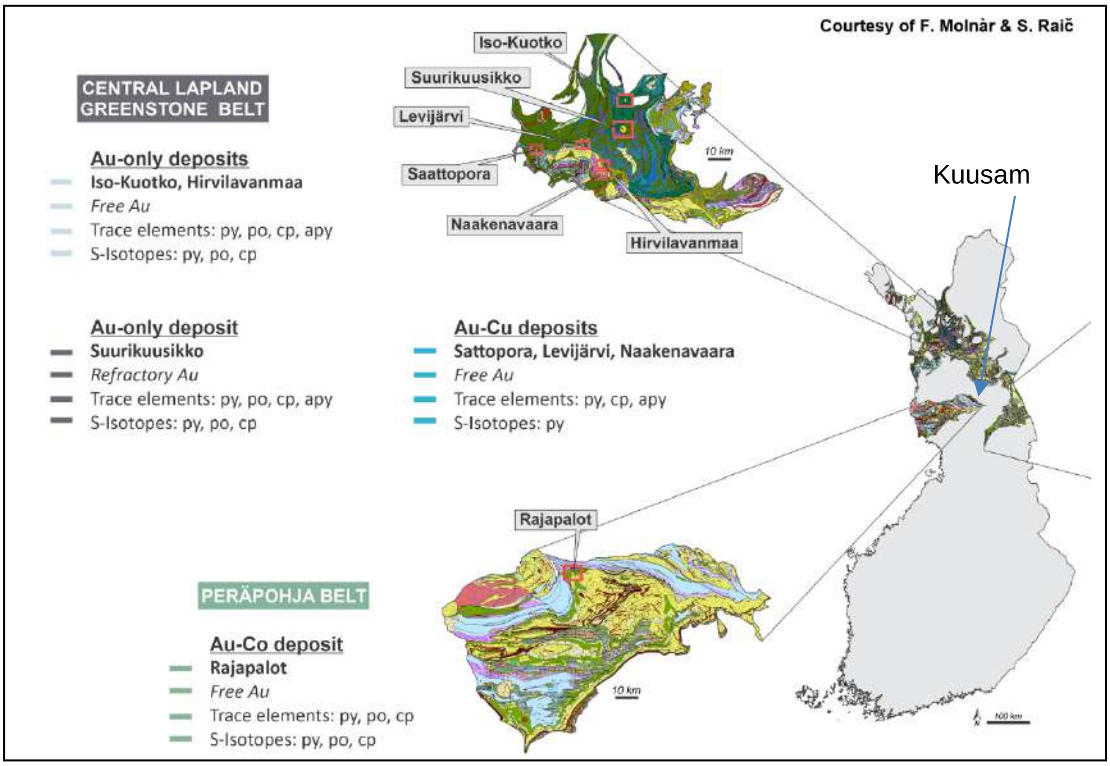


Figure 7-3: Mineralization in the Central Lapland Greenstone belt (CLGB), Peräpohja belt (PB) and Kuusamo belt (KB) (Source: figure modified from Molnár and Raič)

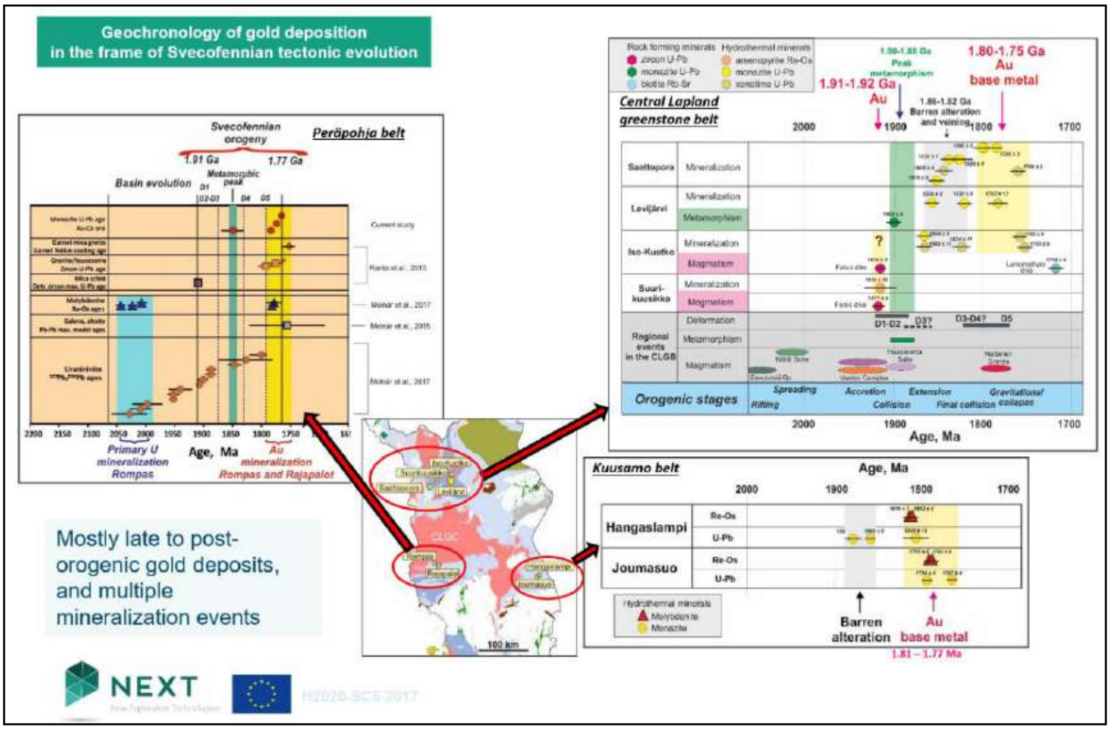


Figure 7-4: Summary of age data on mineralization and host rock minerals from the northern Finland Paleoproterozoic (Source: Molnár and others, supported by New Exploration Technologies (NEXT), a Horizon 2020 project, SC5-2017)

7.2 Local Geology

7.2.1 Rajapalot project area

The host sequence comprises a polydeformed, isoclinally folded package of amphibolite facies metamorphosed Paleoproterozoic rocks. At a local scale Mawson has divided this package into two parts; firstly, a siliciclastic, dolomitic carbonate and albite-altered metasedimentary sequence interpreted as forming in a platformal to continental margin setting. Mawson now recognizes this as correlatives to the Kivalo Group (see Figure 7-2). The second metasedimentary sequence comprises pelitic turbidites, arkosic sands, carbonates, impure and pure quartzitic sandstones and sulphidic bituminous rocks corresponds to the Paakkola Group.

Both groups contain mafic metavolcanics, sills and dykes, but distinctive suite of high-Mg mafic volcanics or sub-volcanic intrusives occur within the Paakkola Group. These high-Mg mafic volcanics are likely komatiites and are locally highly altered to cordierite-anthophyllite schists.

An unconformity between the two sequences representing a boundary between mostly oxidised rocks of the Kivalo Group and generally reduced rocks of the Paakola Group represent the most likely interpretation, although this may also now represent a thrust surface. The mafic rocks, ranging from lava flows, volcaniclastic sediments to dykes and differentiated sills form up to 20% of the total package. Rare, but significant mafic rock-hosted magnetite iron formations up to 20 m thick occur within the Paakkola Group within the Mawson permits.

Outcrop in the Rajapalot area is sparse, with swamps, bouldery till and lakes dominating the terrain. Outcrops where present are dominated by resistant rock types such as quartzite, albitic metasediments and amphibolite which represent more than 99% of exposures. The boulder types reflect the same resistant rock types. A single mineralized outcrop of weathered pyrrhotite bearing Mg-amphibole chlorite rock was found to contain up to 80 g/t Au at Palokas.

Metamorphic grade is largely amphibolite facies throughout the project area, from near the greenschist-amphibolite facies boundary in the south with increasing grade towards the north into sillimanite stability field. Retrograde alteration to chlorite or epidote is relatively common adjacent to quartz veins and fractures. Tourmaline-bearing granitoids (ca. 1.8 Ga) are exposed within 3 km to the north, and recent drilling at southeast of South Palokas prospect has revealed albitised granitoids and diorites interpreted to be 2.10 Ga in the core of the project area (based on monazite dating performed in collaboration with GTK).

The structural interpretation of the Rajapalot project area has evolved over the last three years adding an additional 30 km of drilling. The first deformation stage is dominated by isoclinal folding producing a regional layer-parallel peak metamorphic amphibolite facies foliation. A second deformation event is evident, also largely isoclinal, such that distinguishing S_1 and S_2 foliations is only possible rarely in drill core (and even more so in the sparse outcrops). Drill-core scale small tight folds with overprinting foliations indicate S_2 fabrics were also formed synchronous with peak metamorphic conditions. Mineral lineations, especially those developed in mafic amphibolites are all regarded as synchronous with peak metamorphic conditions.

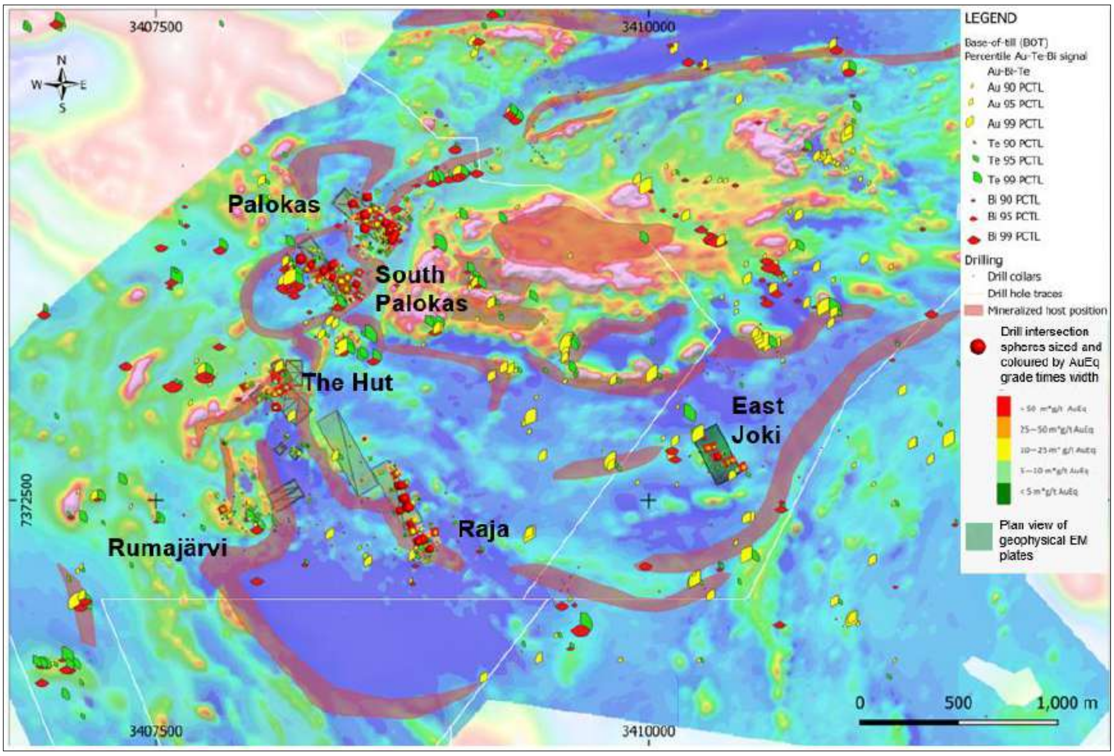


Figure 7-5: Ground magnetics (RTP) showing prospect names, drill intersections (coloured balls, gold equivalent (AuEq) sized and coloured by grade*width), anomalous BOT wedges (yellow corresponds with Au percentile >90%)

A third generation of folds is evident throughout the Rajapalot area, generally tight to close, but may be more open in the most competent host rocks (rarely isoclinal). Crenulations and spaced foliations are common, and all indications are that these folds and foliations developed subsequent to peak metamorphism. Fold axes generally pitch at 70 to 90°producing a reclined fold style. Sulphides are likely concentrated by this fold generation. With enveloping surfaces defined by the dominant S_0-S_1-S_2 parallelism in plan view and the reclined F_3 fold axes, a series of plunging sulphidic units is recognisable as oblique plunging conductors hosting cobalt (and gold) mineralization. Sulphide foliations are clearly folded by the F_3 event. The D_3 deformation may locally develop foliation and fold overprinting through progressive deformation resulting in apparent D_4 folds and planar fabrics. Lineations associated with D_3 deformation are interpreted as intersection fabrics. F_3 axes are not always colinear as the locally more open F_2 fold cause minor perturbations (for example at The Hut prospect). F_3 fold axes do, however, nearly all trend northwesterly in broad parallelism with the enveloping surfaces of the host fabrics (small circles of F3 folds lie about an axis of less than 30°.

A regional overturning of the Rajapalot sequence is indicated by the inversion of the "stratigraphy" between Palokas and Raja prospects (locations shown in Figure 7-5 and Figure 7-6). This overturning event is most likely related to F_1 or F_2 folding events.

Retrograde platy metamorphic minerals (secondary biotite, phengitic muscovite and chlorite) are related to D_3 and subsequent events and fabric-destructive greenschist facies chlorite and white mica are common in association with gold mineralization. Late open to gentle upright folding, fracturing and veining mobilises sulphides and is regarded as the event responsible for the linear high-grade gold trends observed within the prospects. In the most competent and isotropic rocks, fracturing may be the dominant deformation style, locally forming sulphide-matrix breccias.

The mineralization occurs close to the boundary between the Kivalo and Paakkola Groups and may be locally transgressive. The boundary is marked by a strong geochemical gradient, including from sodic (Kivalo) to potassic (Paakkola) and from oxidised (Kivalo) to reduced (Paakkola).

7.2.2 Mineralization

Prospects with high-grade gold and cobalt at Rajapalot occur across 3 km (east-west) by 2 km (north-south) area within the larger Rajapalot project exploration area measuring 4 km by 4 km with multiple mineralized boulders, base-of-till (BOT) and rare outcrops. Figure 7-6 shows the location of the prospects within the Rajapalot project in an oblique view looking northeast. High-grade Au-Co mineralization at Rajapalot has been drilled to 540 m deep at Raja and South Palokas prospects, but is not closed out at depth in any prospect. The only surface exposure of mineralization is at Palokas; however, with the exception of East Joki, all mineralization comes to the top of the bedrock below the till, less than 6 m below the surface. East Joki is 110 m from the surface at its shallowest, but is not drilled yet in the up-dip direction.

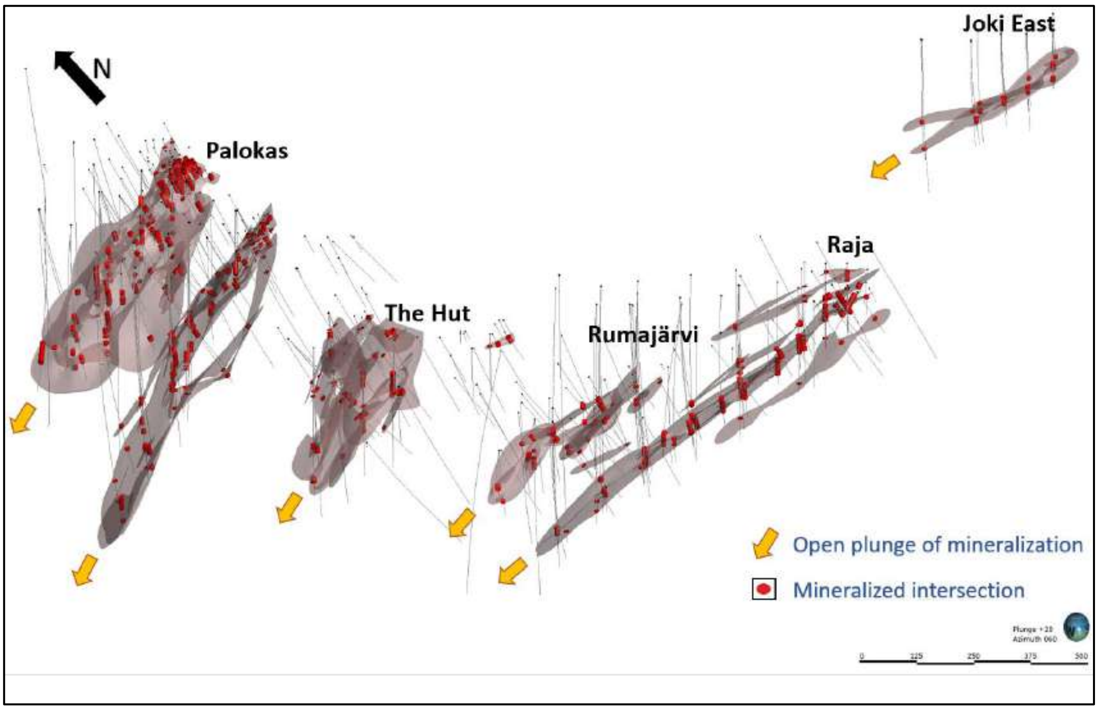


Figure 7-6: Oblique view of Rajapalot project area looking northeast showing prospect names

The Au-Co mineralization at Rajapalot differs markedly from the nuggety Au-U style originally discovered at Rompas. Grade continuity, negligible carbonate and a stratabound potassic or Fe-Mg host to the Au-Co mineralization predominates at Rajapalot, whereas Rompas gold was hosted in and associated with uraninite in dolomite-calcsilicate veins cutting mafic volcanics. The overall stratigraphic position of both the Rajapalot and Rompas mineralization appears the same,occurring near the boundary of the Kivalo and Paakkola Groups.

Mawson's primary target type across the whole Rajapalot-Rompas area is the disseminated Au-Co style, with Mawson's geological team in Finland devoted to uncovering more prospects based on their increased understanding of the host sequence.

The main prospect areas (Figure 7-6) defined by drilling are, from northwest in an anticlockwise direction, Palokas, South Palokas, The Hut, Terry's Hammer, Rumajärvi (also herein referred to as Rumaj), Raja and East Joki (also herein referred to as Joki). Geochemically and spatially associated metals with the gold are As, W, Bi, and Te and with cobalt, Cu, FeO, S, and U.

Two distinct styles of gold mineralization dominate the Rajapalot area. The first, is a variably sulphidic magnesian-iron host, previously referred to internally as “Palokas” style. The magnesian-iron host is most likely an ultramafic volcanic (komatiitic) and occurs within approximately 100 vertical metres of the inferred Kivalo-Paakkola boundary (that is, near the incoming of pelites, calc-pelites and quartz muscovite rocks). A largely retrograde mineral alteration assemblage includes chlorite, Fe-Mg amphiboles (anthophyllite and cummingtonite series), tourmaline and pyrrhotite commonly associated with quartz-veining. Subordinate almandine garnet, magnetite and pyrite occur with bismuth tellurides, scheelite, ilmenite and gold, cobalt pentlandite and cobaltite. Metallurgical testing at Palokas reveals the gold to be non-refractory and 95% pure (with minor Ag and Cu) with excellent recoveries by gravitational circuit with conventional cyanidation and/or flotation. QEMSCAN studies also show that the gold occurs as native grains, found both on grain boundaries and within minerals. Detailed work by Jukka Pekka Ranta of the University of Oulu (plus co-workers) on fluid inclusions and the host rocks to the Fe-Mg mineralization at Palokas indicates weakly saline, methane-bearing fluids at depths as shallow as 5 km and temperatures of approximately 250 ˚C were responsible for deposition of the gold.

The second style of gold-cobalt mineralization at Rajapalot, a potassic-iron (K-Fe) style (formerly referred to internally as “Rumajärvi” type) is characteristically associated with muscovite and / or biotite and chlorite in a diverse range of fabrics. Gold grades of more than 1 g/t Au are associated with pyrrhotite and contained within muscovite-biotite schists, muscovite and biotite-bearing albitic granofels and brecciated, variably micaceous albitic rocks. Magnetite is a common mineral, but not a necessity for anomalous gold grades. The host rocks are grey to white owing to their reduced nature and may be enclosed by light pink to red calcsilicate-bearing albitites. To date, the K-Fe Au-Co mineralization style has been intersected near the muscovite-bearing quartzite at Raja and Rumajärvi, but as other rock types are also mineralized and the clear strong structural control on grade, stratigraphic constraints may locally not be relevant. Spatially associated with the potassic-iron style is subordinate, commonly brecciated, albite-hosted pyrrhotite rich Au-Co mineralization.

The relationship between the Fe-Mg and K-Fe Au-Co systems are not immediately apparent. They do however form in the same general part of the stratigraphic sequence; at South Palokas, the upper mineralized position (upper lenses) correspond geochemically to the Palokas Fe-Mg type, and the main mineralized position at South Palokas is of the K-Fe type. Furthermore, in the deeper parts of Palokas, the base to some of the mineralized intersections are pelitic, more like K-Fe than Fe-Mg type.

The gold, scheelite, bismuth-tellurides and fabrics indicative of a late structural event (at least after the first two isoclinal to tight fold and high temperature events) are all features that are common between the two mineralization styles. Some of the differences in mineral assemblages, such as chlorite, Fe-Mg amphiboles and tourmaline can be related to variations in bulk rock composition and different structural styles reflect competency contrasts from ductile in schistose micaceous rocks to brittle sulphide matrix breccias in albitic granofels. These bulk rock compositional variations are reflected by the host rocks.

Gold mineralization uncovered in boulders and drilling at the Boardwalk prospect (220 m SSE of South Palokas) is a variant on the Palokas style; drilling has not yet revealed the main source of the boulders; however, zones up to 20 m thick of above-detection Au in magnetite-altered mafic rocks ("iron formation") has been intersected. Throughout the entire 10 x 10 km Rajapalot project area, variants of the iron-rich ultramafic rocks (host to Palokas prospect) have been recorded, included in a drill section at South Rompas (some 8 km west of Palokas). New interpretation of the lithogeochemistry with the airborne and ground magnetics is allowing the exploration footprint to broaden.

Exploration for Palokas and Rumajärvi style gold prospects is not restricted to the Rajapalot area. Recognition of the host stratigraphic package (near the boundary of the Kivalo-Paakkola Group boundary) enclosing the 6 km long vein-hosted Rompas Au-U system increases the search space for the pyrrhotite-Au-Co systems to cover Mawson's full permit area. The geochemical characteristics of the ultramafic volcanics and related intrusives are not only present in the southern drill section at South Rompas but have more than 50 km of strike length in Rompas-Rajapalot. It is the interaction of this reactive rock package with late gold-bearing hydrothermal systems driven by ca. 1.8 Ga granitoids, that now form the most highly prospective targets away from the Rajapalot area. The cobalt component of the system is largely stratabound and formed much earlier, most likely from oxidized saline basinal fluids interacting with reduced strata.

8 DEPOSIT TYPES

Within northern Finland are a number of gold deposits of so-called 'atypical metal association' (Molnár 2022.; also "unusual metal association orogenic style"). This unusual metal association is characterized by a suite of redox-critical metals, such as Co, Cu, Mo, U, V, Cr, Fe, S, found more commonly associated with stratabound Kupferschiefer or African copper-belt styles; that is, carried by oxidised basinal fluids and precipitated through reaction with a reductant. The Paleoproterozoic mineralization in the Kuusamo area is also known for this association. A late overprint by tectonic-driven orogenic, or intrusive hydrothermal driven gold systems with the classic Au-W-As-Bi-Te (Hg) association is superimposed on the former stratabound redox-driven metal event. The evidence for these multiple events (see Figure 7-4) is indicated by the long history indicated by the isotopically dated associated minerals (uraninite, hunchunite, galena, altaite monazite, molybdenite).

An earlier gold mineralizing event at Rajapalot is also possible, with evidence at Kittilä of mineralization commencing in the post-orogenic phase of the Lapland-wide regional metamorphism (a more "classic" form of orogenic gold). On a global scale however, tectonism and igneous activity driving ca. 1.8 Ga gold events is relatively common (see Figure 8-1 and Figure 8-2).

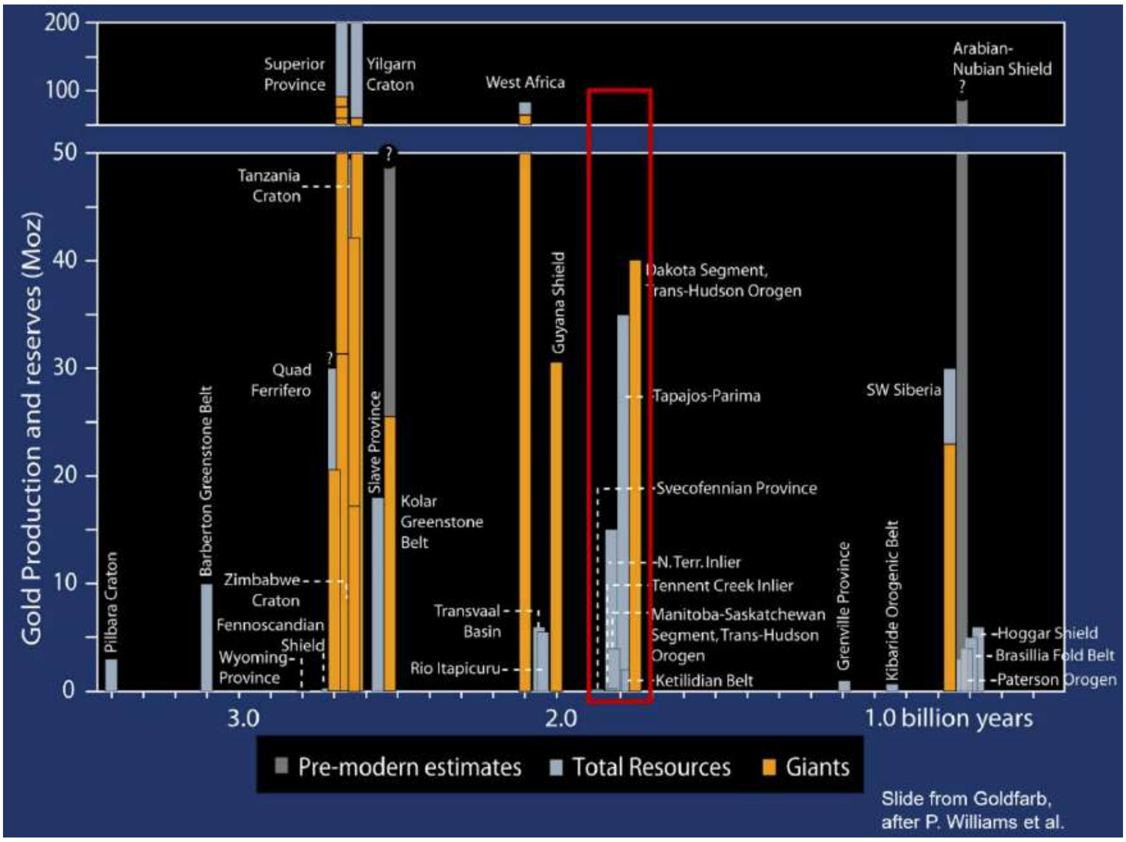


Figure 8-1: Global gold mineralization indicating (inside red rectangle) the abundance of gold produced at ca. 1.8 Ga (Source: Goldfarb R., (n.d))

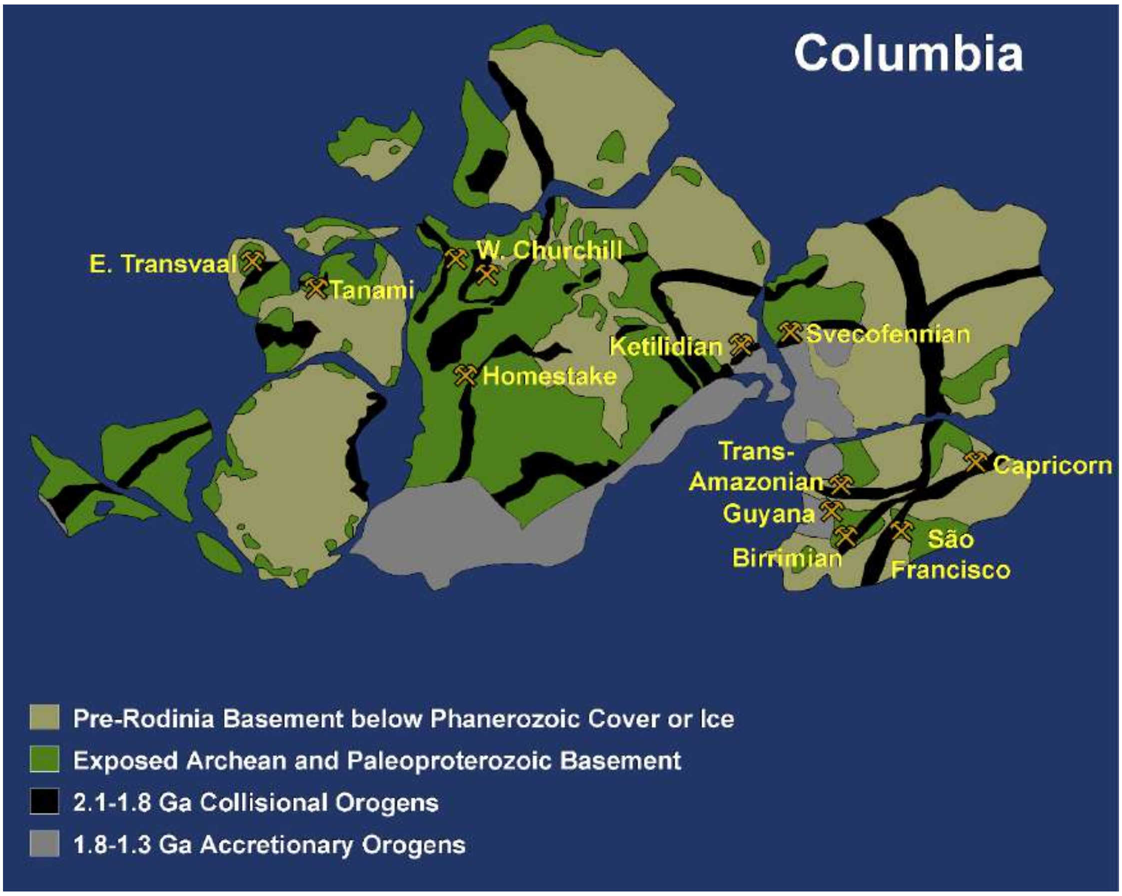


Figure 8-2: Global Paeoproterozoic gold mineralization in the "Columbia" supercontinent (Source: Goldfarb R. (n.d))

The Rajapalot mineralization shows many of the characteristics of larger Proterozoic gold deposits, including examples such as Homestake (USA) and Tanami (Australia), having a predominance of structurally controlled, stratabound occurrences commonly with best grade intercepts associated with fold hinges. Evidence is also emerging suggesting that organic matter also plays an important role in precipitating the high-grade potassic sulphidic gold-cobalt mineralization at Raja. These observations are especially relevant when considering the carbonaceous matter within the Kittilä gold mine operated by Agnico Eagle a system regarded as falling in a similar age bracket of host rocks.

A series of Au-bearing, hydrothermal systems driven by 1.75–1.85 Ga shallowly emplaced granitoids or porphyries is thought to be responsible for the gold prospects discovered to date. Gold mineralization is interpreted to be controlled by a combination of the locations of granitoids and structurally-controlled fluid flow systems either interacting with stratabound iron-magnesium rocks (Palokas type), or in the potassic-iron style where gold precipitation occurred in schistose, breccia and fractured rocks. New unpublished internal research by Mawson indicates that gold event may be associated with minimal sulphide deposition, opening additional opportunities for different gold-mineralized host rocks. Upgrading of the cobalt mineralization through the addition of As to form cobaltite during the deposition of the gold is regarded by Dr. Nick Cook (pers. comm.) of Mawson as the most likely control for the distribution of cobaltite, linnaeite and cobaltian iron sulphides at Rajapalot.

The mineralization at Rajapalot is a stratabound, disseminated, hydrothermal, usually sulphide-associated, and structurally controlled gold-cobalt type. These are generally stratabound pyrrhotite-bearing iron-potassic (K-Fe) or iron+/-magnesium (Fe-Mg) systems with a Cu, Mo, W, B, Bi-Te (Se) association. There are no reasons to assume across the broader prospect area a lack of low-S "simple" Au-W-As gold systems.

9 EXPLORATION

9.1 Geological Mapping and Sampling

Mapping and sampling in the Rajapalot area commenced in 2011 and was increased following the discovery of the Palokas mineralized outcrop in 2012 (first announced by Mawson on September 04, 2012). This outcrop comprises a low rise on the eastern side of a swamp with iron oxide-stained rocks comprising largely randomly oriented chlorite, anthophyllite and pyrrhotite. Initial samples from this outcrop contained up to 80 g/t Au. This outcrop lies approximately 7 km to the east of the earlier explored Rompas vein trend.

Discovery grab samples at Rumajärvi and also a mafic ridgeline also known as “Joki” (separate prospect from Joki East resource area) were also made by Mawson in 2012.

Over 160 boulders and outcrops with >0.1 g/t Au have been uncovered within a 4 x 3 km area with gold grades ranging from 0.1 g/t to 3,870 g/t, with an average of 74.9 g/t Au and a median of 0.71 g/t Au. These samples are selective by nature and will not represent average grades on the property. As thin till covers more than 99% of the Project area, very few in situ outcrops have been found. Large swampy areas are likely underlain by softer rocks and this has become evident as drilling progresses; mineralization is dominated by micaceous rocks in drilling but by isotropic hard albite-quartz rocks in boulders. Total numbers of samples of rock outcrops and boulders by permit are shown in Table 9-1.

Table 9-1: All samples in Mawson database taken across the entire Rajapalot-Rompas area

Exploration area	Number of samples
Rompas	329
Raja	382
Kaitajärvi	50
Uusi Rumavuoma	161
Mäntylaennokka	54
Kultamaat	105
Karsimaat	31
Männistö	748
Kuusivaara	229
Korkiakoivikko	61
Hirvimaa	551
Kairamaat 2/3	516
Petäjäskoski	261
Petäjävaara	5
Takanenvuoma	20
Regional	428
Total	**3,931**

Quaternary mapping of till by experts from the GTK (in particular Pertti Sarala) combined with expert in-house knowledge of Finnish Mawson geologists has allowed determination of likely transport direction and distance of many of the mineralized boulders. Of note is that areas of mineralized boulders whose shape and size indicates short travel distances are still to be sourced in the bedrock.

Project-scale sampling by BOT percussion drilling is discussed in Section 10.

9.2 Geophysics

A series of airborne (VTEM*plus*) and ground geophysical surveys have been conducted since 2013 to locate the conductive, chargeable and magnetic mineralization at Rajapalot. More recent work indicates that a combination of ground magnetic surveys, electromagnetics (both airborne and ground) and IP-resistivity methods are the most promising for locating sulphidic gold-cobalt mineralization. The highly conductive nature of the sulphidic host also makes mise-a-la-masse and important tool for tracing and determining continuity between drillhole intersections and the location of near-surface mineralization, with the ever-present thin glacial till cover. Much of the southeastern portion of Kairamaat 2/3 permit and more than 40% of Hirvimaa permit is now also covered by gradient array IP/chargeability surveys.

Detailed ground magnetic surveys at line spacings between 100 m and 15 m have been completed during 2014 to 2018. The testing has indicated that 25 m line spacing is optimum for discovery and geological interpretation. Geological, primarily structural interpretation of the ground magnetic data indicates a complexly refolded and faulted sequence, but still including distinctive and traceable units. Additional magnetic surveys to infill surveys to 25 m have now been completed across the most prospective portions of Rajapalot.

Magnetic pyrrhotite associated with gold-cobalt mineralization typically shows reverse remanent magnetism (RRM). Thus, combined RRM-conductive-chargeable anomalies usually represent near-surface sulphides. The coincidence of the three geophysical properties was used to successfully locate the mineralization at Raja and The Hut, and corresponding anomalies at Palokas, South Palokas and Terry's Hammer indicate the effectiveness of the programs.

Fixed-loop transient electromagnetic corresponds closely with the Au-Co mineralized wireframes at Palokas, South Palokas and Raja (Figure 4-1) and provide a robust method to target the down-plunge extent to the conductive sulphidic rocks at all prospect areas. Down-hole electromagnetic surveys have also been conducted in drillholes where indications are present of proximity to sulphidic hosts to gold-cobalt mineralization. Mise á la masse surveys track the continuity and map the surface projection of sulphides from deep drilling at Raja, South Palokas and Palokas.

A detailed ground-based 1,070 station gravity survey over 41,700 hectares over the Rompas and Rajapalot project areas was commenced in November 2019 and was completed early in 2020. The survey was divided in two parts: firstly, a detailed survey on a 400 m grid; and secondly, a semi-regional 1,300 to 1,500 m grid outside the detailed area. Results indicate significant low and high density anomalies probably represent felsic and mafic-ultramafic intrusives, respectively. These anomalies remain largely untested.

9.2.1 Exploration discussion

The recent recognition of the stratigraphic host position of the Rajapalot mineralization and the equivalents in the Rompas area has allowed Mawson to rethink the exploration strategy across its permits. New gravity data, combined with regional BOT, electromagnetics and IP-resistivity is further focussing the targeting. It is recommended that the surface sampling database be re-visited to combine it more effectively with the new geophysics and understanding of the stratigraphic location Rajapalot resources.

10 DRILLING

10.1 Drill Program Summary

From mid-2011 to June 2021, Mawson has drilled 84,085 m in 544 diamond drillholes at Rajapalot with an average length of 155 m (Table 10-1). Mawson has used up to six drill rigs in any one season. Of the 544 drillholes, 330 have been used in the Inferred Mineral Resource estimate (Section 14). No holes have been drilled into the direct project area since June 2021.

Table 10-1: Total Rajapalot drill programs to 30 June 2021

Drill program	Number of holes	Years	Drilled (m)	Cumulative average hole length (m)	Core diameter	Drill company
PAL0001-PAL0007	8	2013	757	95	NQ=47.6 mm, HQ=63.5 mm	Arctic Drilling Company OY (ADC)
PRAJ0001-PRAJ0120	120	2013-16	3,431	33	EW=25.2 mm	Mawson
LD0001-LD0120	120	2014	874	20	BQ=36.4 mm	Ludvika Borrteknik AB
PAL0008-PAL0025	18	2015-16	3,290	31	NTW=56.0 mm	Energold
PAL0026-PAL0082	57	2017	11,139	60	NQ2=50.7 mm, NTW=56.0 mm	ADC, MSJ Drilling, KATI Oy
PAL0083-PAL0147	65	2018	14,743	88	NQ2=50.7 mm, WL76=57.7 mm	ADC, MK Core Drilling Oy, KATI Oy
PAL0148-PAL0201D	44	2019*	16,851	118	NQ2=50.7 mm	ADC, MK Core Drilling Oy, KATI Oy
PAL0202-PAL0236	36	2020*	14,000	139	NQ2=50.7 mm	ADC, KATI Qy
PAL0235-PAL0311	76	2020-21	19,447	155	NQ2=50.7 mm	KATI Oy, MK Core, NTK
Total	**544**		**84,532**			

10.2 Drilling Procedure and Progression

Hand-portable drill rigs (Winkie and JKS4M; PRAJ drillholes PRAJ0001-PRAJ00120) were operated by Mawson and contract staff and obtained diamond drill core of 25.2 mm diameter. The ‘PRAJ’ series of holes were drilled from 2013 to 2016 with EW (25.2 mm) diameter core. The ‘Ludvika’ series of short diamond drillhole tails were completed during 2014 by Ludvika Borrteknik AB with a GM100 drill rig. The Ludvika drillholes were short, averaging only 7.3 m depth, with a core diameter of 36.4 mm. The ‘PAL’ series of drillholes were drilled from 2013 onwards. Drillholes PAL0001-PAL0007 were completed in 2013 by ADC with a core diameter of 47.6 or 63.5 mm. Later programs (drillholes PAL0008-PAL0147) were operated with larger drill rigs (NQ2, NTW and WL-76 size) provided by multiple contractors.

Energold completed drillholes PAL0008-PAL0025 drilled during the winter of 2015 to 2016. During the winter 2016, 17 drillholes, PAL0026-PAL0084, were completed. Four diamond drill rigs were utilized (two diamond drill rigs (K1 & K2) from the ADC, one diamond drill rig from KATI OY and a single diamond drill rig from Mason and St John (MSJ)). Water recirculation and drill cuttings collection systems were used. Core diameter was NQ2 (50.6 mm), NTW (56.0 mm) and WL-76 (57.5 mm). During the winter of 2017-18, up to six drill rigs with water recirculation and cuttings collection systems were utilized (two diamond drill rigs from MK Core Drilling OY; two from ADC; two from KATI OY). Drill seasons 2019 and 2020 continued the NQ2 diamond drill programs utilizing drill companies employed at site in 2018.

The 2020-21 drill season commenced in September 2020 at Joki East prospect before moving to the prospects within Kairamaat 2/3 permit. MK Core Drilling OY and two drill rigs from KATI OY were joined by Nivalan Timanttikairaus (NTK) to complete the 19.7 km of NQ2 diameter diamond coring for the season.

The programs have progressed well with a trend of increasing mineralized intervals each season (see Figure 10-1 and Table 10-1).

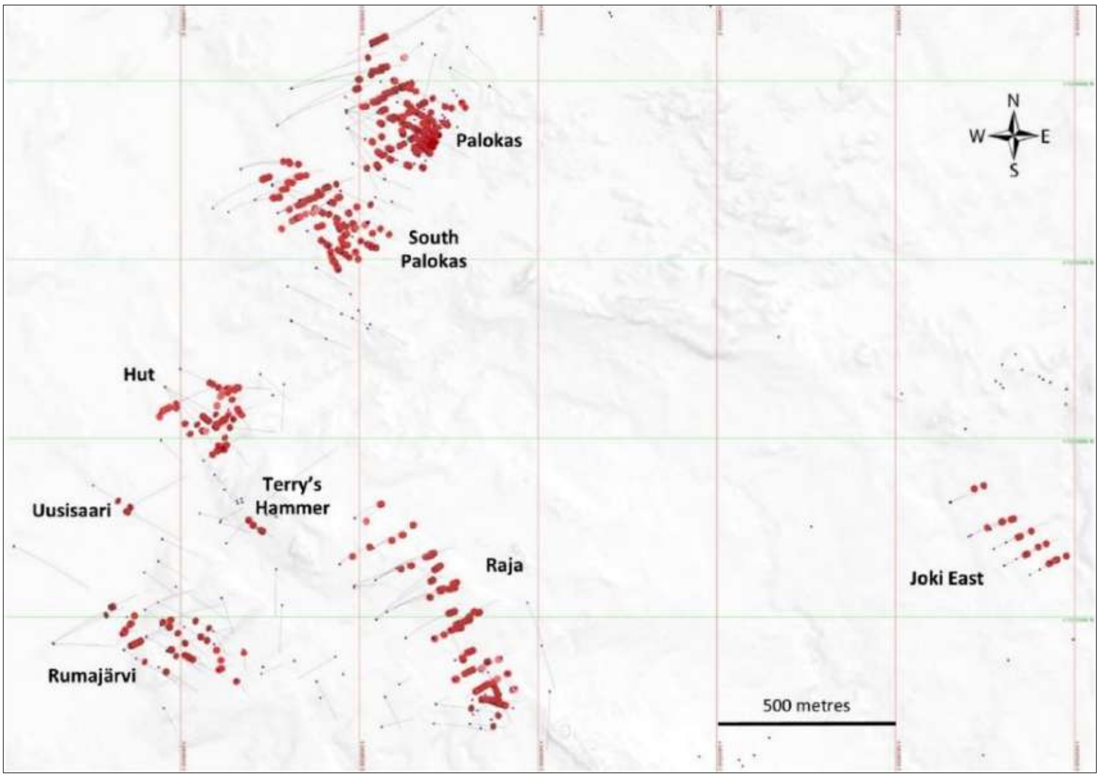


Figure 10-1: Map of Rajapalot project prospects over Lidar image. Red disks correspond to drillholes / wireframes intersecting

Table 10-2: Resource definition drill progress 2013 - 2021

Drill season		Sum of mineralized intervals	# holes with mineralized intersections	Total drilling of mineralized intersections	# holes defining resource	# metres defining resource
2013-14	Season_1	288.9	19	467.9	28	573.7
2014-15	Season_2	165.5	19	782.4	40	1,625.0
2015-16	Season_3	316.2	17	2,714.8	23	3,473.2
2017	Season_4	261.2	19	4,042.6	42	8,302.0
2018	Season_5	461.8	27	7,084.5	45	10,132.6
2019	Season_6	389.6	27	10,341.1	43	14,861.6
2019-20	Season_7	628.8	32	13,300.8	36	14,798.8
2020-21	Season_8	808.1	60	16,592.7	73	19,082.4
Grand Total		**3,320.2**	**220**	**55,326.6**	**330**	**72,849.3**

Drill rigs are actively supervised by Mawson geologists to ensure efficient operation, drillhole commencement and termination. Core is delivered daily to the Rovaniemi core logging facility (co-located with the office), quick logged for lithology and mineralization and entered immediately into Leapfrog Geo software to manage the program.

All drill core of NQ size and larger is oriented by the drilling company. The orientation line is marked on the base of the drill core. Mawson confirms continuity of drill core orientation lines by joining core on V-rails on all of the logging tables (example shown in Figure 10-2). The blue dashed orientation line indicates moderate certainty in orientation (a solid red line is used for good orientation). Sampling intervals, blanks after a sample interval with VG and ¼ core duplicates are all evident in the red sampling notes on the core tray. The blue metre marks on the core and red sampling lines are also present.

Mawson collects the following information on diamond drill core (on 1 m intervals unless otherwise stated):

- magnetic susceptibility;
- resistivity;
- core recovery;
- Rock Quality Designation (RQD);
- radiometrics (total counts);
- geologic logs, made at the scale appropriate for rock type variation;
- structural logging with measurements made using the standard alpha/beta technique, or by direct measurement using a drill core orientation holder;
- alteration logging;
- key mineral logging, including sulphides, silicates, oxides, visible gold (VG);
- veins, with infill minerals and textures ;
- sampling of core is generally at 1 metre intervals in mineralized rocks and 2 metres in interpreted barren rocks where sampling of drill core is always such that 1/2 core is taken from the same side, leaving the trace of the orientation line in the drill tray;
- specific gravity measurements are made as required (3,345 measurements to date); and
- point load tests total 658.

Data are recorded directly into Mawson's MX Deposit database (link to MX Deposit website).

Figure 10-2: Example of marked up drill core tray prior to cutting (PAL0093, core tray 57)

10.3 Core Recovery

Diamond drill core recovery since 2013 at Rajapalot has been excellent. In a total of 78,003 measurements, the mean of core recovery is 99.87%. There are no changes to the core recoveries with either grade (using AuEq), or depth (Figure 10-3 and Figure 10-4, respectively). Dots are coloured by AuEq grade in Figure 10-3 and >99% of samples have 100% recovery. The core recovery data are collected by Mawson geotechnical staff at the Rovaniemi logging facility prior to geological logging.

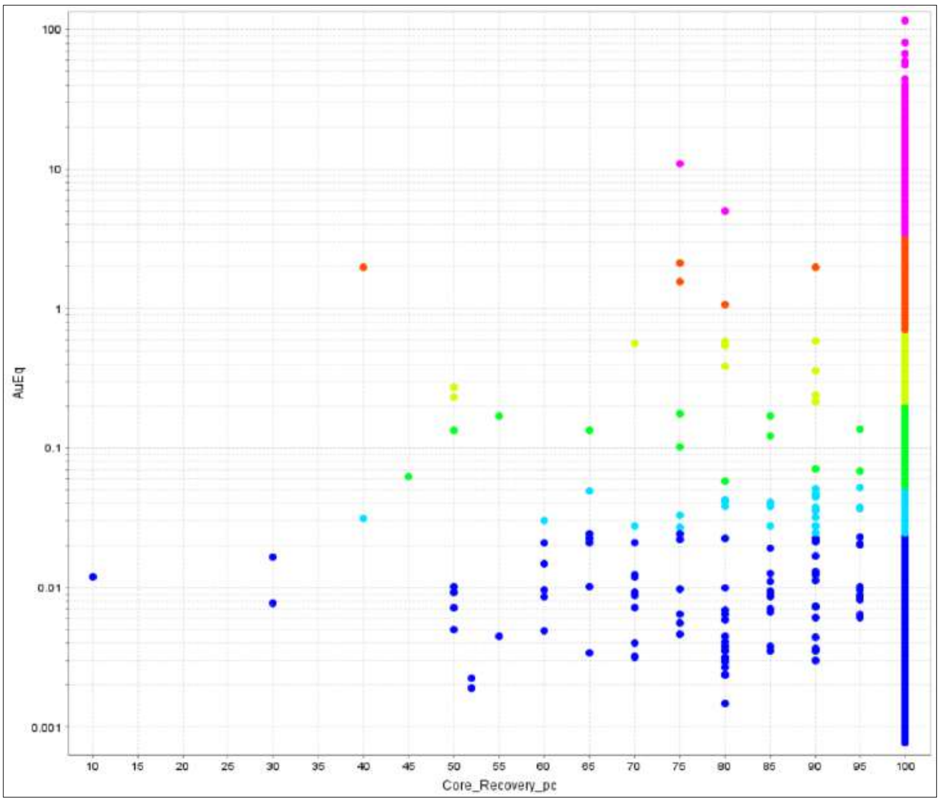


Figure 10-3: Graph indicating the lack of correlation of AuEq content (g/t) against core recovery with dots coloured by AuEq grade

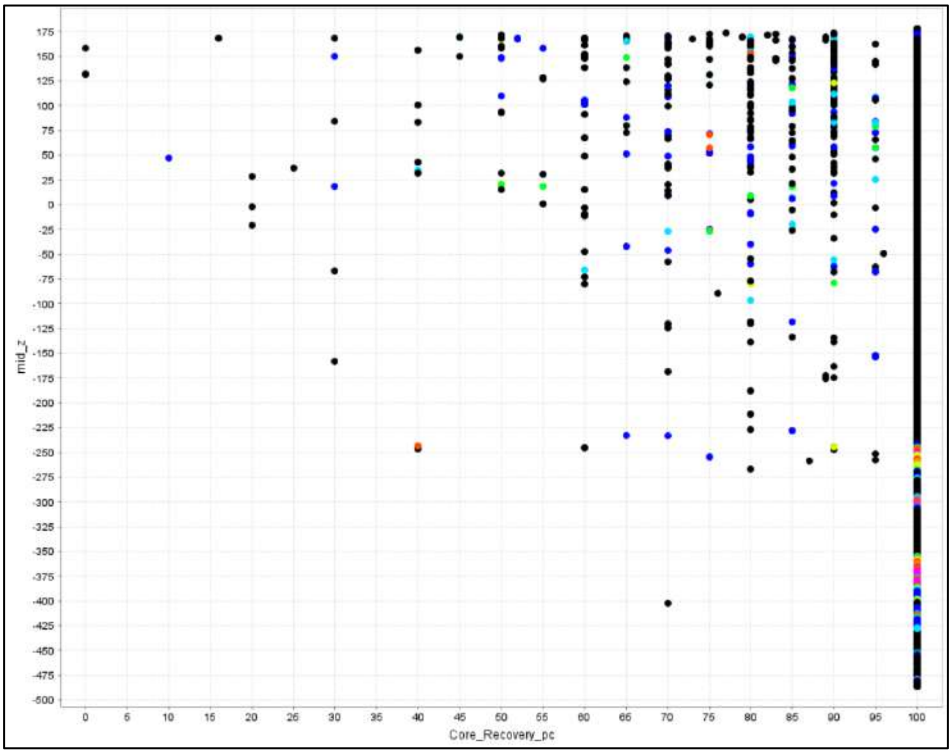


Figure 10-4: Graph showing the lack of correlation of core recovery (x axis) with drill depth (y axis "mid_z")

10.4 Other Drilling Information

10.4.1 Drillhole collar coordinates

Mawson hires a differential GPS for each drill season to accurately locate each collar position.

10.4.2 Drillhole surveying

Diamond drilling companies have used a variety of down-hole surveying tools. In addition, Mawson has checked down-hole surveys independently using continuous survey gyroscopic tools. Checking for closure on up- and down-hole surveys is part of the validation procedure. Each drill collar in the last three drill seasons has been checked for starting orientation with a north-seeking gyro.

10.4.3 Drill collar maintenance

Each drillhole is cased at the surface, cut off at or just below ground level and capped. Drill collars and sites are checked after the snow melt for water leakage (approximately 1 to 2% of drillholes require rubber plugs to be inserted to prevent water flow). Each site is cleaned of any drill cuttings if required (minimal, as these are collected during drilling).

10.4.4 Logging notes, turn-around times

Mawson staff completed logging the 14.7 km of drill core within two months of completion of the drill season. Logging is prioritized following the daily quick logging of drill core and drill management is improved by a 5 to 7 day standard turn-around for gold assays.

10.4.5 Drillhole prospect maps

Figure 10-5 to Figure 10-10 show prospect maps with drill collar information for each prospect. Appendices B and C provide drill collar and intersection information, respectively.

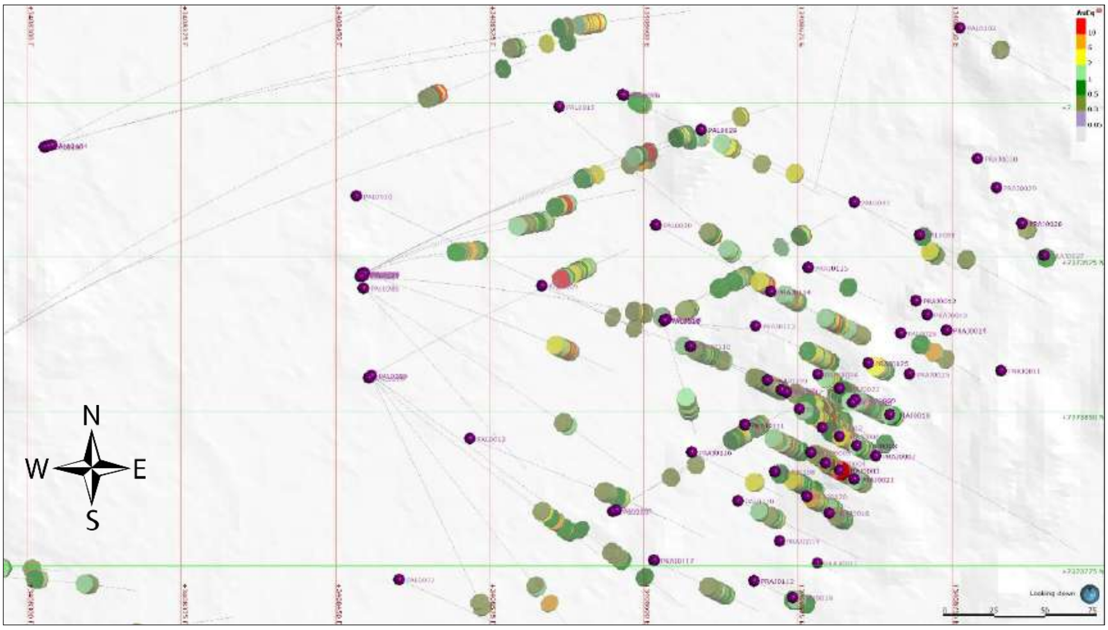


Figure 10-5: Drillhole map for Palokas prospect with collars as purple balls, and gold plotted on traces above 0.3 g/t Au; South Palokas drill string traces visible in SW

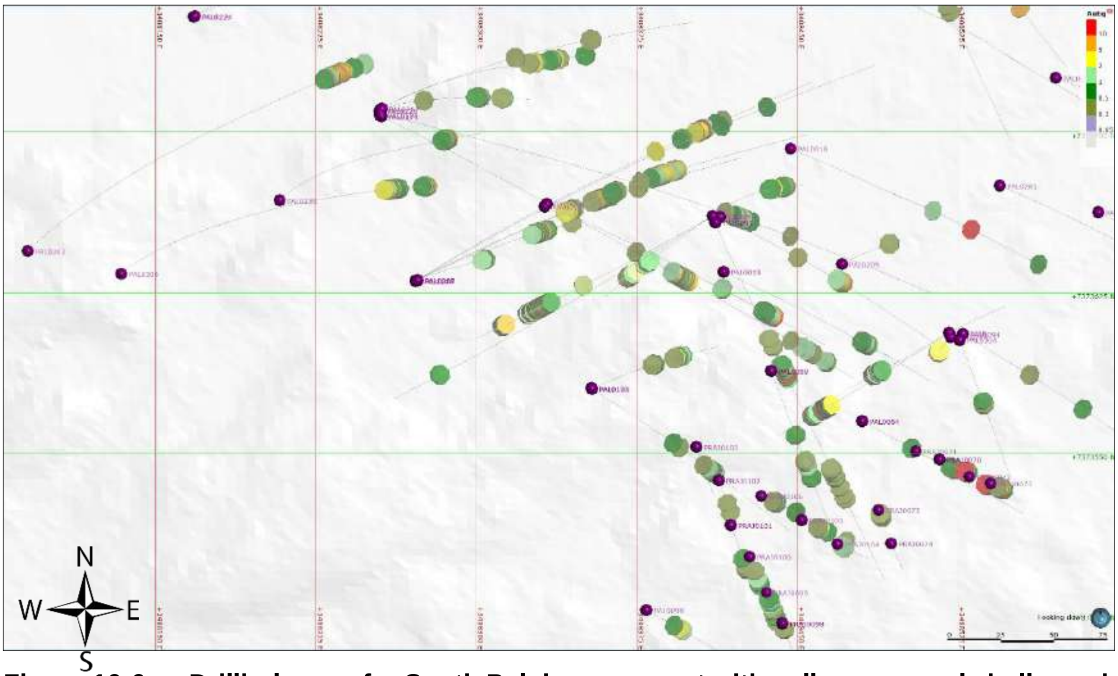


Figure 10-6: Drillhole map for South Palokas prospect with collars as purple balls, and gold plotted on traces above 0.3 g/t Au; Palokas prospect drill traces are present in NE

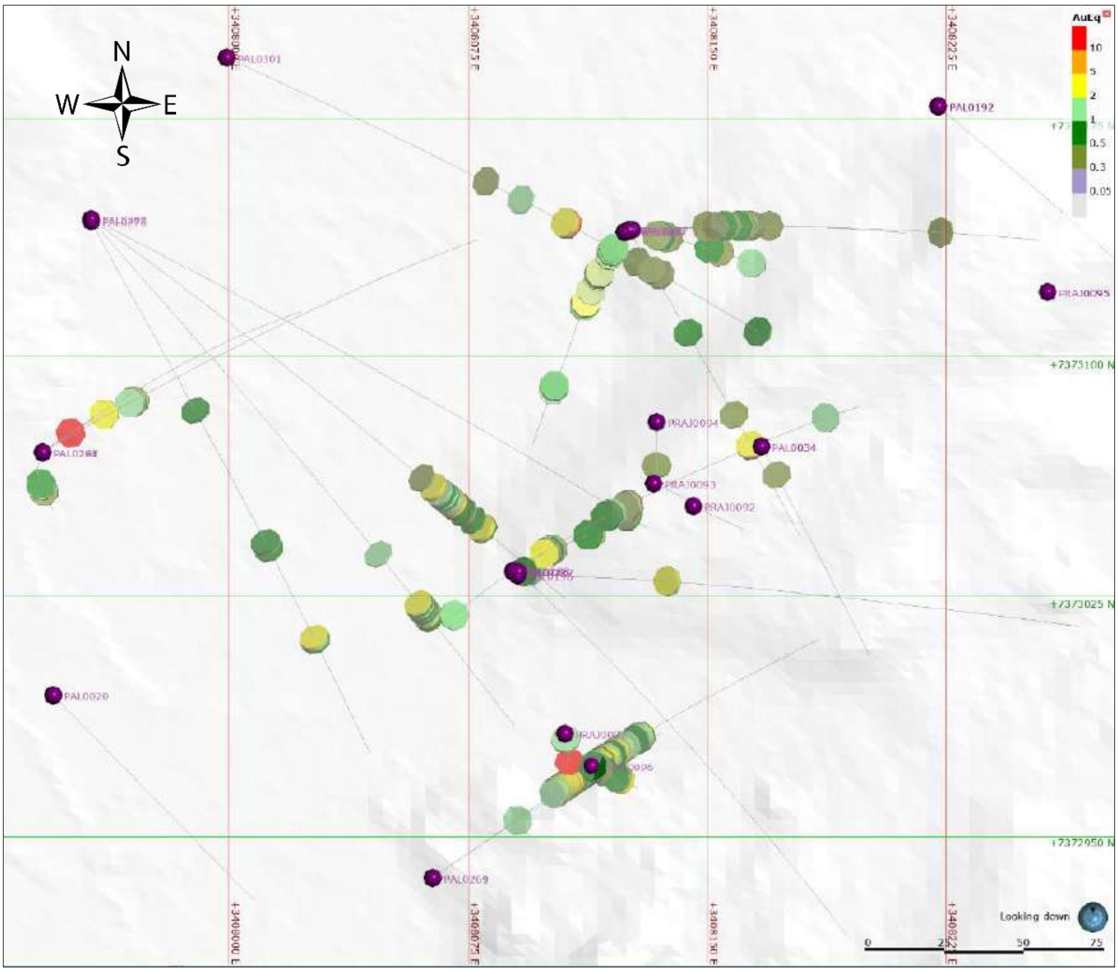


Figure 10-7: Drillhole map for The Hut prospect with collars as purple balls, and gold plotted on traces above 0.3 g/t Au

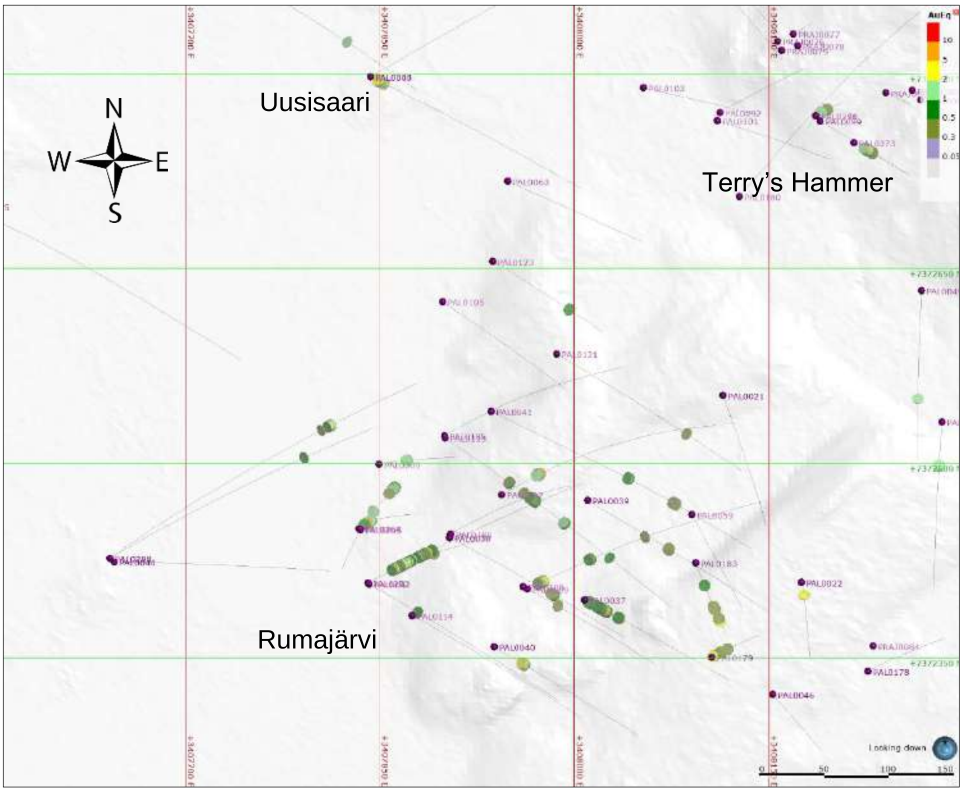


Figure 10-8: Drillhole map for Rumajärvi, Terry's Hammer and Uusisaari prospects with collars as purple balls, and gold plotted on traces above 0.3 g/t Au

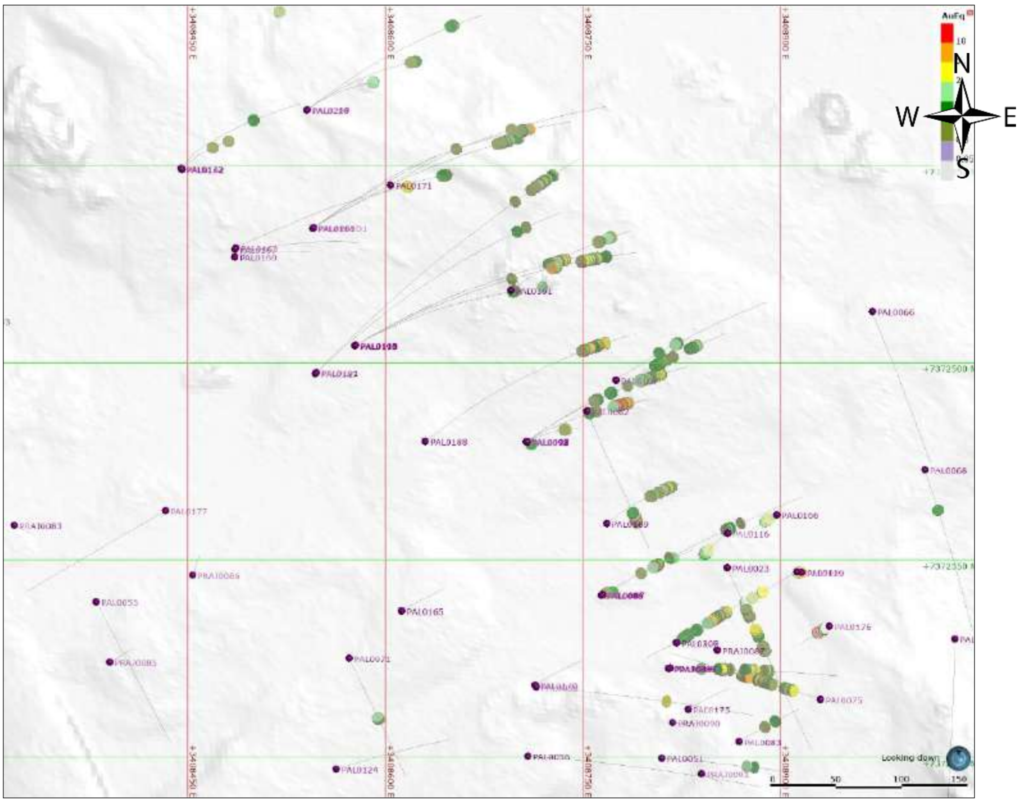


Figure 10-9: Drillhole map for Raja prospect with collars as purple balls, and gold plotted on traces above 0.3 g/t Au

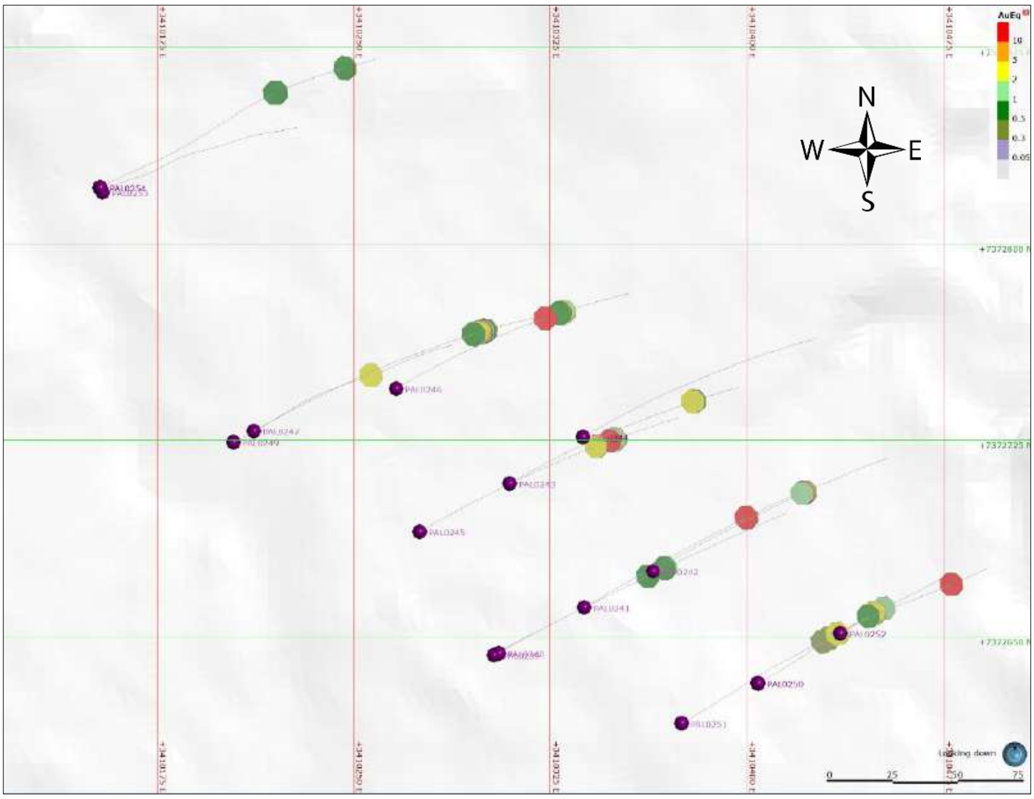


Figure 10-10: Drillhole map for Joki East prospect with collars as purple balls, and gold plotted on traces above 0.3 g/t Au

10.5 Drilling Discussion

Drilling in sections across the linear to moderately oblate mineralized lenses has resulted in reasonable control on the geometry of the host rocks and the mineralization itself. With dips and plunges ranging from just over 20° to 55° the true thicknesses vary, but in general are better than 80% of the drill intersection thickness. Drill orientations across the prospects have been varied appropriately to return the best approximation to true thickness.

Drilling in swampy ground and Natura 2000 designated areas is possible in winter months with good frozen ground and snow cover, and year-round drilling is possible on drier ground.

Down-hole orientation variation has been carefully managed, as most drillholes intersect close to the desired target. This may become more difficult to manage as the depth increases below 500 m.

The QP finds that drill core management, sampling, orientation and core loss are at industry best practice levels and will not materially affect the assay results reported, or the estimation of the Mineral Resource.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

Sample preparation, analytical work, and security of samples have been reviewed by the QP and found to follow industry best-practice guidelines. Mawson reviews the quality of their assays regularly and holds internal Quality Assurance Quality Control (QAQC) reports and data covering all drilling at Rajapalot.

11.1 Core and Sample Logging and Preparation for Assay

Core recoveries average over 99.8% as discussed in Section 10.3. Core logging is detailed and covers up to 13 criteria. Continuous logging is achieved for geological features, RQD, recovery, radiometrics and alteration. Key minerals, structural features, veining, density measurements and point loading testing are all conducted on an as-needs basis.

On completion of key logging, in particular, detailed geological logging, sampling intervals are determined, recorded on the core trays and incorporated into the MX Deposit database. Included in the sampling stage is the introduction of standards, blanks after visible gold and ¼ core duplicates in mineralized zones. An example of the sampling process for assays is evident in Figure 10.2 on drillhole PAL0093. Wet and dry core photography takes place after the sample numbers are written on the core trays. Samples are always taken from the same side of the drill core, retaining the orientation line in the core tray.

Drill core is cut at the GTK facility in Lepikontie, Rovaniemi (all core has been cut by the same staff at this facility since 2013). Mawson regards the core cutting work as exemplary, having never discovered an error during the core cutting process. Assay samples are bagged by GTK staff along with standards and duplicates and shipped via commercial transport directly to assay laboratories.

11.2 Security of Drill Core and Shipping

Drill core is collected from the drill rig by employees of drilling contractors and transported to a storage area next to Mawson's field hut. In winter a sled behind a skidoo is used and when there is no snow, a 4 or 6 wheel all-terrain vehicle. A contractor collects the drill core once or twice each day and drives it to Rovaniemi to Mawson's core logging and office facilities at Ahjotie 7.

Once logged and marked for cutting the core boxes are placed on pallets and transported a few kilometres to GTK's cutting facility. Stored wooden core boxes are clearly marked, are clean and can be readily recovered later from either cold or warm facilities. Mawson employees do not handle the drill core or samples after cutting as they are shipped directly to the laboratory or laboratory preparation facility, typically CRS in Kempele, or to ALS in Sodankylä.

Identification of samples is clear as bags are numbered with unique sample numbers and water-resistant numbered tags are placed inside the bags. Standards are treated in the same way, although some blanks are prepared by Mawson and delivered to GTK already mixed with a known concentration of blank ("sauna" rock = olivine diorite) and commercial certified standard.

11.3 Certified Standards, Blanks and Duplicates

A total of 31,943 entries with sample numbers are stored in the MX Deposit database for drillholes in the PAL and PRAJ series (derived from Palokas and Portable Rajapalot). Standard assay samples total 29,155 in addition to 2,380 standards (recorded as control samples) and 407 duplicates (9% of total samples are control samples). Within the group of standards, blanks of various types total 585, Ore Research & Exploration Pty Ltd Standards (OREAS) commercial certified standards total 764 and there are 987 spiked standards (description follows later).

Duplicates are achieved by quartering the core and recording the same interval as FDUP in MX Deposit. The laboratories also insert standards and some duplicates into the analytical runs.

11.4 Laboratory Procedures and Analytical Methods

The following four laboratories independent of Mawson have been used in the preparation and analysis of samples.

- CRS Laboratory, Kempele, Finland. Sample preparation (pulps) for MS Analytical and sample preparation and PAL1000 assay technique for gold. Laboratory certification testing Lab T342 (FINAS); SFS-EN ISO/IEC 17025:2017.
- MS Analytical Laboratory, Langley, Canada. Multi-element analyses and fire assays for gold. Laboratory certification ISO 9001:2015.
- ALS Laboratory, Piteå, Sweden; Sodankylä, Finland; Loughrea, Ireland; Vancouver, Canada. Fire assays for Au and multi-element analyses. All ALS geochemical hub laboratories are accredited to ISO/IEC 17025:2017.
- Labtium Laboratory, Rovaniemi, Finland. Fire assay and multi-element analyses. Laboratory certification (this is now Eurofins Labtium; accredited according to ISO/IEC 17025 by FINAS, the Finnish accreditation service. Testing laboratory T025).

Early drillholes in the PRAJ series were analysed using the Labtium facility in Rovaniemi. After some 20 drillholes the quality of the duplicates declined and many samples were re-assayed using the ALS Laboratory in Piteå, Sweden. Once the Sodankylä ALS facility opened, Mawson sent samples there as required. CRS became the primary provider of gold assays for Mawson late in 2015 who then partnered with MS Analytical to provide multi-element assays (formerly nearly all multi-element analyses were conducted by ALS). Over 24,000 gold assays by the PAL1000 technique have been completed by CRS.

Mawson's standard sampling procedures and descriptions of the check assay procedures are shown in Figure 11-1 to Figure 11-3.

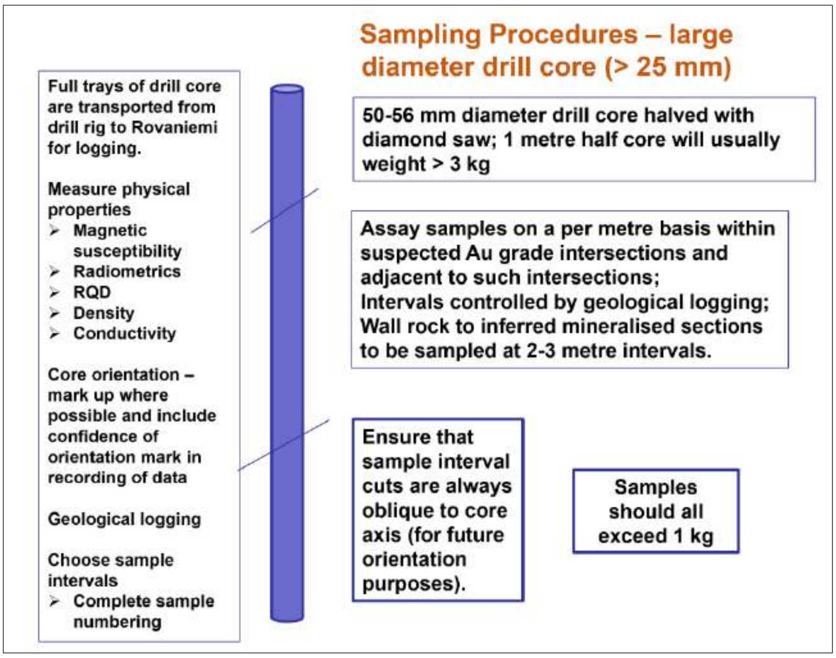


Figure 11-1: Mawson's sampling procedures

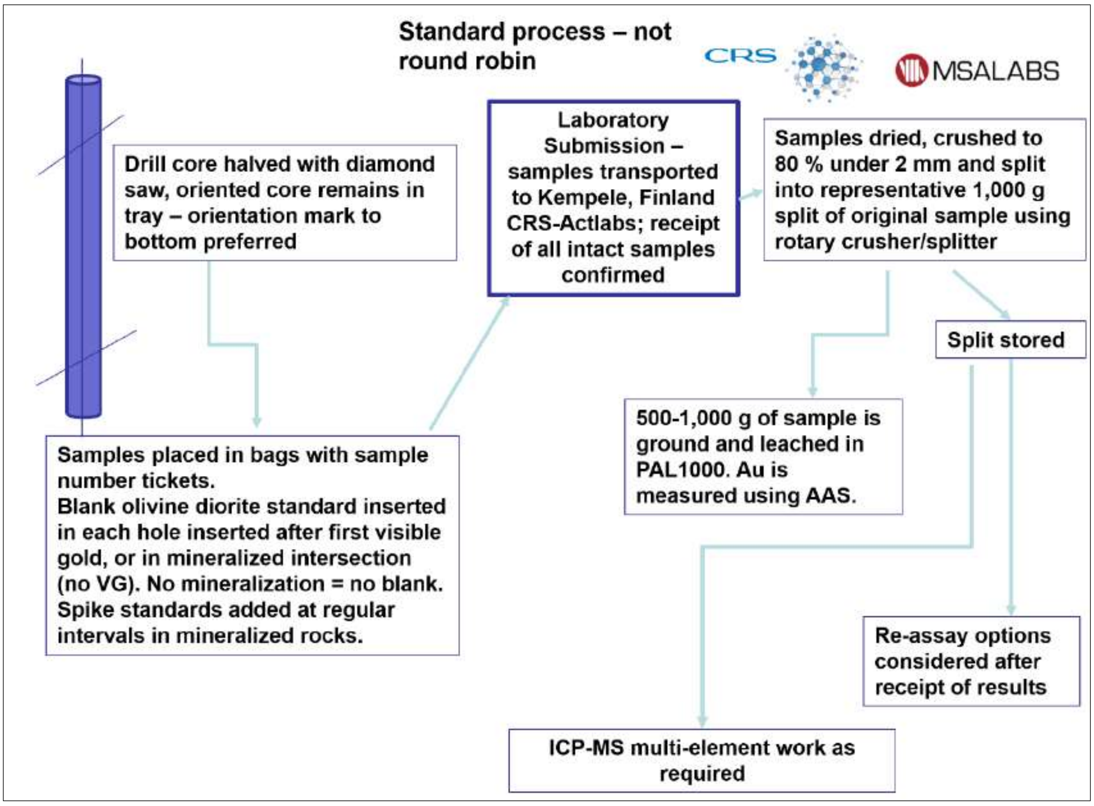


Figure 11-2: PAL1000 sampling process, blanks and standards

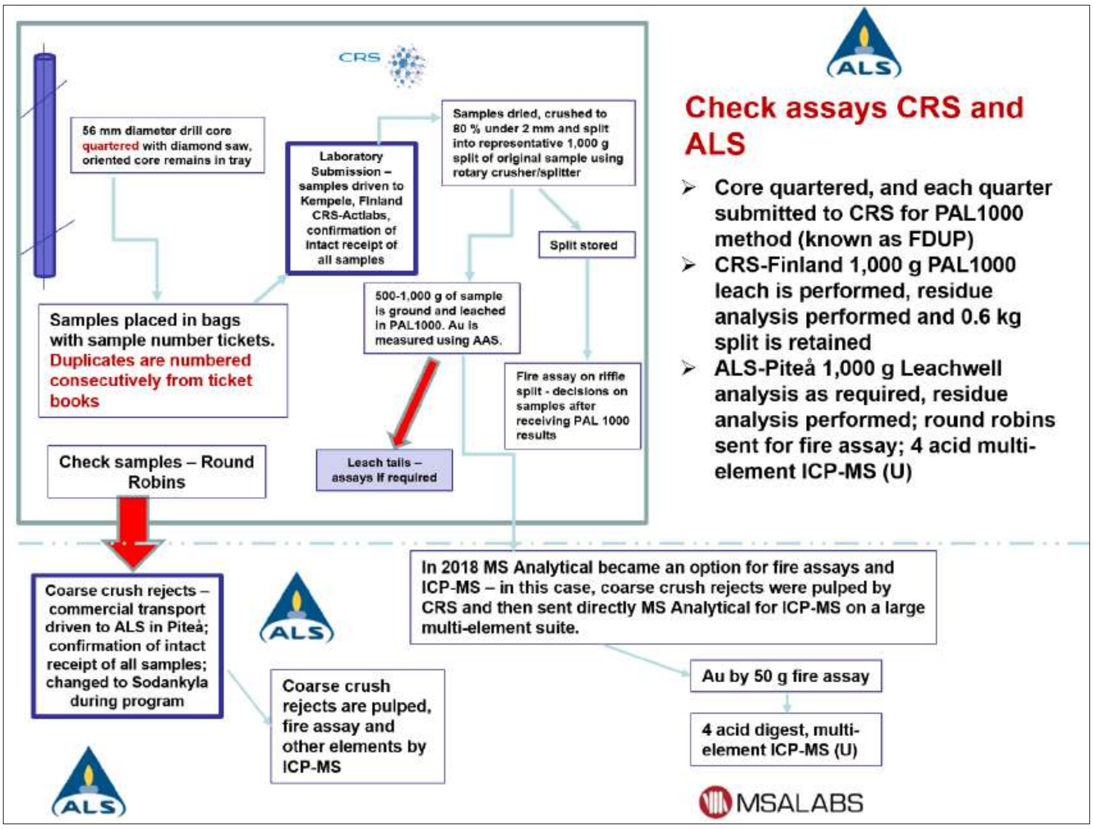


Figure 11-3: Multi-element work, check assays, round robins and considerations for checking leach tails

Gold has been analysed by two different main methods. The PAL1000 method in CRS Laboratory in Kempele, Finland (Figure 11-4) and fire assay method Au-ICP22 is done at ALS Laboratory. Multi-element assays are mainly done by method ME-MS61 in ALS Laboratory or by method IMS-230 in MS Analytical Laboratory.

CRS laboratory method PAL1000 involves grinding the sample in steel pots with abrasive media in the presence of cyanide, followed by measuring the gold in solution with flame AAS equipment. PAL1000 detection limit for gold is 0.05 ppm. In order to improve the detection limit of the PAL1000 technique from 0.05 g/t to 0.01 g/t Au for a 1 kg sample, gold concentration using the DiBK (di-isobutyle ketone) extraction method was also used (Figure 11-5).

ALS Laboratory method Au-ICP22 is a fire assay and inductively coupled plasma atomic emission spectroscopy (ICP-AES) method with 50 g subsample. Method detection limit for gold is 0.001 ppm. ALS fire assay method Au-ICP21 with a 30 g subsample was used for some holes during the Rajapalot early exploration phases. Gold detection limit for Au-ICP21 is 0.001 ppm. Upper limit of methods Au-ICP22 and Au-ICP21 is 10 ppm and over-limit samples were re-assayed by Au-GRA method which is fire assay with gravimetric finish (50 g subsample).

Labtium laboratory gold assay methods 705P and 704P are fire assays with 50 g and 25 g subsamples, respectively, and ICP-OES finish. Gold detection limits for 705P are 0.005 ppm and 704P, 0.01 ppm. Method 704P was used only if the sample amount was too small for 705P or the 705P assay failed.

Some single samples were assayed for gold at ActLabs by methods FA-AA (fire assay with AA finish) and its over limit method FA-GRA.

ALS multi-element method ME-MS61 is a four-acid digest with ICP-MS finish (0.25 g subsample). 48 elements are reported. Part of the multi-element assays were done by ME-MS61U which is the same method as ME-MS61 but optimized for uranium with specific laboratory certified reference materials (CRM) for superior quality.

MS Analytical multi-element method IMS-230 is a four-acid digest followed by ICP-AES/MS finish (0.2 g subsample). The IMS-230 method records 48 elements.

Single multi-element assays were also completed at Labtium, by a whole rock XRF method (175X) using pressed powder pellet.

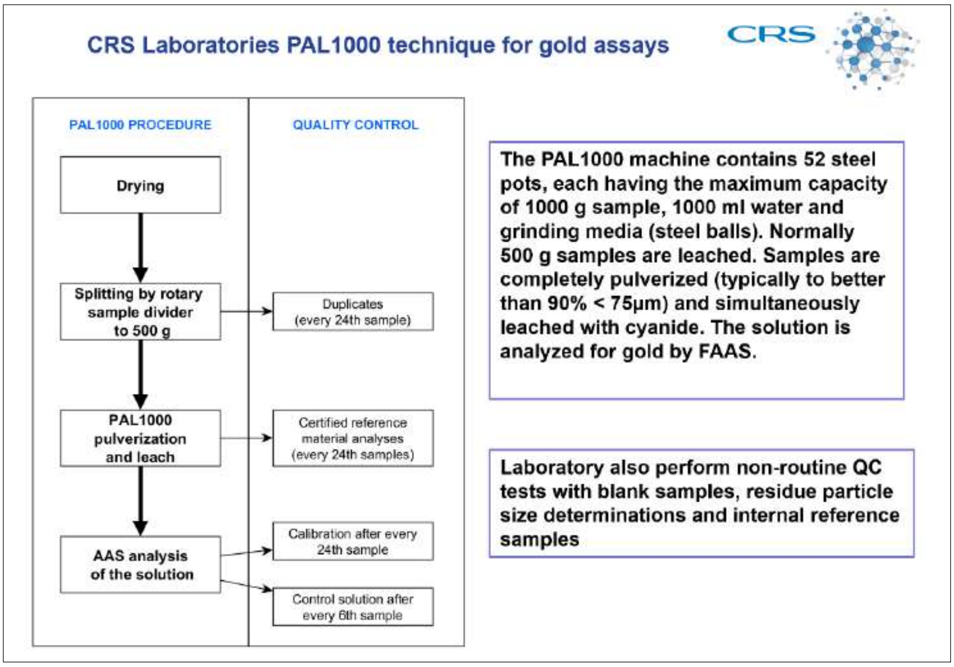


Figure 11-4: Details of the PAL1000 technique provided by CRS

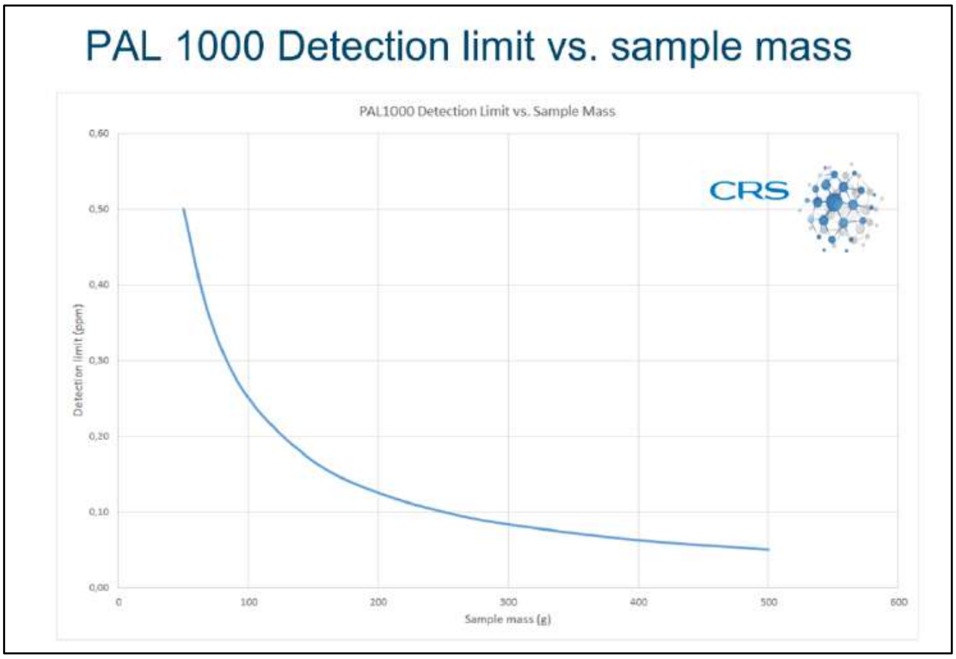


Figure 11-5: Details of detection limit for sample sizes (provided by CRS)

11.5 Quality Control Data

11.5.1 Introduction

Aqua regia digests by ALS were trialled on 30 samples to ensure that the cobalt assays represented cobalt held in sulphide, rather than silicates. All cobalt samples with >50 ppm cobalt were confirmed with sulphide minerals as the host.

11.5.2 Quality control data 2013-2019

This section covers all drilling in the PAL series through to and including PAL201D. Drillhole series PRAJ and LD are also presented (LD is not applicable to this resource, but these samples were being generated during the same period that other PRAJ samples were submitted to the laboratories). These quality control data are separated into two sections, as from late 2019, two primary laboratories were involved (CRS and MS Analytical) simplifying the reporting of QAQC data.

Assays of the 604 standards created using the spiked blank approach described in Figure 11-6. The aim is to match the laboratory sample preparation and analysis methods as closely as possible. Results of this indicate an appropriate method for standards analysis. Two sets of anomalous results are highlighted (Figure 11-7). Weighing errors are highlighted in orange and result in lack of correlation of expected values with measured assays. The first set involves weighing errors created by the sampling technician where the measured laboratory weight versus the received weight at the laboratory exceeds 10% difference (comparison shown in Figure 11-7).

The second set of anomalous results required checking involved drillholes PAL0038, PAL0017 and PAL0043. All drillholes remain outside the resource published in December 2018. PAL0038 has subsequently been analysed by fire assay and PAL0043 and PAL0017 was reassessed in late 2019 and assay results were found to be satisfactory.

The PAL1000 "spiking" process as a substitute for small pulp laboratory standards

- As the PAL1000 technique ideally uses a 1.0 kg charge in the pot, and standards are distributed in 50 g sealed packets, a bulking up method using blank coarse crush reject was created. This is a geochemical "spiking" process, in a similar way to that used in mass spectrometry.
- Typical pulp samples are only 50-60 g and are a different medium to the partly crushed material placed in pots for the PAL1000 leach
- Pulp standards between 0.5 g/t Au and 11 g/t Au are available in sealed bags up to 60 g.
- Olivine diorite ("sauna rock") is used as a blank commencing with assays from PAL0027
- If we spike a 500 g partly crushed blank with a known standard – already suitably crushed for the PAL1000 pots – we can place a measured amount of a certified 2 g/t Au or 11 g/t Au standard pulp into the plastic bag. With a measured amount of blank, the concentration may be calculated.
- For example: 50 g of 11 g/t Au in 450 g of blank ….

  50 g std /500 total x 11 = 1.1 g/t; or,

  100 g std would be 2.2 g/t Au etc
- See detection limit versus sample mass graph – we should always be trying to keep below 0.1 g/t as the acceptable detection limit.
- The most likely source of error in this process happens during the weighing and dat entry stages by the geologist or sample technician. These errors can be detected by inverse slope of trend lines (in the case of blank versus standard entered in incorrect columns), or by a mis-match in laboratory received weight versus total submitted weight.

Figure 11-6: Mawson notes on preparation of standards for PAL1000

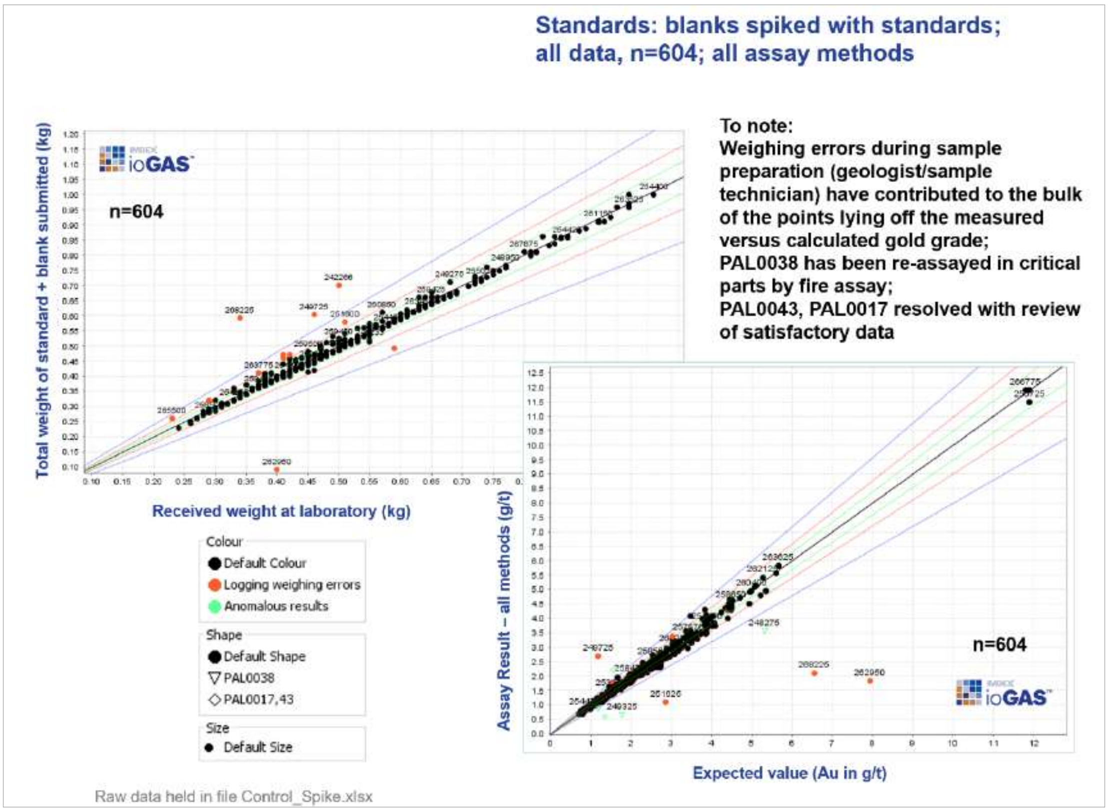
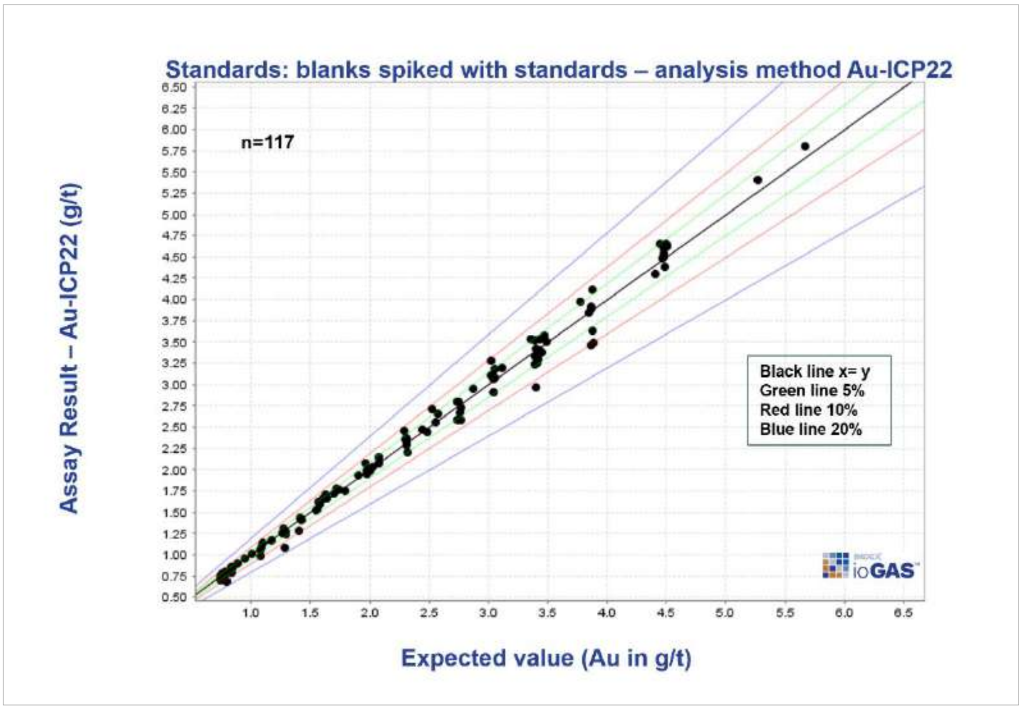


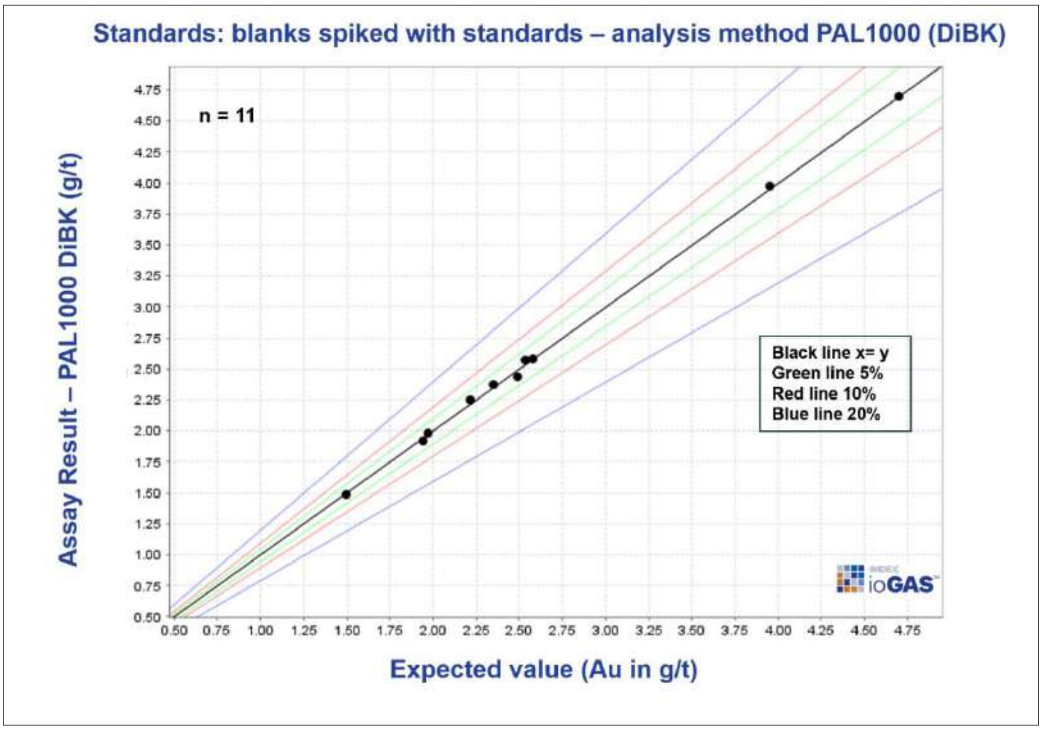
Figure 11-7: Weight of submitted blank plus standard (in known proportions) versus received weight, and assay results versus expected values

Figure 11-8 to Figure 11-10 show the results of the different analytical methods used (Au-ICP22, PAL1000 with DiBK and standard PAL1000). Figure 11-10 highlights the four anomalous samples lying outside the 20% difference line (blue) with the reasoning discussed in Figure 11-7.



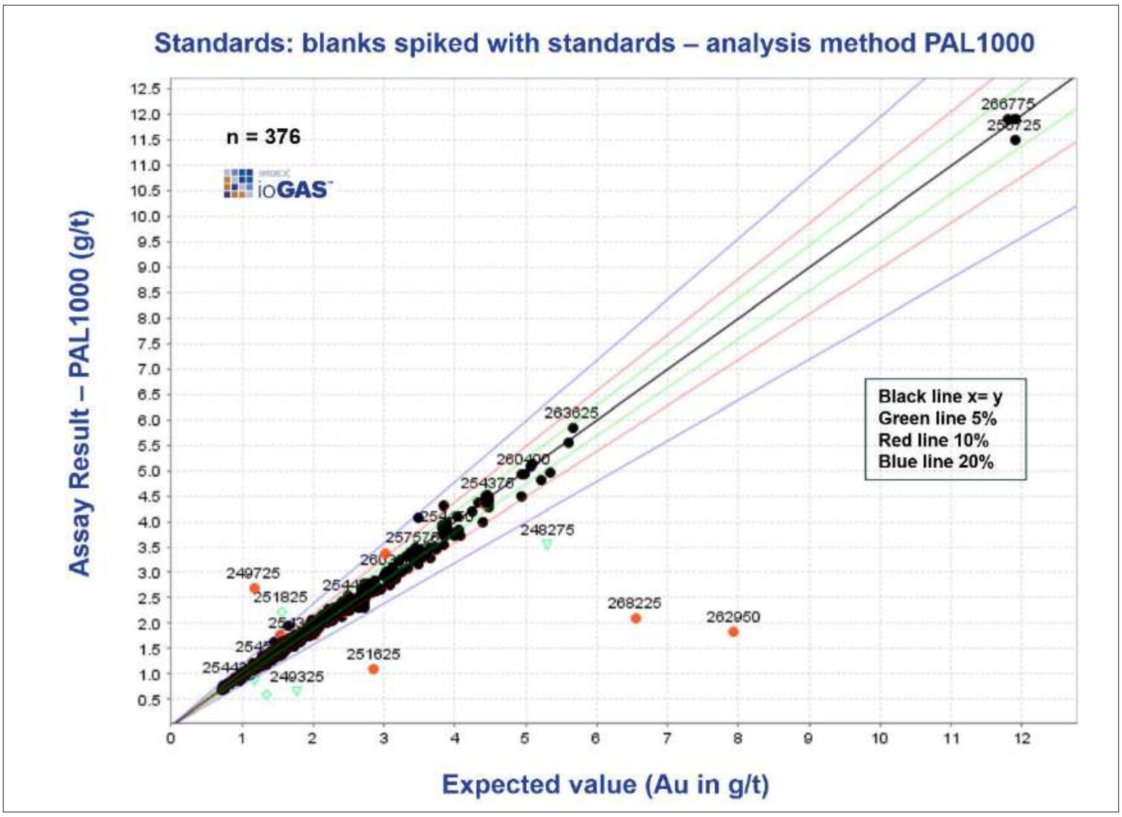
Source: ALS

Figure 11-8: Assay results versus expected value for fire assay method AU-ICP22



Source: CRS

Figure 11-9: Assay results versus expected value for PAL1000 with DiBK pre-concentration



Source: CRS

Figure 11-10: Assay results versus expected value for standard PAL1000 method

Figure 11-11 to Figure 11-13 address the performance of blank samples. These data are satisfactory as only a couple of samples reported positive values (all detection limits samples are recorded in the MX Deposit database as negative).

Note on Figure 11-13 that there is a single 0.02 g/t Au result (Labtium) and a couple at 0.01 g/t Au or less. The anomalous result is from very early in the program where some issues with the Labtium laboratory were identified. Nearly all significant drillholes were reanalysed using ALS methods and preparation following recognition of the problems. The box plots for OREAS22c and 22d showed significant improvement when Labtium data were removed from the data. A separate spreadsheet covering the comparison of Labtium and ALS data is available, but does not impact on the resource as very few Labtium data are included.

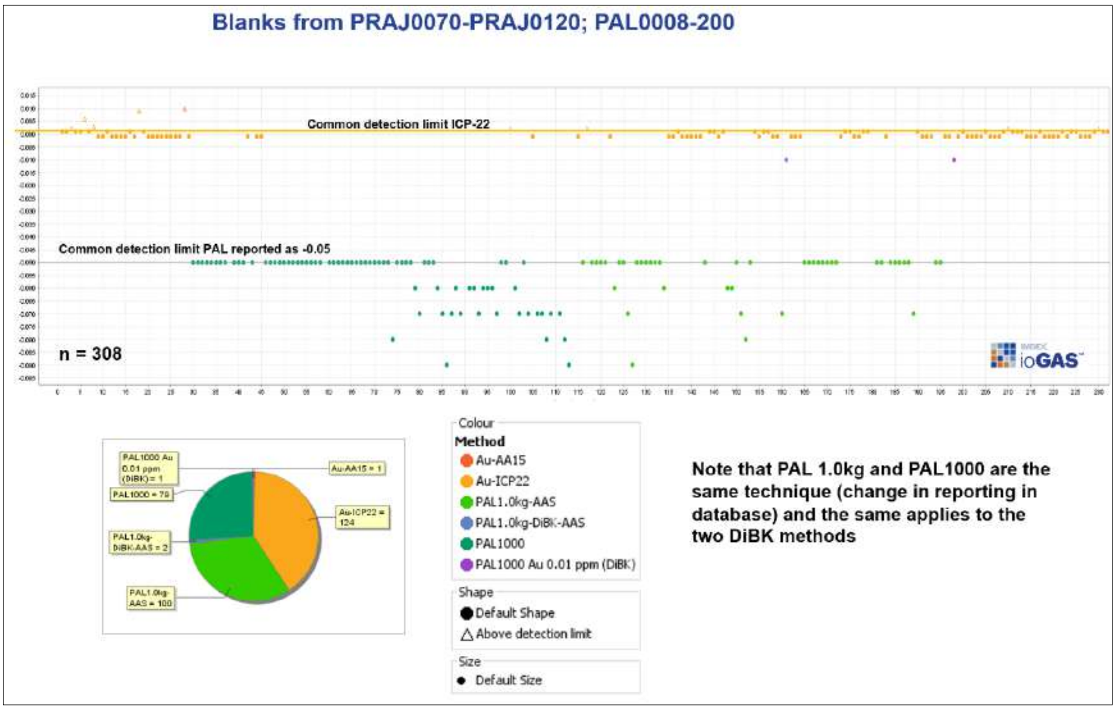


Figure 11-11: Data on analysis of blanks during drilling programs covering PRAJ0070 to PRAJ0120 and PAL0008 to PAL0200

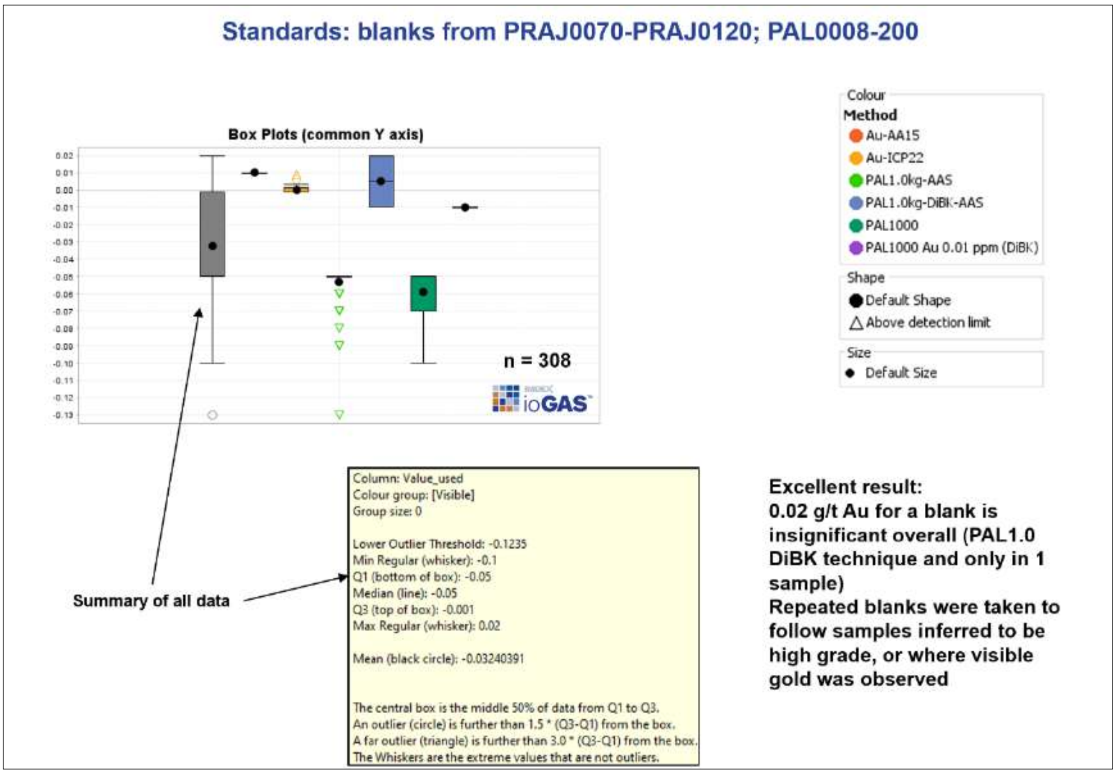


Figure 11-12: Data on analysis of standards during drilling programs covering PRAJ0070 to PRAJ0120 and PAL0008 to PAL0200

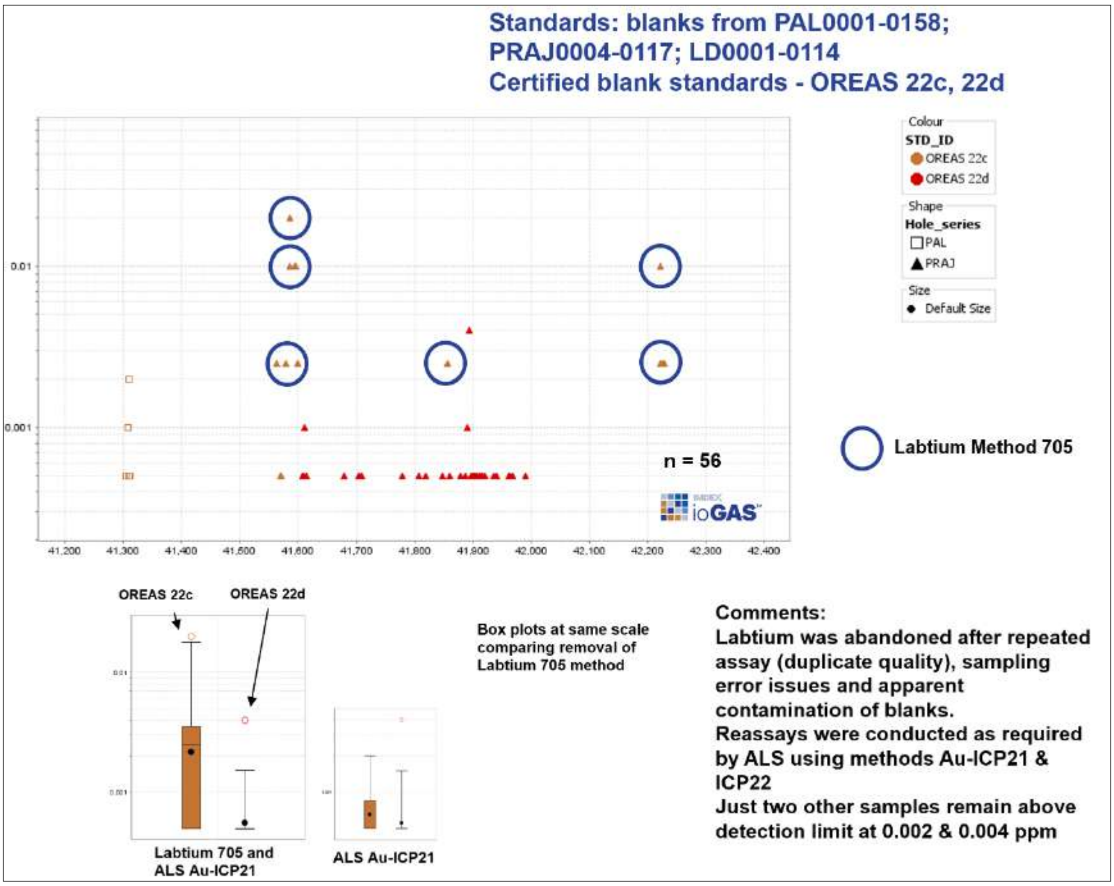


Figure 11-13: Data on analysis of blank standards during drilling programs covering PRAJ0004 to PRAJ0117 and PAL0001 to PAL0158 and LD0001-LD0114 (not included in resource work)

The coarse crush and pulp duplicates (n=633) from all of the PRAJ (25 mm core) and the PAL drillholes up to PAL0082 (NQ, NTW and NQ2 core) are shown in Figure 11-14. X-Y scatterplots with linear and logarithmic axes are shown. A very different picture emerges between the graphs. PRAJ0004 appears the worst anomaly in the linear plot (a Labtium assay) at 20 g/t Au versus duplicate of 12.5 g/t Au which is either a nugget effect, or laboratory issue.

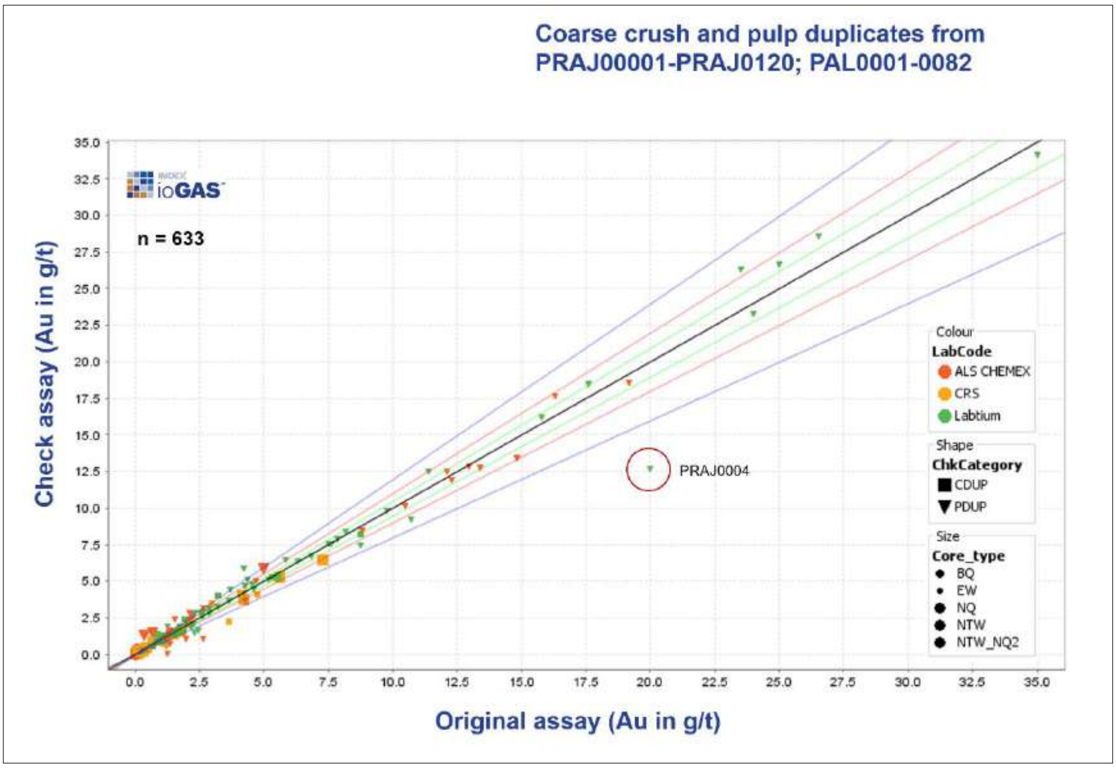


Figure 11-14: Coarse crush and pulp duplicates (n=633) from all of the PRAJ (25 mm core) and the PAL drillholes up to PAL0082

Changing the view of above plot (Figure 11-14) to a log-log plot (Figure 11-15) shows more detail in the 0.37-5.0 g/t Au range. Two samples, – PAL0009 and PRAJ0107, required follow-up. As a population, the fire assay pulp duplicates from ALS show the most variation and the CRS (PAL1000) sample population becomes tighter towards the X=Y line, converging above the 0.30 g/t Au value.

PAL0009 requires resampling for ¼ core and the 25 mm diameter PRAJ drillholes will be superseded by larger diameter diamond drilling to upgrade the quality of the Inferred Mineral Resource estimate.

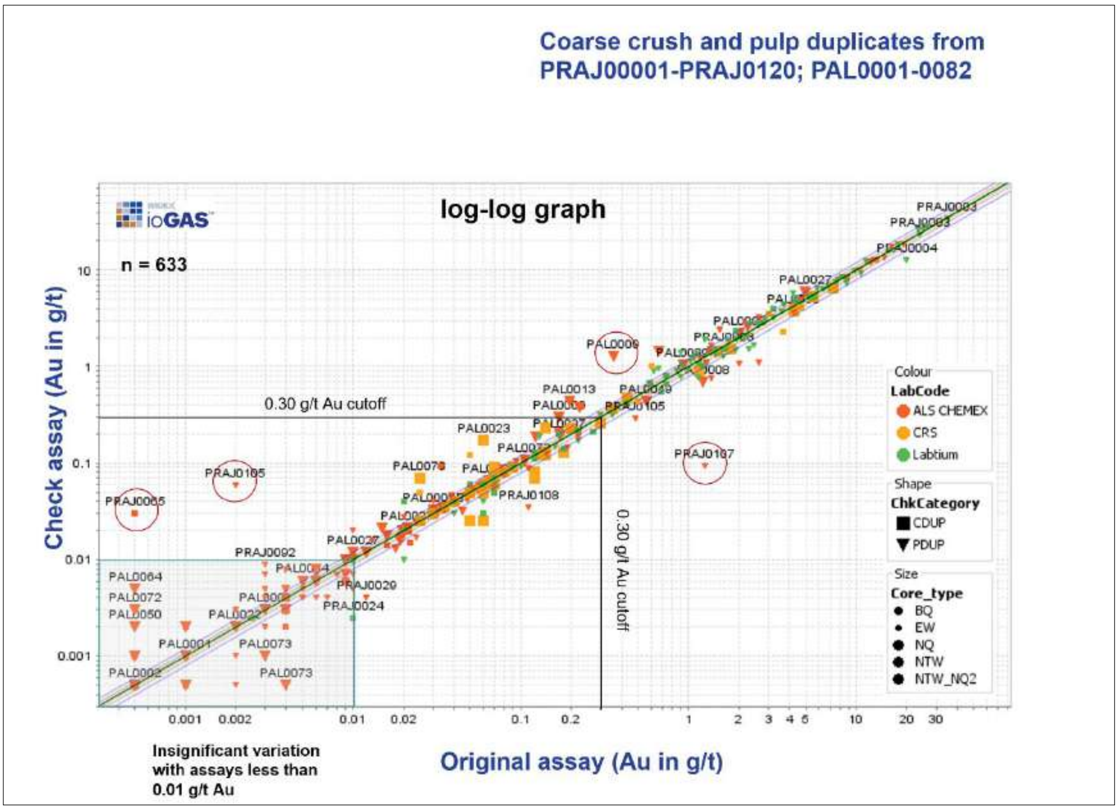
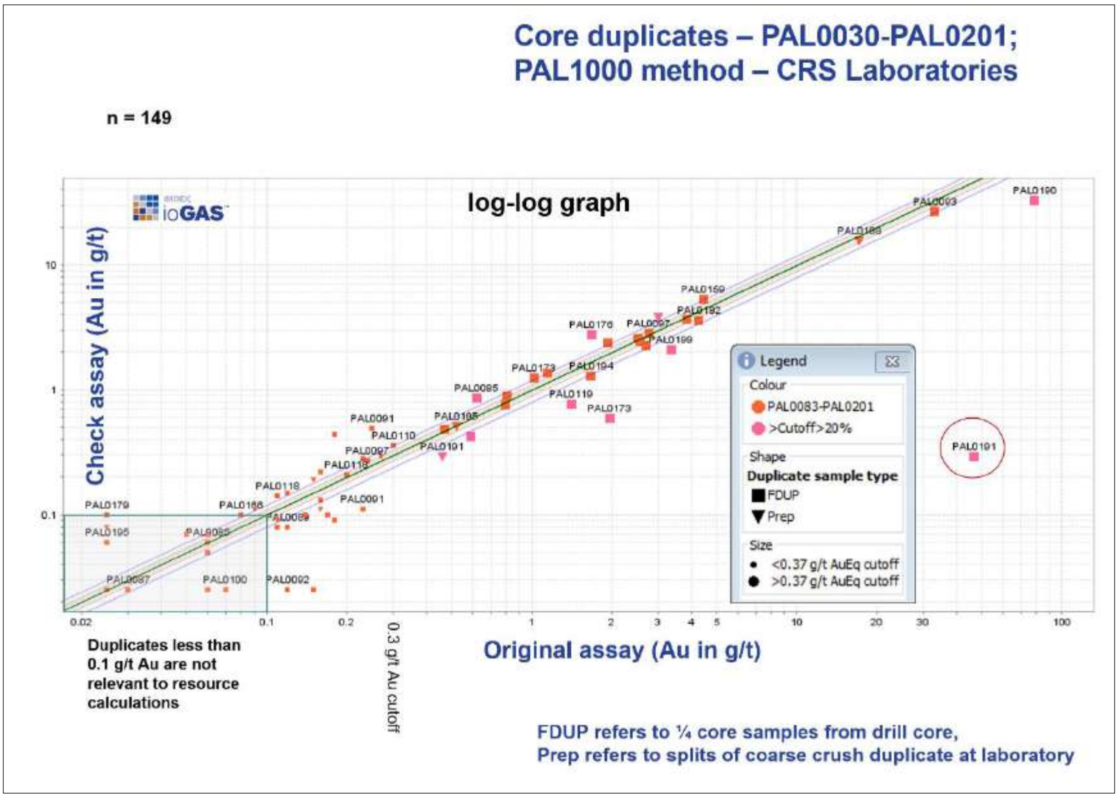


Figure 11-15: Detail of the lower 0.3 to 5.0 g/t Au range shown in a log-log plot

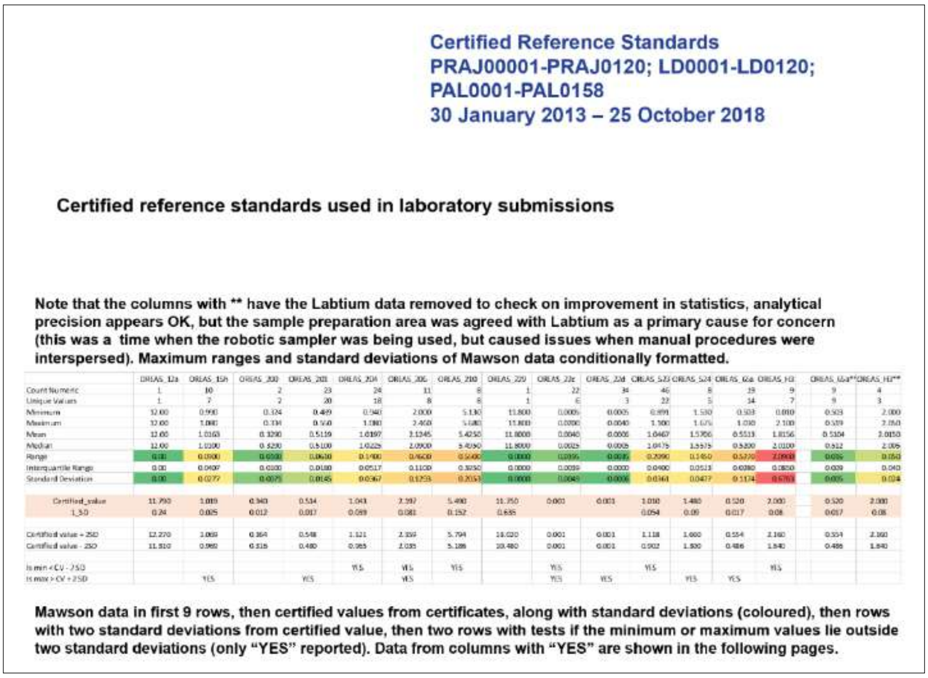


Source: CRS

Figure 11-16: Comparison of ¼ core duplicates using the PAL1000 method

Figure 11-16 covers the PAL1000 technique (CRS Laboratory) from PAL0030 to PAL0201 (last drillhole completed in April 2019). The correlation with X=Y is reasonably good, given some visible gold is evident in the drill core, but the samples outside 20% variance are to be submitted to ALS for LeachWELL tests as a comparison.

Figure 11-17 to Figure 11-22 cover the analysis of Certified Reference standards with the primary list of standards and their performance assessed in Figure 11-17. Note again the Labtium issues and significant improvement in statistics when Labtium data are removed from the population. The base of the table assesses if any samples lie outside 2 Standard Deviations (SD) based on the certificate of the reference sample. Descriptions of any anomalies are described and commented on in Figure 11-18 to Figure 11-22.


Certified Reference Standards
PRAJ00001-PRAJ0120; LD0001-LD0120;
PAL0001-PAL0158
30 January 2013 – 25 October 2018

Certified reference standards used in laboratory submissions

Note that the columns with ** have the Labtium data removed to check on improvement in statistics, analytical precision appears OK, but the sample preparation area was agreed with Labtium as a primary cause for concern (this was a time when the robotic sampler was being used, but caused issues when manual procedures were interspersed). Maximum ranges and standard deviations of Mawson data conditionally formatted.

	OREAS_12a	OREAS_15h	OREAS_200	OREAS_201	OREAS_204	OREAS_206	OREAS_210	OREAS_229	OREAS_22e	OREAS_224	OREAS_523	OREAS_524	OREAS_62a	OREAS_H3	OREAS_62a**	OREAS_H3**
Count Numeric	1	10	2	23	24	11	8	1	22	34	46	8	19	9	9	4
Unique Values	1	7	2	20	18	8	8	1	6	3	22	5	14	7	9	3
Minimum	12.00	0.990	0.324	0.489	0.940	2.000	5.130	11.800	0.0005	0.0005	0.891	1.530	0.503	0.010	0.503	2.000
Maximum	12.00	1.080	0.334	0.550	1.080	2.460	5.680	11.800	0.0200	0.0040	1.100	1.675	1.030	2.100	0.539	2.050
Mean	12.00	1.0163	0.3290	0.5119	1.0197	2.1345	5.4250	11.8000	0.0040	0.0006	1.0467	1.5706	0.5513	1.8156	0.5104	2.0150
Median	12.00	1.0100	0.3290	0.5100	1.0225	2.0900	5.4050	11.8000	0.0025	0.0005	1.0475	1.5575	0.5200	2.0100	0.512	2.005
Range	0.00	0.0900	0.0100	0.0610	0.1400	0.4600	0.5400	0.0000	0.0195	0.0035	0.2090	0.1450	0.5270	2.0900	0.036	0.050
Interquartile Range	0.00	0.0407	0.0100	0.0180	0.0517	0.1100	0.3250	0.0000	0.0039	0.0000	0.0400	0.0513	0.0380	0.0850	0.009	0.040
Standard Deviation	0.00	0.0277	0.0071	0.0145	0.0367	0.1259	0.2051	0.0000	0.0049	0.0006	0.0361	0.0477	0.1174	0.6703	0.005	0.024
Certified_value	11.790	1.019	0.340	0.514	1.043	2.197	5.490	11.750	0.001	0.001	1.010	1.480	0.520	2.000	0.520	2.000
1_SD	0.24	0.025	0.012	0.017	0.039	0.081	0.152	0.635			0.054	0.09	0.017	0.08	0.017	0.08
Certified value + 2SD	12.270	1.069	0.364	0.548	1.121	2.359	5.794	13.020	0.001	0.001	1.118	1.660	0.554	2.160	0.554	2.160
Certified value - 2SD	11.310	0.969	0.316	0.480	0.965	2.035	5.186	10.480	0.001	0.001	0.902	1.300	0.486	1.840	0.486	1.840
Is min < CV - 2SD					YES	YES	YES		YES		YES			YES		
Is max > CV + 2SD		YES		YES		YES			YES	YES		YES	YES			

Mawson data in first 9 rows, then certified values from certificates, along with standard deviations (coloured), then rows with two standard deviations from certified value, then two rows with tests if the minimum or maximum values lie outside two standard deviations (only "YES" reported). Data from columns with "YES" are shown in the following pages.

Figure 11-17: Statistical data on reference standards used and determinations of values exceeding 2SD

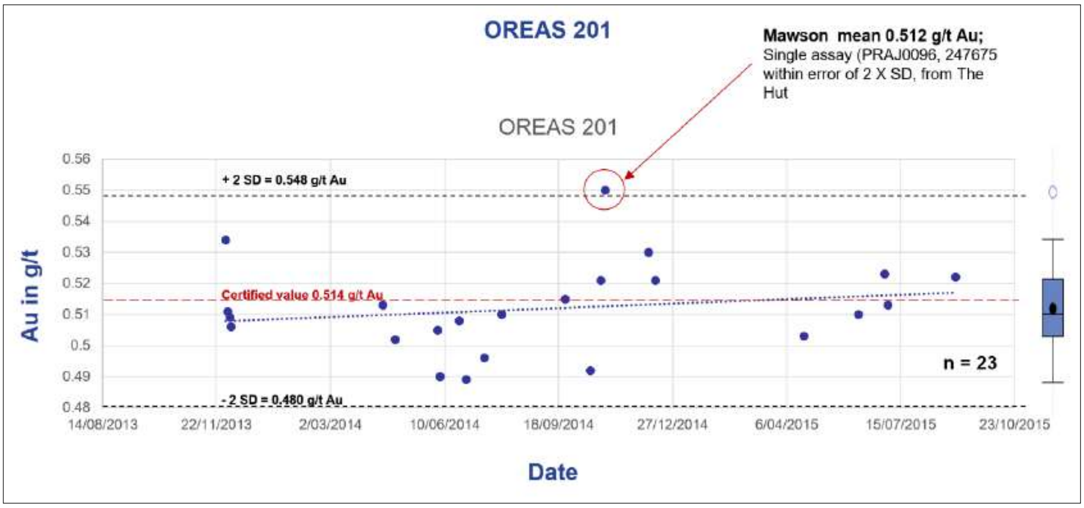


Figure 11-18: Comparison with Certified reference standard OREAS 201 data

PRAJ0096 is a small diameter (25 mm) hole that will require follow-up with a NQ or greater diamond drill hole.

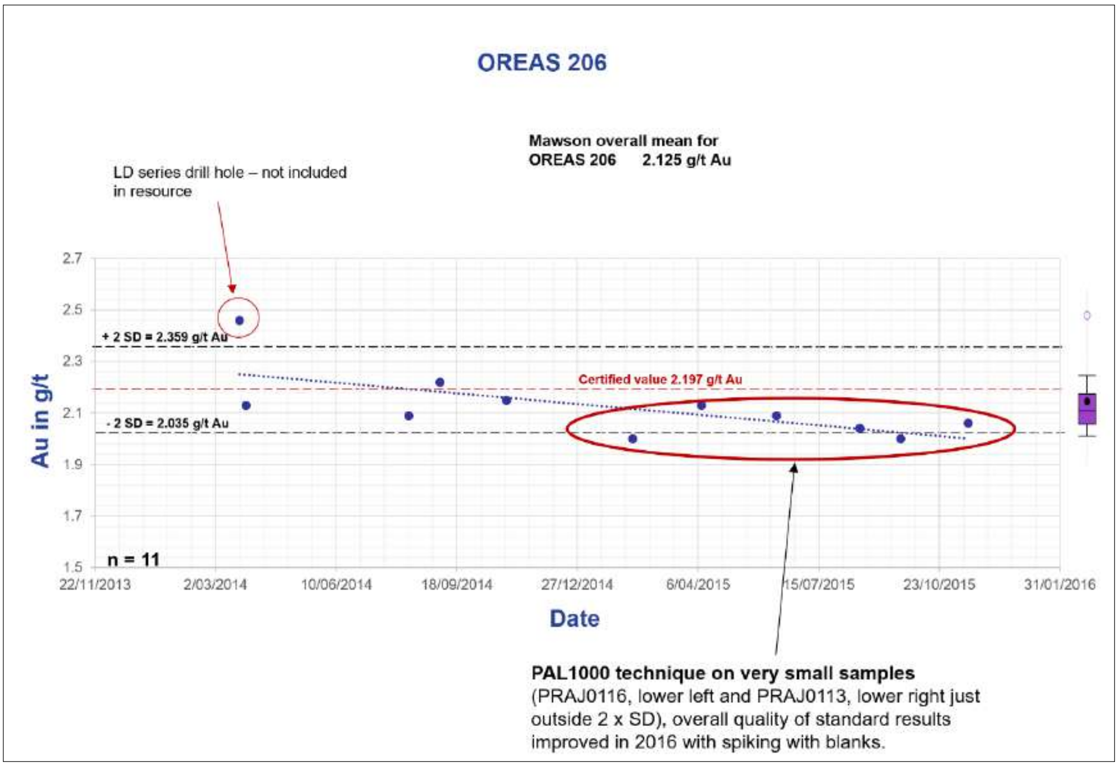


Figure 11-19: Comparison with Certified reference standard OREAS 206 data

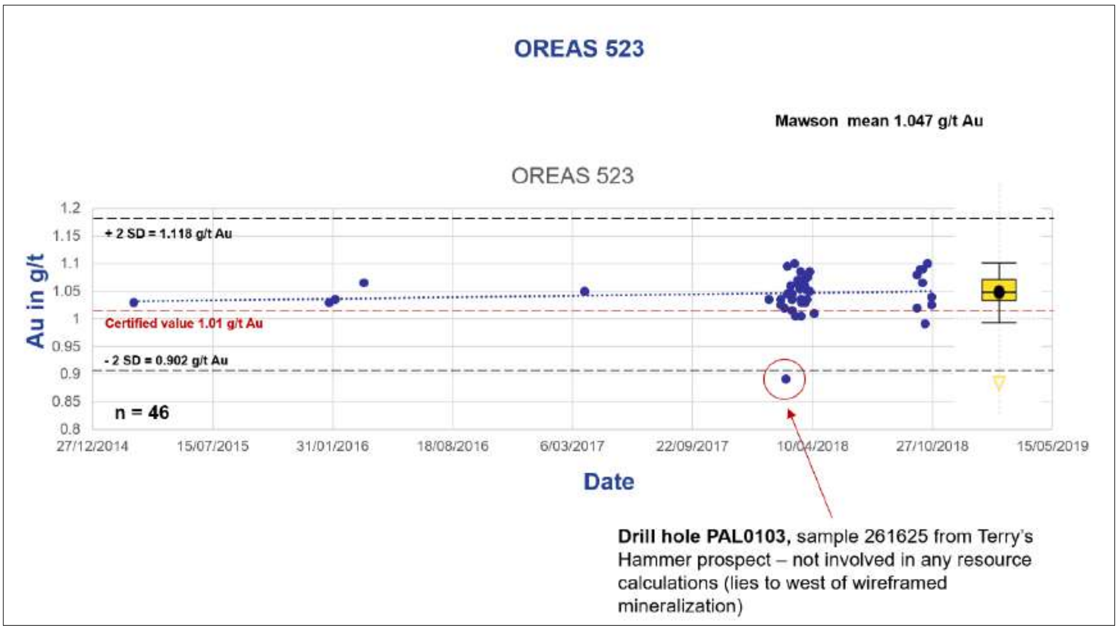


Figure 11-20: Comparison with Certified reference standard OREAS 523

A single sample is shown to lye just outside 2SD and it is recommended that Mawson checks core trays and considers 1/4 core samples as a check.

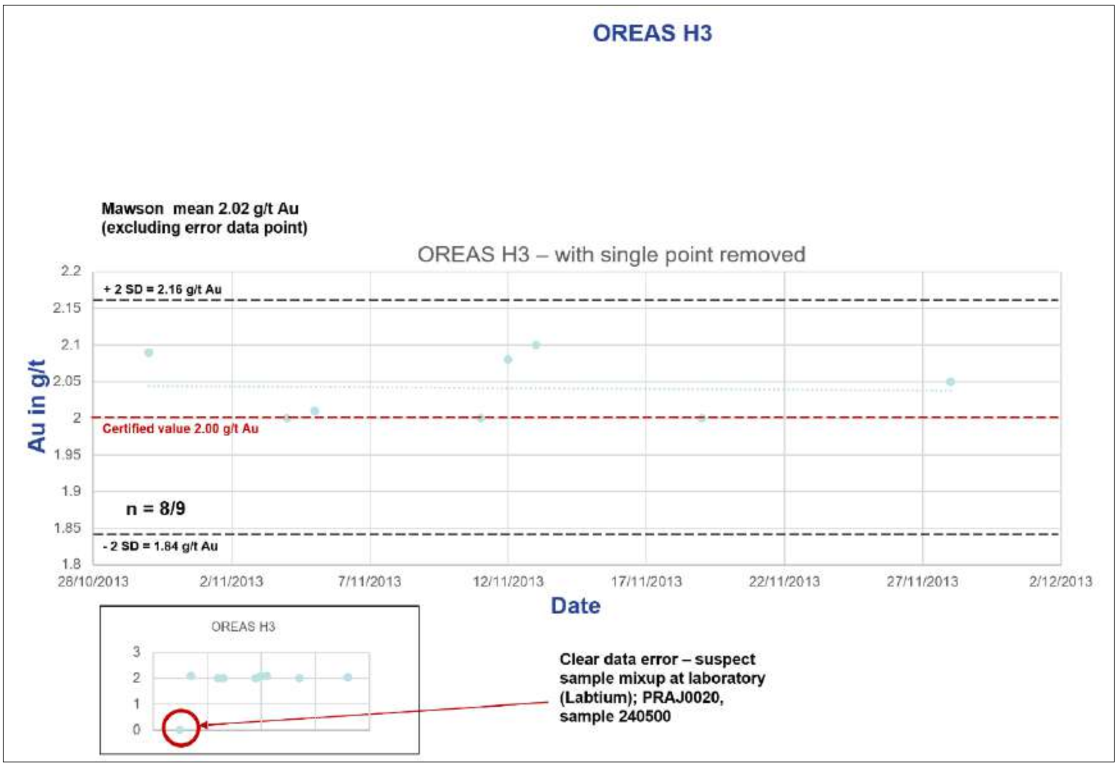


Figure 11-21: Comparison with Certified reference standard OREAS H3 data

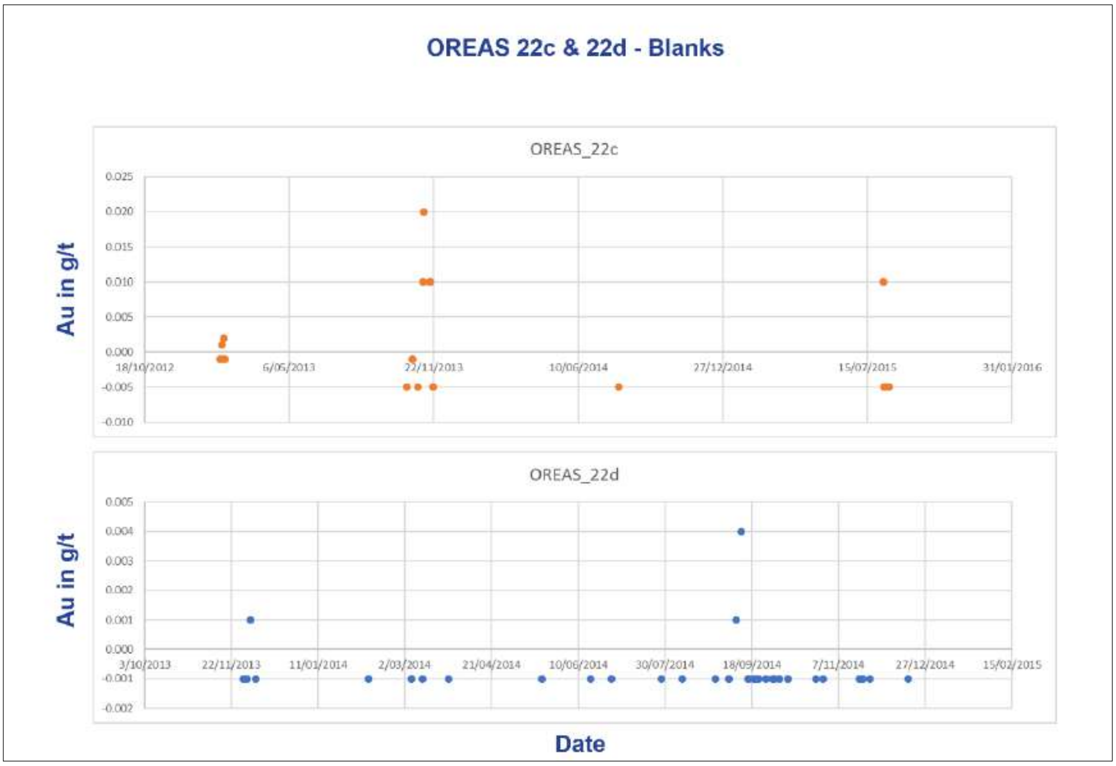


Figure 11-22: OREAS 22c and 22d results

11.5.3 Quality control data November 2019-2021 drilling

This section covers all drilling in the PAL series from PAL0202 to PAL0311 with assays and QAQC completed during the period November 2019 through to 30 July 2021.

Blanks inserted after visible gold are a key to recognizing the quality of laboratory preparation (cleaning, cross contamination, incorrectly labelled samples). Figure 11-23 indicates no concerns with sample identification or preparation at CRS. The coarsely crushed olivine diorite enjoys the same preparation as the drill core in the preparation routine at the laboratory. All results are negative; that is, below the detection limit plotted held as a negative number in the Mawson database.

There are no concerns with cobalt blank data (Figure 11-24). A single sample of 2.8 ppm lies above the mean of 1.0 ppm Co, but is insignificant given a cut-off of approximately 300 ppm for Co.

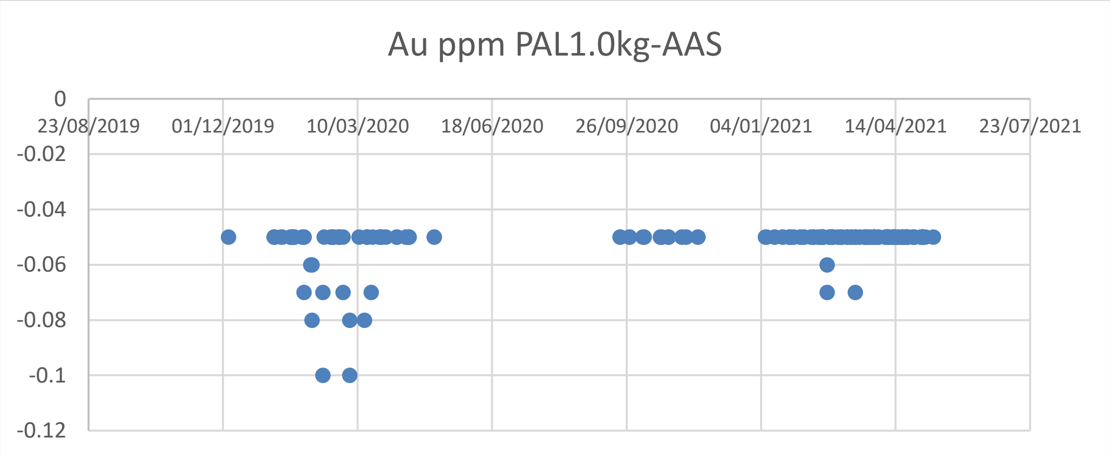


Figure 11-23: Assay results from olivine diorite blank samples

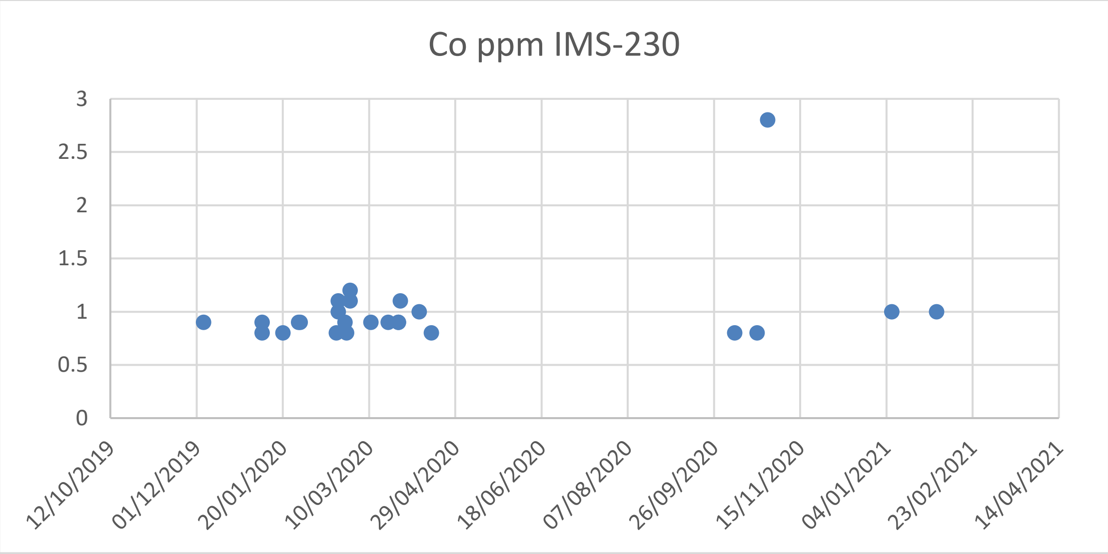


Figure 11-24: Cobalt blank data

The PAL1000 assay quality is tested from the spiked standards. The first check is to confirm that the laboratory received weight matches the combined weight of olivine diorite blanks (sauna rock) and the weight of standard combined (Figure 11-25). Off-linear data are marked in colour to check against the plot of the expected gold (calculated) versus measured. Six orange and two green samples lie outside the acceptable measurement limits. N=367 samples (refer to Figure 11-26 for impact on measured versus expected assay results). Note that in Figure 11-26 the only points significantly off the X=Y are those with weighing errors.

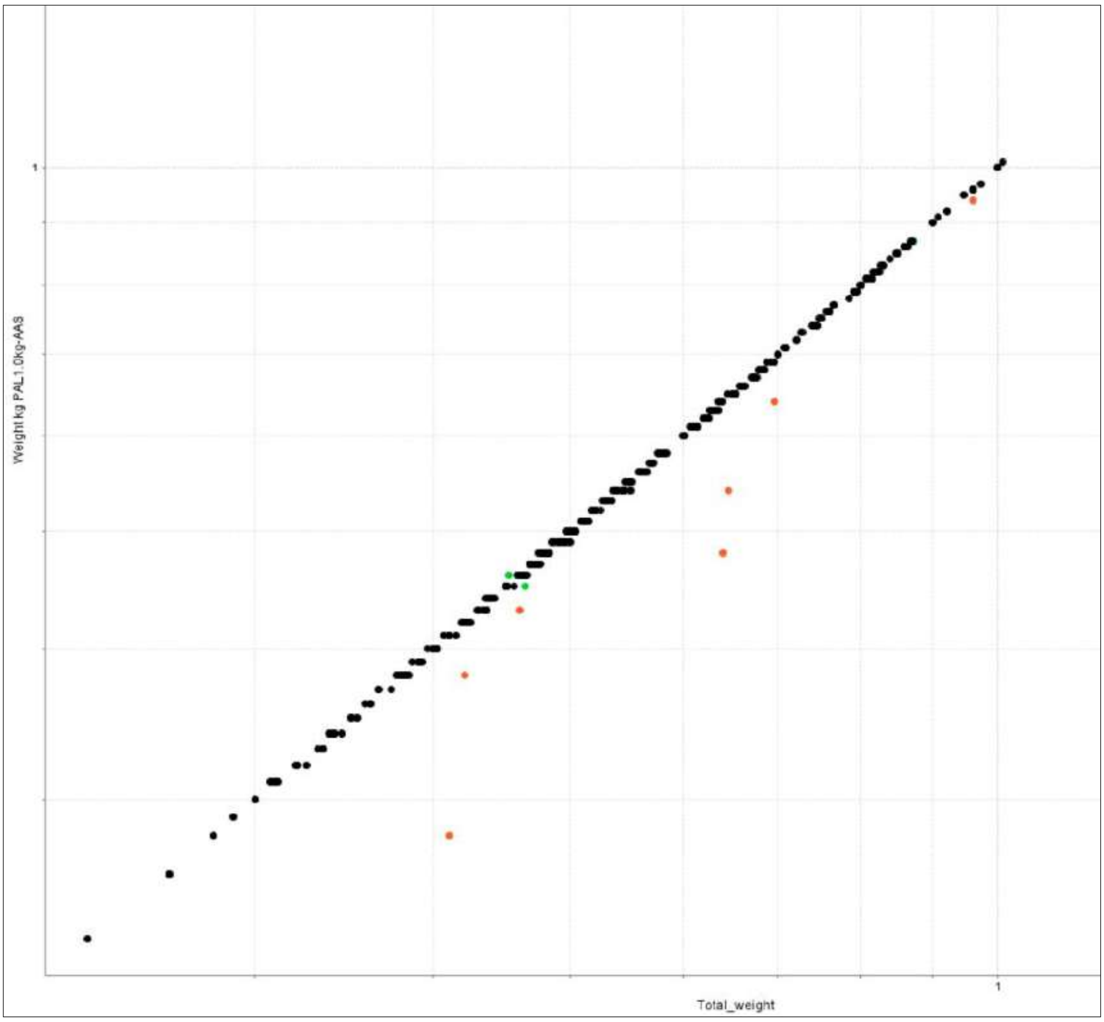


Figure 11-25: Plot of weight received by laboratory on the Y axis versus weight measured in spike preparation by technical staff at Mawson core facility

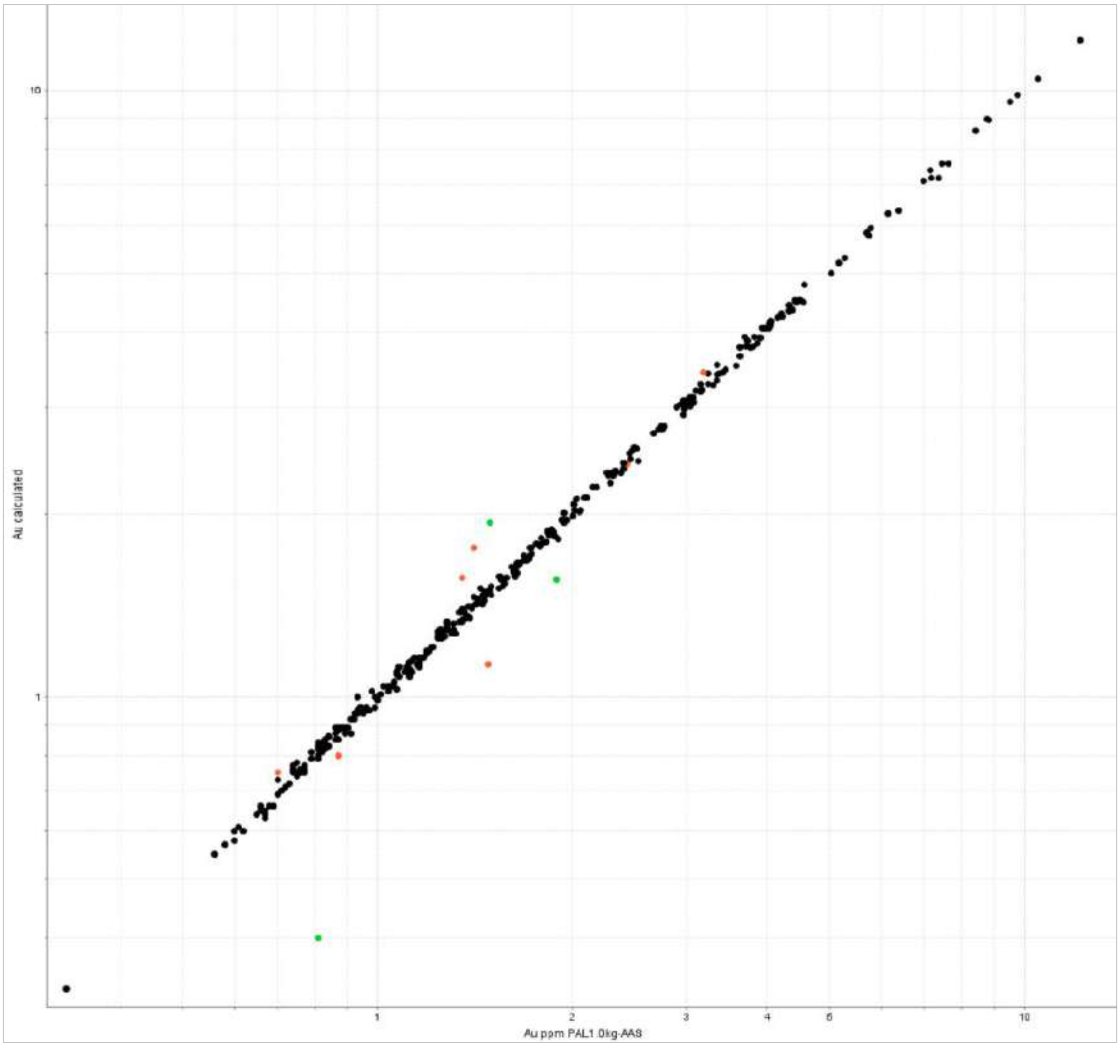


Figure 11-26: Log-log plot of expected gold concentration versus measured gold (g/t Au CRS PAL1000 method)

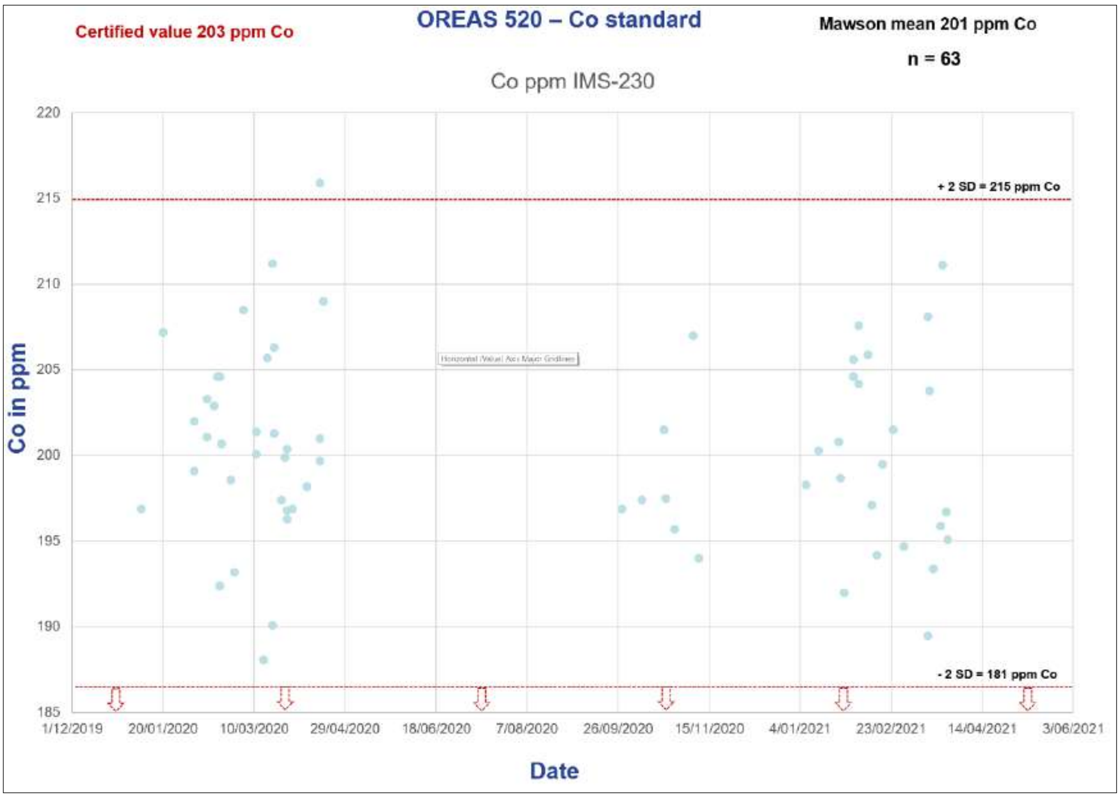


Figure 11-27: OREAS 520 cobalt standard (period covers PAL0202 to PAL0311). 4 acid digest determinations

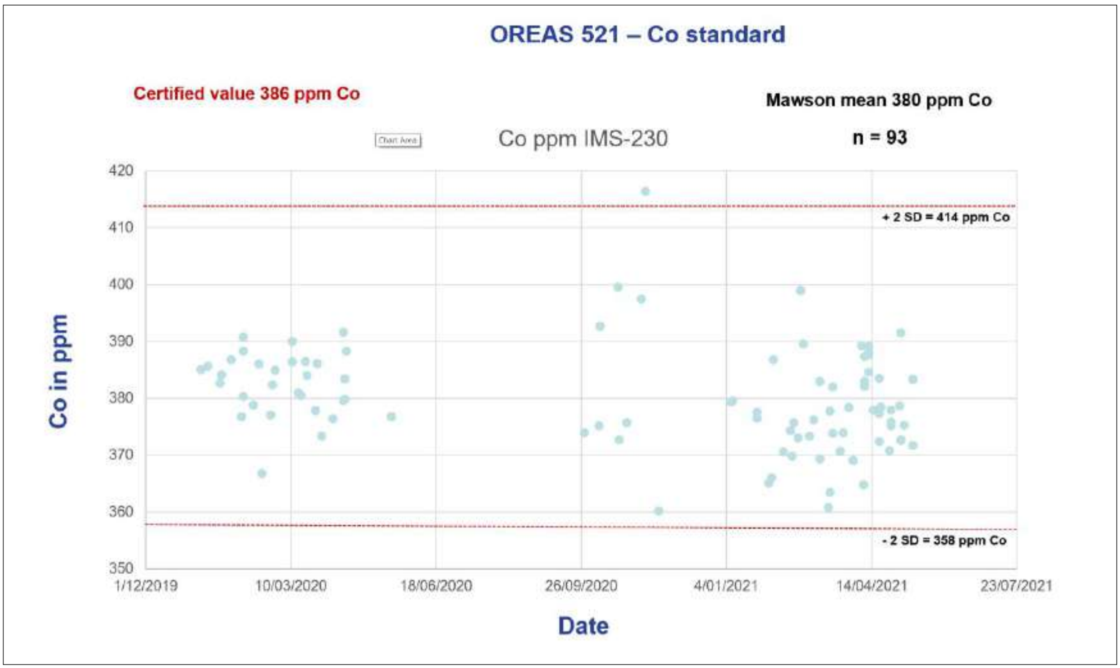


Figure 11-28: OREAS 521 cobalt standard (period covers PAL0202 to PAL0311). 4 acid digest determinations

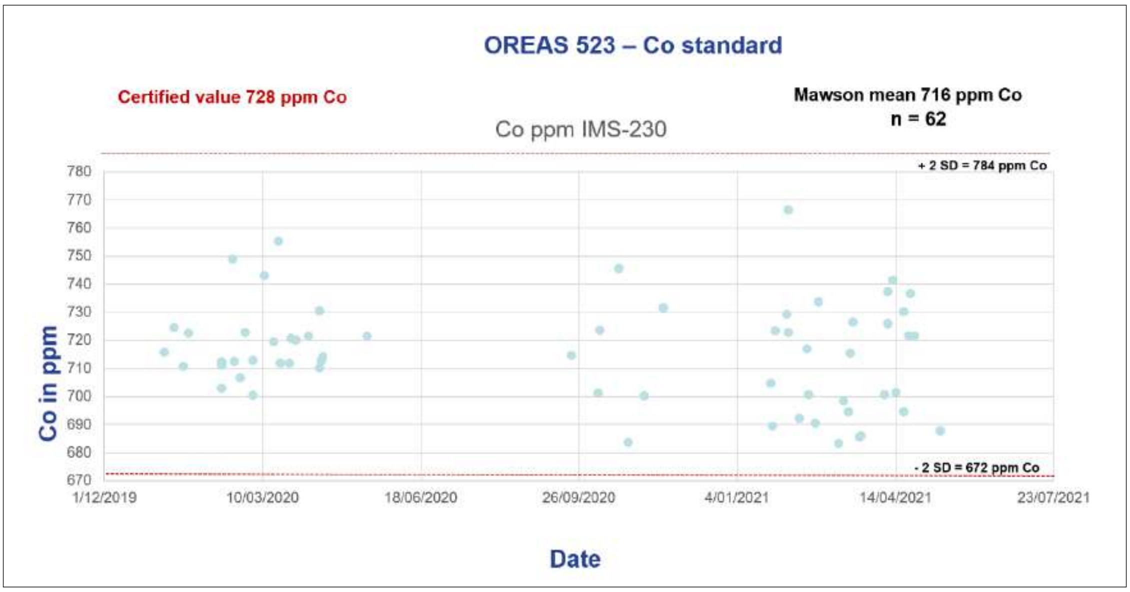


Figure 11-29: OREAS 523 cobalt standard (period covers PAL0202 to PAL0311). 4 acid digest determinations

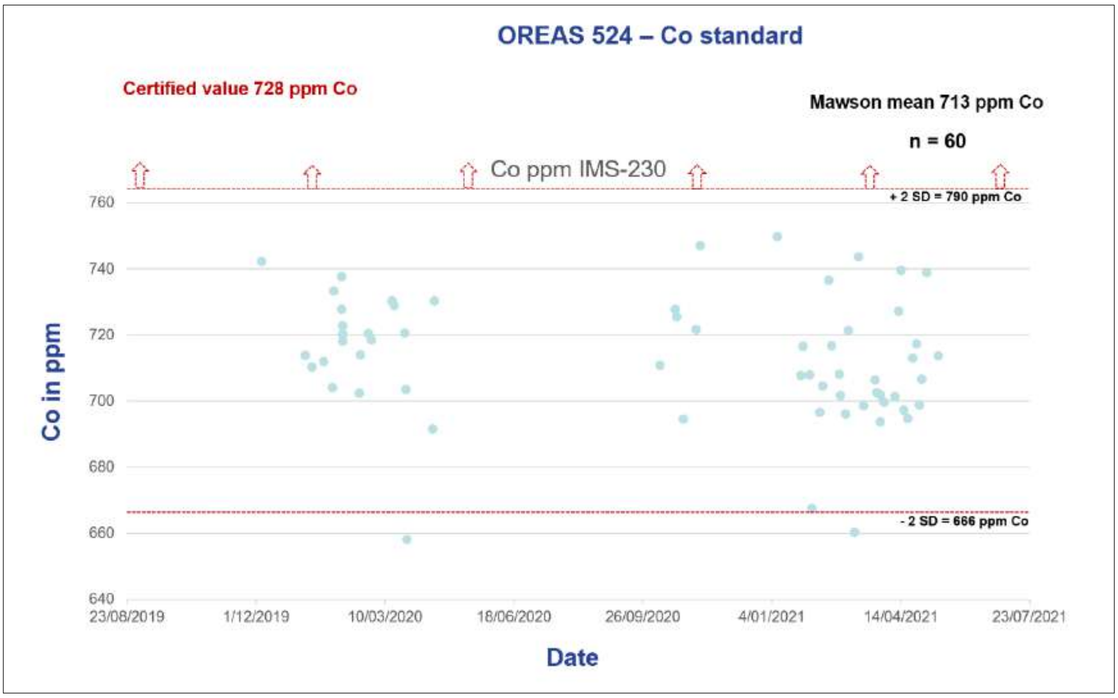


Figure 11-30: Oreas 524 cobalt standard (period covers PAL0202 to PAL0311). 4 acid digest determinations

Following the recommendation in AMC, 2018 by Rod Webster (acting as QP), Mawson adopted cobalt standards (instead of non-specific standards with a lower average Co contents). The mean values (Figure 11-27 to Figure 11-30) for the Mawson assays for the four standards all fall just on the low side of all corresponding certified reference materials, and thus on average, Mawson may be very slightly under-reporting. A verification check ("round robins"; Figure 11-31) by sending 25 samples with a range from 5 ppm to 2110 ppm Co to ALS for independent assay (ME-ICP41, aqua regia digest) reveals no systematic error – in fact, quite the opposite with a clear X=Y trend with a correlation coefficient (R^2) of 0.9975. Note that there is no discernible difference between the aqua regia and 4 acid digest determinations for cobalt; cobalt is most likely held in sulphides.

Figure 11-31 shows inter-laboratory cobalt determinations. 4 acid digest MS Analytical IMS-230 cobalt versus ALS ME-ICP41 aqua regia digest cobalt. Note the linear X=Y trend of slope 1 with an excellent correlation.

Figure 11-32 shows interlaboratory gold determinations. Slope of line of X=Y should be compared with linear regression line which is dragged to the right by the weight of high values. Note a better correlation at lower gold concentrations with the greater spread to higher values, most likely driven by nugget effect in coarse crush duplicates. Further work on the homogeneity of the coarse crush duplicates should be considered prior to, or as part of any metallurgical testwork on these materials.

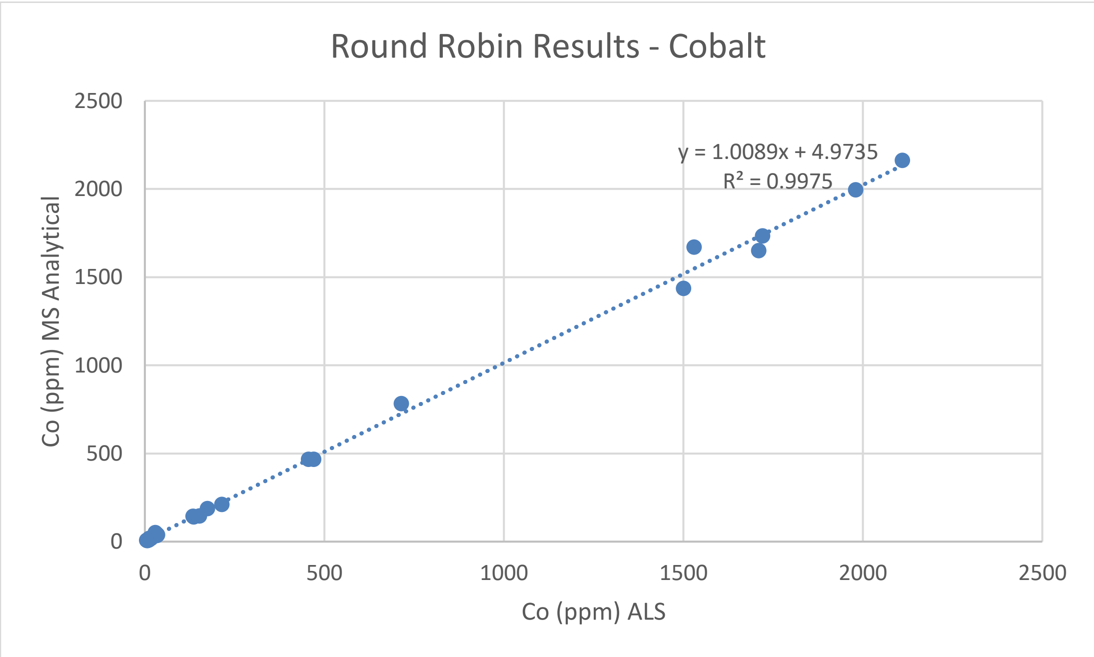


Figure 11-31: Inter-laboratory cobalt determinations

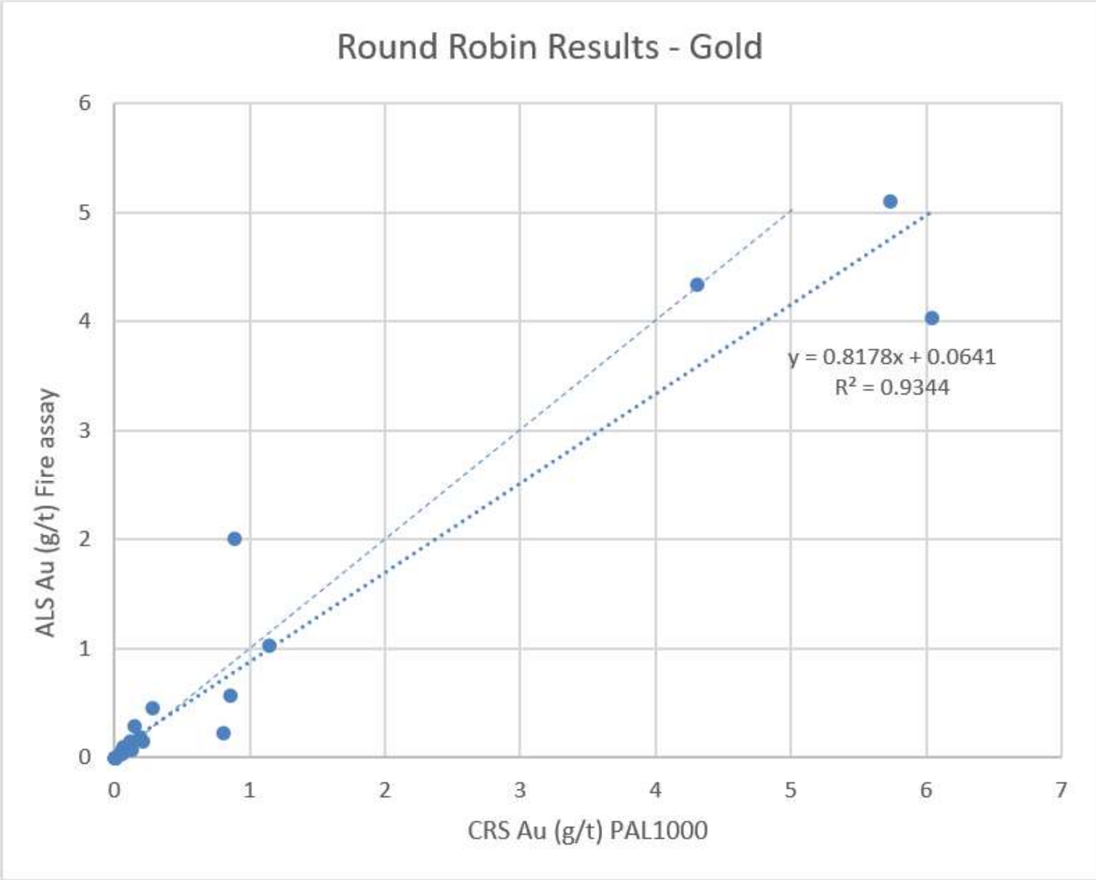


Figure 11-32: Interlaboratory gold determinations

11.5.4 Summary and recommendations

The QAQC results are considered to be of a high standard and are suitable for the reporting of an Inferred Mineral Resource estimate. There is no evidence for significant systematic under- or over-reporting of gold or cobalt. The spiked gold standards and blanks assays show no significant problems. Variation in cobalt assay results outside 2 SD are minimal and insignificant, and the inter-laboratory standards have a very strong linear X=Y relationship.

The following matters should be addressed to further understand minor variations in inter-laboratory results.

- Consider pulp duplicate tests on inter-laboratory samples and more riffle split tests on coarse crush duplicates to determine the source of the variation (Figure 11-32) in reporting of gold in the CRS results. The cause of the variation may simply be a nugget effect appearing in the sampling of the coarse crush duplicates.
- Test some riffle splits of coarse crush duplicates for gold (1 kg samples) by another leach technique (such as the ALS method known as LeachWELL).

12 DATA VERIFICATION

The QP (Ove Klavér, AFRY) accompanied by, and supervising, a second independent geologist from AFRY conducted the following work in the verification of the Mawson data and interpretations:

- visited the Rovaniemi core logging and office facility;
- observed geological and structural logging of the drill core, compared this with information held in the MX Deposit database;
- understood and discussed the geotechnical logging process;
- checked assay results against half core remaining in core trays;
- confirmed field locations of drillholes and prospects;
- understood and validated QAQC for sampling of drill core;
- viewed the download process of data from the MX Deposit database;
- compared drill core data against three dimensional models and wireframes of the mineralization; and
- inspected mineralized drill core from the each of the prospects that are included in the Inferred Mineral Resource estimate.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

Initial metallurgical evaluation of mineralization from the Rajapalot deposits was undertaken as part of Finland's Circular Ecosystem of Battery Metals (BATCircle) project. The samples for the initial evaluation were selected to represent distinct end members of particular silicate constituents of the mineralization rather than to be representative of potential processing feeds.

The samples were characterised by mineralogical investigation and a series of sighter metallurgical tests undertaken to investigate responses to standard metallurgical processes.

The results of the preliminary testwork (refer to AFRY, 2021) highlighted that several possible processing scenarios are suited to producing a range of gold and cobalt products from Rajapalot mineralization. A testwork plan was developed to investigate responses to different processes to enable selection of the preliminary flowsheet, and to provide inputs to elements of this PEA study.

For the purposes of the PEA, only the testwork relevant to the selected process flowsheet is presented herein.

13.2 Samples for Testing

13.2.1 Mineralization

Two distinct styles of gold mineralization dominate the Rajapalot area and demonstrate distinct geometallurgical features with a clear spatial aspect.

The first style of gold-cobalt mineralization at Rajapalot is termed 'Raja', a potassic-iron (K-Fe) style and is characteristically associated with muscovite and/or biotite and chlorite in a diverse range of fabrics. Gold grades of more than 1 g/t Au are associated with pyrrhotite, and contained within muscovite-biotite schists, muscovite and biotite-bearing albitic granofels, and brecciated, variably micaceous albitic rocks.

The second is a variably sulphidic magnesian-iron host, referred to as 'Palokas' style. The magnesian-iron host is most likely an ultramafic volcanic (komatiitic) and occurs within approximately 100 vertical metres of the inferred Kivalo-Paakkola boundary (that is, near the incoming of pelites, calc-pelites and quartz muscovite rocks).

A key distinguishing feature is the greater presence of arsenic bearing cobaltite (CoAsS) in the 'Raja' type. Cobalt in the 'Palokas' type is predominantly fine-grained linnaeite (Co_3S_4), intimately associated with the iron sulphide mineral pyrrhotite.

The mineralization from the South Palokas area is of the Raja style.

13.2.2 Sample Selection

Considerable mass of crushed 'rejects' of drill core was available from the sampling and assaying procedure. It was determined that this material would be suitable for the majority of the testwork to be conducted. Samples of intact half-core were also selected to be used in the testing of comminution characteristics.

Mawson geologists reviewed available data from resource drilling to identify intervals that would provide appropriate representation of mineable grades, lithologies, and spatial distributions for the two mineralization styles.

The half-core selected was cut in half again to generate the necessary samples.

Two samples were generated in each form of crushed and ¼ core. They were designated 'Raja-SthPal', which was representative of the arsenic containing cobalt mineralization, and 'Palokas', which was representative of the arsenic deficient mineralization.

The drillholes, intervals, calculated grades, and designation for comminution testing are tabulated in Table 13-1 and Table 13-2. The locations of drillholes are shown in Figure 13-1 – Palokas and Sth Palokas in the north, Raja in the south.

Table 13-1: Raja-South Palokas sample selection

Prospect	Hole Number	From, m	To, m	Mass, kg	Au, ppm	Co, ppm	As, ppm	Fe, %	S, %	Comm
South Pal	PAL0173	263	269	6.9	5.24	391	262	5.76	1.66	Y
South Pal	PAL0173	273	282	13.4	2.46	1131	799	8.28	3.21	Y
Raja	PAL0188	299.25	316.6	24.7	3.31	1251	752	10.30	4.01	
Raja	PAL0188	319.55	330.05	14.1	8.88	1178	1170	6.92	2.50	
South Pal	PAL0204	92.7	104	13.1	4.31	861	455	9.44	3.37	Y
South Pal	PAL0235	439.45	455.7	23.1	2.52	950	580	10.64	3.37	
Weighted Average				**95.4**	**4.10**	**1034**	**702**	**9.15**	**3.26**	

Table 13-2: Palokas sample selection

Prospect	Hole Number	From, m	To, m	Mass, kg	Au, ppm	Co, ppm	As, ppm	Fe, %	S, %	Comm
Palokas	PAL0210	132.3	150.05	25.95	1.39	587	41	19.41	3.82	Y
Palokas	PAL0210	152.55	158.85	8.96	3.53	265	6	14.40	2.42	Y
Palokas	PAL0222	265.9	280	21.19	5.71	1065	180	20.91	5.14	
Palokas	PAL0227	293.1	300.1	10.25	3.41	483	55	13.52	3.35	
Palokas	PAL0227	310.7	315.7	7.21	1.57	414	54	9.59	2.74	
Palokas	PAL0227	321.8	339.7	29.22	0.68	409	16	9.68	1.90	
Weighted Average				**102.8**	**2.48**	**584**	**62**	**15.24**	**3.30**	

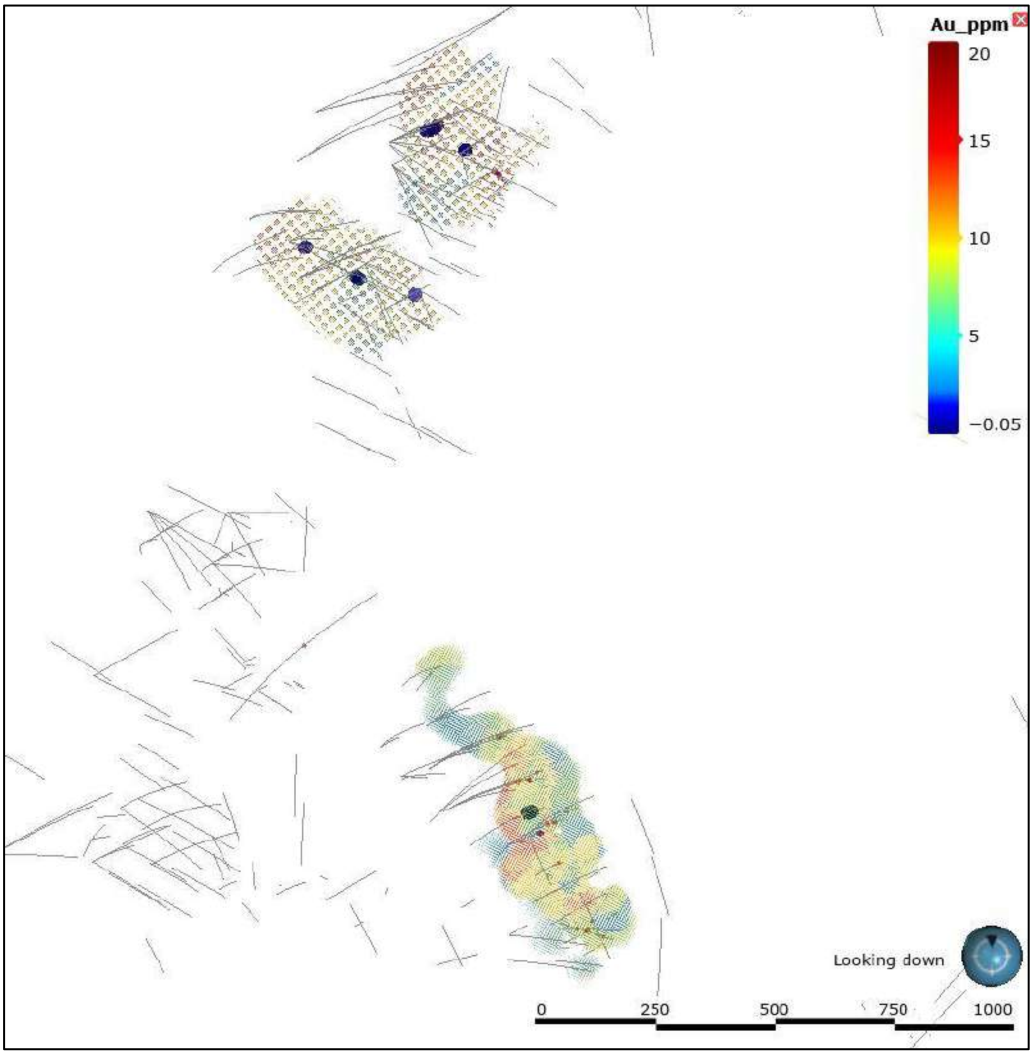


Figure 13-1: Drillhole plan for selected samples

13.3 Metallurgical Testing Plans and Laboratory Selection

Analysis of the early series of testwork as part of the BATCircle 1.0 research program (REM, 2021) provided direction for development of preliminary metallurgical flowsheets for processing mineralization from the Rajapalot deposits. Key objectives for processing were defined:

1. Highest value is in gold, therefore high recovery of gold to high payable product is primary objective.

2. Recovery of cobalt into saleable products.

The concept that forms the basis of this study is Whole-of-Ore (WoO) leach for gold recovery, followed by recovery of cobalt in sulphides. The concept flowsheet is presented in Figure 13-2.

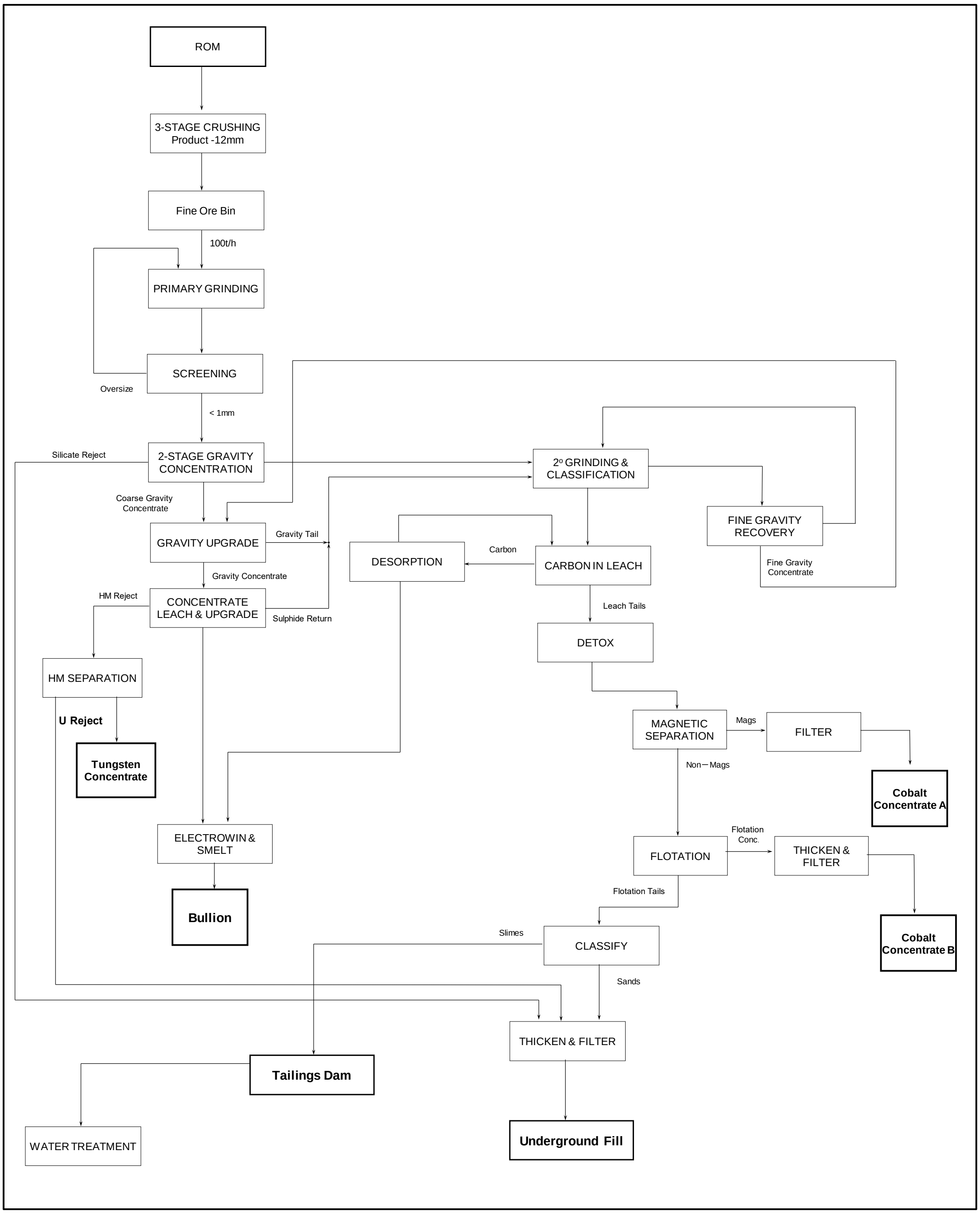


Figure 13-2: Whole-of-Ore leach first concept flowsheet

To gain insight to other potential treatment scenarios, some parallel testwork was undertaken. The results for this work, relevant to alternate flowsheet options, are not presented in this report, but will remain under consideration for future study stages

As the valuable constituents are associated with higher density minerals, it was considered that there was potential for up-front coarse gravity separation to reject a barren silicate tail.

Fine gravity-based recovery was considered appropriate to maximise potential gold recovery. If warranted, further processing of gravity concentrates could be deployed to upgrade and separate constituents.

The initial testing in the BATCircle program had shown that Low Intensity Magnetic Separation (LIMS) was a viable process for recovering a considerable proportion of the cobalt containing pyrrhotite in the different feed types. Testing was planned to assess this as an adjunct to flotation recovery of sulphide minerals.

The elements of the conceptual flowsheet incorporated into the metallurgical testing plan to support the current PEA were:

- Size reduction to <1 mm.
- Bulk Gravity-based separation; ‘barren’ reject assessment.
- Size reduction to achieve liberation for leach and flotation.
- Fine Gravity-based Separation; coarse gold recovery.
- Upgrading and processing of gravity concentrates; investigate recovery of gold to bullion and other heavy minerals to separate products if warranted.
- WoO Cyanide Leach; maximum gold recovery.
- LIMS to recover pyrrhotite.
- WoO Sulphide flotation; recovery of Co bearing sulphide minerals.

The flotation testing conditions in the BATCircle testwork did not produce a good response from the pyrrhotite content, so it was considered that pyrrhotite needed to be recovered by LIMS ahead of flotation. However, in the early stages of the PEA testing campaign, flotation conditions more suited to pyrrhotite recovery gave a good response and so WoO flotation became the basis of investigating cobalt recovery.

13.3.1 Specific Testing Elements

Testing for each mineralization type selected included:

1. **Comminution Testing - Head sample**

   Bond Rod Mill Wi Test

   Bond Ball Mill Wi Test – 106 µm closing size

   Abrasion Test

2. **Grind Establishment - Head Sample**

   Crush to <3.35 mm

   Separate to 1 kg sub-samples

   Laboratory grind for various times

   Size products and establish grind time to product size relationship

3. **Coarse Gravity Separation of Feed - Crushed Head Sample**

   Crush/grind to <1 mm

   Classify at approximately 75 µm silica

Separate sands on laboratory shaking table; concentrate, middlings & tails

4. **Fine Gravity Recovery from Feed - Crushed Head Sample**

   Grind to P80 of 0.1 mm

   Feed to laboratory Knelson Concentrator (or similar)

5. **WoO Cyanide Leach - Crushed Head Sample**

   Range of grind sizes and leach times

6. **WoO LIMS - Crushed Head Sample**

   Grind to P80 of 0.075 mm

   Feed to laboratory LIMS

7. **WoO Flotation - Crushed Head Sample**

   Grind to P80 of 0.075 mm

   Sulphide flotation tests; selection of reagent and flotation time conditions

   Evaluation to compare against LIMS

13.3.2 Laboratory Selection

WAI was selected to undertake the metallurgical testing elements associated with comminution assessment, mineralogical characterisation, gravity separation and WoO cyanide leaching. The detailed results are presented in their report MM1573 September 2022.

The laboratory of the Circular Economy Solutions unit of the GTK was selected to undertake the metallurgical testing elements associated with magnetic separation and flotation. The detailed results are presented in their report (GTK, 2021).

This split of the program recognised differences in expertise of the laboratories and their personnel and built on the previous experience gained by the GTK laboratory in flotation of the feed types. However, it meant that a fully integrated procedure, based on optimised conditions for each stage, to match the selected flowsheet has not been able to be undertaken at the time of this reporting.

13.4 Sample Preparation

The selected coarse rejects were assembled at the laboratory of the regular Mawson assaying provider, CRS, where they were blended and split into two samples for each of the mineralization types, Raja-South Palokas and Palokas. One split of each was then despatched to the separate testing laboratories. The prepared core samples were sent separately to the WAI laboratory.

Each laboratory undertook standard protocols on receipt. These included homogenising and sub-sampling to produce appropriate samples for characterisation and testing.

13.5 Head Assay and Mineralogy

13.5.1 Head assay

Each laboratory undertook analysis of a prepared head sample. WAI used internal laboratory resources, utilising Aqua Regia digest for Au, Co, As, Fe, and S analysis, and ICP for multi-element analysis. GTK used external provider, CRS, utilising Fire Assay with ICP finish for gold analysis and XRF for multi-element analysis.

Both laboratories found repeatability for gold assaying was challenging. The inference was that the presence of coarse gold particles persisted through the size reduction process and resulted in inconsistencies resulting from sub-sampling practices. Of note is that the drill core assaying for resource definition utilises the PAL1000 method which involves processing of larger sample sizes. This method is not suitable for laboratory metallurgical testing in which only small sample masses may be available.

Table 13-3 presents assay comparisons of the different methods and laboratories for the major constituents and others of interest.

In general, there is good agreement between the sources of data apart from the lower-than-expected cobalt grade from the GTK-CRS XRF analysis – particularly for the Raja-South Palokas (R-SP) sample.

The differences between the different mineralization types are well demonstrated:

- The R-SP sample had considerably higher As and Co grades, representing the presence of cobaltite.
- The Palokas (Pal) sample was much higher in Fe, Mg and Ca content though similar in S grade, indicating significantly different types and abundances of silicates.
- In contrast, the R-SP sample was higher in K, Na and SiO_2.
- Both samples had only minor accessory base metals content of Cu and Ni.
- The R-SP sample had higher U and W.
- Both samples had not insignificant Ti content.

13.5.2 Size-by-Size Analysis

WAI undertook particle size analysis of a sub-sample of each mineralization type with assaying of size fractions for Au, Co, As, Fe, S, U and W (see Table 13-4):

- Coarser fractions lower grade in sulphides; namely, higher in silicates as expected based on probable breakage rates.
- Au slightly enriched in coarse as well as fines indicating Au association with silicates, particularly for Pal.

Table 13-3: Comparison of Head Sample Assaying

Rajapalot Head Sample Analyses

		Drill Core Assays Calc Estimate		WAI Aqua Regia analysis				GTK-CRS FA-ICP, XRF	
Element	Unit	Raja-South Palokas	Palokas	Raja-South Palokas	Palokas	Element	Unit	Raja-South Palokas	Palokas
Au	ppm	4.10	2.48	3.97	2.36	Au	ppm	3.81	2.60
Co	%	0.103	0.058	0.120	0.055	Co	%	0.075	0.049
As	%	0.070	0.006	0.078	0.005	As	%	0.060	0.005
Fe	%	9.15	15.2	9.13	14.7	Fe	%	8.74	14.4
S	%	3.26	3.30	4.10	3.71	S	%	3.81	3.67

		Calc Estimate		ICP Head Analysis				XRF Head Analysis	
Element	Unit	Raja-South Palokas	Palokas	Raja-South Palokas	Palokas	Element	Unit	Raja-South Palokas	Palokas
Al	%	7.69	6.76	7.51	6.23	Al2O3	%	15.29	12.72
Ca	%	0.71	1.32	0.70	1.34	CaO	%	1.00	1.99
Cu	ppm	397	366	413	376	Cu	ppm	379	379
K	%	1.92	0.40	2.01	0.35	K2O	%	2.38	0.44
Mg	%	2.40	5.97	2.25	5.59	MgO	%	4.49	10.07
Mo	ppm	75.4	30.6	92.1	40.9	MoO3	ppm	170	103
Na	%	2.38	1.23	2.46	1.28	Na2O	%	4.25	2.19
Ni	ppm	166	183	182	200	Ni	ppm	165	191
						SiO2	%	55.1	47.3
Ti	%	0.256	0.300	0.243	0.276	TiO2	%	0.479	0.510
U	ppm	72	11.5	67	10.3	U	ppm	60	-
W	ppm	277	105	359	114	W	ppm	373	98

Table 13-4: Size-by-Size Analysis

Sample: **Raja-South Palokas**

Size	Weight	% Retained		Cumulative	Assay (ppm)	Assay (%)				Distribution (%)				
µm	grams	Individual	Cumulative	% Passing	Au	Co	As	Fe	S	Au	Co	As	Fe	S
600	715.0	14.43	14.43	85.57	3.58	0.104	0.063	8.08	2.91	11.9	12.7	13.0	12.7	10.4
425	610.1	12.31	26.74	73.26	3.91	0.107	0.059	9.83	3.79	11.1	11.2	10.4	13.2	11.6
300	485.0	9.79	36.53	63.47	2.81	0.118	0.056	11.55	5.05	6.3	9.9	7.9	12.3	12.3
212	440.8	8.90	45.43	54.57	6.40	0.116	0.052	11.70	5.18	13.1	8.7	6.7	11.3	11.5
150	567.7	11.46	56.89	43.11	2.84	0.104	0.048	10.10	4.71	7.5	10.2	7.9	12.6	13.4
106	583.1	11.77	68.65	31.35	3.92	0.095	0.045	8.36	3.72	10.7	9.6	7.7	10.7	10.9
75	607.5	12.26	80.91	19.09	4.84	0.100	0.057	7.48	3.36	13.7	10.4	10.0	10.0	10.3
53	461.2	9.31	90.22	9.78	5.52	0.129	0.088	7.66	3.64	11.9	10.2	11.8	7.7	8.4
38	219.0	4.42	94.64	5.36	5.12	0.157	0.126	7.94	3.90	5.2	5.9	8.0	3.8	4.3
-38	265.5	5.36	100.00		6.94	0.244	0.216	9.93	5.20	8.6	11.1	16.6	5.8	6.9
Calc Feed	**4954.9**	**100.00**			*4.33*	*0.118*	*0.069*	*9.20*	*4.02*	*100.0*	*100.0*	*100.0*	*100.0*	*100.0*
Measured Feed					*3.97*	*0.120*	*0.078*	*9.13*	*4.10*					

Sample: **Palokas**

Size	Weight	% Retained		Cumulative	Assay (ppm)	Assay (%)				Distribution (%)				
µm	grams	Individual	Cumulative	% Passing	Au	Co	As	Fe	S	Au	Co	As	Fe	S
600	994.7	19.97	19.97	80.03	2.26	0.041	0.006	13.45	2.19	20.1	14.3	29.5	17.7	12.0
425	590.1	11.85	31.82	68.18	2.78	0.044	0.006	13.50	2.38	14.7	9.1	17.3	10.6	7.7
300	464.1	9.32	41.14	58.86	1.86	0.051	0.003	14.60	3.32	7.7	8.3	7.3	9.0	8.5
212	414.7	8.33	49.47	50.53	1.42	0.056	0.003	15.45	4.05	5.3	8.1	5.0	8.5	9.2
150	402.0	8.07	57.54	42.46	1.33	0.059	0.003	15.95	4.48	4.8	8.2	5.4	8.5	9.9
106	887.9	17.83	75.37	24.63	2.52	0.061	0.002	16.10	4.53	20.0	18.7	8.8	19.0	22.1
75	695.0	13.95	89.32	10.68	2.35	0.079	0.003	16.45	4.63	14.6	18.9	11.3	15.2	17.7
53	362.6	7.28	96.60	3.40	2.38	0.079	0.005	16.45	4.52	7.7	10.0	8.3	7.9	9.0
38	82.1	1.65	98.25	1.75	3.00	0.061	0.007	15.95	3.96	2.2	1.7	2.6	1.7	1.8
-38	87.2	1.75	100.00		3.69	0.093	0.010	16.70	4.54	2.9	2.8	4.3	1.9	2.2
Calc Feed	**4980.4**	**100.00**			*2.24*	*0.058*	*0.004*	*15.14*	*3.65*	*100.0*	*100.0*	*100.0*	*100.0*	*100.0*
Measured Feed					*2.36*	*0.055*	*0.005*	*14.70*	*3.71*					

13.5.3 Head Sample Mineralogy

Selected size fractions of each sample were submitted for detailed scanning electron microscopy (SEM) mineralogical analysis (Petrolab, 2022).

The modal mineralogy of the two samples is presented in Figure 13-3 and Table 13-5.

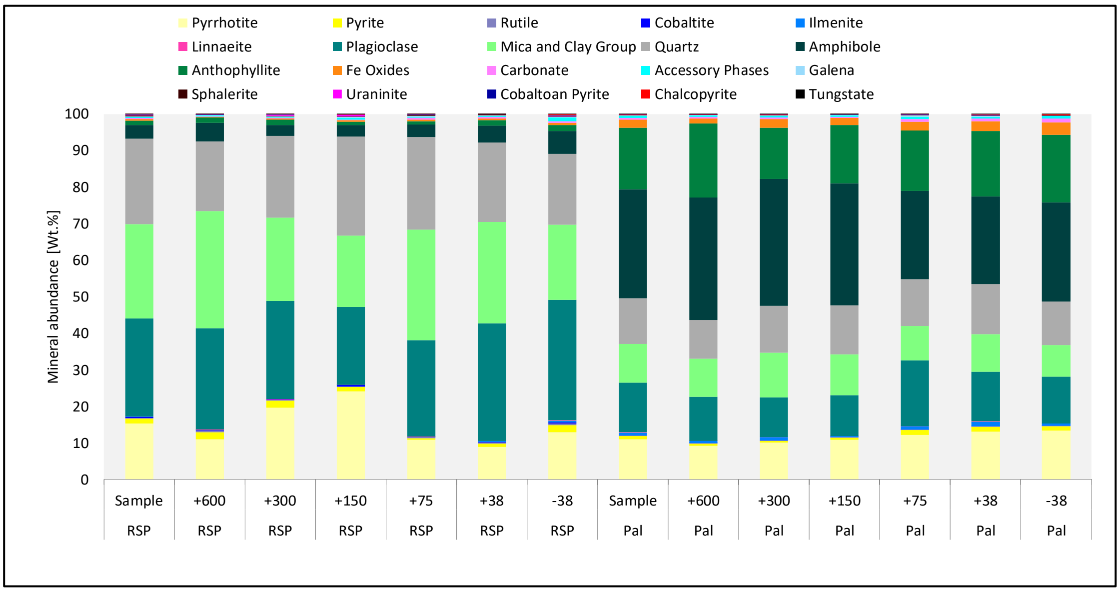


Figure 13-3: Mineral Abundance data by size fraction and overall

Table 13-5: Mineral Abundance data by size fraction and overall

[Wt.%]	RSP	RSP	RSP	RSP	RSP	RSP	RSP	Pal	Pal	Pal	Pal	Pal	Pal	Pal
	Sample	+600	+300	+150	+75	+38	-38	Sample	+600	+300	+150	+75	+38	-38
Pyrrhotite	15.3	11.0	19.6	24.1	10.9	8.8	12.9	11.0	9.2	10.1	10.8	12.3	13.0	13.4
Pyrite	1.3	1.9	2.1	1.2	0.5	1.0	2.0	0.90	0.60	0.48	0.65	1.3	1.4	1.2
Rutile	0.21	0.35	0.13	0.10	0.22	0.25	0.36	0.12	0.13	0.11	0.11	0.12	0.12	0.20
Cobaltite	0.19	0.17	0.07	0.45	0.05	0.20	0.33	0.01	0.00	0.02	0.01	0.02	0.01	0.00
Ilmenite	0.11	0.11	0.10	0.02	0.02	0.27	0.52	0.75	0.65	0.88	0.32	0.81	1.26	0.58
Linnaeite	0.06	0.04	0.06	0.06	0.08	0.02	0.07	0.06	0.04	0.03	0.04	0.10	0.06	0.05
Plagioclase	26.8	27.7	26.8	21.3	26.4	32.1	32.9	13.7	12.0	10.9	11.2	17.9	13.6	12.7
Mica and Clay Group	25.8	32.1	22.8	19.5	30.2	27.7	20.6	10.6	10.6	12.2	11.2	9.4	10.3	8.6
Quartz	23.4	19.1	22.5	27.1	25.4	21.8	19.4	12.5	10.6	12.8	13.3	12.7	13.8	12.0
Amphibole	3.8	5.1	3.0	3.2	3.4	4.6	6.2	29.8	33.5	34.6	33.4	24.2	23.9	27.1
Anthophyllite	1.2	1.5	1.4	0.84	0.94	1.6	1.6	16.8	20.3	14.0	16.0	16.5	17.9	18.4
Fe Oxides	0.37	0.18	0.28	0.42	0.39	0.47	0.71	2.2	1.4	2.4	1.9	2.4	2.7	3.5
Carbonate	0.27	0.13	0.13	0.26	0.37	0.43	0.42	0.56	0.47	0.50	0.40	0.67	0.73	0.99
Accessory Phases	0.32	0.28	0.26	0.37	0.27	0.22	0.97	0.49	0.41	0.52	0.33	0.58	0.54	0.68
Galena	0.22	0.09	0.18	0.28	0.32	0.20	0.17	0.24	0.07	0.12	0.17	0.46	0.29	0.10
Sphalerite	0.22	0.07	0.14	0.25	0.36	0.24	0.23	0.14	0.05	0.11	0.09	0.25	0.15	0.08
Uraninite	0.17	0.00	0.41	0.32	0.01	0.03	0.15	0.00	0.00	0.02		0.00	0.00	
Cobaltoan Pyrite	0.10	0.11	0.09	0.09	0.15	0.03	0.08	0.04	0.05	0.04	0.03	0.03	0.04	0.05
Chalcopyrite	0.10	0.09	0.08	0.18	0.04	0.05	0.27	0.13	0.17	0.15	0.11	0.07	0.21	0.28
Tungstate	0.01	0.00	0.01	0.01	0.01	0.02	0.10	0.03	0.00	0.01	0.02	0.04	0.05	0.06

The R-SP sample was predominantly approximately equal proportions of plagioclase, the mica and clay group, and quartz. The Pal sample had significantly more amphibole, including anthophyllite. Pyrrhotite was a significant minor phase of both samples.

As anticipated from assay data, the R-SP sample had considerably more cobaltite than the Pal sample and the Pal sample had significantly more Fe oxides. No uraninite was detected in the Pal sample.

The distributions of selected elements across the identified minerals are presented in Figure 13-4.

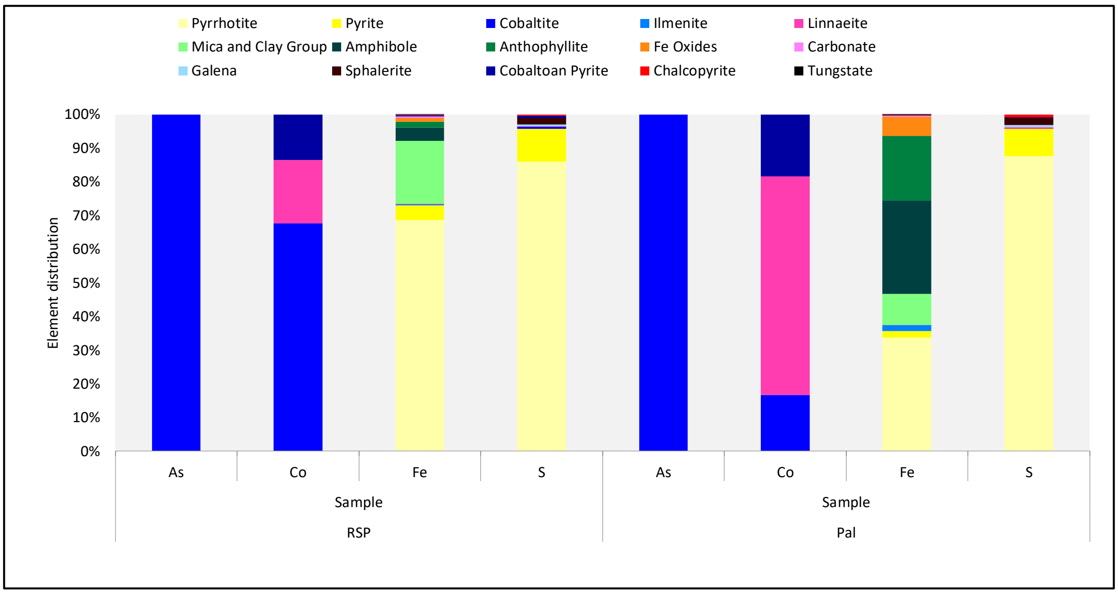


Figure 13-4: Distributions of selected elements across the identified minerals

Arsenic was exclusively hosted in cobaltite. Most of the cobalt present in the R-SP sample was hosted in cobaltite whilst the cobalt in the Pal sample was predominantly hosted in linnaeite.

As noted previously, Fe occurrence was significantly different for the two samples.

Sulphur was almost exclusively associated with pyrrhotite with minor pyrite.

Detailed liberation analysis indicated that cobaltite in the R-SP sample was quite coarse, d80 approximately 200 µm, and was largely liberated in sizes below 150 µm but much finer and poorly liberated in the Pal sample.

Linnaeite was almost exclusively locked and associated with pyrrhotite at all particle sizes in both samples. The pyrrhotite itself was quite coarse grained and well liberated at sizes below 150 µm. Grade-recovery predictions for pyrrhotite separation indicated that high recovery to high grade of mineral is possible.

13.6 Comminution

13.6.1 Abrasion Index

Bond Abrasion Index testing was performed. Based on the standard classification criteria, both samples were characterised as being “slightly abrasive”. Results and indications of liner and grinding media wear rates are:

- Raja-South Palokas
  - Bond Abrasion Index: 0.290
  - Wet Ball Mill Media Wear Rate: 104 g/kWh
  - Wet Ball Mill Liner Wear Rate: 8.0 g/kWh
  - Crusher Liner Wear Rate: 21.0 g/kWh

- Palokas
  - Bond Abrasion Index: 0.295
  - Wet Ball Mill Media Wear Rate: 104 g/kWh
  - Wet Ball Mill Liner Wear Rate: 8.1 g/kWh
  - Crusher Liner Wear Rate: 21.2 g/kWh

Based on the standard classification criteria, both samples were characterised as being “slightly abrasive”.

13.6.2 Bond work index

Bond Work Index tests were undertaken on each sample. Both samples can be classified as “medium” with respect to coarse and fine grindability. Closing size for Rod Mill test was 850 µm, and 106 µm for the Ball Mill test.

- Raja-South Palokas
  - Rod Mill Work Index: 9.65 kWh/t
  - Ball Mill Work Index; 14.0 kWh/t
- Palokas
  - Rod Mill Work Index: 12.9 kWh/t
  - Ball Mill Work Index: 14.1 kWh/t

Both samples can be classified as “medium” with respect to coarse and fine grindability.

13.7 Gravity Testwork

13.7.1 Coarse gravity

Both samples were tested using a laboratory shaking table to assess upgrading potential for Raja-South Palakos (Table 13-6) and Palakos (Table 13-7). Samples were crushed to pass 1.0 mm and screened at 75 µm. The objective of the testing was to assess whether a ‘barren’ reject could be produced ahead of fine grinding.

Table 13-6: Coarse gravity testwork for Raja-South Palokas

Product	Mass		Assay (ppm)	Assay (%)				Distribution (%)				
	(g)	(%)	Au	Co	As	Fe	S	Au	Co	As	Fe	S
Concentrate	460.8	4.85	44.8	0.391	0.203	30.60	20.43	47.60	19.14	14.55	20.57	28.40
Middlings 1	922.4	9.70	3.34	0.172	0.064	17.39	10.56	7.10	16.85	9.16	23.41	29.39
Middlings 2	3,372.0	35.46	2.08	0.055	0.053	3.97	1.32	16.14	19.71	27.82	19.54	13.46
Tailings	2,400.7	25.25	1.82	0.033	0.031	3.57	0.43	10.07	8.43	11.74	12.51	3.11
Shaking Table Feed	**7,155.9**	**75.25**	**4.91**	**0.084**	**0.0569**	**7.28**	**3.44**	**80.91**	**64.14**	**63.26**	**76.03**	**74.35**
Sub-Total C + M1	**1,383.2**	**14.5**	**17.2**	**0.245**	**0.110**	**21.8**	**13.8**	**54.7**	**36.0**	**23.7**	**44.0**	**57.8**
Sub-Total M2 + Tail	**5,772.7**	**60.7**	**1.97**	**0.046**	**0.044**	**3.81**	**0.95**	**26.2**	**28.1**	**39.6**	**32.1**	**16.6**
-75µm	2,353.0	24.75	3.52	0.144	0.100	6.98	3.61	19.09	35.86	36.74	23.97	25.65
Feed	9,508.9	100.00	4.57	0.099	0.068	7.21	3.49	100.00	100.00	100.00	100.00	100.00
Head			3.97	0.120	0.078	9.38	4.10					

Results for the R-SP sample indicated that Au, Co and particularly S can be upgraded to a small mass gravity concentrate but significant values remained in the tailings stream. The results are consistent with the sized assay and mineralogical data, which indicated coarse liberation of cobaltite and pyrrhotite and presence of gold associated with coarse silicates.

Table 13-7: Coarse gravity testwork for Palokas

Product	Mass		Assay (ppm)	Assay (%)				Distribution (%)				
	(g)	(%)	Au	Co	As	Fe	S	Au	Co	As	Fe	S
Concentrate	746.1	8.08	7.0	0.167	0.017	21.35	13.12	20.86	23.35	15.44	23.07	29.83
Middlings 1	925.3	10.02	2.14	0.063	0.011	8.55	4.24	7.94	10.89	11.69	11.46	11.95
Middlings 2	3,345.0	36.22	2.34	0.027	0.010	4.24	1.46	31.41	16.82	38.09	20.52	14.91
Tailings	1,212.7	13.13	2.96	0.014	0.008	3.28	0.58	14.43	3.24	11.57	5.75	2.13
Shaking Table Feed	**6,229.1**	**67.46**	**2.98**	**0.047**	**0.0104**	**6.74**	**3.10**	**74.64**	**54.29**	**76.80**	**60.80**	**58.82**
Sub-Total C + M1	**1,671.4**	**18.1**	**4.29**	**0.110**	**0.014**	**14.3**	**8.2**	**28.8**	**34.2**	**27.1**	**34.5**	**41.8**
Sub-Total M2 + Tail	**4,557.7**	**49.4**	**2.50**	**0.024**	**0.009**	**3.98**	**1.23**	**45.8**	**20.1**	**49.7**	**26.3**	**17.0**
-75µm	3,005.0	32.54	2.10	0.081	0.006	9.01	4.50	25.36	45.71	23.20	39.20	41.18
Feed	9,234.1	100.00	2.70	0.058	0.009	7.48	3.55	100.00	100.00	100.00	100.00	100.00
Head			2.36	0.055	0.005	8.31	3.71					

As indicated from the mineralogical data, the Pal sample exhibited less liberation than the R-SP sample.

The test results do not demonstrate any advantage would be gained by including a coarse gravity stage in the process flowsheet.

13.7.2 Fine gravity

Both samples were tested to evaluate the gravity recoverable gold content and the deportment of other heavy minerals (see Table 13-8 and Figure 13-5). A 10 kg sample of each was ground to 80% passing 106 µm and processed in a laboratory Knelson centrifugal concentrator. The Knelson concentrate was separated on a Mozley table. Unfortunately, insufficient sample masses were produced to be able to analyse all samples for the full range of elements of interest.

Table 13-8: Fine gravity testwork results for Raja-South Palokas and Palokas

	Raja-South Palokas				Palokas			
Product	Mass		Assay (ppm)	Distribution (%)	Mass		Assay (ppm)	Distribution (%)
	(g)	(%)	Au	Au	(g)	(%)	Au	Au
Mozley Concentrate 1	0.64	0.0064	3,070	5.51	0.23	0.0024	5,121	7.47
Mozley Concentrate 2	6.58	0.066	998	18.43	4.90	0.051	107	3.33
Mozley Concentrate 3	8.05	0.081	41.6	0.94	16.20	0.168	11.1	1.14
Mozley Tailings	51.78	0.521	41.2	5.98	62.00	0.643	20.3	8.00
Calc Knleson Concentrate	67.05	0.674	164	30.87	83.33	0.864	37.7	19.94
Knelson Tailings	9,880.0	99.33	2.49	69.13	9,560.0	99.14	1.32	80.06
Calc. Feed	9,947.1	100.00	3.58	100.00	9,643.3	100.00	1.63	100.00
Head			3.97				2.63	
Calculated products								
Conc 1		0.006	3,070	5.5		0.002	5,121	7.5
Conc 1 + 2		0.073	1,182	23.9		0.053	332	10.8
Conc 1 + 2 + 3		0.154	581	24.9		0.221	88	11.9
Knelson Conc		0.674	164	30.9		0.864	38	19.9

Figure 13-5: Fine gravity testwork results for Raja-South Palokas and Palokas

The results confirm the presence of gravity recoverable gold in both samples. The primary concentrates from each sample were able to be upgraded to high grade concentrates suitable for further treatment to produce bullion.

The R-SP showed considerably greater gold liberation than Pal with higher recovery and grade for same mass recovery, summarised as follows:

- 0.033% Mass Recovery (1.2 tpd for 3,600 tpd feed)
- Raja: approximately 18% Au recovery to approximately 1,800 ppm Au
- Palokas: approximately 10% Au recovery to approximately 500 ppm Au

Insufficient sample mass was available to determine grade-recovery relationships for Co, As, U, W and Ti, though there was indication that the Raja sample showed more significant upgrading of Co and As to gravity concentrates, but at very low recoveries.

13.8 Gold Leaching

Kinetic cyanide leach testing was performed on samples of each mineralization employing a range of grind sizes, initial cyanide concentrations, and leach times. Results showed anomalous outcomes with inconsistent outcomes in terms of trends related to grind size or leach time.

The results spread was interpreted as demonstrating the effect of coarse particles of gold in some feed samples which exhibited slower leaching kinetics and resulted in higher Au grade residues regardless of grind size, and independent of cyanide concentration.

Leaching of Knelson tails samples, both without and with a regrind was tested. Results clearly demonstrated that low residue grades, similar to those achieved by long residence time and high cyanide concentration, could be achieved by this approach and that gold extraction was improved by grinding of Knelson tailings.

Two samples of high Au grade residue from each type were tested by an intensive cyanide leach. High Au extraction was achieved, demonstrating that the gold was not present in a refractory form.

Results for both samples are presented in the following:

- Leach test results for Raja-South Palokas sample (Table 13-9 and Figure 13-6).
- Leach test results for Palokas sample (Table 13-10 and Figure 13-7).

Table 13-9: Leach test results for Raja-South Palokas sample

Raja-South Palokas Test	Grind 80% Pass	NaCN Conc g/l	Calc Feed Au, g/t	Leach time hr	Residue Au, g/t	Final Soln Au, ppm	Calc Extract %	NaCN kg/t	Lime kg/t
Assay Head			3.97						
LT1	150	1.00	4.71	24	1.54	2.12	67.4	1.43	0.44
LT2	100	1.00	4.32	24	0.51	2.52	88.3	1.55	0.48
LT3	**75**	**1.00**	**3.98**	**24**	**0.50**	**2.31**	**87.4**	**1.39**	**0.57**
LT9	150	2.00	4.31	48	0.32	2.54	92.6	3.57	1.12
LT10	100	2.00	4.30	48	0.22	2.64	95.0	3.94	0.77
LT11	50	2.00	4.60	48	0.11	2.89	97.6	4.69	1.00
LT15	100	1.00	4.88	32	0.67	2.60	86.2	1.73	0.65
LT16	75	1.00	4.68	32	0.79	2.41	83.1	1.96	0.61
LT17	**50**	**1.00**	**4.93**	**32**	**0.27**	**2.87**	**94.6**	**2.10**	**0.66**
			4.52						
Knelson Tail			2.49						
LT7	106	1.00	2.53	48	0.25	1.41	90.1	2.25	1.30
LT21	**75**	**1.00**	**2.76**	**32**	**0.16**	**1.57**	**94.2**	**1.68**	**0.72**
Intensive Leach									
LT16 Residue	**75**	**2.00**	**0.96**	**48**	**0.10**	**0.53**	**89.6**		

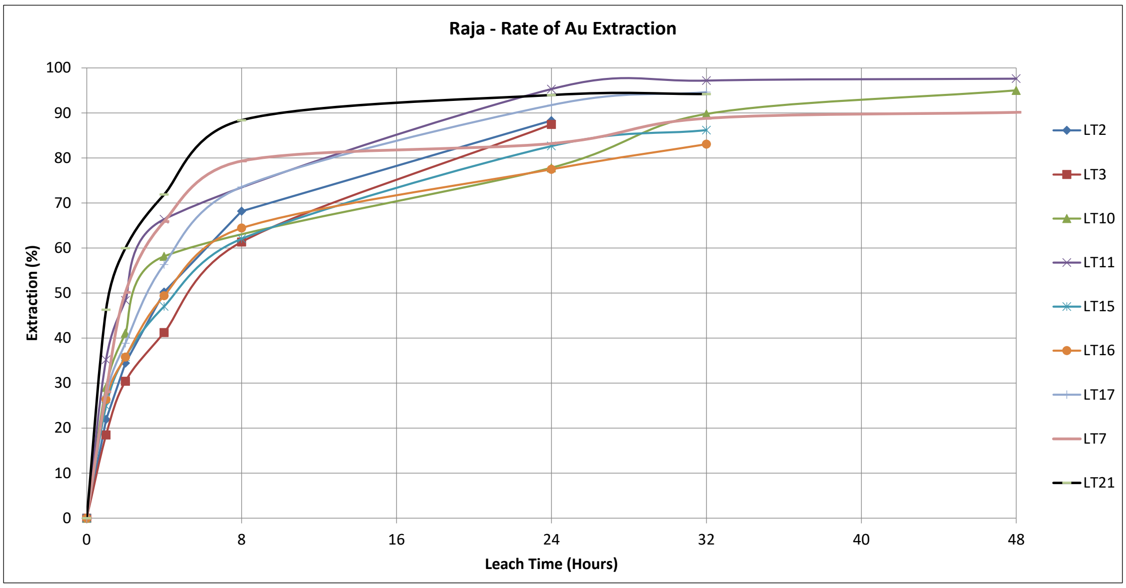


Figure 13-6: Leach test results for Raja-South Palokas sample

Table 13-10: Leach test results for Palokas sample

Palokas Test	Grind 80% Pass	NaCN Conc g/l	Calc Feed Au, g/t	Leach time hr	Residue Au, g/t	Final Soln Au, ppm	Calc Extract %	NaCN kg/t	Lime kg/t
Assay Head			**2.36**						
LT4	150	1.00	2.60	24	0.70	1.26	73.0	1.28	0.24
LT5	100	1.00	3.05	24	0.26	1.84	91.6	0.94	0.50
LT6	**75**	**1.00**	**2.93**	**24**	**0.25**	**1.75**	**91.6**	**1.20**	**0.52**
LT12	150	2.00	2.68	48	0.18	1.59	93.5	2.96	1.31
LT13	100	2.00	2.66	48	0.09	1.66	96.6	2.86	1.34
LT14	50	2.00	3.33	48	0.04	2.12	98.8	3.35	0.64
LT18	100	1.00	3.04	32	0.74	1.42	75.7	1.09	0.36
LT19	75	1.00	2.85	32	0.09	1.72	96.8	1.34	0.42
LT20	**50**	**1.00**	**2.64**	**32**	**0.24**	**1.48**	**90.8**	**1.39**	**0.36**
			2.86						
Knelson Tail			**1.32**						
LT8	106	1.00	1.76	48	0.18	1.01	89.8	2.47	0.28
LT22	**75**	**1.00**	**1.56**	**32**	**0.08**	**0.89**	**94.9**	**1.36**	**0.50**
Intensive Leach									
LT18 Residue	**100**	**2.00**	**0.82**	**48**	**0.09**	**0.46**	**89.0**		

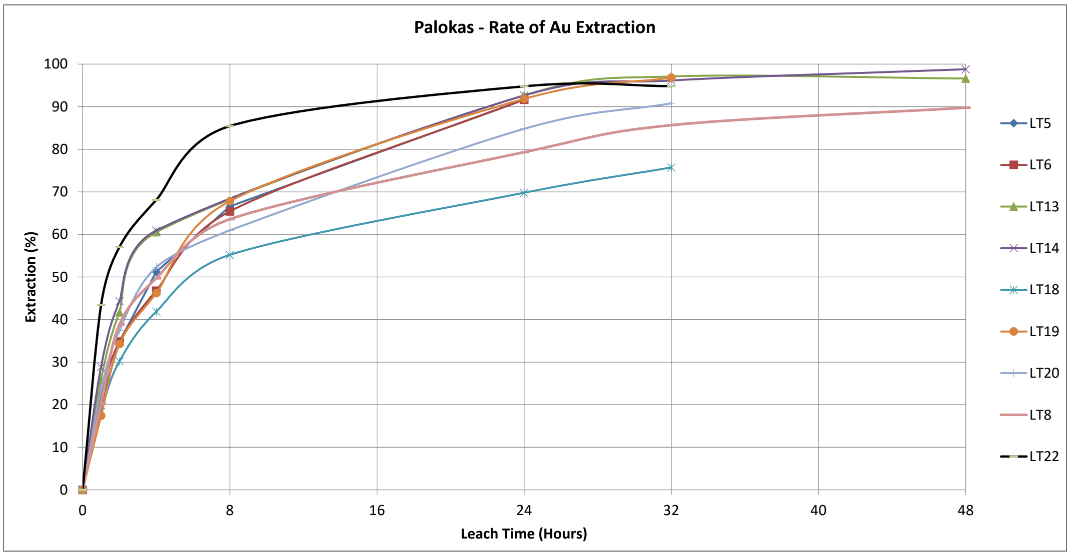


Figure 13-7: Leach test results for Palokas sample

It was concluded that gold recovery is higher when subjected to a combination of gravity recovery and cyanide leaching. Good extraction, low residue gold grade, can be achieved at moderate grind size and leach time.

The presence of coarse gold particles is a product of both the nature of the gold occurrence, and an artifact of the batch grinding undertaken for laboratory testwork. In a process plant, grinding will be in closed-circuit, with an integral gravity recovery circuit, reducing the likelihood of coarse gold reaching the leaching stage.

Therefore, for both potential feed types, there is confidence in predicting high total gold extraction, approximately 95%, in moderate leach times at standard cyanide concentration in a 'typical' commercial operation incorporating relatively fine grinding of silicate host with integrated coarse gold recovery by gravity separation and intensive leaching.

Reagent consumptions were in the normal range of expectations. Lime consumption was quite low, reflecting the relatively high natural pH of the samples tested. Cyanide consumptions in the testing were generally related to leach time. At high cyanide concentration there was slightly elevated Cu extraction, which would have added to cyanide consumption in this series of tests. There was no indication of any adverse effects related to sulphide content in the feed samples. The pyrrhotite appears to be unreactive in terms of oxidation and cyanide consumption.

13.9 Magnetic Separation and Flotation

Magnetic separation and flotation testing was performed by GTK.

13.9.1 Magnetic Separation

Initial testing was conducted on each sample after grinding to 80% passing 75 µm. Two separate tests were conducted: a Low Intensity separation at 0.07T field strength; and a Medium Intensity Separation at 0.3T field strength.

Results for the R-SP sample are presented in Table 13-11.

Table 13-11: Raja-South Palokas Magnetic Separation Test Results

Raja_M1	Wt	ASSAYS (%)								DISTRIBUTION (%)							
LIMS 0.07 T	%	Au (g/t)	Co	As	Fe	S	SiO2	MgO	Al2O3	Au	Co	As	Fe	S	SiO2	MgO	Al2O3
Mag Con	4.2	2.38	0.281	0.015	43.8	31.2	3.2	0.44	1.18	5.2	17.1	1.2	20.9	38.1	0.3	0.3	0.3
Tail	95.8	1.91	0.060	0.055	7.3	2.23	55.6	5.58	17.17	94.8	82.9	98.8	79.1	61.9	99.7	99.7	99.7
Calc Feed	100.0	1.93	0.069	0.053	8.8	3.45	53.4	5.36	16.50	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Assay Head		3.81	0.075	0.060	8.7	3.81	55.1	4.5	15.3								

Raja_M2	Wt	ASSAYS (%)								DISTRIBUTION (%)							
MIMS 0.3 T	%	Au (g/t)	Co	As	Fe	S	SiO2	MgO	Al2O3	Au	Co	As	Fe	S	SiO2	MgO	Al2O3
Mag Con	6.1	6.10	0.323	0.039	41.9	27.2	7.2	1.02	2.54	12.0	28.4	4.4	29.0	49.2	0.8	1.2	0.9
Tail	93.9	2.91	0.053	0.055	6.7	1.83	56.5	5.70	17.35	88.0	71.6	95.6	71.0	50.8	99.2	98.8	99.1
Calc Feed	100.0	3.11	0.070	0.054	8.9	3.38	53.5	5.41	16.44	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Assay Head		3.81	0.075	0.060	8.7	3.81	55.1	4.5	15.3								

The tests demonstrated that at the grind size tested, although a high S grade concentrate could be produced with very low dilution from silicates, there was still potential for significant gold loss to the magnetic concentrate when processing a WoO sample.

For the R-SP mineralization it appeared that a significant proportion of the Fe-S mineralization was other than magnetic pyrrhotite. This is likely to be a mixture of non-magnetic pyrrhotite and pyrite.

Cobalt recovery to magnetic product reflected the deportment of cobalt between non-magnetic cobaltite and linnaeite in magnetic pyrrhotite.

Results for the Pal sample are presented in Table 13-12.

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Table 13-12: Palokas Magnetic Separation Test Results

Palokas_M1	Wt	ASSAYS (%)								DISTRIBUTION (%)							
LIMS 0.07 T	%	Au (g/t)	Co	As	Fe	S	SiO2	MgO	Al2O3	Au	Co	As	Fe	S	SiO2	MgO	Al2O3
Mag Con	9.2	1.11	0.283	0.001	45.1	28.1	3.8	1.14	1.30	7.2	57.8	2.0	28.8	84.8	0.7	1.0	0.9
Tail	90.8	1.46	0.021	0.005	11.4	0.51	51.7	11.48	13.97	92.8	42.2	98.0	71.2	15.2	99.3	99.0	99.1
Calc Feed	100.0	1.43	0.045	0.005	14.5	3.05	47.3	10.52	12.80	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Assay Head		2.60	0.049	0.005	14.4	3.67	47.3	10.1	12.7								

Palokas_M2	Wt	ASSAYS (%)								DISTRIBUTION (%)							
MIMS 0.3 T	%	Au (g/t)	Co	As	Fe	S	SiO2	MgO	Al2O3	Au	Co	As	Fe	S	SiO2	MgO	Al2O3
Mag Con	11.0	2.38	0.261	0.002	41.2	24.7	9.0	2.70	2.83	18.9	64.2	4.0	31.2	88.7	2.1	2.8	2.4
Tail	89.0	1.26	0.018	0.006	11.2	0.39	51.9	11.63	13.95	81.1	35.8	96.0	68.8	11.3	97.9	97.2	97.6
Calc Feed	100.0	1.38	0.045	0.006	14.5	3.06	47.2	10.65	12.73	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Assay Head		2.60	0.049	0.005	14.4	3.67	47.3	10.1	12.7								

The Palokas sample produced a higher deportment of Au to the magnetic concentrate at both field strengths tested in line with the higher mass recovery.

For the Palokas sample there was a much higher recovery of S and cobalt to the magnetic concentrates indicating a high proportion of the Fe-S mineralization was magnetic pyrrhotite and a higher proportion of the cobalt was as linnaeite within pyrrhotite.

For both feed types, the ratio of S to Co in the magnetic concentrates indicate that the grade of Co in a high grade pyrrhotite concentrate, say 35% S, would be of the order of 0.35% Co.

13.9.2 Flotation

Testing was undertaken to investigate WoO flotation. To maximise recovery of all gold and cobalt containing constituents, bulk rougher flotation was initially applied. Two series of testing were conducted. The testing schedule was protracted as it was necessary to receive assays from a test series before deciding on conditions for the next series, and assay turnaround was slow to begin with.

Test conditions and outcomes are presented and discussed below, and summary results presented in the following tables and figures, by feed type. The test numbers reflect other flotation investigations occurring in parallel with the design concept study.

Raja-South Palokas

- **FT-03** – Addition of H_2SO_4 to achieve pH of 6.0, $CuSO_4$ as activator, and combination of Danafloat 245 and SEX as collector for first flotation stages with further $CuSO_4$ and SEX for later stages (see Table 13-13 and Figure 13-8).
    - Significant recovery of all sulphide components, although gold recovery was lower at 83%.
    - Cobaltite exhibited faster kinetics than Fe-S components.
- **FT-07** – As per test FT-03 but without Danafloat and reduced SEX additions (see Table 13-14 and Figure 13-9).
    - Results very similar to FT-03 to slightly lower final recoveries apart from Au.

Palokas

- **FT-04** – Addition of H_2SO_4 to achieve pH of 6.0, $CuSO_4$ as activator, and combination of Danafloat 245 and SEX as collector for first flotation stages with further $CuSO_4$ and SEX for later stages (see Table 13-15 and Figure 13-10).
    - Very high S recovery but lower recovery of cobalt constituents.
    - Cobaltite exhibited faster kinetics but only a minor constituent of feed.
    - Final Au recovery of 85%.
- **FT-08** – As per test FT-04 but without Danafloat and reduced SEX additions (see Table 13-16 and Figure 13-11).
    - Results very similar to FT-04 to lower final recoveries.

The kinetic results for the individual elements of gold, cobalt, arsenic and sulphur for each test sample are presented respectively in Figure 13-12 to Figure 13-15.

Table 13-13: Bulk Float test results (FT-03) for Raja-South Palokas

BULK FLOAT Series

FT-03: Raj-IIA **Raja - South Palokas**

P80 = 75 µm

	Reagents (g/t)				
	H2SO4	CuSO4	SEX	nafloat 2	MIBC
Tot.	2190	350	130	60	50

	Flot min
RC1-5	15

Comments

Bulk float - High reagents

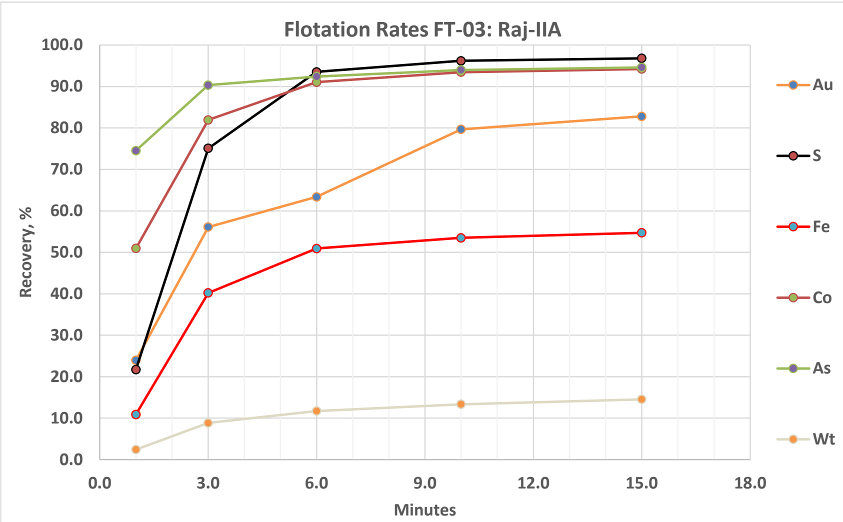
Very high recovery of As, Co & S.

Poor initial recovery of Au.

Poor rejection of SiO2 & MgO.

Cobaltite faster kinetics

FT-03: Raj-IIA	Wt	ASSAYS (%)									DISTRIBUTION (%)								
	%	Au (g/t)	Co	As	Cu	Fe	S	U	MgO	SiO2	Au	Co	As	Cu	Fe	S	U	MgO	SiO2
Con 1 - 4	13.3	21.9	0.518	0.335	0.218	34.2	25.3	0.006	2.31	15.4	79.7	93.5	94.0	82.1	53.5	96.2	15.1	7.0	3.7
Calc Tails	*86.7*	*0.86*	*0.006*	*0.003*	*0.007*	*4.58*	*0.15*	*0.005*	*4.76*	*62.0*	*20.3*	*6.5*	*6.0*	*17.9*	*46.5*	*3.8*	*84.9*	*93.0*	*96.3*
Calc Feed	100.0	3.68	0.074	0.048	0.035	8.52	3.51	0.005	4.43	55.8	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head Assay		3.81	0.075	0.060	0.038	8.74	3.81	0.006	4.49	55.1									



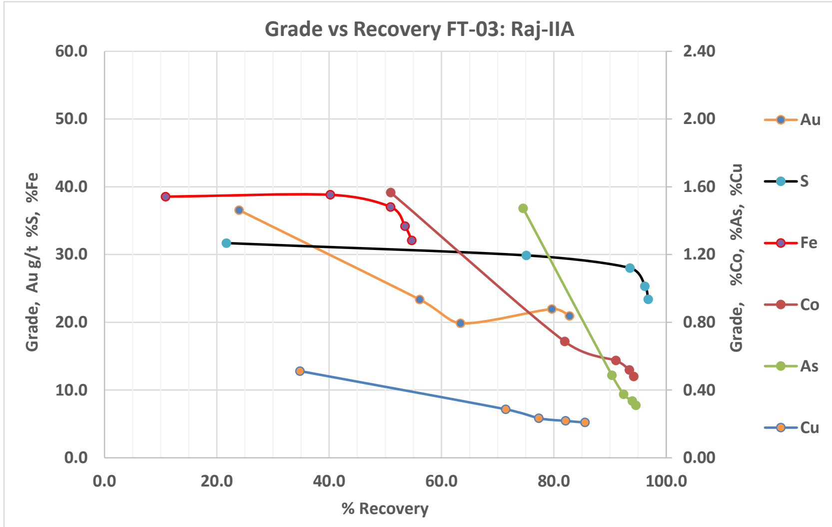


Figure 13-8: Bulk Float test results (FT-03) for Raja-South Palokas

Table 13-14: Bulk Float test results (FT-07) for Raja-South Palokas

FT-07: Raj-IIB **Raja - South Palokas**

P80 = 75 µm

	Reagents (g/t)				
	H2SO4	CuSO4	SEX	nafloat 2	MIBC
Tot.	1960	350	100	0	70

	Flot min
RC1-5	15

Comments

Bulk float - Low reagents

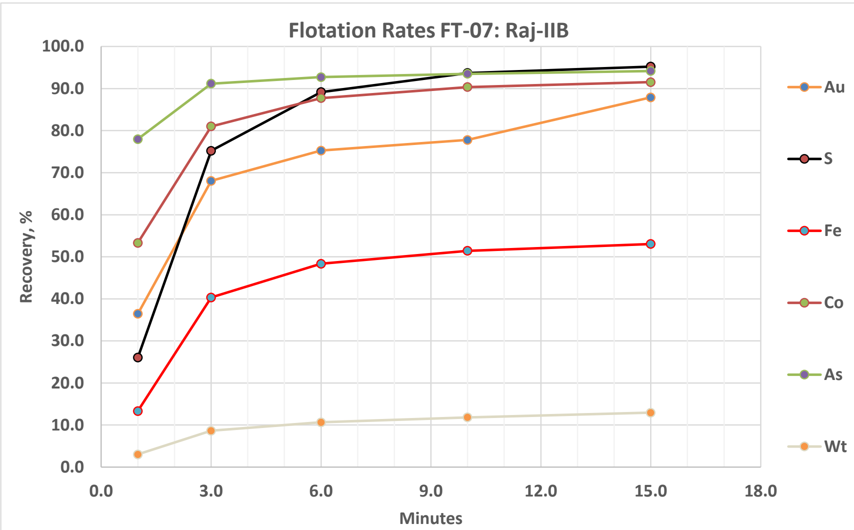
Very high recovery of As, Co & S.

Poor initial recovery of Au.

Better rejection of SiO2 & MgO.

Initial cobaltite selectivity

FT-07: Raj-IIB	Wt	ASSAYS (%)									DISTRIBUTION (%)								
	%	Au (g/t)	Co	As	Cu	Fe	S	U	MgO	SiO2	Au	Co	As	Cu	Fe	S	U	MgO	SiO2
Con 1 - 4	**11.8**	**21.8**	**0.550**	**0.356**	**0.244**	**36.3**	**27.1**	**0.002**	**2.37**	**11.9**	**77.8**	**90.4**	**93.5**	**78.6**	**51.4**	**93.7**	**4.3**	**6.1**	**2.5**
Calc Tails	***88.2***	*0.83*	*0.008*	*0.003*	*0.009*	*4.6*	*0.24*	*0.006*	*4.91*	*61.7*	*22.2*	*9.6*	*6.5*	*21.4*	*48.6*	*6.3*	*95.7*	*93.9*	*97.5*
Calc Feed	**100.0**	3.31	0.072	0.045	0.037	8.3	3.42	0.006	4.61	55.8	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head Assay		3.81	0.075	0.060	0.038	8.74	3.81	0.006	4.49	55.1									



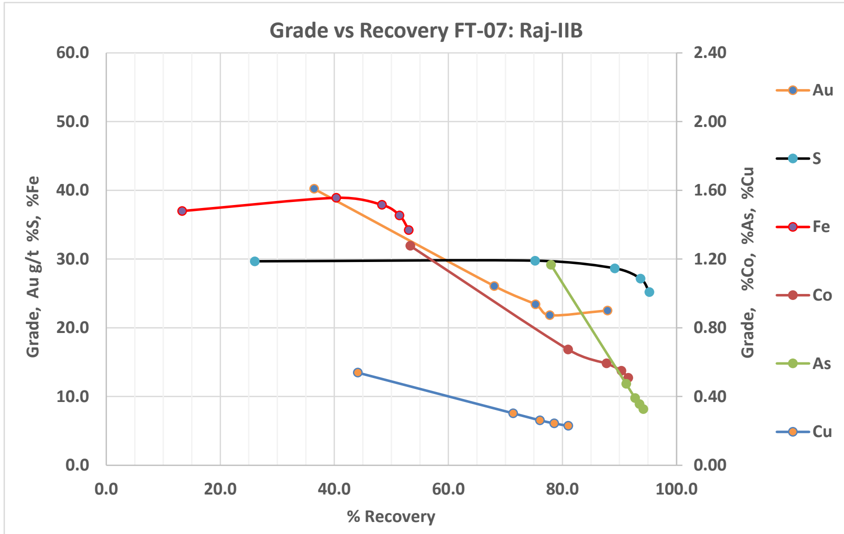


Figure 13-9: Bulk Float test results (FT-07) for Raja-South Palokas

Table 13-15: Bulk Float test results (FT-04) for Palokas

FT-04: Pal-IIA **Palokas**

P80 = 74 µm

	Reagents (g/t)				
	H2SO4	CuSO4	SEX	nafloat 2	MIBC
Tot.	2920	350	130	60	70

	Flot min
RC1-5	15

Comments

Bulk float - High reagents

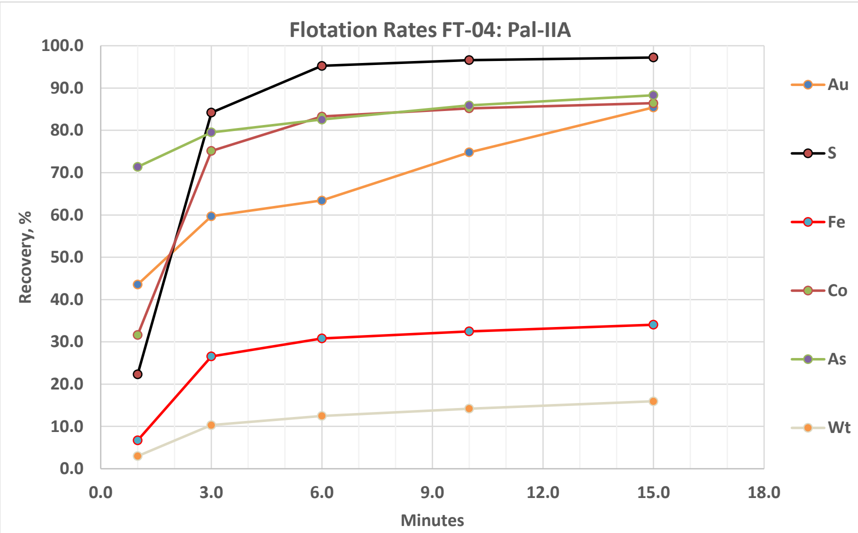
Very high recovery of As, Co & S.

Poor initial recovery of Au.

Poor rejection of SiO2 & MgO.

Cobaltite faster kinetics

FT-04: Pal-IIA	Wt	ASSAYS (%)									DISTRIBUTION (%)								
	%	Au (g/t)	Co	As	Cu	Fe	S	U	MgO	SiO2	Au	Co	As	Cu	Fe	S	U	MgO	SiO2
Con 1 - 4	**14.2**	**12.1**	**0.260**	**0.022**	**0.189**	**32.2**	**21.6**		**6.6**	**19.6**	**74.8**	**85.2**	**85.9**	**76.3**	**32.5**	**96.6**		**9.3**	**5.8**
Calc Tails	***85.8***	*0.68*	*0.007*	*0.001*	*0.010*	*11.1*	*0.13*		*10.6*	*52.8*	*25.2*	*14.8*	*14.1*	*23.7*	*67.5*	*3.4*		*90.7*	*94.2*
Calc Feed	**100.0**	2.31	0.043	0.004	0.035	14.1	3.18		10.1	48.1	100.0	100.0	100.0	100.0	100.0	100.0		100.0	100.0
Head Assay		2.60	0.049	0.005	0.038	14.36	3.67		10.07	47.3									



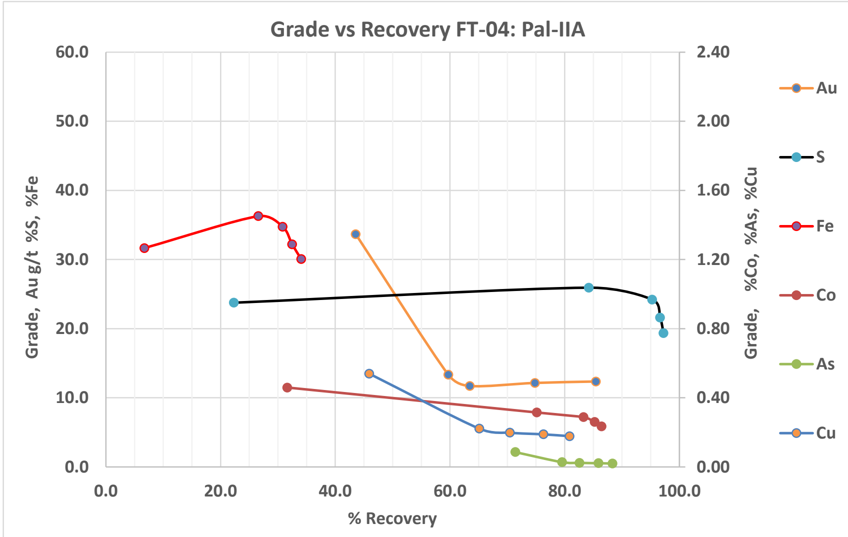


Figure 13-10: Bulk Float test results (FT-04) for Palokas

Table 13-16: Bulk Float test results (FT-08) for Palokas

FT-08: Pal-IIB Palokas

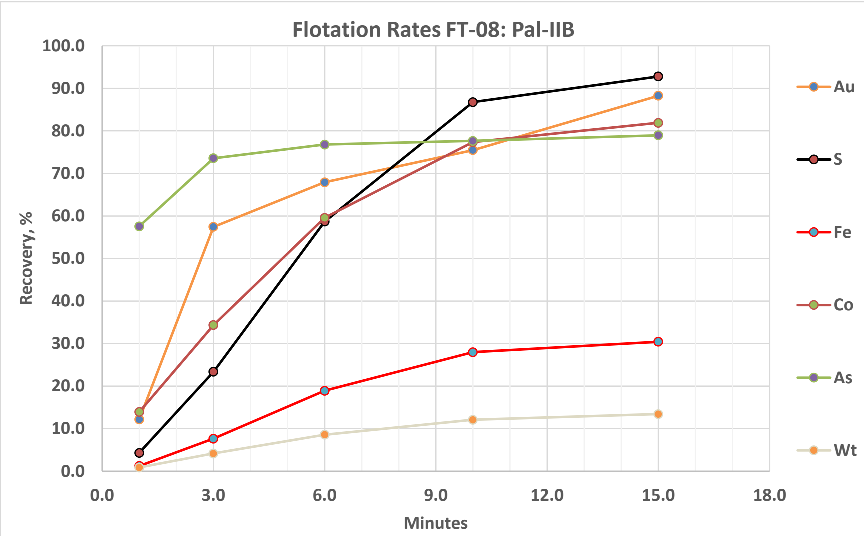
P80 = 75 µm

	Reagents (g/t)				
	H2SO4	CuSO4	SEX	nafloat 2	MIBC
Tot.	2430	350	100	0	70

	Flot min
RC1-5	15

Comments
Bulk float - Low reagents
Reduced recovery of As, Co & S.
Poor initial recovery of Au.
Better rejection of SiO2 & MgO.
Initial cobaltite selectivity

FT-08: Pal-IIB	Wt	ASSAYS (%)									DISTRIBUTION (%)								
	%	Au (g/t)	Co	As	Cu	Fe	S	U	MgO	SiO2	Au	Co	As	Cu	Fe	S	U	MgO	SiO2
Con 1 - 4	**12.1**	**15.2**	**0.276**	**0.026**	**0.212**	**32.2**	**21.9**		**7.1**	**19.3**	**75.5**	**77.4**	**77.6**	**70.9**	**28.0**	**86.8**		**8.3**	**4.8**
Calc Tails	***87.9***	*0.68*	*0.011*	*0.001*	*0.012*	*11.4*	*0.46*		*10.7*	*52.2*	*24.5*	*22.6*	*22.4*	*29.1*	*72.0*	*13.2*		*91.7*	*95.2*
Calc Feed	**100.0**	2.43	0.043	0.004	0.036	13.9	3.06		10.3	48.2	100.0	100.0	100.0	100.0	100.0	100.0		100.0	100.0
Head Assay		2.60	0.049	0.005	0.038	14.36	3.67		10.07	47.3									



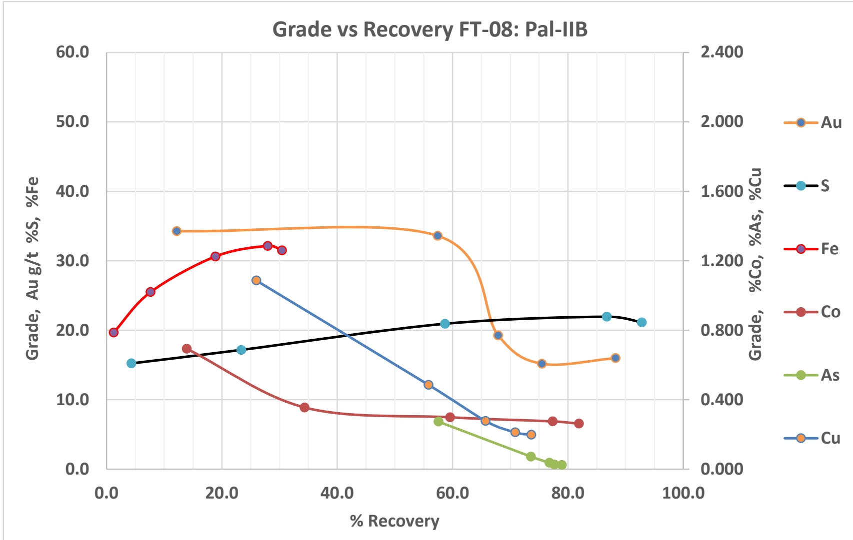


Figure 13-11: Bulk Float test results (FT-08) for Palokas

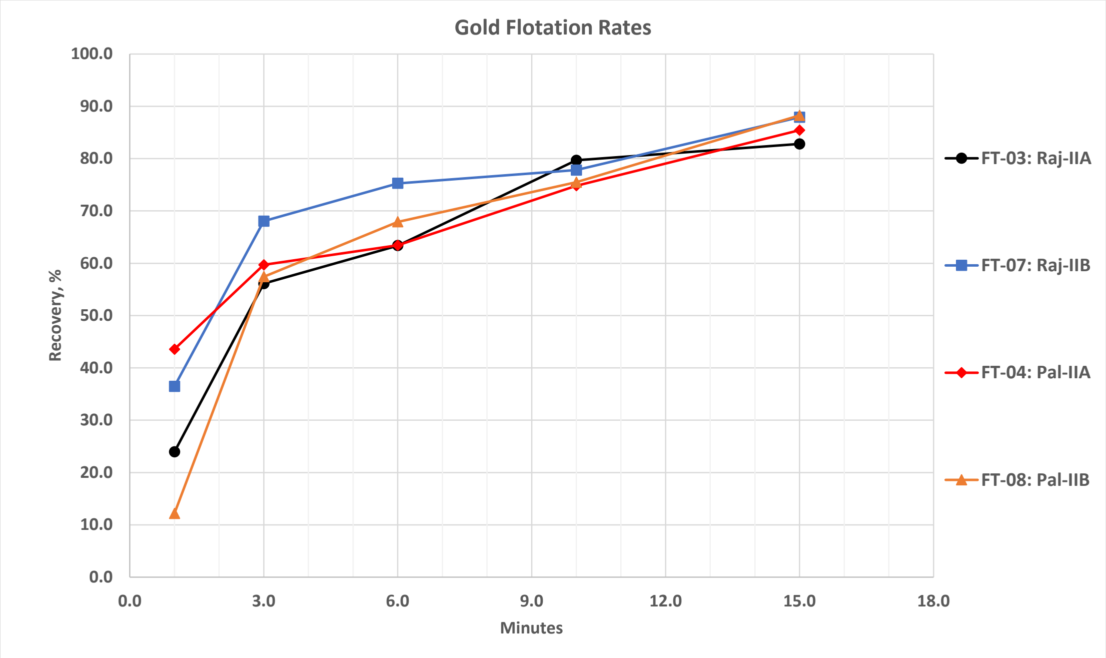


Figure 13-12: Kinetic test results for gold flotation rates

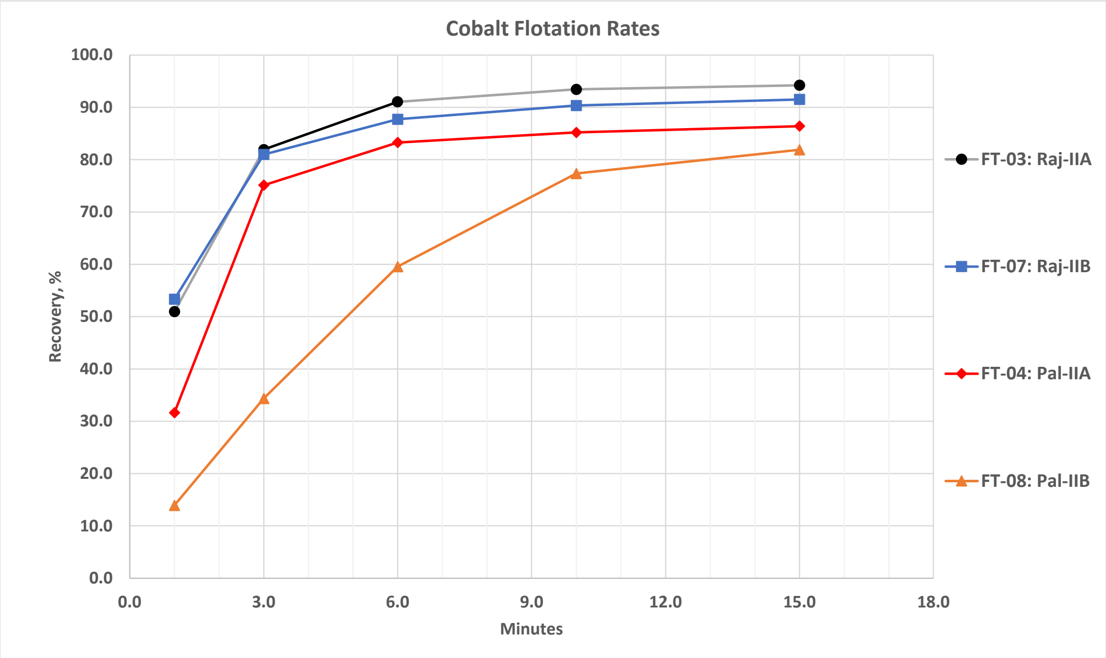


Figure 13-13: Kinetic test results for cobalt flotation rates

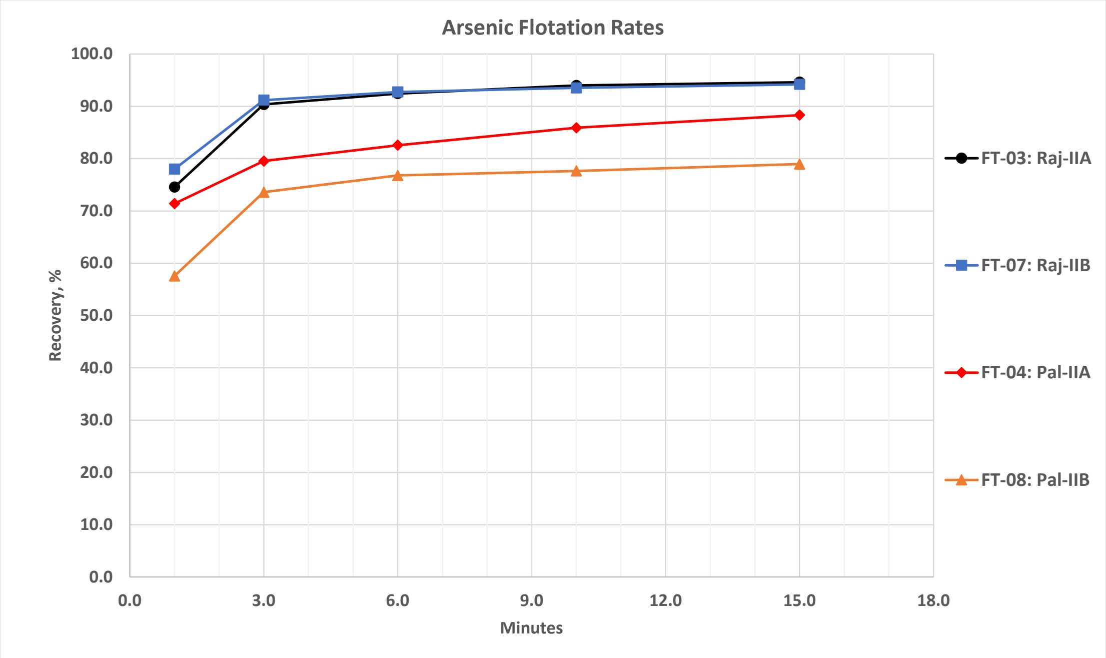


Figure 13-14: Kinetic test results for arsenic flotation rates

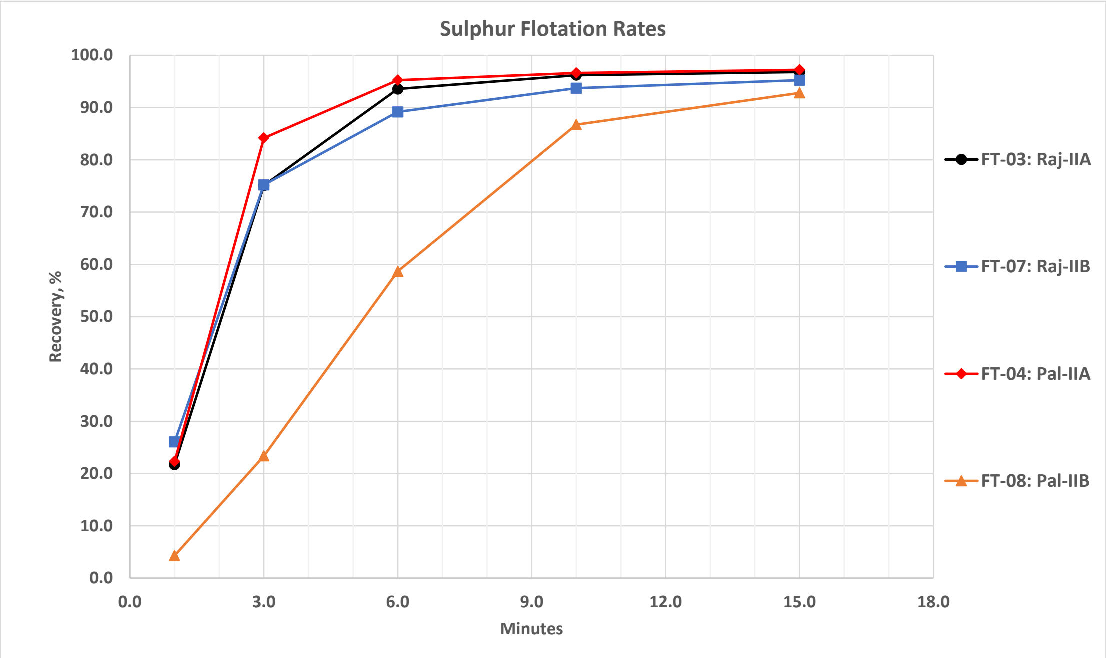


Figure 13-15: Kinetic test results for sulphur flotation rates

Discussion

Testing of both mineral types demonstrated that high recoveries of all components can be achieved with appropriate pulp chemistry and collector additions. Gold recovery kinetics appeared to be affected by the competing Fe-S species, requiring longer flotation time to achieve target recoveries.

The highest final rougher gold recoveries were approximately 88% for both sample location types. The best cobalt recovery for the Raja-South Palokas sample was 94% and 86% for Palokas. This reflects the different mineralogy of the two samples, and the difference in observed grain size of cobaltite. Arsenic recoveries for Palokas were also lower, though sulphur recoveries were high for both types.

The Palokas sample appeared to be more sensitive to flotation conditions, with performance deteriorating at reduced collector doses. Also, the Palokas sample exhibited higher rougher recovery of silicate species.

Cleaner Flotation

A simple one-stage cleaner test was conducted on each sample type. Rougher stage flotation conditions were chosen to maximise sulphur recovery. Videos of the cleaner stages showed that reagent dosing for the cleaner stage was probably higher than optimum. Froth was very sticky with little signs of drainage necessary for removal of entrained silicates.

Results are provided for Raja-South Palokas in Table 13-17 and Figure 13-16 and Palokas in Table 13-18 and Figure 13-17.

Table 13-17: Cleaner Float test results (FT-R-C1: Raj) for Raja-South Palokas

FT-R-Cl-1: Raj **Raja - South Palokas**

P80 = 75 µm

	Reagents (g/t)				
	H2SO4	CuSO4	SEX	nafloat 2	MIBC
Ro	1000	350	140	65	60
Cl	197	150	40	20	15
Tot.	1197	500	180	85	75

	Flot min
RC1-3	12
CC1-3	8

Comments

Bulk float - Higher reagents

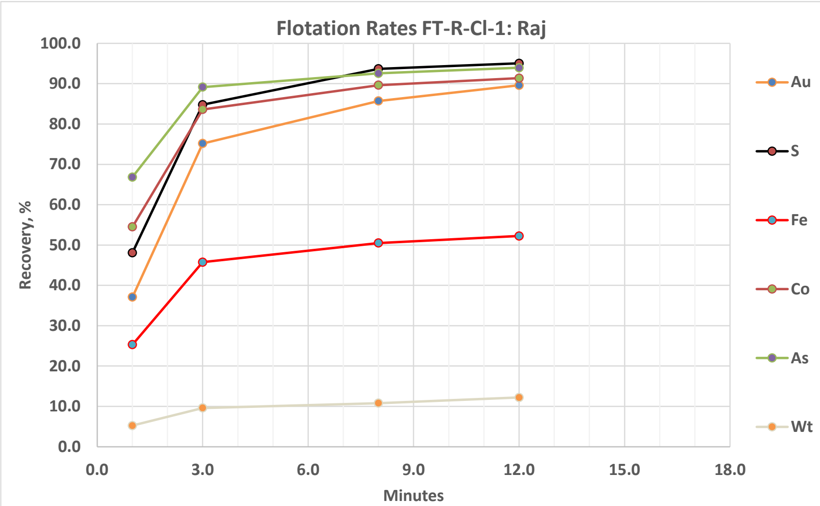
High recovery of As, Co, S & Au.

Cleaner - Hi reagents

Negligible cobaltite selectivity

Higher loss of gold

FT-R-Cl-1: Raj	Wt	ASSAYS (%)									DISTRIBUTION (%)								
	%	Au (g/t)	Co	As	Cu	Fe	S	U	MgO	SiO2	Au	Co	As	Cu	Fe	S	U	MgO	SiO2
Rougher Con	12.2	31.6	0.533	0.335	0.297	37.1	25.5	0.005	2.93	11.9	89.6	91.4	93.9	85.5	52.2	95.0	10.9	6.4	2.7
Cl Con 1 - 3	**10.8**	**34.2**	**0.591**	**0.373**	**0.321**	**40.5**	**28.4**	**0.003**	**2.33**	**6.8**	**85.7**	**89.6**	**92.6**	**81.9**	**50.5**	**93.7**	**5.1**	**4.5**	**1.4**
Calc Tails	***89.2***	*0.69*	*0.008*	*0.004*	*0.009*	*4.80*	*0.23*	*0.006*	*5.99*	*59.1*	*14.3*	*10.4*	*7.4*	*18.1*	*49.5*	*6.3*	*94.9*	*95.5*	*98.6*
Calc Feed	**100.0**	4.31	0.071	0.043	0.042	8.66	3.27	0.006	5.59	53.4	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0	100.0
Head Assay		3.81	0.075	0.060	0.038	8.74	3.81	0.006	4.49	55.1									



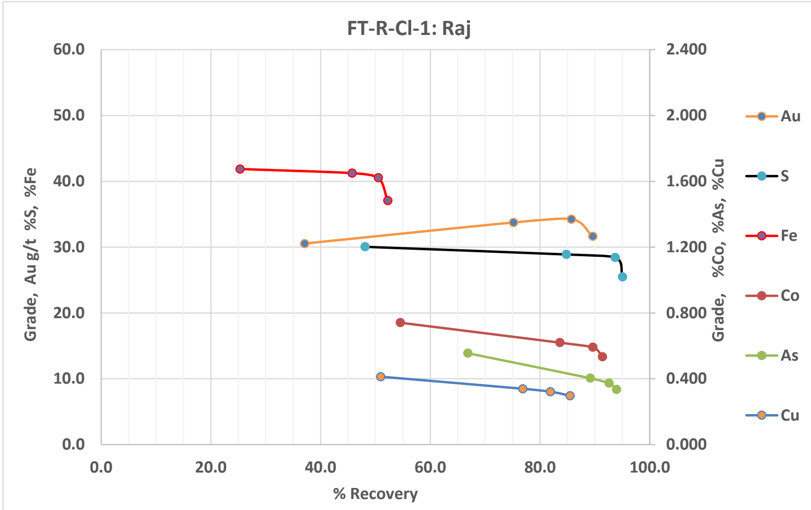


Figure 13-16: Cleaner Float test results (FT-R-C1: Raj) for Raja-South Palokas

Table 13-18: Cleaner Float test results (FT-R-C2: Pal) for Palokas

FT-R-Cl-2: Pal **Palokas**

P80 = 75 µm

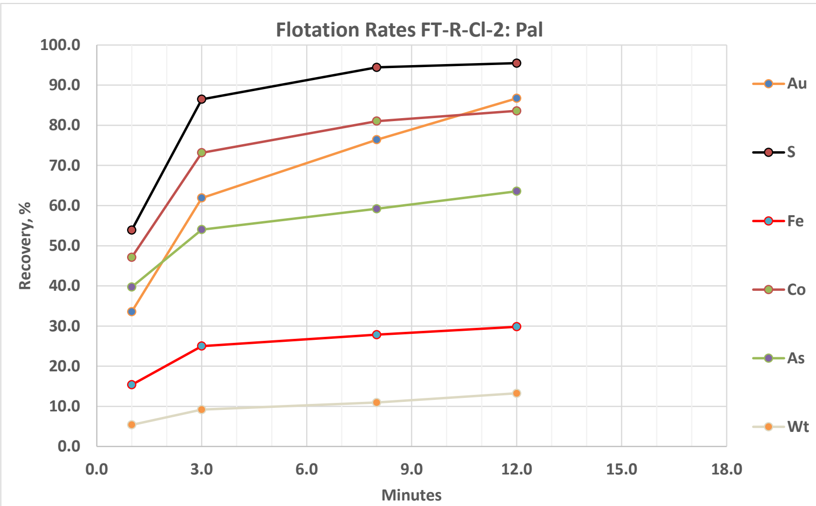
	Reagents (g/t)				
	H2SO4	CuSO4	SEX	nafloat 2	MIBC
Ro	1904	350	140	65	60
Cl	272	150	40	20	15
Tot.	2176	500	180	85	75

	Flot min
RC1-3	12
CC1-3	8

Comments
Bulk float - Higher reagents
- High recovery of S.
- Lower recovery of Co.
- High recovery of Si & Mg
- Low S and Co con grade

Cleaner - Hi reagents
- Negligible cobaltite selectivity
- Higher loss of gold

FT-R-Cl-2: Pal	Wt	ASSAYS (%)									DISTRIBUTION (%)								
	%	Au (g/t)	Co	As	Cu	Fe	S	U	MgO	SiO2	Au	Co	As	Cu	Fe	S	U	MgO	SiO2
Rougher Con	13.3	15.4	0.267	0.023	0.250	31.9	19.7		8.41	20.5	86.7	83.6	63.6	80.9	29.8	95.5		10.2	5.7
Cl Con 1 - 3	**10.9**	**16.4**	**0.313**	**0.026**	**0.273**	**36.1**	**23.6**		**6.82**	**14.2**	**76.4**	**81.0**	**59.2**	**73.1**	**27.9**	**94.4**		**6.8**	**3.3**
Calc Tails	***89.1***	*0.62*	*0.009*	*0.002*	*0.012*	*11.5*	*0.17*		*11.5*	*51.4*	*23.6*	*19.0*	*40.8*	*26.9*	*72.1*	*5.6*		*93.2*	*96.7*
Calc Feed	**100.0**	2.35	0.042	0.005	0.041	14.2	2.73		10.9	47.4	100.0	100.0	100.0	100.0	100.0	100.0		100.0	100.0
Head Assay		2.60	0.049	0.005	0.038	14.4	3.67		10.1	47.3									



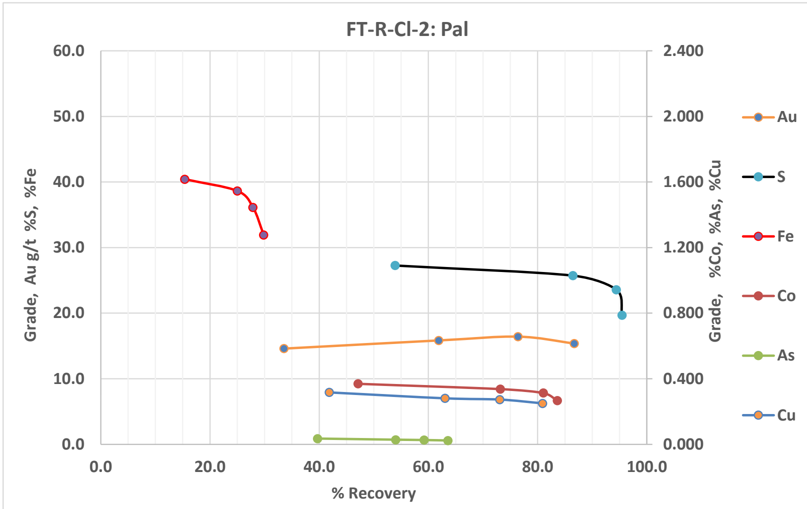


Figure 13-17: Cleaner Float test results (FT-R-C2: Pal) for Palokas

Cleaner ***Flotation Discussion***

The cleaner flotation achieved high stage recoveries for all components, apart from gold for the Palokas sample. Further upgrading of sulphides is expected by controlling chemical additions and additional cleaning stages.

For the Raja-South Palokas sample, 10% total mass recovery to cleaner concentrate produced 90% S recovery, 90% As, 88% Co and 82% Au, to a concentrate of approximately 29% S grade and approximately 0.6% Co grade.

For the Pal sample, 10% total mass recovery to cleaner concentrate produced 91% S recovery, 58% As, 78% Co, 70% Au, to a concentrate of approximately 25% S grade and approximately 0.3% Co grade.

Results are consistent with the mineralogy of the samples – specifically the cobaltite content and grain size as determined in the Petrolab investigation. The Pal material appears only capable of producing a low-Co grade Fe-S concentrate. The Raja-South Palokas material has scope to produce an upgraded Co concentrate by a degree of separation of the cobaltite from the low-Co Fe-S mineralization by use of mixture of selective flotation and/or magnetic separation, though at significantly reduced overall cobalt recovery.

13.10 Processing Discussion

The testwork conducted relevant to the current study has demonstrated the effectiveness of gravity recovery for coarse gold plus cyanide leaching to produce high gold recoveries for both feed type samples tested.

Standard flotation at typical conditions to provide high recovery of Fe-S constituents was effective for both feed types. Associated cobalt recovery from the Palokas sample was less effective than for the Raja-South Palokas sample, which reflected the different mineralogy for the two types related to cobaltite presence in Raja-South Palokas being significantly higher than in Pal.

The Co grade of concentrate achieved from processing the Palokas sample was less than 0.4% Co. This was in-line with expectations, as it is controlled by the inherent concentration of cobalt in pyrrhotite.

Higher Co grade flotation concentrate was achieved for the Raja-South Palokas sample, reflecting the presence of liberated cobaltite in the feed.

Based on this series of testwork, a simpler process flowsheet was developed to support the current PEA; see Figure 13-18.

Work is ongoing to investigate possible enhancements such as:

- Upgrading cobalt concentrate by focussing on cobaltite recovery.
- Reducing capital and operating costs through a flotation first process followed by gold leaching from concentrate.

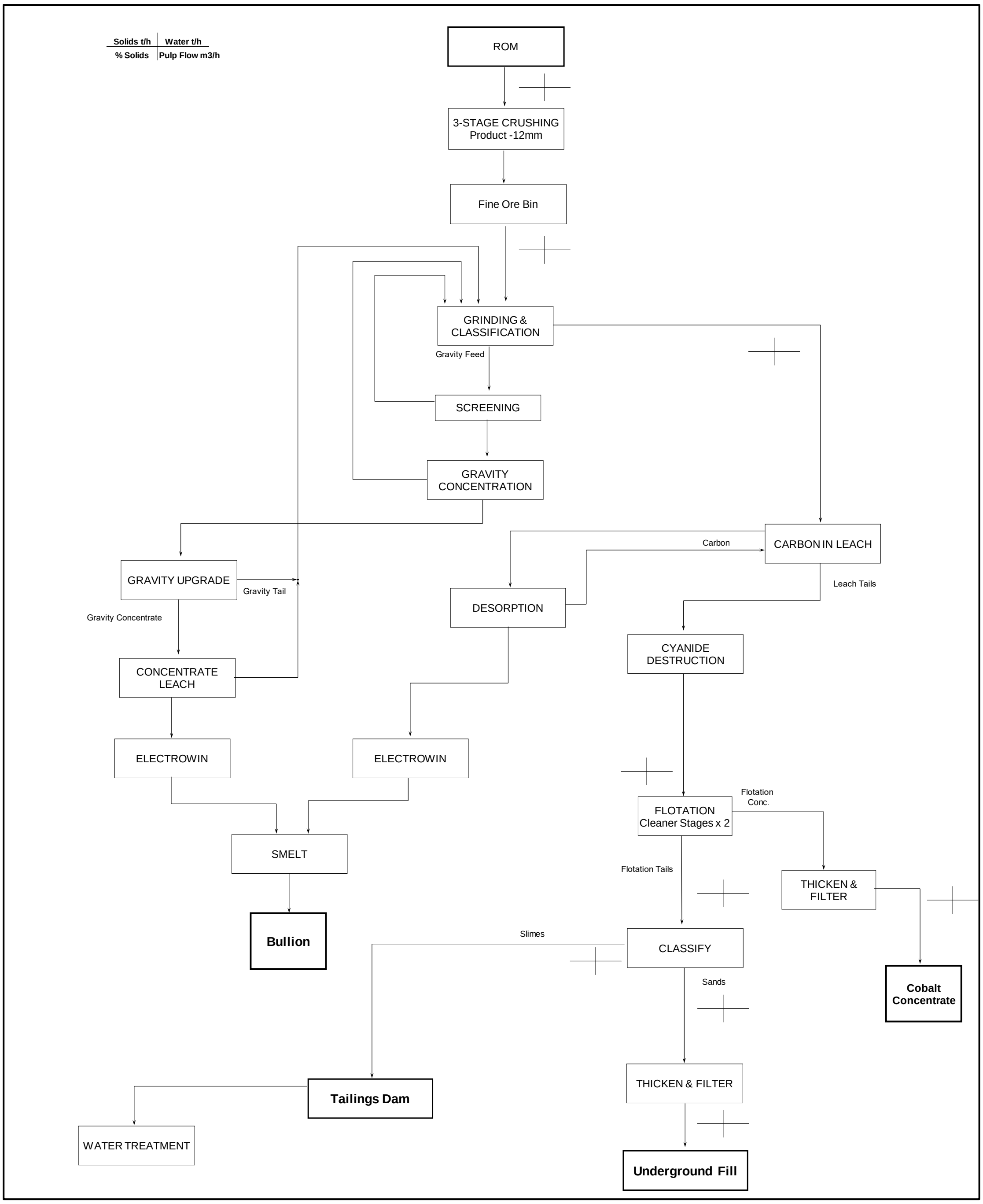


Figure 13-18: Simplified Process Flowsheet for Rajapalot PEA

As the gold recovery and cobalt recovery sections of the process have been investigated separately, there is a reliance in the process that the Cyanide Destruction stage will not adversely affect the cobalt flotation stage. Although it is yet to be proven for the Rajapalot feed types, this has been successfully demonstrated in other projects, both at laboratory and production scales, for a range of sulphide minerals.

13.11 Concentrate Dewatering

No assessment of concentrate dewatering was undertaken as part of this study.

13.12 Environmental Characterisation

No assessment of tailings characterisation was undertaken as part of this study.

13.13 Recovery Assumptions

For the purpose of the current PEA, the process design is based on maximizing the possible recovery of gold and cobalt by gravity recoverable gold recovery and WoO leach for gold recovery as bullion, followed by bulk sulphide flotation from the leach tailings to produce a cobalt concentrate.

Based on the results from the testing, gold recovery of 95% is anticipated from the combination of gravity recovery for coarse gold followed by gold leaching at p80 of 75 µm and leach residence time of 30 hours.

Cobalt recovery and product grade are dependent on feed source and target concentrate product. Flotation testing of both mineral types demonstrated that high recoveries of cobalt can be achieved with appropriate pulp chemistry and collector additions.

As demonstrated by the test results, however, the mineralogical type represented by the Pal sample, with low cobalt head grade and only minor cobaltite content, will only produce a low Co grade concentrate, such that flotation processing after gold recovery may not always meet current economic criteria.

For production forecasting, a simplified model for flotation response was developed from interpretation of the test results for the higher Co feed grade (Raja-South Palokas sample), to apply to the proportion of feed material meeting the criteria for flotation processing for cobalt recovery. It is anticipated that optimised conditions and plant design will enable production of a final flotation concentrate with a grade of 35% sulphur, effectively 90% sulphide minerals, predominantly pyrrhotite. Related anticipated element recoveries are 88% for sulphur and 87.6% for cobalt. Therefore, for the average feed grade (of the high cobalt grade proportion of the mining inventory) of 2.07% S and 0.053% Co, the calculated mass recovery to flotation concentrate is 5.19% to a grade of 0.89% Co.

14 MINERAL RESOURCES

14.1 Introduction

The PEA is based on the Inferred Mineral Resource estimate (MRE) outlined in the technical report titled "Mineral Resource Estimate NI 43-101 Technical Report – Rajapalot Property" dated 26 August 2021 ("Previous MRE"), available on SEDAR. Owing to the underground only mining scenario selected in the PEA, the Mineral Resource estimate utilizes the "All underground Model" as the base case, rather than the "Open Pit-Underground Model" selected previously. All other resource estimation methodologies remain the same. Open pit potential remains however, the selected case change reflects a more 'reasonable prospect of eventual economic extraction' determination that matches the potential development case outlined in this report.

The original gold equivalent ($AuEq^1$) stated in Previous MRE was calculated on each block using long term projected prices of USD1,590 per troy ounce and USD23.07 per pound for gold and cobalt respectively. This results in AuEq = Au (g/t) + Co/1,005 (ppm).

The updated gold equivalent ($AuEq^2$) and ounces stated here are based on the updated PEA metal prices of USD1,700/oz Au and USD60,000/t Co and recovery assumptions of 95% Au and 87.6% Co. ($AuEq^2$ = Au (g/t) x 95% + Co (ppm) x 87.6% / 911).

Optimization of the resource was undertaken on the Previous MRE and conducted using Whittle software based on the criteria for pit optimization on Palokas, South Palokas, Raja, The Hut, Rumajärvi, Uusisaari, Terry's Hammer and Joki East prospect wireframes to define the mineralization falling within the confines of an open pit (demonstrating reasonable prospects for eventual economic extraction, RPEEE) and resources below the pit optimization shells were considered for RPEEE for future underground mining potential. Five block models were created covering the 8 prospects.

14.2 Data Used in the Formulation of Mineral Resources, Wireframing Methods and Domains

The QP was provided with drillhole data for Rajapalot prospects by Mawson through shared access to a joint Dropbox folder or sharing of Leapfrog Geo and Edge models via Seequent's "Central" server.

A full database download included all parameters measured on drill core and assay results as outlined in Section 10. A total of 48 separate wireframes were created by Mawson (Table 14-1), using gold and cobalt assay intersections, with shapes of wireframes modified by geologic factors including lithology, major and trace element geochemistry and structural measurements. Internal waste was created in wireframes where more than 2 m of <0.3 g/t Au or cobalt <300 ppm was included in the drill intersections.

A topographic wireframe derived from Lidar data and a base of till wireframe was also utilized in the resource estimation.

Table 14-1: Details of the wireframes created by Mawson and used as domains in the estimation of the Inferred Mineral Resource

Note: The details of the polylines created to make the orientation meshes are presented in columns 3 & 4. The estimation of either Au or Co is displayed in column 5. Column 6 covers the estimation of additional elements that are related to the individual gold and cobalt wireframes built for each prospect.

Mesh_name	Volume (m^3)	Polyline	Orientation mesh	Au and/or Co	Other elements
Raja_gold_main: Raja-gold-main	938,660	Raja-main_Au	Raja-main_Au	Au	As, W, FeO
Raja_gold_main: Raja-minor_lower	19,047	Raja_lower_Au	Raja_lower_Au	Au	As, W, FeO (merged)
Raja_V2_upper_eastern	68,294	Raja_Au_upper_eastern	Raja_Au_upper_eastern	Au	As, W, FeO
Raja_V2_upper_central	39,727	Raja_Au_upper_central	Raja_Au_upper_central	Au	As, W, FeO
Raja_cobalt_main: Raja_cobalt_main	779,710	Raja_Co_main	Raja_Co_main	Co	Cu, Ni, FeO, S, U
Raja_Co_upper: Co_upper-2	57,217	Raja_Co_upper_2	Raja_Co_upper_2	Co	Cu, Ni, FeO, S, U
Raja_Co_upper: Raja_Co_upper	146,010	Raja_Co_upper	Raja_Co_upper	Co	Cu, Ni, FeO, S, U
Palokas_gold_main: Palokas_main	1,476,200	Palokas_Au_main_trend	Palokas_Au_main_trend	Au	As, W, FeO
Cobalt_Palokas: Pal_Cobalt_wireframe	875,930	Palokas_Co_main_trend	Palokas_Co_main_trend	Co	Cu, Ni, FeO, S, U
Gold_Sth_Pal: Sth_Pal_Au_Main	594,530	South Palokas Main trend	South Palokas Main trend	Au	As, W, FeO
Gold_Sth_Pal: Sth_Pal_lower	145,340	Sth_Pal_Au_lower	Sth_Pal_Au_lower	Au	As, W, FeO
Gold_Sth_Pal: Sth_Pal_Au_upper	67,856	Sth-Pal_upper_trend	Sth-Pal_upper	Au	As, Cu, Ni, FeO, S, U, W
Cobalt_wireframing: Sth_Palokas_Main	839,480	South Palokas Main trend	South Palokas Main trend	Co	As, Cu, Ni, FeO, S, U, W
Cobalt_wireframing: Sth_Pal_upper_Co	26,229	Sth-Pal_upper_trend	Sth-Pal_upper	Co	As, Cu, Ni, FeO, S, U, W
Cobalt_wireframing: Sth_Pal_lower_Co	80,467	Sth_Pal_Co_lower_trend	Sth_Pal_Co_lower_trend	Co	Cu, Ni, FeO, S, U
Hut_Au_lenses: Hut_Au_1_lens	169,880	Hut_Au_1_lens	Hut_Au_1_lens	Au	W, FeO
Hut_Au_lenses: Hut_Au_1_upper_1	1,744	Hut_Au_upper_1	Hut_Au_upper_1	Au	W, FeO
Hut_Au_lenses: Hut_Au_1_upper_2	11,256	Hut_Au_upper_2	Hut_Au_upper_2	Au	W, FeO
Hut_Au_lenses: Hut_Au_1a_lens	18,800	Hut_Au_1a_lens	Hut_Au_1a_lens	Au	W, FeO
Hut_Au_lenses: Hut_Au_2a_lens	187,560	Hut_Au_2a_lens	Hut_Au_2a_lens	Au	W, FeO
Hut_Co_lenses: Hut_Lens_1	71,319	Hut_Co_lens_1	Hut_Co_lens_1	Co	As, Cu, Ni, FeO, S, U, W
Hut_Co_lenses: Hut_Lens_2	101,270	Hut_Co_lens_2	Hut_Co_lens_2	Co	As, Cu, Ni, FeO, S, U, W
Hut_Co_lenses: Hut_Lens_2a	8,527	Hut_Co_lens_2a	Hut_Co_lens_2a	Co	As, Cu, Ni, FeO, S, U, W
Hut_Co_lenses: Hut_lens_3a	5,187	Hut_Co_lens_3a	Hut_Co_lens_3a	Co	As, Cu, Ni, FeO, S, U, W
Hut_Co_lenses: Hut_Lens_4a	21,687	Hut_Co_lens_4a	Hut_Co_lens_4a	Co	As, Cu, Ni, FeO, S, U, W
Hut_Co_lenses: Hut_lens_4	26,296	Hut_Co_lens_4	Hut_Co_lens_4	Co	As, Cu, Ni, FeO, S, U, W

Mesh_name	Volume (m^3)	Polyline	Orientation mesh	Au and/or Co	Other elements
Hut_Co_lenses: Hut_top_lens	97,601	Hut_Co_top_lens	Hut_Co_top_lens	Co	As, Cu, Ni, FeO, S, U, W
Joki_gold: Joki_1_lens	3,859	Joki_1_lens	Joki_1_lens	Au	As, W, FeO
Joki_gold: Joki_2_lens	13,635	Joki_2_lens	Joki_2_lens	Au	As, W, FeO
Joki_gold: Joki_main	132,710	Joki_main_trend	Joki_main_trend	Au	As, W, FeO
Joki_cobalt: Joki_Main_Co	56,558	Joki_main_Co_trend	Joki_main_Co_trend	Co	Cu, Ni, FeO, S, U, W
Rumaj_Au: Rumaj_Au_1	75,529	Rumaj_1_lens	Rumaj_1_lens	Au	Combined mesh all others
Rumaj_Au_2	10,993	Rumaj_Au_2	Rumaj_Au_2	Au	Combined mesh all others
Rumaj_Au_3	20,804	Rumaj_Au_3	Rumaj_Au_3	Au	Combined mesh all others
Rumaj_Au: Rumaj_Au_4	18,074	Rumaj_Au_4	Rumaj_Au_4	Au	Combined mesh all others
Rumaj_Au_6	34,831	Rumaj_Au_6	Rumaj_Au_6	Au	Combined mesh all others
Rumaj_Au_7	11,667	Rumaj_Au_7	Rumaj_Au_7	Au	Combined mesh all others
Rumaj_Au_8	3,234	Rumaj_Au_8	Rumaj_Au_8	Au	Combined mesh all others
Rumaj_Co: Rumaj_Co_1	123,950	Rumaj_Co_1	Rumaj_Co_1	Co	Combined mesh all others
Rumaj_Co_2	30,359	Rumaj_Co_2	Rumaj_Co_2	Co	Combined mesh all others
Rumaj_Co_3	53,433	Rumaj_Co_3	Rumaj_Co_3	Co	Combined mesh all others
Rumaj_Co_4	9,906	Rumaj_Co_4	Rumaj_Co_4	Co	Combined mesh all others
Rumaj_Co_upper	3,980	Rumaj_Co_upper	Rumaj_Co_upper	Co	Combined mesh all others
Rumaj_Co_6	45,853	Rumaj_Co_6	Rumaj_Co_6	Co	Combined mesh all others
Rumaj_Co: Rumaj_Co_5	20,439	Rumaj_Co_5	Rumaj_Co_5	Co	Combined mesh all others
Rumaj_Co_8	3,744	Rumaj_Co_8	Rumaj_Co_8	Co	Combined mesh all others
Rumaj_Co_lower_single	6,128	Rumaj_Co_lower_single	Rumaj_Co_lower_single	No estimation	No estimation
T_Hammer_Au	8,993	T_Hammer_Au_Co	T_Hammer_Au_Co	Au, Co	As, Cu, Ni, FeO, S, U, W

An individual orientation mesh was created for each wireframe by first building a set of polylines through sectional cuts and then forming the orientation mesh from the groups of polylines built for that wireframe. Thus, during the estimation process, the orientation of the variogram could be varied parallel to the central plane of each wireframe. Separate estimations were made for Au and Co within each wireframe (with the exception of Terry's Hammer).

FeO was determined for each wireframe by combining the Au and Co wireframes so that all domains for estimation were filled to create a satisfactory density model (see Section 14.4).

Estimations of other elements (As, W, Cu, Ni, S, and U) were made into wireframes determined through the understanding of the whole-rock geochemistry and their inter-relationships of gold and cobalt. Thus, As and W are generally spatially and statistically related to gold, and Cu, Ni, S and U are more spatially and statistically associated with cobalt.

An overall view of the Rajapalot wireframes (all one colour) and wireframes for each prospect are shown in oblique views in Figure 14-1 to Figure 14-7. Note that gold wireframes are "gold" and cobalt wireframes are in blue. Red zones on drill traces correspond to mineralized intervals in drill holes included in wireframes

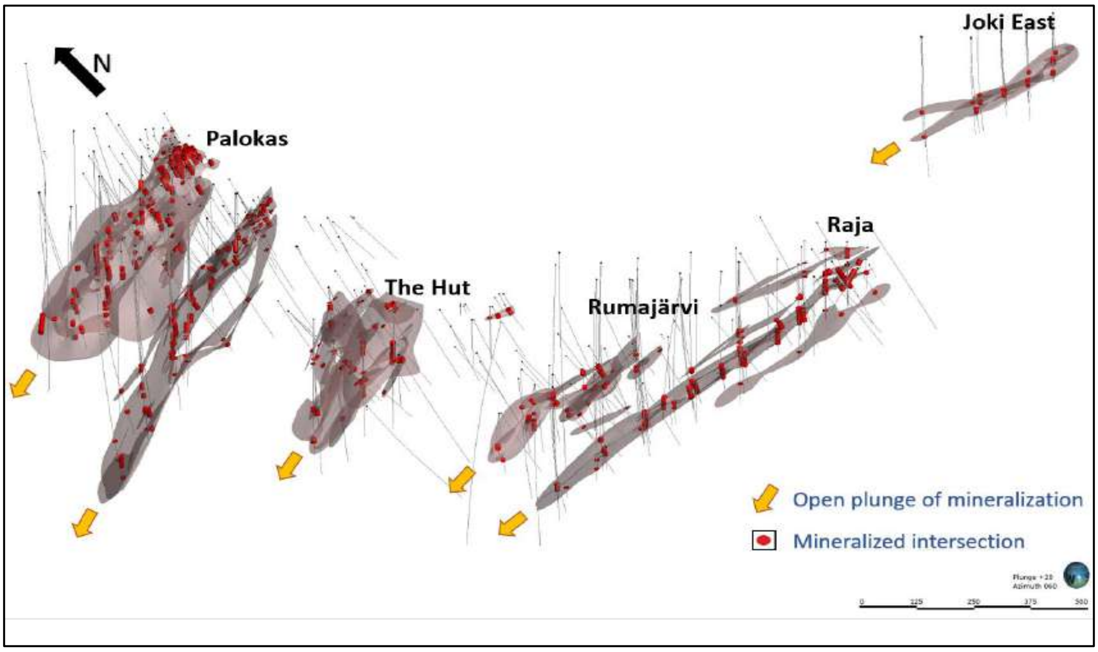


Figure 14-1: Mineralized wireframes at Rajapalot

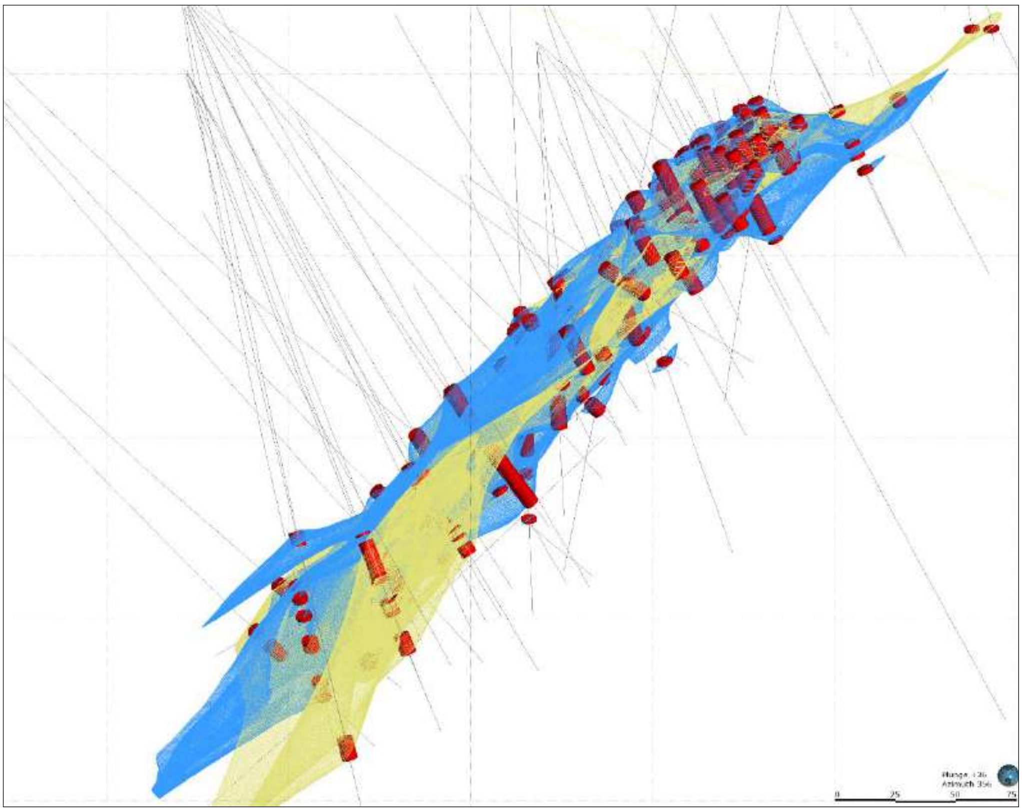


Figure 14-2: Gold and cobalt wireframes at Palokas prospect; view looking 36° towards 356 (North)

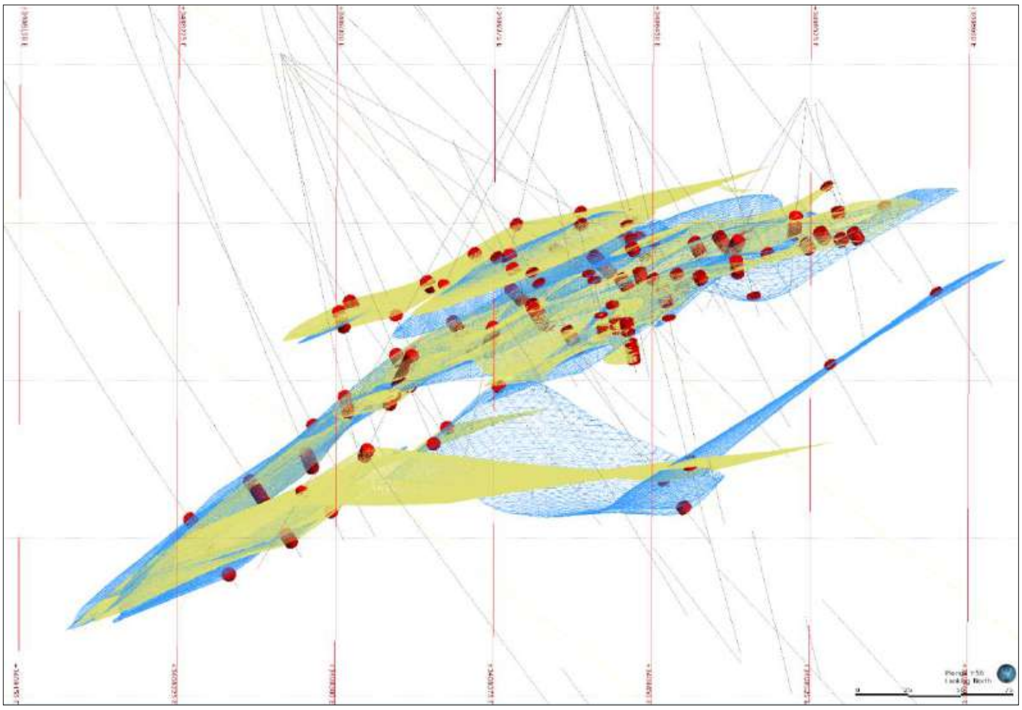


Figure 14-3: Gold and cobalt wireframes at South Palokas prospect; view looking 56° towards 000 (North)

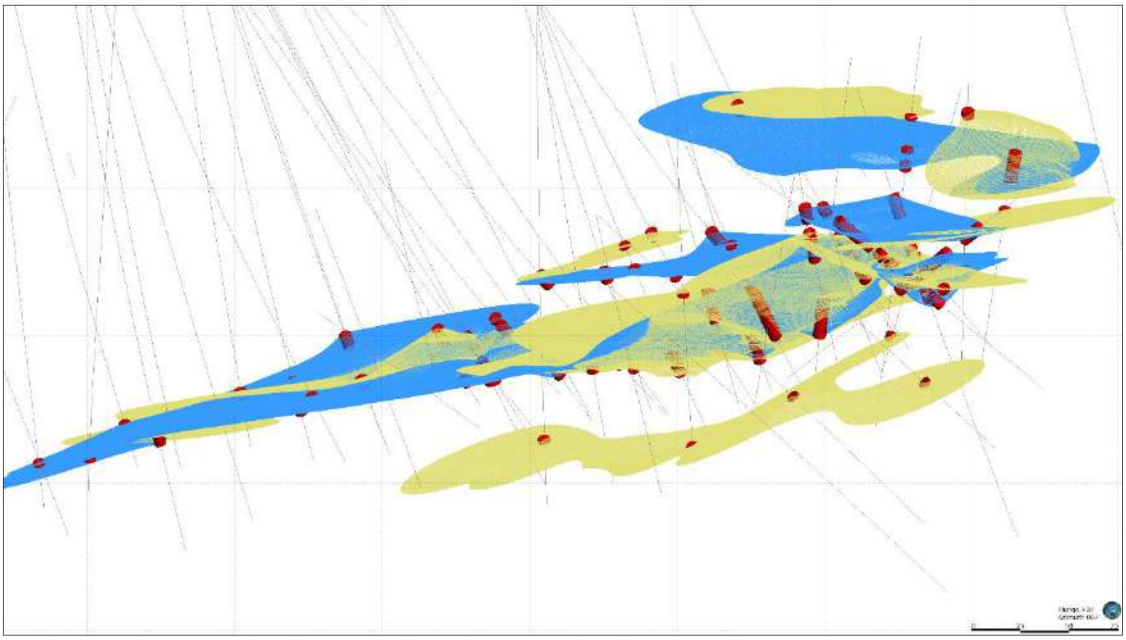


Figure 14-4: Gold and cobalt wireframes at Raja prospect; view looking 37° towards 007 (North)

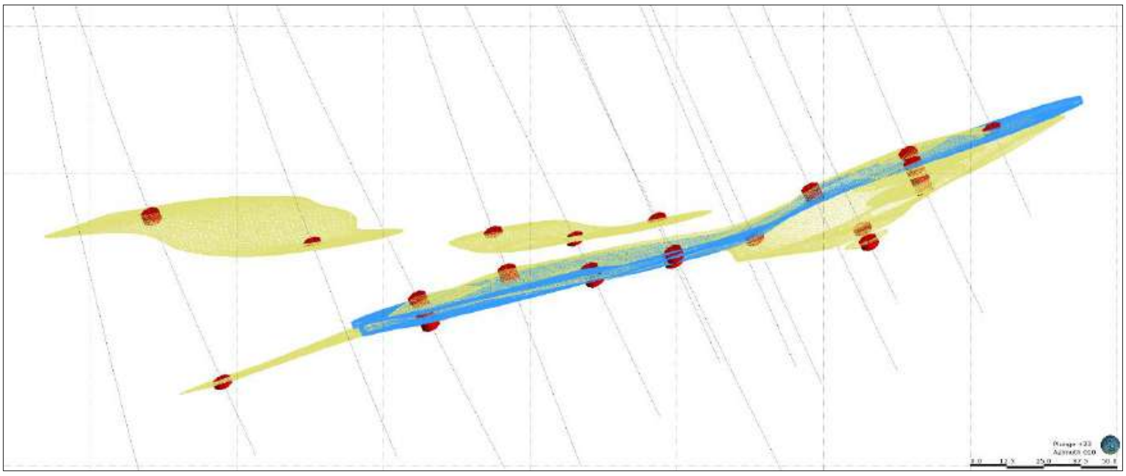


Figure 14-5: Gold and cobalt wireframes at Joki prospect; view looking 23° towards 010 (North)

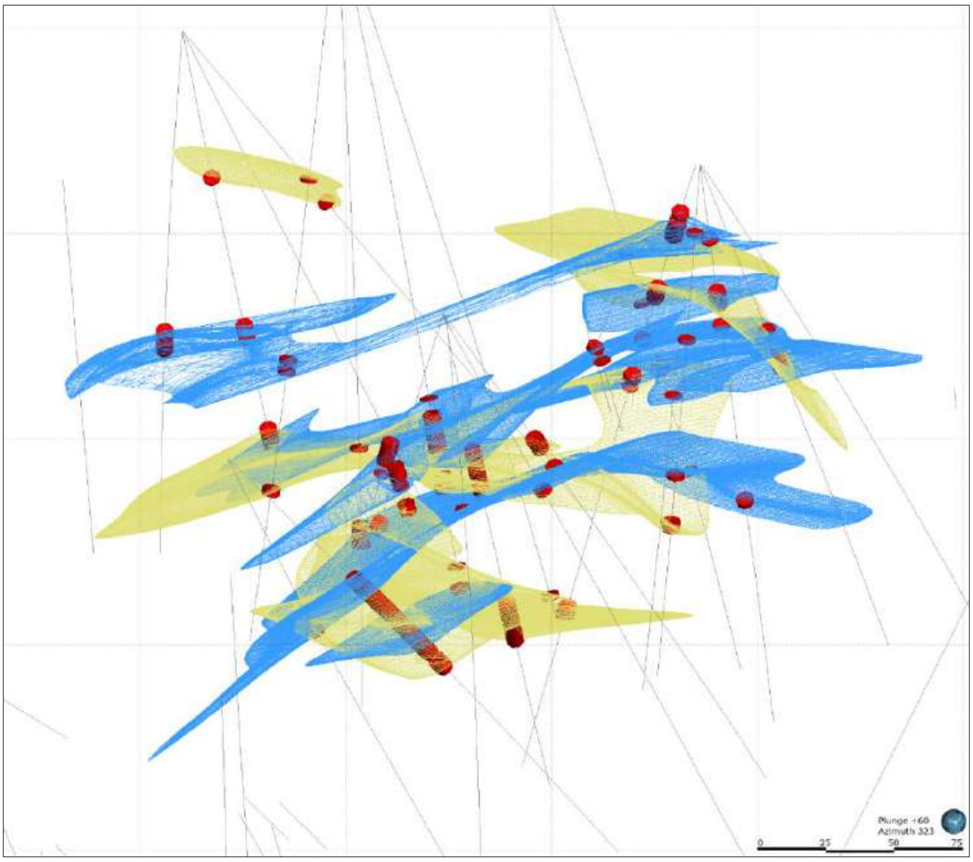


Figure 14-6: Gold and cobalt wireframes at The Hut prospect; view looking 60° towards 323 (NW)

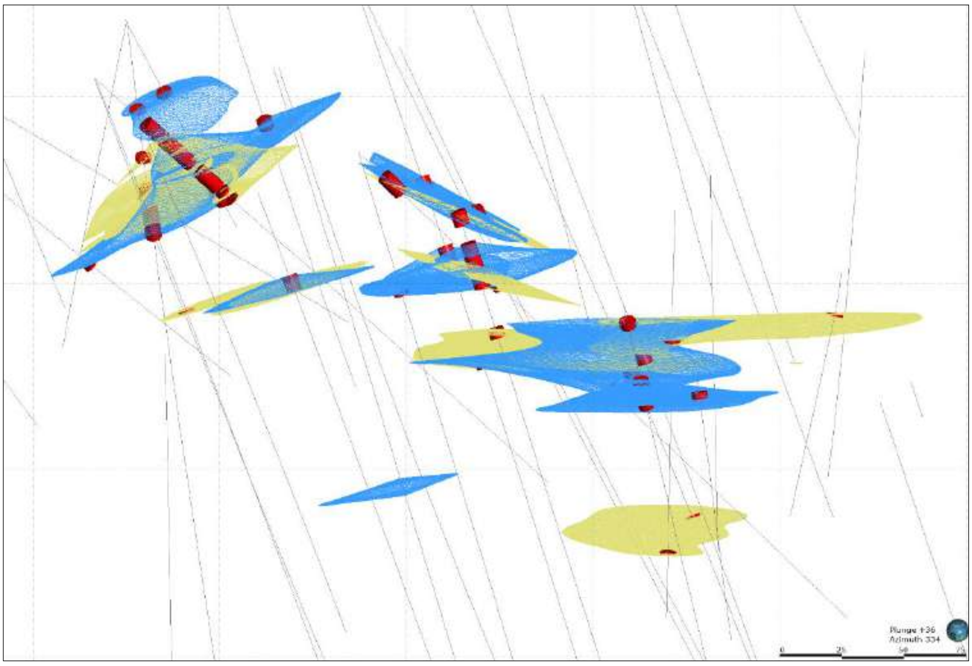


Figure 14-7: Gold and cobalt wireframes at Rumajärvi prospect; view looking 36° towards 334 (NW)

14.3 Sample Statistics

Assay data for 330 drillholes was used in the creation of the gold and cobalt wireframes at Rajapalot. A random subset of these data (n=1083 samples) was selected to test for any systematic variation of grade (AuEq) with depth (Table 14-2). There is no surface enrichment evident with the highest mean value occurring between -400 and -300 m below sea level (mRL). The 220 drillholes which contain mineralization are included within the wireframes (domains). Statistics on each of these domains is presented in Table 14-3 and Table 14-4.

Inspection of continuity of the gold data indicates that a reasonable top-cut is 50 g/t Au. Selected histograms and mean values are plotted in Figure 14-8 to Figure 14-11 to show the strong demarcation of wireframe boundaries (hard) with respect to gold and cobalt grades.

Table 14-2: Randomly sub-sampled set of AuEq assay data plotted in depth slices to test for systematic variation of grade with depth

Depth range (mRL)	count	Min	Max	mean	median
below -400	11	0.31	1.68	0.99	1.01
from -400 to -300	66	0.33	31.22	4.53	2.92
from -300 to -200	205	0.30	12.64	1.74	0.84
from -200 to -100	507	0.30	80.54	3.75	1.14
from -100 to 0	785	0.30	116.03	3.38	0.93
from 0 to 100	1,112	0.30	36.02	2.21	1.02
> 100	1,083	0.30	190.15	2.74	1.01

Table 14-3: Gold domain statistics (24 wireframes) top cut to 50 g/t Au

Gold domain	Count	Mean	SD	CV	Variance	Minimum	Maximum	Q1	Q2	Q3
Au_Palokas_main	737	2.8	6	2.1	35.73	0.0005	50	0.24	0.84	2.57
Au_South_Palokas_main	325	2.8	4.9	1.7	24.22	0.0005	42.8	0.44	1.05	3.05
Au_South_Palokas_lower	25	2	3.3	1.7	10.93	0.18	16.1	0.37	0.6	1.54
Au_South_Palokas_upper	52	0.9	1	1.1	0.94	0.0005	3.9	0.32	0.51	1.08
Au_Raja_main	405	3.2	7.3	2.3	53.19	0.0005	50	0.14	0.62	3.01
Au_Raja_minor_lower	11	1.1	0.9	0.8	0.84	0.116	3	0.5	0.57	1.87
Au_Raja_Upper_central	26	1.8	2.3	1.3	5.19	0.0005	10	0.48	0.88	1.76
Au_Raja_upper_eastern	28	1.6	3.3	2.1	10.59	0.0005	16.8	0	0.62	1.31
Au_Joki_main	52	4	8.3	2.1	69.12	0.0005	35.5	0.31	0.64	2.55
Au_Joki_1_lens	4	13.6	18.1	1.3	327.53	0.85	36.6	0.85	10.3	36.55
Au_Joki_2_lens	4	13.6	18.1	1.3	327.53	0.85	36.6	0.85	10.3	36.55
Au_The Hut_lens_1	84	2.1	3.2	1.5	10.55	0.0005	19.5	0.36	1.05	2.16
Au_The Hut_lens_1a	15	0.6	0.4	0.7	0.19	0.06	1.6	0.33	0.47	0.94
Au_The Hut_lens_2a	85	1	1.3	1.3	1.58	0.0005	7.6	0.19	0.61	1.17
Au_The Hut_lens_upper_1	3	3.9	6	1.6	36.16	0.51	11.2	0.51	0.7	11.2
Au_The Hut_lens_upper_2	2	0.3	0	0.1	0	0.28	0.3	0.28	0.28	0.33
Au_Terry’s_Hammer	16	1.1	0.9	0.9	0.88	0.0005	2.6	0.26	0.67	2.02
Au_Rumaj_1	52	1.3	2.9	2.2	8.47	0.0005	19.5	0.19	0.52	1.18
Au_Rumaj_2	6	1.1	0.8	0.7	0.58	0.0005	1.8	0.29	1.61	1.67
Au_Rumaj_3	21	2.3	2.1	0.9	4.43	0.0005	6.4	0.39	2.01	3.91
Au_Rumaj_4	16	1.1	1.3	1.2	1.6	0.015	5	0.47	0.64	1.27
Au_Rumaj_6	11	2	2.4	1.2	5.59	0.09	6.7	0.42	1.06	2.49
Au_Rumaj_7	18	0.9	1.1	1.2	1.25	0.06	3.8	0.12	0.38	1.31
Au_Rumaj_8	2	1.6	0.7	0.5	0.5	1.2	2.2	1.2	1.2	2.2

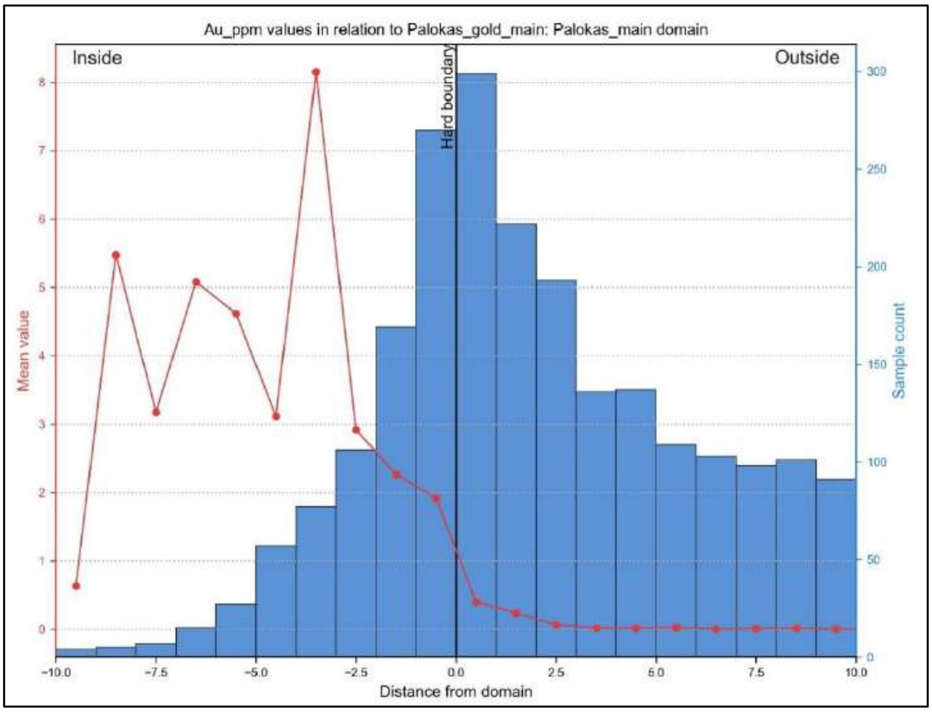


Figure 14-8: Palokas prospect, main gold wireframe (Au_Palokas_main). Histograms (n=737) of gold (g/t) and mean values in 1 m intervals with respect to domain boundaries

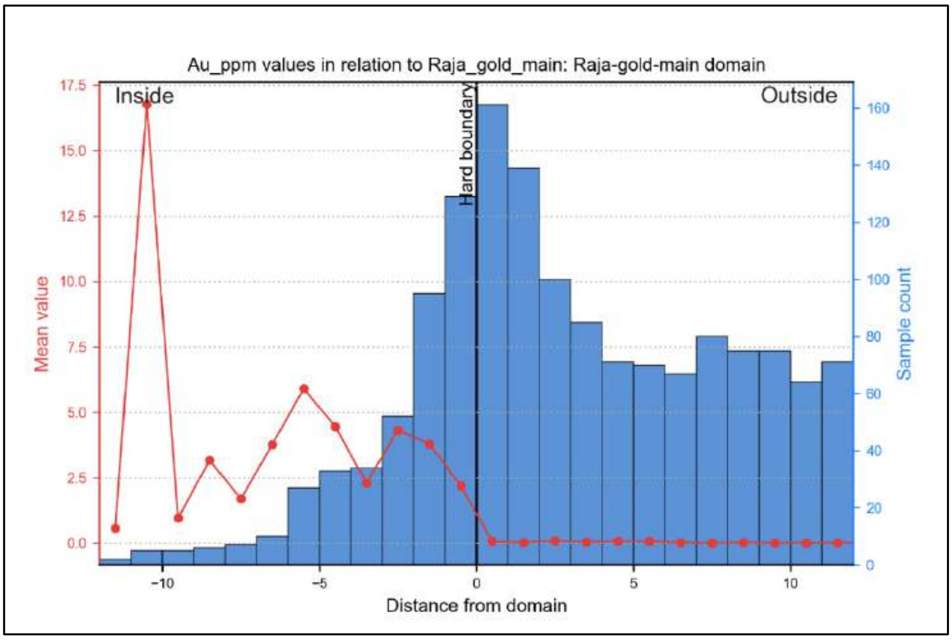


Figure 14-9: Raja prospect, main gold wireframe (Au_raja_main). Histograms (n=405) of gold (g/t) and mean values in 1 m intervals with respect to domain boundaries

Table 14-4: Cobalt domain statistics (24 wireframes)

Domain	Count	Length	Mean	SD	CV	Var.	Min.	Max.	Q1	Q2	Q3
Co_Palokas_main	1,022	1,038.5	603	702	1.16	493,305	1.4	14,620.0	262.0	460.0	755.0
Co_Sth_Palokas_main	406	400.6	839	694	0.83	481,265	9.4	4,723.7	368.3	640.0	1,103.5
Co_Sth_Palokas_lower	24	22.5	760	489	0.64	239,304	44.9	1,903.7	435.8	649.4	996.0
Co_Sth_Palokas_upper	25	24.7	400	159	0.4	25,307	72.5	799.7	315.8	381.0	478.0
Co_Raja_main	415	415.2	755	840	1.11	706,422	3.3	9,492.7	271.0	521.0	965.0
Co_Raja_upper	114	118.8	515	471	0.91	221,717	10.1	1,940.0	119.0	405.0	715.0
Co_Raja_upper_2	23	22.4	333	347	1.04	120,417	3.5	958.0	33.4	152.4	642.6
Co_Joki_main	33	25.9	914	805	0.88	647,505	43.7	3,974.4	216.6	695.3	1,530.3
Co_Hut_lens_1	51	43.8	416	480	1.16	230,812	4.7	2,790.0	90.7	371.0	524.2
Co_Hut_lens_2	40	39.7	940	1,004	1.07	1,008,527	19.1	3,609.4	195.0	525.0	1,644.0
Co_Hut_lens_4	27	27.4	1,048	731	0.7	534,108	9.1	2,293.4	187.9	1,347.2	1,671.0
Co_Hut_lens_2a	10	8.7	310	77	0.25	5,880	163.8	397.6	254.1	316.3	379.8
Co_Hut_lens_3a	6	5.4	363	86	0.24	7,391	311	638.0	313.8	339.0	382.2
Co_Hut_lens_4a	11	10.6	1,144	1,214	1.06	1,473,547	156.4	3,716.3	350.0	470.3	1,291.1
Co_Hut_upper	33	33.0	589	513	0.87	263,195	38.9	2,194.9	146.9	618.0	745.2
Co_Terrys_Hammer	16	16.6	326	236	0.72	55,811	42.7	746.0	119.6	253.5	545.2
Co_Rumaj_1	63	63.5	820	518	0.63	268,049	11.5	1,995.9	419.4	656.5	1,199.7
Co_Rumaj_2	32	29.2	601	269	0.45	72,159	62.3	999.9	408.3	629.9	809.0
Co_Rumaj_3	34	33.0	523	247	0.47	60,780	126.5	1,160.0	373.7	490.9	610.0
Co_Rumaj_4	8	8.0	370	280	0.76	78,147	51.1	822.0	106.0	383.0	487.0
Co_Rumaj_5	10	8.7	309	168	0.55	28,371	52.1	592.3	257.0	311.1	407.0
Co_Rumaj_6	29	27.1	400	333	0.83	111,060	10.7	1,650.0	81.3	313.0	708.6
Co_Rumaj_8	5	5.0	255	75	0.29	5,557	196	382.0	206.0	242.0	248.0
Co_Rumaj_upper	7	7.4	733	594	0.81	352,266	58.8	1,595.2	64.4	675.0	1,489.0

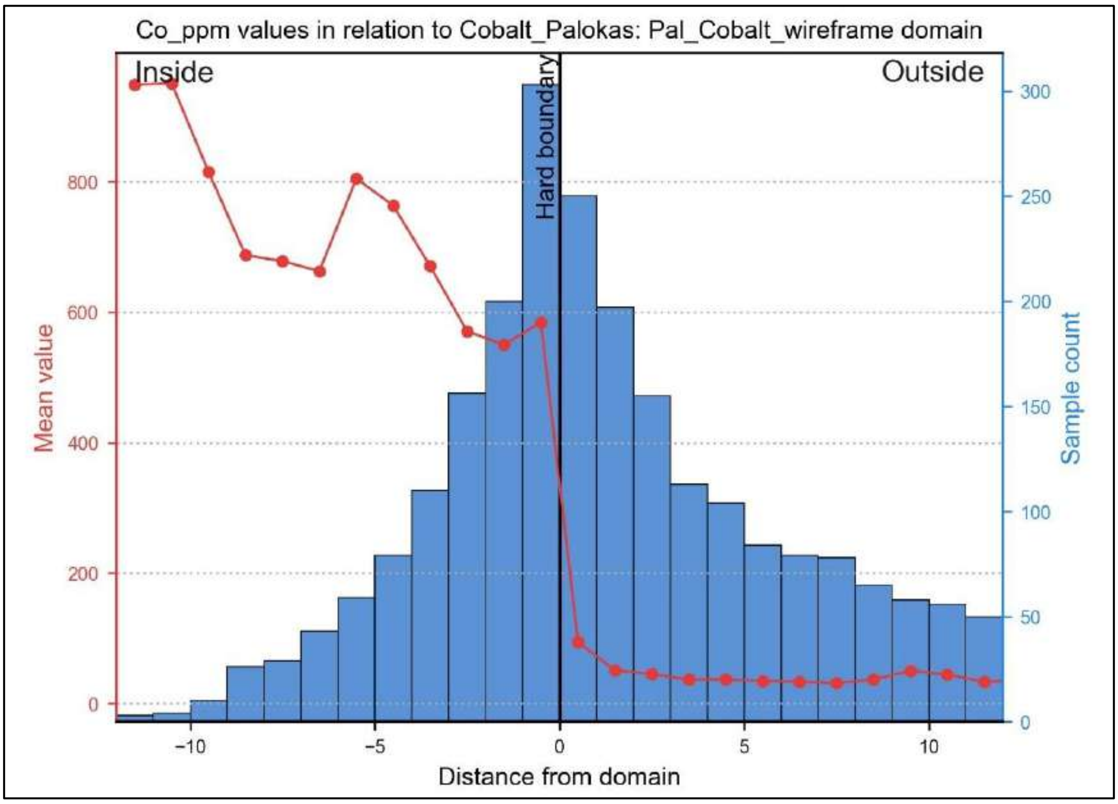


Figure 14-10: Palokas prospect, main cobalt wireframe (Co_Palokas_main). Histograms (n=1022) of cobalt (ppm) and mean values in 1 m intervals with respect to domain boundaries

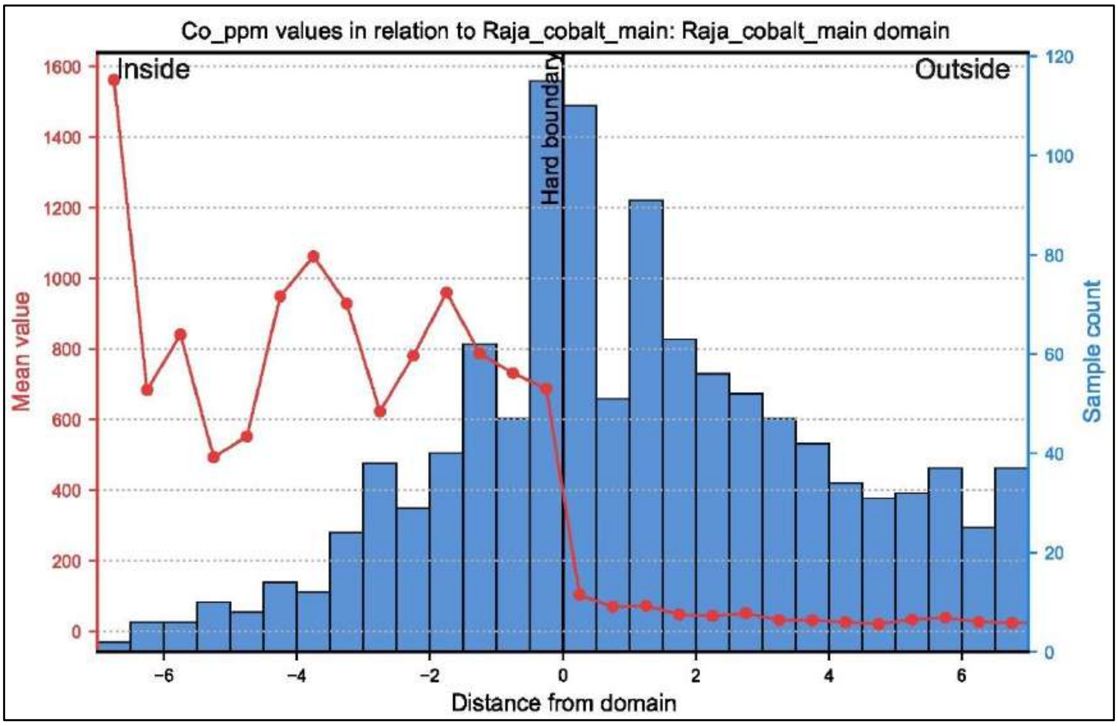


Figure 14-11: Raja prospect, main cobalt wireframe (Co_raja_main). Histograms (n=415) of cobalt (ppm) and mean values in 1 m intervals with respect to domain boundaries

14.4 Density Data – Measurements and Calculation

14.4.1 Introduction

A total of 3,345 density measurements have been calculated for Rajapalot drill core. Of these measurements, 1,103 fall within the category of AuEq >0.3 g/t AuEq. Density is determined by the standard method of measuring the weight of drill core in air and water, then using the following formula:

- Density (bulk) = mass in air / (mass in air – mass in water)

These data were then combined with assay interval data to determine any relationships with bulk rock chemistry. There is a clear relationship between rock bulk density and FeO content, and this relationship improves when using a cut-off of 0.3 g/t AuEq (Figure 14-12 for histogram of the samples and Figure 14-13 for linear relationship of density with FeO content for all samples).

Estimation methodology uses Ordinary Kriging to produce a block estimate for FeO. The best fit linear relationship of density with FeO is then used to provide the most realistic determination of density on a block-by-block basis. This is regarded as more accurate than using a single averaged density across each domain. In each case, the separate gold and cobalt wireframes were combined for estimation purposes. This ensured that FeO estimation was made across all blocks.

Searches for “without value” FeO blocks were made across all prospects to ensure that density estimation was made across the full volume of the combined wireframes.

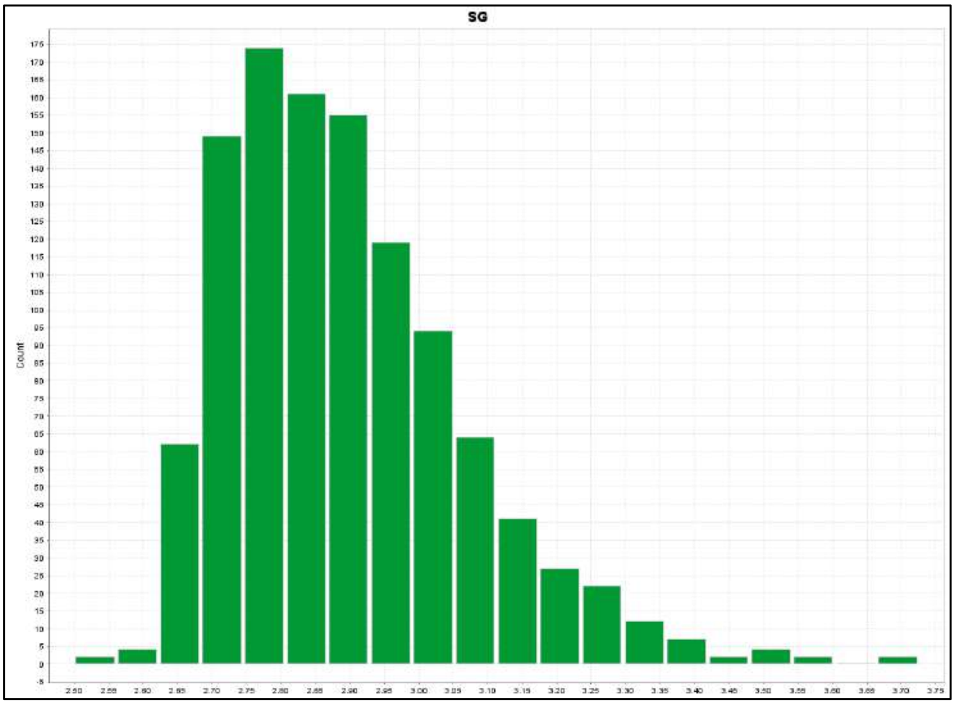


Figure 14-12: Histogram (n=1103) of density measurements for Rajapalot mineralized rocks (>0.3 g/t AuEq)

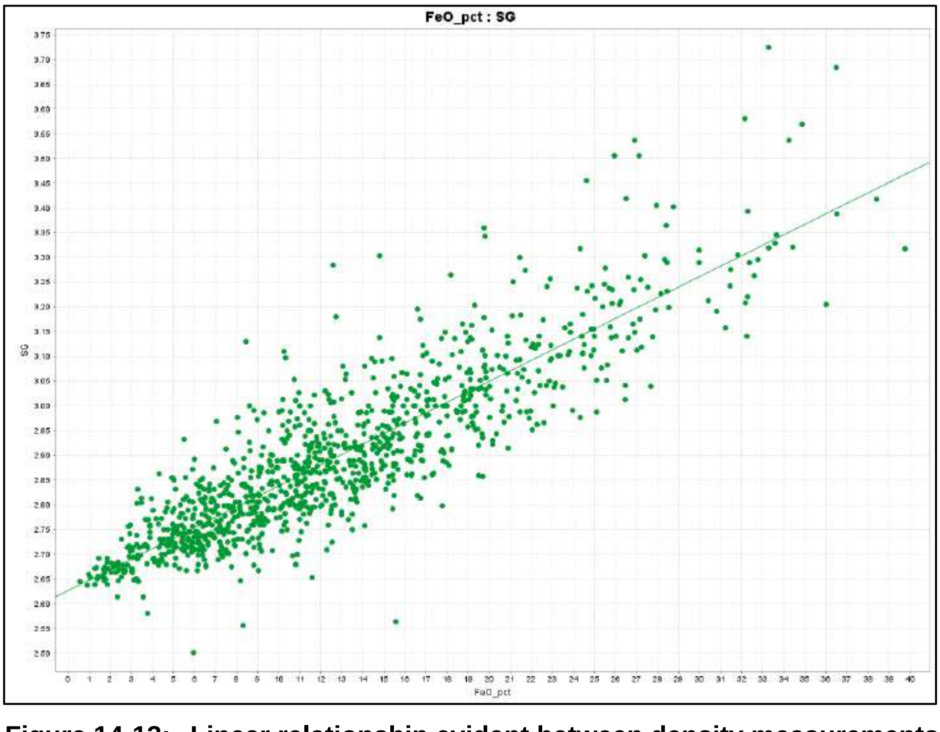


Figure 14-13: Linear relationship evident between density measurements and FeO%

14.4.2 Palokas prospect

Palokas prospect had 918 density determinations, of which 242 occur in mineralized rocks of >0.3 g/t AuEq and is the highest density mineralized mean for any of the prospects. The mean density is 2.98 t/m^3 with a range of 2.5 t/m^3 to 3.57 t/m^3. Figure 14-14 and Figure 14-15 show the distribution of density measurements as a histogram and the relationship between density and FeO.

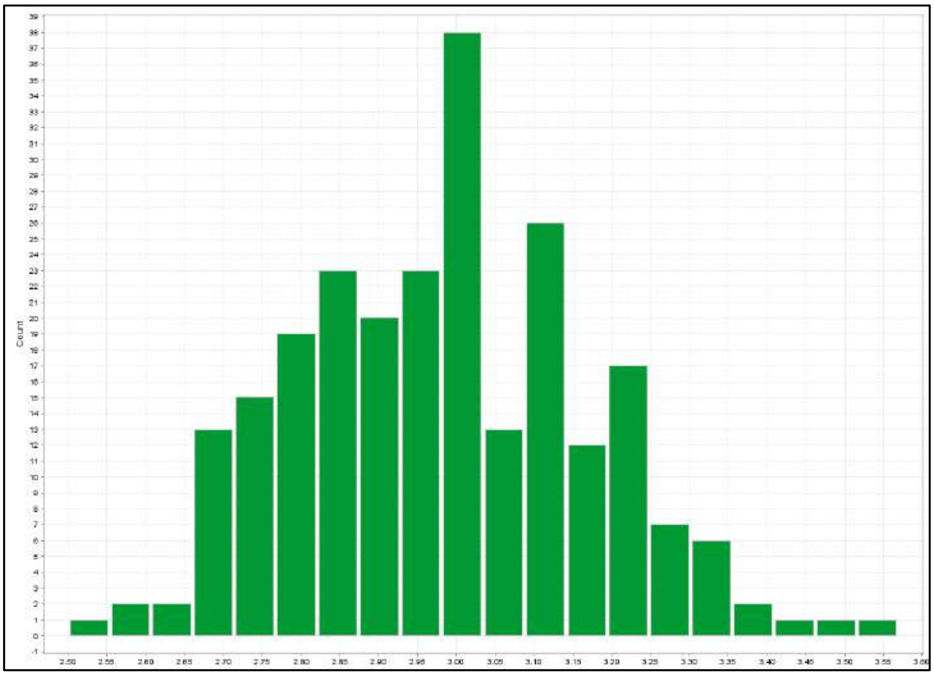


Figure 14-14: Palokas prospect density histogram for samples >0.3 g/t AuEq. Total number of samples 242

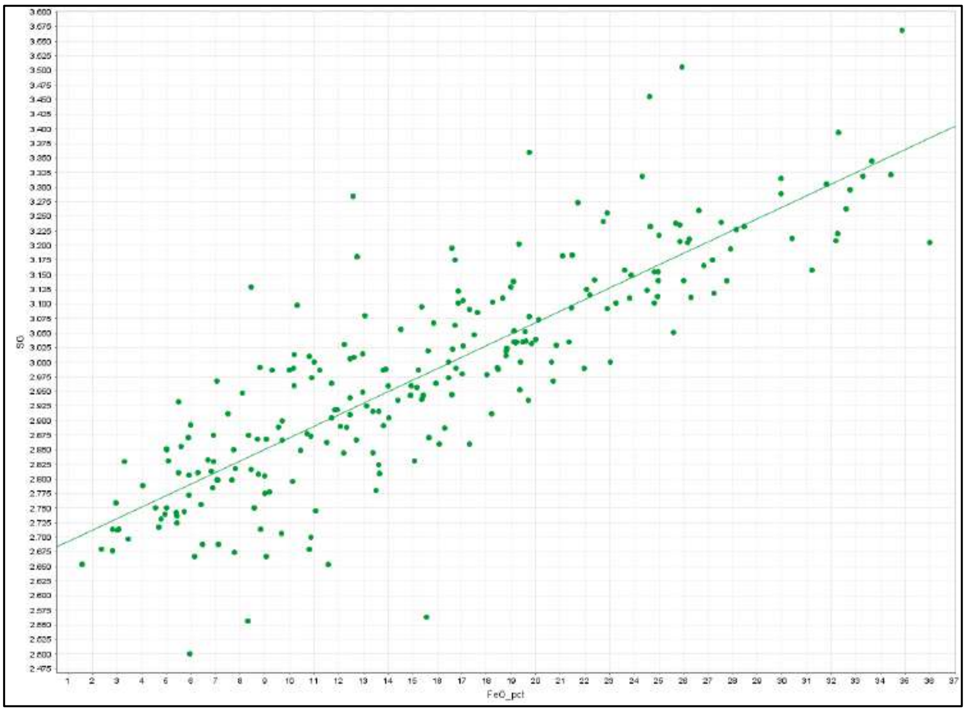


Figure 14-15: Palokas prospect density versus FeO. This produces a best fit line of density=2.6722+FeO*0.01979

14.4.3 South Palokas prospect

South Palokas prospect had 679 density determinations, of which 284 occur in mineralized rocks of >0.3 g/t AuEq. Three sets of combined Au and Co mineralized lenses occur at South Palokas. The upper lens is regarded as equivalent to Palokas and thus the linear Palokas correlation was applied to that determination. The mean density is 2.90 t/m^3 with a range of 2.67 t/m^3 to 3.73 t/m^3. Figure 14-16 and Figure 14-17 show the distribution of density measurements as a histogram and the relationship between density and FeO.

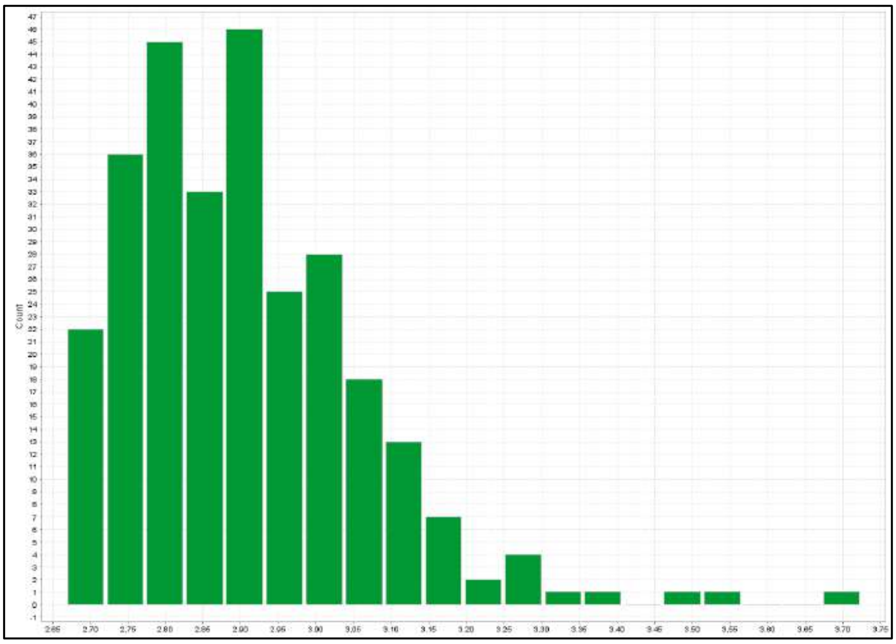


Figure 14-16: South Palokas prospect density histogram for samples >0.3 g/t AuEq. Total number of samples 284

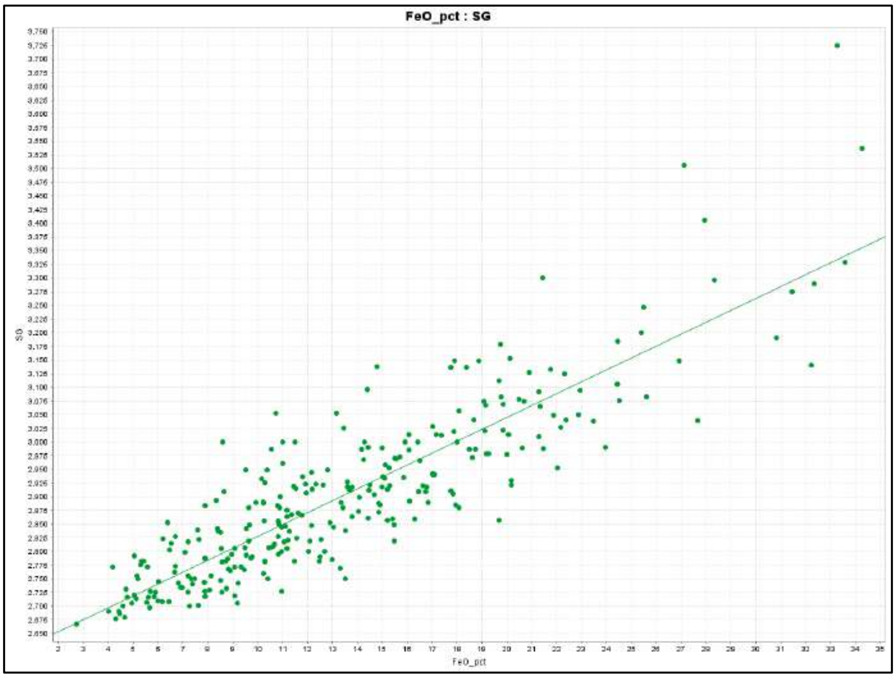


Figure 14-17: South Palokas prospect density versus FeO. This produces a best fit line of density=2.6107+FeO*0.02172

14.4.4 Raja prospect

Raja prospect had 670 density determinations, of which 260 occur in mineralized rocks of >0.3 g/t AuEq. Seven independent Au and Co mineralized lenses occur at Raja. The mean density is 2.83 t/m³ with a range of 2.61 t/m³ to 3.42 t/m³. Figure 14-18 and Figure 14-19 show the distribution of density measurements as a histogram and the relationship between density and FeO.

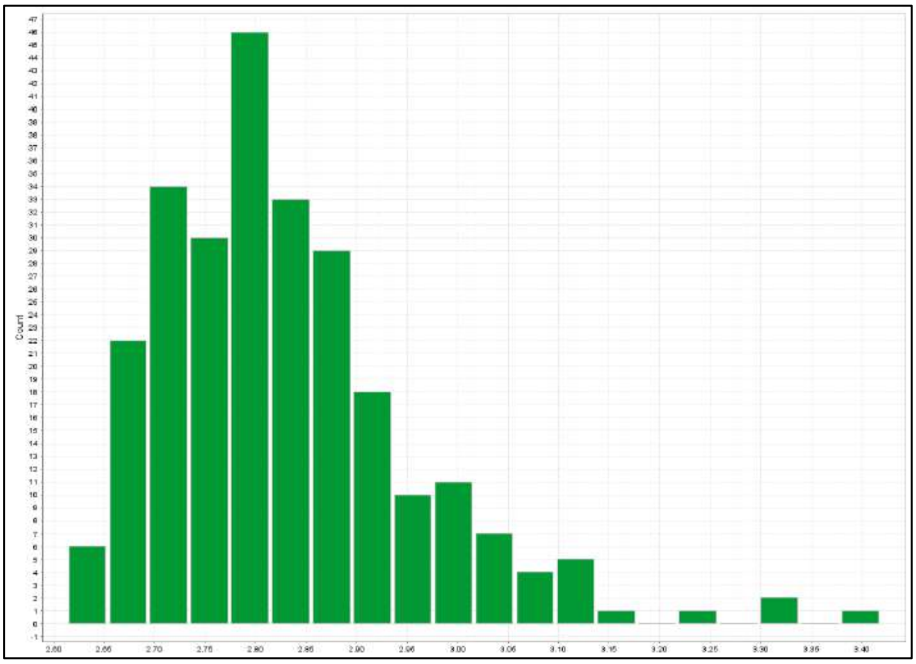


Figure 14-18: Raja prospect density histogram for samples > 0.3 g/t AuEq. Total number of samples 260

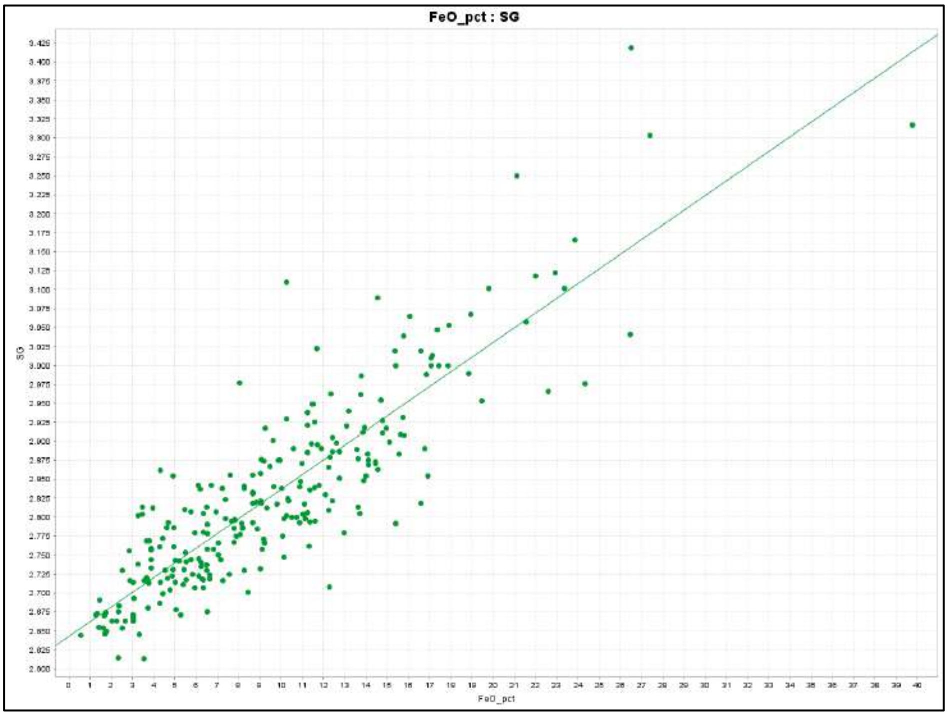


Figure 14-19: Raja prospect density versus FeO. This produces a best fit line of density=2.6426+FeO*0.01939

14.4.5 The Hut prospect

The Hut prospect had 423 density determinations, of which 153 occur in mineralized rocks of >0.3 g/t AuEq. Twelve independent Au and Co mineralized lenses occur at The Hut. The mean density is 2.91 t/m^3 with a range of 2.64 t/m^3 to 3.67 t/m^3. Figure 14-20 and Figure 14-21 show the distribution of density measurements as a histogram and the relationship between density and FeO.

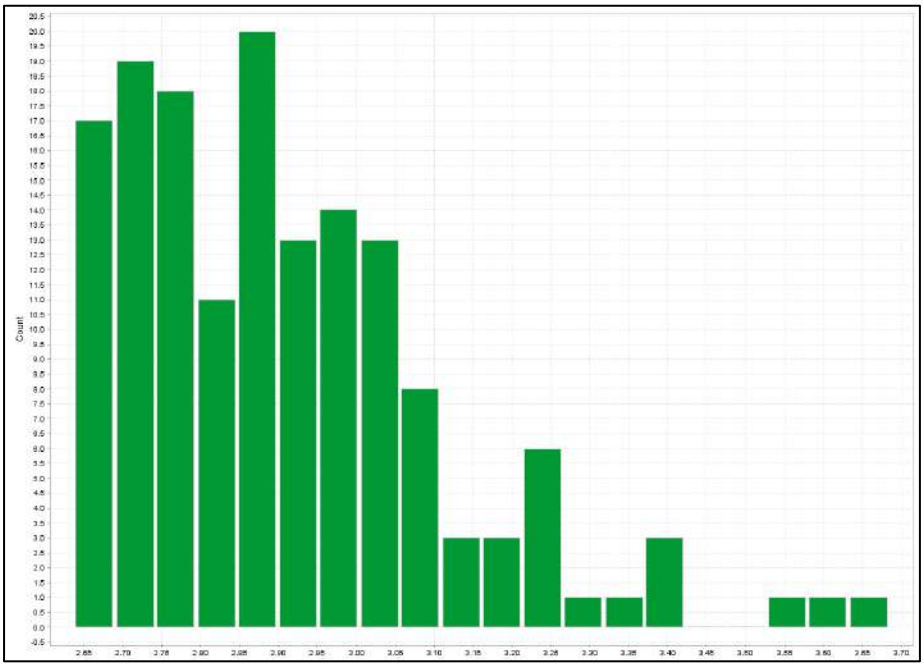


Figure 14-20: The Hut prospect density histogram for samples > 0.3 g/t AuEq. Total number of samples 153.

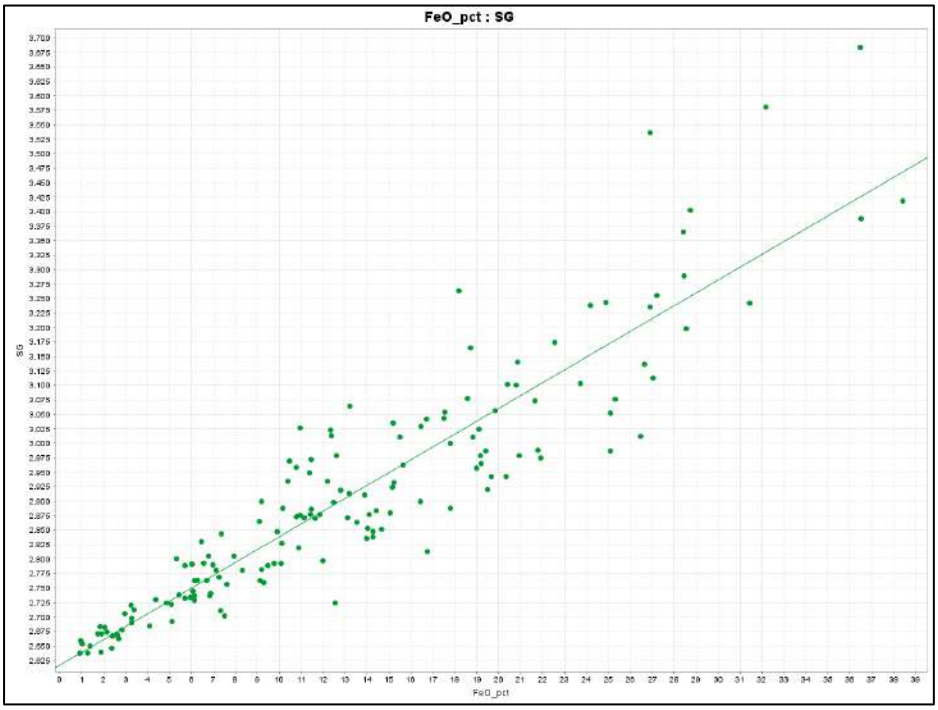


Figure 14-21: The Hut prospect density versus FeO. This produces a best fit line of density=2.6171+FeO*0.02217

14.4.6 Joki prospect

Joki prospect had 98 density determinations, of which 23 occur in mineralized rocks of >0.3 g/t AuEq. Four Au and Co mineralized lenses occur at Joki. The mean density is 2.91 t/m^3 with a range of 2.69 t/m^3 to 3.16 t/m^3. Figure 14-22 and Figure 14-23 show the distribution of density measurements as a histogram and the relationship between density and FeO.

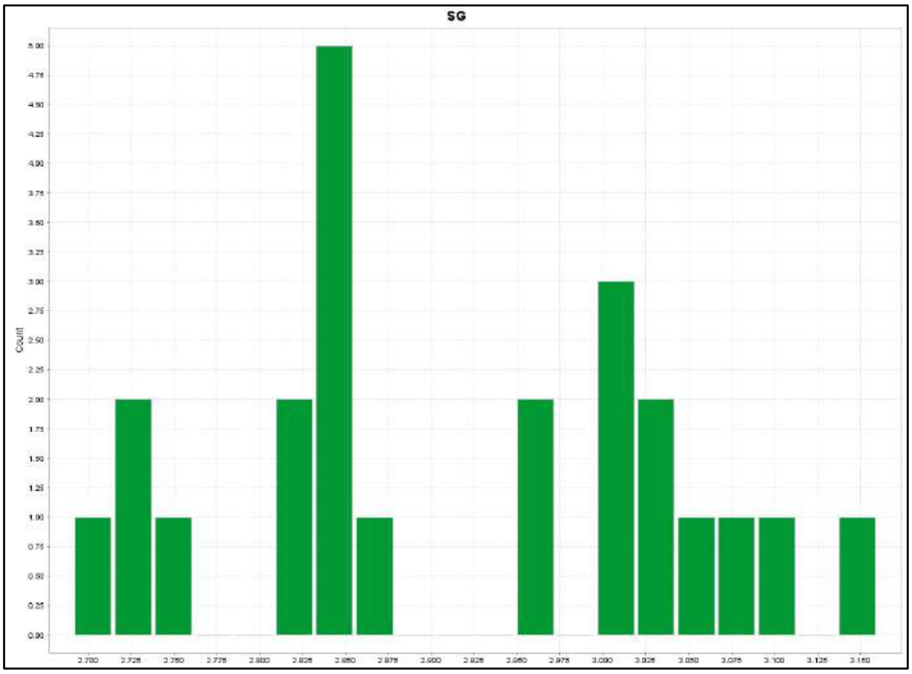


Figure 14-22: Joki prospect density histogram for samples > 0.3 g/t AuEq. Total number of samples 23.

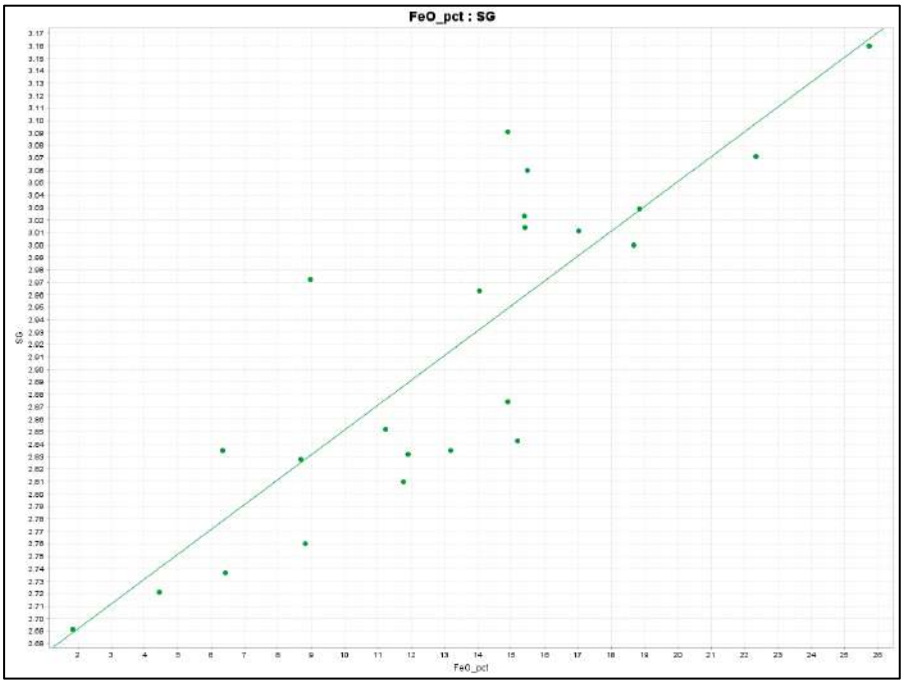


Figure 14-23: Joki prospect density versus FeO. This produces a best fit line of density=2.6519+FeO*0.01995

14.4.7 Rumajärvi prospect

Rumajärvi prospect had 557 density determinations, of which 141 occur in mineralized rocks of >0.3 g/t AuEq. Seventeen Au and Co mineralized lenses occur at Rumajärvi. The mean density is 2.84 t/m^3 with a range of 2.58 t/m^3 to 3.34 t/m^3. Figure 14-24 and Figure 14-25 show the distribution of density measurements as a histogram and the relationship between density and FeO. Terry's Hammer and Uusisaari prospects are included in Rumajärvi data.

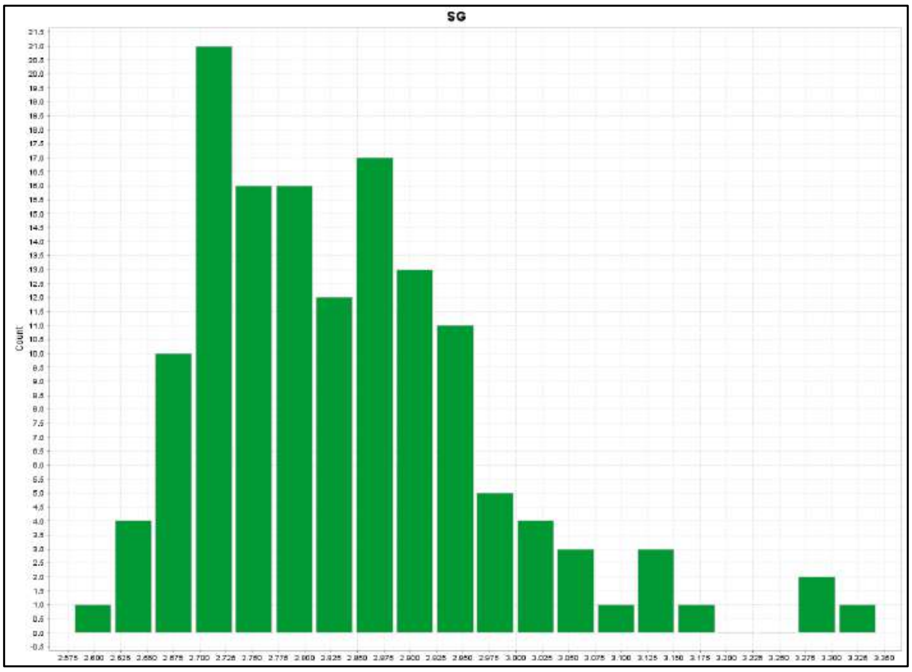


Figure 14-24: Rumajärvi prospect density histogram for samples >0.3 g/t AuEq. Total number of samples 141

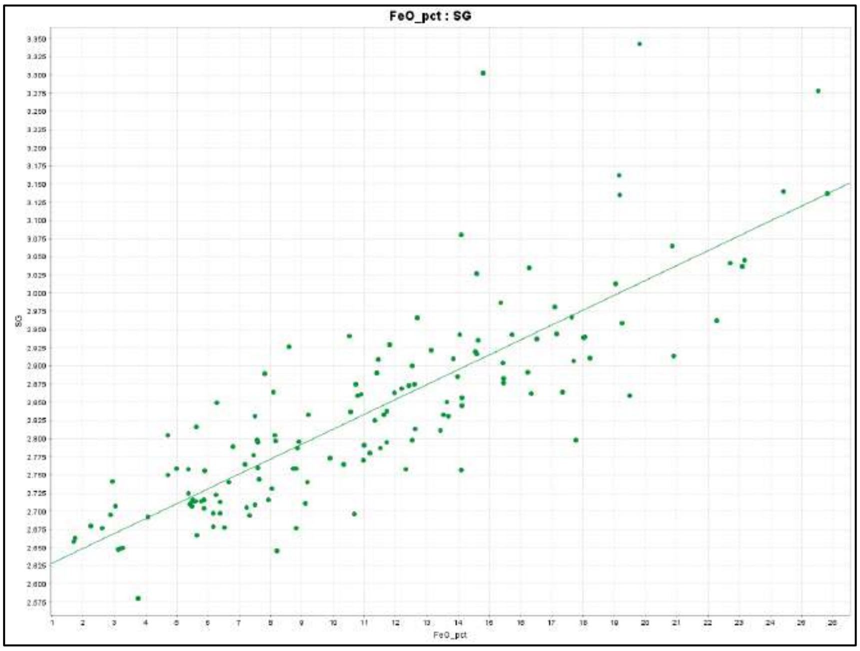


Figure 14-25: Rumajärvi prospect density versus FeO. This produces a best fit line of density=2.6089+FeO*0.02044

14.5 Block Model and Estimation Parameters

Estimation of the Rajapalot Inferred Mineral Resource estimate was completed using Leapfrog Edge across 8 prospects including 48 individual wireframes. Five sub-block models were created; the Palokas sub-block model includes Palokas and South Palokas prospects as convergence of pits occurs in some pit optimization models; Rumajärvi sub-block model includes the small Terry's Hammer and Uusisaari wireframes (domains) in addition to Rumajärvi wireframes (domains).

14.5.1 Block model parameters

Figure 14-26 to Figure 14-30 show the parent and sub-block model parameters copied directly from Leapfrog with the included map showing the limits of the model in each case. Sub-block triggers in each case were created using the gold and cobalt wireframes, the base of till and lidar surface wireframes were also used to control the density model for "air" and till blocks (till density is set to 2 t/m^3). Parent blocks were used in all cases for grade estimation. A range of parent block sizes was tested with an optimal 12 x 12 x 4 m size determined (>20% of the drillhole spacing) as suitable. Sub-blocking down to 4 x 4 x 0.5 m was optimal for geologic control on volumes, thinner and moderately dipping wireframes (testing of options up to the parent block size showed less than 5% overall variation in the Mineral Resource estimate).

For creation of the Selective Mining Unit (SMU) model for pit optimization, the sub-block model was copied and controlled to regular 5 x 5 x 2.5 m blocks. There was less than 0.5% difference in the total Mineral Resource estimate created during the change to regularized blocks.

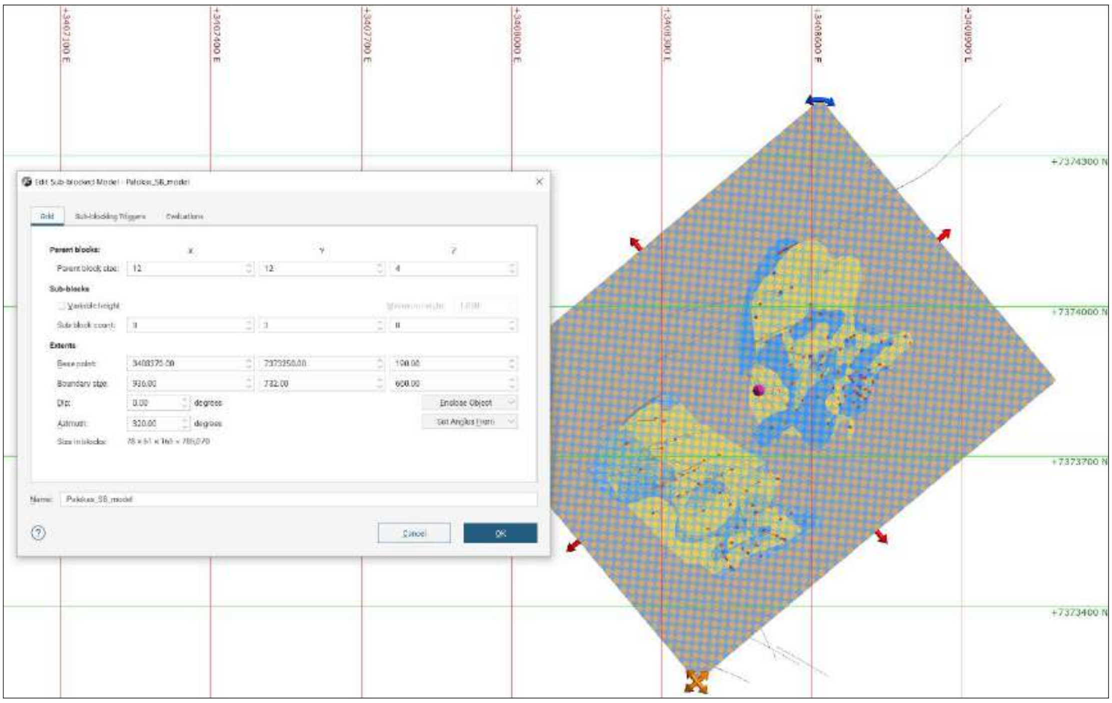


Figure 14-26: Details of Palokas sub-block model with plan view including outline of block model and gold and cobalt wireframes (785,070 blocks in total)

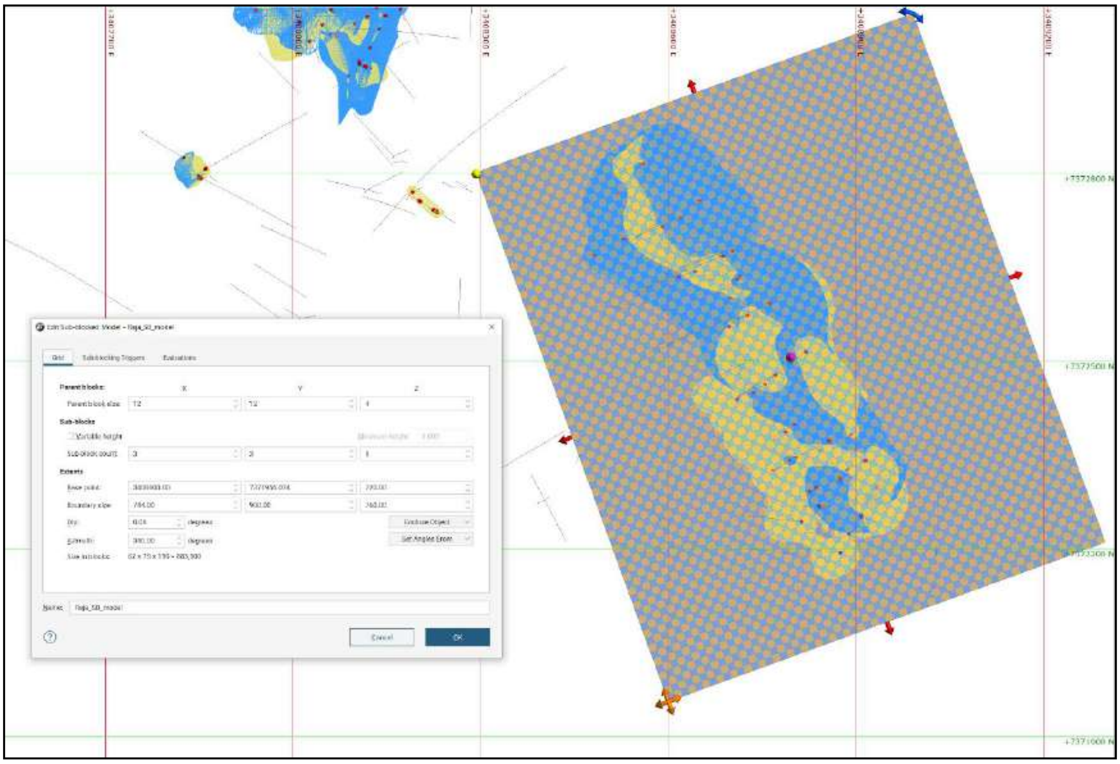


Figure 14-27: Details of Raja sub-block model with plan view including outline of block model and gold and cobalt wireframes (883,500 blocks in total)

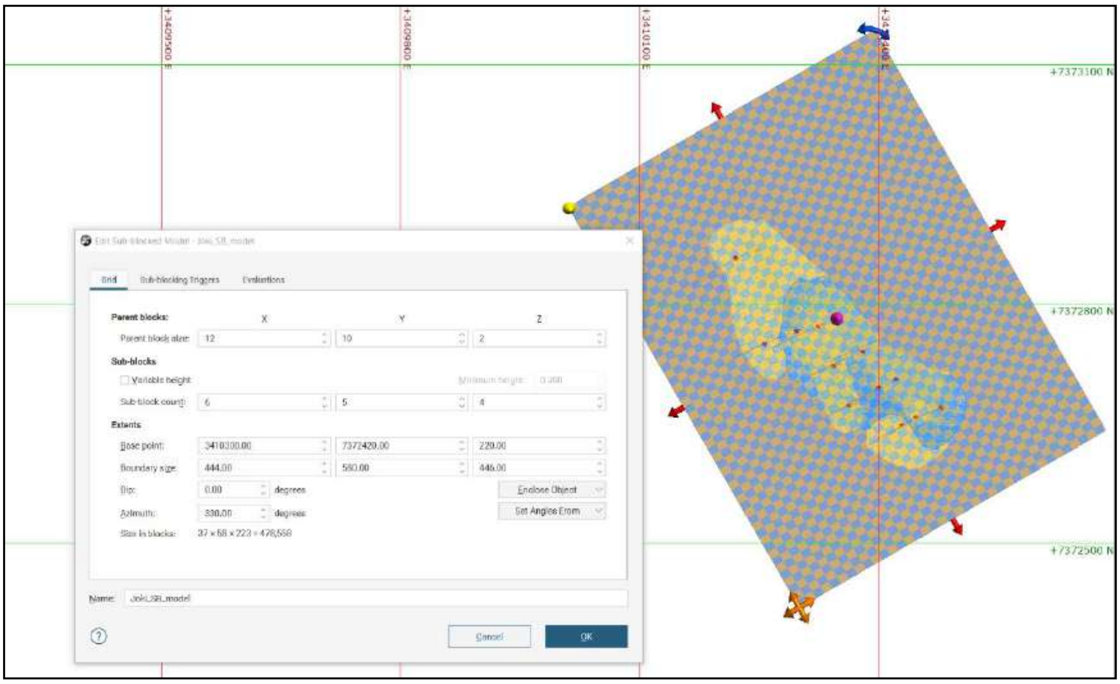


Figure 14-28: Details of Joki East sub-block model with plan view including outline of block model and gold and cobalt wireframes (478,558 blocks in total)

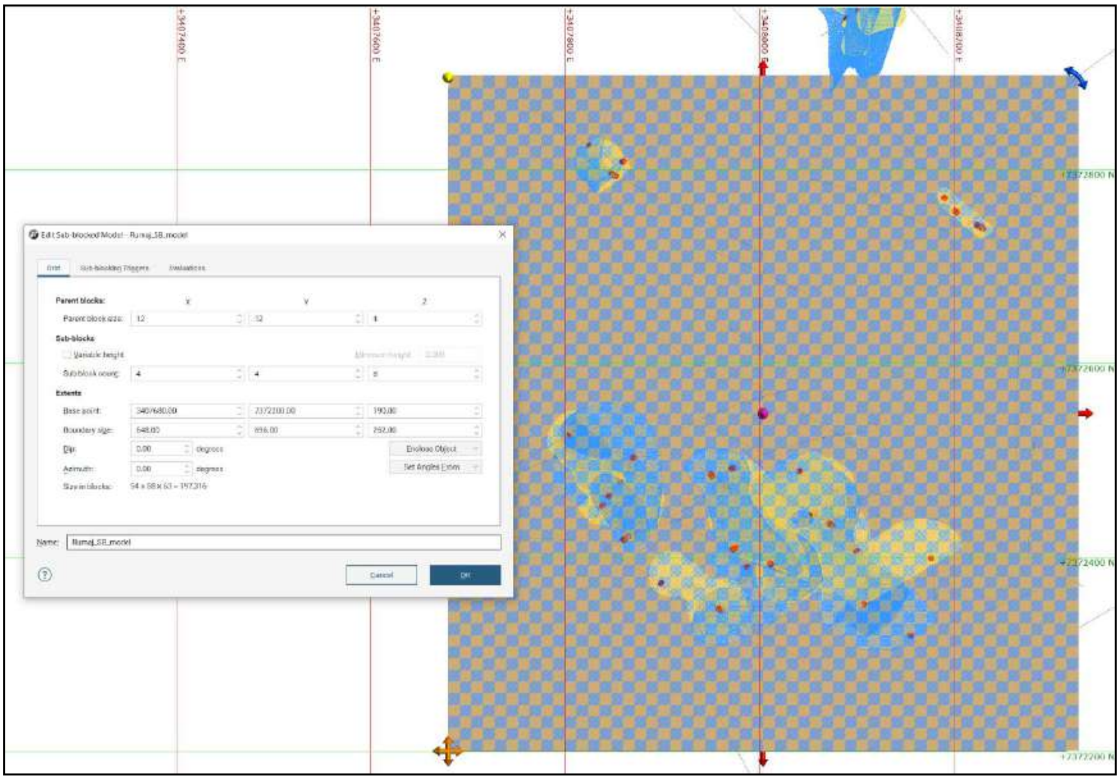


Figure 14-29: Details of Rumajärvi sub-block model with plan view including outline of block model and gold and cobalt wireframes (197,316 blocks in total)

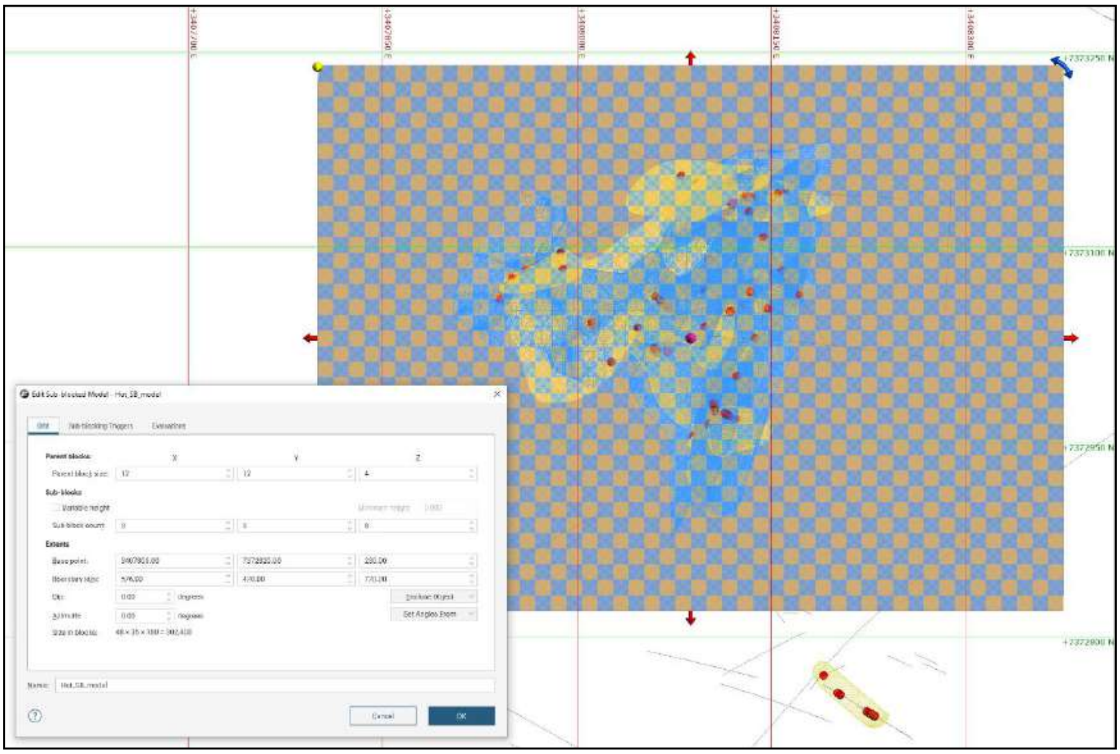


Figure 14-30: Details of The Hut sub-block model with plan view including outline of block model and gold and cobalt wireframes (302,400 blocks in total)

14.5.2 Variography

Log normalised assay data for gold (normal score) was used in the production of the variograms for gold domains and raw cobalt assay data were appropriate for cobalt domain variography. Where support is good, for example in Raja and Palokas (Figure 14-31) domains, coherent variograms reflect the geological understanding of the mineralization. In each domain, geologic control was used in the first step of the orientation of the axes of the variograms; the major axis of the variogram generally corresponds with the inferred down-plunge orientation of mineralization, the minor axis is sub-parallel to the down-hole direction. The coherence of the intermediate axis of the variogram is largely controlled by the number of drillholes on a section; more drilling on sections is a recommendation to improve the estimation at some prospects. The Palokas variography presented in Figure 14-31 as an example is encouraging in terms of the continuity and understanding of mineralization. Variogram domains, orientations of major, intermediate and minor axes and variogram statistics are presented in Table 14-5.

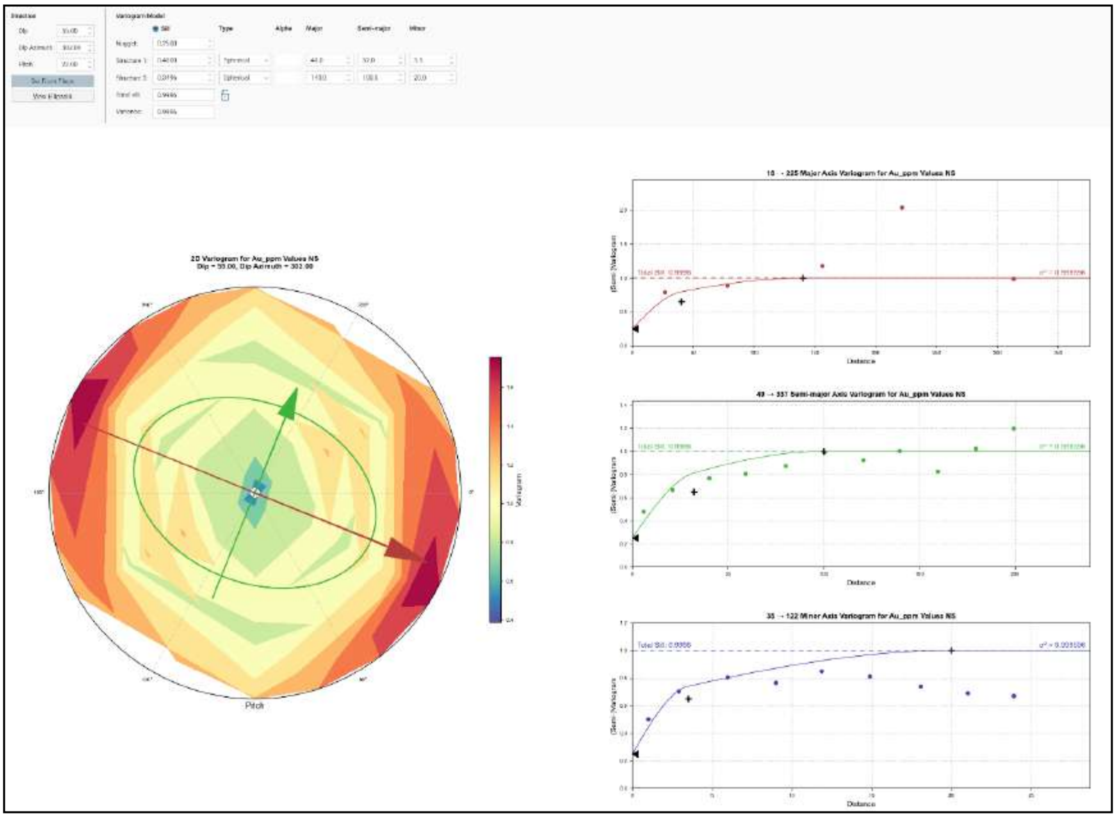


Figure 14-31: Major, intermediate and minor axis variograms for the main gold wireframe at Palokas prospect

The creation of the orientation meshes to control the variogram searches are described in Section 14.2 and detailed in Table 14-1. Further estimation parameters (search ellipses and discretization) are presented in Table 14-6 and Table 14-7.

Discussion on the quality of the estimations can be found in Section 14.10.

Table 14-5: Domain variography data for gold and cobalt

Variogram Name	Dip	Dip Azimuth	Pitch	Model space	Variance	Nugget	Normalised Nugget
Au_Hut_lens_1: Transformed Variogram Model	50	299	92	Normal score	1.0	0.20	
Au_Hut_lens_1: Transformed Variogram Model	50	299	92	Data	10.1	2.88	0.3
Au_Hut_lens_1a: Transformed Variogram Model	50	300	90	Normal score	1.0	0.20	
Au_Hut_lens_1a: Transformed Variogram Model	50	300	90	Data	0.2	0.04	0.2
Au_Hut_lens_2a: Transformed Variogram Model	56	340	80	Data	1.8	0.46	0.3
Au_Hut_lens_2a: Transformed Variogram Model	56	340	80	Normal score	1.0	0.20	
Au_Hut_lens_Upper_1: Transformed Variogram Model	71	340	94	Data	37.4	12.75	0.3
Au_Hut_lens_Upper_1: Transformed Variogram Model	71	340	94	Normal score	0.9	0.19	
Au_Hut_lens_Upper_2: Transformed Variogram Model	50	300	90	Data	0.2	0.04	0.2
Au_Hut_lens_Upper_2: Transformed Variogram Model	50	300	90	Normal score	0.9	0.18	
Au_ppm_GM-Sth_Palokas_Lower: Sth_Pal transformed VM	50	330	160	Data	0.3	0.07	0.2
Au_ppm_GM-Sth_Palokas_Lower: Sth_Pal transformed VM	50	330	160	Normal score	1.0	0.10	
Au_ppm_GM-Sth_Palokas_Main: South_Palokas_transf_VM	50	328	144	Data	18.1	6.81	0.4
Au_ppm_GM-Sth_Palokas_Main: South_Palokas_transf_VM	50	328	144	Normal score	1.0	0.25	
Au_ppm_GM-Sth_Palokas_Upper: Sth_Pal transformedVM_upper	50	330	160	Data	0.6	0.25	0.4
Au_ppm_GM-Sth_Palokas_Upper: Sth_Pal transformedVM_upper	50	330	160	Normal score	1.0	0.20	
Au_ppm_Joki_1_lens: Au_Joki_Tr_Variogram	23	343	90	Data	280.2	79.67	0.3
Au_ppm_Joki_1_lens: Au_Joki_Tr_Variogram	23	343	90	Normal score	0.9	0.19	
Au_ppm_Joki_2_lens: Au_Joki_Tr_Variogram	23	343	90	Data	280.2	79.67	0.3
Au_ppm_Joki_2_lens: Au_Joki_Tr_Variogram	23	343	90	Normal score	0.9	0.19	
Au_ppm_Joki_main: Au_Joki_Tr_Variogram	28	311	84	Normal score	1.0	0.20	
Au_ppm_Joki_main: Au_Joki_Tr_Variogram	28	311	84	Data	69.9	22.58	0.3
Au_ppm_Palokas_Main: Palokas_Transf_VM	55	302	22	Data	89.9	42.36	0.5
Au_ppm_Palokas_Main: Palokas_Transf_VM	55	302	22	Normal score	1.0	0.25	
Au_ppm_Raja_Main: Raja_Trans_Variog_Model	34	330	106	Normal score	1.0	0.10	
Au_ppm_Raja_Main: Raja_Trans_Variog_Model	34	330	106	Data	94.6	18.54	0.2
Au_ppm_Raja_Upper_central: Raja_Trans_Variog_Model	38	9	68	Data	4.8	1.34	0.3
Au_ppm_Raja_Upper_central: Raja_Trans_Variog_Model	38	9	68	Normal score	1.0	0.20	
Au_ppm_Raja_Upper_eastern: Raja_Trans_Variog_Model	38	9	68	Data	10.8	2.00	0.2
Au_ppm_Raja_Upper_eastern: Raja_Trans_Variog_Model	38	9	68	Normal score	1.0	0.10	
Au_ppm_Raja_lower_minor: Raja_Trans_Variog_Model	38	9	68	Data	0.9	0.20	0.2
Au_ppm_Raja_lower_minor: Raja_Trans_Variog_Model	38	9	68	Normal score	1.0	0.20	
Au_ppm_Rumaj_1: Transformed Variogram Model	46	330	90	Normal score	1.0	0.20	
Au_ppm_Rumaj_1: Transformed Variogram Model	46	330	90	Data	8.3	2.92	0.4
Au_ppm_Rumaj_2: Transformed Variogram Model	46	330	90	Data	8.3	2.92	0.4
Au_ppm_Rumaj_2: Transformed Variogram Model	46	330	90	Normal score	1.0	0.19	

Variogram Name	Dip	Dip Azimuth	Pitch	Model space	Variance	Nugget	Normalised Nugget
Au_ppm_Rumaj_3: Transformed Variogram Model	46	330	90	Data	8.3	2.92	0.4
Au_ppm_Rumaj_3: Transformed Variogram Model	46	330	90	Normal score	1.0	0.20	
Au_ppm_Rumaj_4: Transformed Variogram Model	46	330	90	Data	8.3	2.92	0.4
Au_ppm_Rumaj_4: Transformed Variogram Model	46	330	90	Normal score	1.0	0.20	
Au_ppm_Rumaj_6: Transformed Variogram Model	46	330	90	Data	8.3	2.92	0.4
Au_ppm_Rumaj_6: Transformed Variogram Model	46	330	90	Normal score	1.0	0.20	
Au_ppm_Rumaj_7: Transformed Variogram Model	60	321	90	Data	1.5	0.37	0.2
Au_ppm_Rumaj_7: Transformed Variogram Model	60	321	90	Normal score	1.0	0.20	
Au_ppm_Rumaj_8: Transformed Variogram Model	31	338	90	Data	0.5	0.16	0.3
Au_ppm_Rumaj_8: Transformed Variogram Model	31	338	90	Normal score	0.9	0.18	
Au_ppm_T_Hammer: Variogram Model	19	312	90	Data	0.9	0.18	0.2
Co_ppm_Joki_Main_Co: Joki_Co_VM	25	310	92	Data	787,384	314,954	0.4
Co_ppm_Palokas_Main: Palokas Co Vario_Model	55	302	156	Data	499,671	49,967	0.1
Co_ppm_Raja_Main: Co Raja Variogram Model	37	9	61	Data	722,227	144,445	0.2
Co_ppm_Raja_Upper: Co Raja Variogram Model	34	335	86	Data	225,738	-	0.0
Co_ppm_Raja_Upper_2: Co Raja Variogram Model	35	0	67	Data	117,447	-	0.0
Co_ppm_Rumaj_1: Variogram Model	43	325	68	Data	264,441	52,888	0.2
Co_ppm_Rumaj_2: Variogram Model	33	337	68	Data	72,651	14,530	0.2
Co_ppm_Rumaj_3: Variogram Model	33	337	68	Data	59,284	11,857	0.2
Co_ppm_Rumaj_4: Variogram Model	33	337	68	Data	78,356	15,671	0.2
Co_ppm_Rumaj_5: Variogram Model	33	337	68	Data	26,701	5,340	0.2
Co_ppm_Rumaj_6: Variogram Model	33	337	68	Data	150,722	30,144	0.2
Co_ppm_Rumaj_8: Variogram Model	33	337	68	Data	5,557	1,111	0.2
Co_ppm_Rumaj_lower_single: Variogram Model	33	337	68	Data	355,706	71,141	0.2
Co_ppm_Sth_Palokas_Lower: Sth Palokas_Co_VM_lower	55	320	90	Data	249,964	49,993	0.2
Co_ppm_Sth_Palokas_Main: Sth_Palokas_Co_Variogram Model	55	320	90	Data	489,271	48,927	0.1
Co_ppm_Sth_Palokas_Upper: Sth Palokas_Co_VM_upper	50	329	75	Data	24,984	2,498	0.1
Co_ppm_T_Hammer: Variogram Model	19	312	90	Data	55,798	11,160	0.2
Co_ppm_Upper: Variogram Model	23	345	68	Data	378,943	75,789	0.2
Hut_Cobalt_Upper: Variogram Model	60	300	90	Data	263,204	52,641	0.2
Hut_Cobalt_lens_1: Variogram Model	60	300	90	Data	287,513	57,503	0.2
Hut_Cobalt_lens_2: Variogram Model	60	300	90	Data	979,903	195,981	0.2
Hut_Cobalt_lens_2a: Variogram Model	60	300	90	Data	5,300	1,060	0.2
Hut_Cobalt_lens_3a: Variogram Model	60	300	90	Data	15,291	3,058	0.2
Hut_Cobalt_lens_4: Variogram Model	60	300	90	Data	527,899	105,580	0.2
Hut_Cobalt_lens_4a: Variogram Model	60	300	90	Data	1,437,099	287,420	0.2

Table 14-6: Domained estimation names and discretization inputs.

Interpolant name	Domained estimation name	Discr. X	Discr. Y	Discr. Z
Kr, Au_Hut_lens_1	Au_Hut_lens_1	2	1	1
Kr, Au_Hut_lens_1a	Au_Hut_lens_1a	2	2	1
Kr, Au_Hut_lens_2a	Au_Hut_lens_2a	3	3	1
Kr, Au_Hut_lens_Upper_1	Au_Hut_lens_Upper_1	2	2	1
Kr, Au_Hut_lens_Upper_2	Au_Hut_lens_Upper_2	2	2	1
Kr, Au_ppm_GM-Sth_Palokas_Lower	Au_ppm_GM-Sth_Palokas_Lower	5	5	2
Kr, Au_ppm_GM-Sth_Palokas_Main	Au_ppm_GM-Sth_Palokas_Main	5	5	2
Kr, Au_ppm_GM-Sth_Palokas_Upper	Au_ppm_GM-Sth_Palokas_Upper	5	5	2
Kr, Au_ppm_Joki_1_lens	Au_ppm_Joki_1_lens	2	2	1
Kr, Au_ppm_Joki_2_lens	Au_ppm_Joki_2_lens	2	2	1
Kr, Au_ppm_Joki_main	Au_ppm_Joki_main	2	2	1
Kr, Au_ppm_Palokas_Main	Au_ppm_Palokas_Main	4	4	2
Kr, Au_ppm_Raja_Main	Au_ppm_Raja_Main	2	2	1
Kr, Au_ppm_Raja_Upper_central	Au_ppm_Raja_Upper_central	2	2	1
Kr, Au_ppm_Raja_Upper_eastern	Au_ppm_Raja_Upper_eastern	2	2	1
Kr, Au_ppm_Raja_lower_minor	Au_ppm_Raja_lower_minor	2	2	1
Kr, Au_ppm_Rumaj_1	Au_ppm_Rumaj_1	2	2	1
Kr, Au_ppm_Rumaj_2	Au_ppm_Rumaj_2	2	2	1
Kr, Au_ppm_Rumaj_3	Au_ppm_Rumaj_3	2	2	1
Kr, Au_ppm_Rumaj_4	Au_ppm_Rumaj_4	2	2	1
Kr, Au_ppm_Rumaj_6	Au_ppm_Rumaj_6	2	2	1
Kr, Au_ppm_Rumaj_7	Au_ppm_Rumaj_7	2	2	1
Kr, Au_ppm_Rumaj_8	Au_ppm_Rumaj_8	2	2	1
Kr, Au_ppm_T_Hammer	Au_ppm_T_Hammer	2	2	1
Kr, Co_ppm_Joki_Main_Co	Co_ppm_Joki_Main_Co	2	2	1
Kr, Co_ppm_Palokas_Main	Co_ppm_Palokas_Main	4	4	2
Kr, Co_ppm_Raja_Main	Co_ppm_Raja_Main	2	2	1
Kr, Co_ppm_Raja_Upper	Co_ppm_Raja_Upper	5	5	2
Kr, Co_ppm_Raja_Upper_2	Co_ppm_Raja_Upper_2	5	5	2
Kr, Co_ppm_Rumaj_1	Co_ppm_Rumaj_1	2	2	1
Kr, Co_ppm_Rumaj_2	Co_ppm_Rumaj_2	2	2	1
Kr, Co_ppm_Rumaj_3	Co_ppm_Rumaj_3	2	2	1
Kr, Co_ppm_Rumaj_4	Co_ppm_Rumaj_4	2	2	1
Kr, Co_ppm_Rumaj_5	Co_ppm_Rumaj_5	2	2	1
Kr, Co_ppm_Rumaj_6	Co_ppm_Rumaj_6	2	2	1
Kr, Co_ppm_Rumaj_8	Co_ppm_Rumaj_8	2	2	1
Kr, Co_ppm_Rumaj_lower_single	Co_ppm_Rumaj_lower_single	2	2	1
Kr, Co_ppm_Sth_Palokas_Lower	Co_ppm_Sth_Palokas_Lower	5	5	2
Kr, Co_ppm_Sth_Palokas_Main	Co_ppm_Sth_Palokas_Main	5	5	2
Kr, Co_ppm_Sth_Palokas_Upper	Co_ppm_Sth_Palokas_Upper	5	5	2
Kr, Co_ppm_T_Hammer	Co_ppm_T_Hammer	2	2	1
Kr, Co_ppm_Upper	Co_ppm_Upper	2	2	1
Kr, Hut_Cobalt_Upper	Hut_Cobalt_Upper	5	5	2
Kr, Hut_Cobalt_lens_1	Hut_Cobalt_lens_1	5	5	2
Kr, Hut_Cobalt_lens_2	Hut_Cobalt_lens_2	5	5	2
Kr, Hut_Cobalt_lens_2a	Hut_Cobalt_lens_2a	5	5	2
Kr, Hut_Cobalt_lens_3a	Hut_Cobalt_lens_3a	5	5	2
Kr, Hut_Cobalt_lens_4	Hut_Cobalt_lens_4	5	5	2
Kr, Hut_Cobalt_lens_4a	Hut_Cobalt_lens_4a	5	5	2

Table 14-7: Domain details for search and minimum and maximum sample numbers used in estimation

Note that none of the "main" wireframing domains have fewer than 2 samples, single samples per drill hole were only used in searches in very thin wireframes or where geological reasoning showed grade continuity into the next drill section (to stop unreasonable estimation pinch-outs). Less than 5 % of the estimation falls within the 1 sample per drill hole category. These single sample per drill hole gold occurrences typically lie within cobalt wireframes, thus improving the AuEq estimation in a block.

Domain name	Numeric Values	Search Max.	Search Int.	Search Min.	Sample# Min.	Sample# Max
Refined_Gold_Hut_1: Hut_Au_lenses: Hut_Au_1_lens	Au_ppm	140	60	20	2	20
Refined_Gold_Hut_1a: Hut_Au_lenses: Hut_Au_1a_lens	Au_ppm	120	70	20	1	20
Refined_Gold_Hut_2a: Hut_Au_lenses: Hut_Au_2a_lens	Au_ppm	140	60	20	1	20
Refined_Gold_Hut_1: Hut_Au_lenses: Hut_Au_upper_1	Au_ppm	150	68	5	1	20
Refined_Gold_Hut_1: Hut_Au_lenses: Hut_Au_upper_2	Au_ppm	120	70	10	1	20
Refined-Sth_Pal_Gold: Gold_Sth_Pal: Sth_Pal_lower	Au_ppm	264	187	20	2	20
Refined-Sth_Pal_Gold: Gold_Sth_Pal: Sth_Pal_Au_Main	Au_ppm	120	80	10	2	20
Refined-Sth_Pal_Gold: Gold_Sth_Pal: Sth_Pal_Au_upper	Au_ppm	120	85	40	4	20
Refined GM-Au_Joki: Joki_gold: Joki_1_lens	Au_ppm	100	40	10	2	20
Refined GM-Au_Joki: Joki_gold: Joki_2_lens	Au_ppm	100	40	10	1	20
Refined GM-Au_Joki: Joki_gold: Joki_main	Au_ppm	100	40	10	2	20
Refined-Palokas_Gold: Palokas_gold_main: Palokas_main	Au_ppm	140	70	25	2	20
Refined_Raja_main_gold_only: Raja_gold_main: Raja-gold-main	Au_ppm	150	80	40	1	20
Raja_gold_upper: Raja_V2_upper_central	Au_ppm	150	80	50	1	20
Raja_gold_upper: Raja_V2_upper_eastern	Au_ppm	150	80	50	1	20
Refined_Raja_main_gold_only: Raja_gold_main: Raja_minor_lower	Au_ppm	150	80	50	1	20
Refined GM_Rumaj_Au_1: Rumaj_Au: Rumaj_Au_1	Au_ppm	140	60	20	2	20
GM_Rumaj_Au: Rumaj_Au_2	Au_ppm	140	60	20	2	20
GM_Rumaj_Au: Rumaj_Au_3	Au_ppm	140	60	20	2	20
Refined GM_Rumaj_Au_4: Rumaj_Au: Rumaj_Au_4	Au_ppm	140	60	20	2	20
GM_Rumaj_Au: Rumaj_Au_6	Au_ppm	180	100	20	2	20
GM_Rumaj_Au: Rumaj_Au_7	Au_ppm	140	60	20	2	20
GM_Rumaj_Au: Ruma_Au_8	Au_ppm	140	60	20	2	20
GM_T_Hammer_Au-Co: T_Hammer_Au	Au_ppm	120	80	25	2	20
Joki_Main_Co: Joki_Main_Co	Co_ppm	120	80	20	2	20
Refined-Palokas_Cobalt: Cobalt_Palokas: Pal_Cobalt_wireframe	Co_ppm	200	200	140	2	20
Refined_Raja_cobalt_main: Raja_cobalt_main: Raja_cobalt_main	Co_ppm	200	100	60	2	20
Raja_Upper_Co: Raja_Co_upper	Co_ppm	200	100	60	2	20
Raja_Upper_Co: Co_upper-2	Co_ppm	200	100	60	2	20
Refined GM_Rumaj_Co_1: Rumaj_Co: Rumaj_Co_1	Co_ppm	120	80	25	2	20
GM_Rumaj_Co: Rumaj_Co_2	Co_ppm	120	80	25	2	20
GM_Rumaj_Co: Rumaj_Co_3	Co_ppm	120	80	25	2	20
GM_Rumaj_Co: Rumaj_Co_4	Co_ppm	120	80	25	2	20
Refined GM_Rumaj_Co_5: Rumaj_Co: Rumaj_Co_5	Co_ppm	120	80	25	2	20
GM_Rumaj_Co: Rumaj_Co_6	Co_ppm	120	80	25	2	20
GM_Rumaj_Co: Rumaj_Co_8	Co_ppm	120	80	25	2	20
GM_Rumaj_Co: Rumaj_Co_lower_single	Co_ppm	120	80	25	2	20

Domain name	Numeric Values	Search Max.	Search Int.	Search Min.	Sample# Min.	Sample# Max
Refined_Sth_Pal_Cobalt: Cobalt_wireframing: Sth_Pal_lower_Co	Co_ppm	87	71	62	4	20
Refined_Sth_Pal_Cobalt: Cobalt_wireframing: Sth_Palokas_Main	Co_ppm	150	75	80	4	20
Refined_Sth_Pal_Cobalt: Cobalt_wireframing: Sth_Pal_upper_Co	Co_ppm	87	71	62	4	20
GM_T_Hammer_Au-Co: T_Hammer_Au	Co_ppm	120	80	25	2	20
GM_Rumaj_Co: Rumaj_Co_upper	Co_ppm	120	80	25	2	20
Refined_Cobalt_Hut_upper: Hut_Co_lenses: Hut_top_lens	Co_ppm	140	80	15	2	20
Refined_Cobalt_Hut: Hut_Co_lenses: Hut_Lens_1	Co_ppm	140	80	15	2	20
Refined_Cobalt_Hut: Hut_Co_lenses: Hut_Lens_2	Co_ppm	140	80	15	2	20
Refined_Cobalt_Hut: Hut_Co_lenses: Hut_Lens_2a	Co_ppm	140	80	15	2	20
Refined_Cobalt_Hut: Hut_Co_lenses: Hut_lens_3a	Co_ppm	140	80	15	2	20
Refined_Cobalt_Hut_4: Hut_Co_lenses: Hut_lens_4	Co_ppm	140	80	15	2	20
Refined_Cobalt_Hut: Hut_Co_lenses: Hut_Lens_4a	Co_ppm	140	80	15	2	20

14.6 Block Calculation Routines

Calculations and filters used, following the estimation of gold and cobalt onto the block model and subsidiary sub-blocks, include:

- Ensuring negative blocks created as rare artefacts of estimation are not exported (fewer than 5 negative blocks were created in gold estimation and fewer than 10 in other elements).
- Density models were calculated from FeO data (Figure 14-32), assignment of density to blocks outside the wireframes was made using the average value determined from the non-mineralized blocks for each prospect.
- A “waste” versus “ore” category was made for use in the export for pit optimization modelling.
- Gold equivalent calculations for each block (AuEq = Au + (Co/1005).
- A pit model calculator was included to allow classification of blocks that lie inside or outside any pits (Figure 14-33). The calculation for an “all underground” option is also included. Filters can then be applied to determine block tonnes, average densities and grade, in addition to total metal in any particular category.

Figure 14-32: Example of density calculations used in block models (image from Leapfrog Edge)

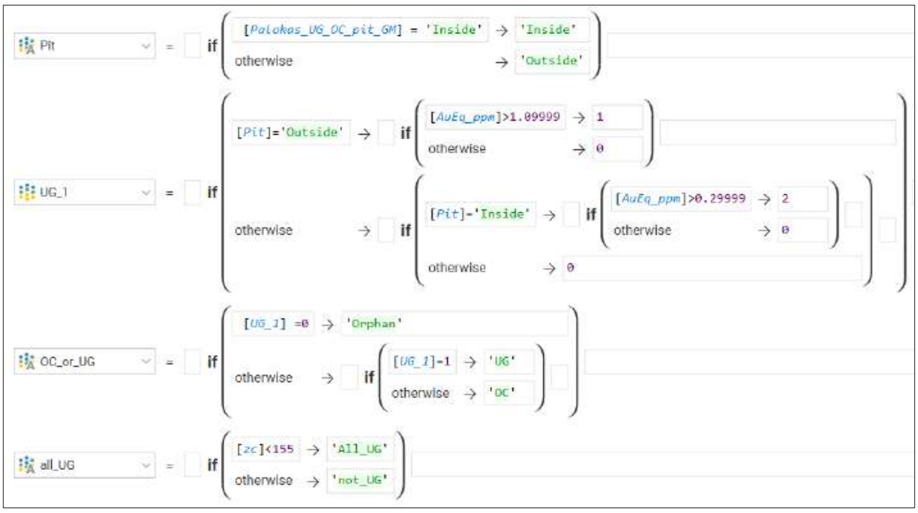


Figure 14-33: Example of calculations made to determine if blocks fall inside or outside any pit models, or, in the case of the "All underground" model, occur in solid rock more than 20 m below the base of the till (image from Leapfrog Edge)

14.7 Pit Optimization and RPEEE

Optimization of the resource was conducted using Whittle software based on the criteria for pit optimization, using input parameters are from Previous MRE (Table 14-8). Fixed cut-off grades were then used as follows:

- 1.1 g/t AuEq outside the optimal pits, potentially to be accessed by underground (UG) methods.
- 0.3 g/t AuEq for the deposits within each optimal pit (OC).

Whittle software (version 4.7.3) was used in the optimization on Palokas, South Palokas, Raja, The Hut, Rumajärvi, Uusisaari, Terry's Hammer and Joki prospect wireframes to define the mineralization falling within the confines of an open pit (demonstrating RPEEE). Five block models were created covering the 8 prospects. Mineralization falling outside these pits above the cut-off grade of 1.1 g/t AuEq was then defined as underground resources with RPEEE.

Criteria for pit optimization were based on knowledge of current and recent Finnish and European operations. The optimization process was conducted considering three scenarios:

- Whittle optimization for a pit of Revenue Factor 1 (Rev-F-1);
- changeover from OC to UG based on the estimated differential operating expenses of OC and UG (model OC-UG); and
- UG scenario where a depth of 20 m below the base of solid rock was regarded as the near-surface limit of potential mining (UG only).

These three scenarios were developed to allow consideration of RPEEE without further consideration of economic viability. The MRE for this report, utilizes the "All underground Model" as the base case, rather than the "Open Pit-Underground Model" selected in the Previous MRE.

Table 14-8: Criteria used for pit optimization models in Previous MRE

Property	Details
Gold price	USD1,590/oz
Cobalt price	USD23.07/lb
Processing cost	USD12.00/t
Processing recovery gold	97%
Processing recovery cobalt	80%
G & A costs	USD2.35/t
Selling cost	USD0.75/oz Au
Royalty	0.15% of revenue
Mining cost at the surface	USD1.50
Mining cost increased	USD0.02 per 5 m bench
Geologic model	Regularized to 5 m x 5 m x 2.5 m to account for dilution
Whittle pit shells	Created using 5 m benches
Overall slope angle	50°
No allowance for capital included	
Additional cost for mining ore	USD0.60/t
Estimate for underground mining	USD30/t

14.8 Prospect Cross Sections

A series of 25 m thick sections with gold equivalent grades on blocks and drill traces in addition to the previously preferred OC-UG model are presented here (Figure 14-34 to Figure 14-39) noting the following:

- Orientations of view and the scale are all situated in the lower right of each section.
- All sections are vertical and in each case the surface is marked with the Lidar topography mesh.
- Legend for drill hole and block model grades (AuEq) are the same.
- Pit outlines where shown are the OC-UG model.

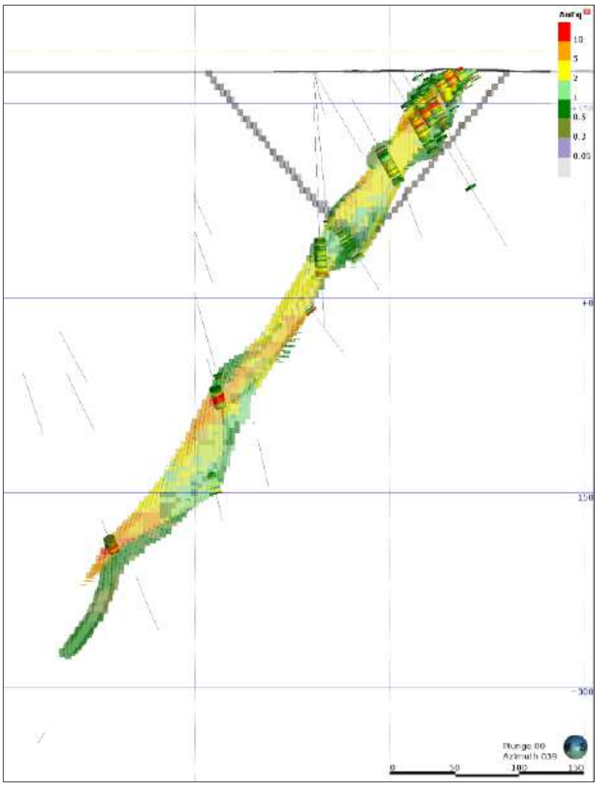


Figure 14-34: Palokas Prospect OC-UG model thick section (25 m) looking due north at Palokas showing good continuity of grade (AuEq) in drilling near surface

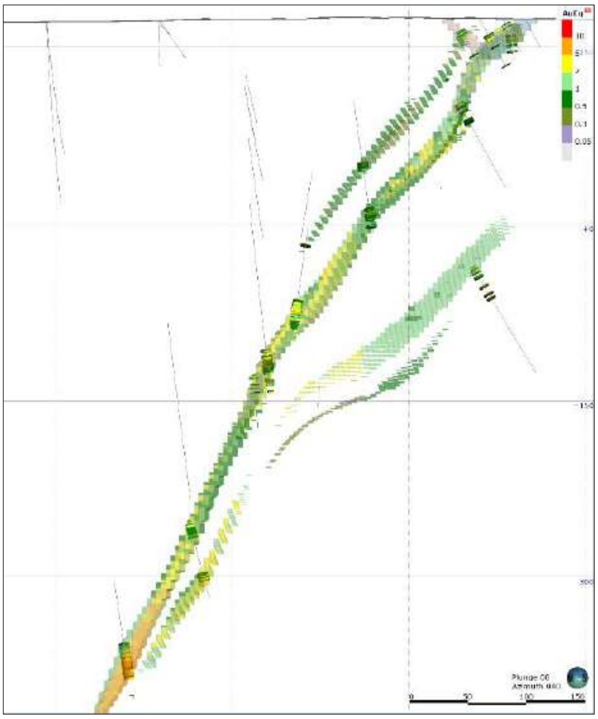


Figure 14-35: South Palokas prospect OC-UG model thick section (25 m) looking northeast showing increase of of grade (AuEq) at depth

Note the three main lenses (the lower lens looks split as the gold and cobalt wireframes diverge slightly in the plane of the section).

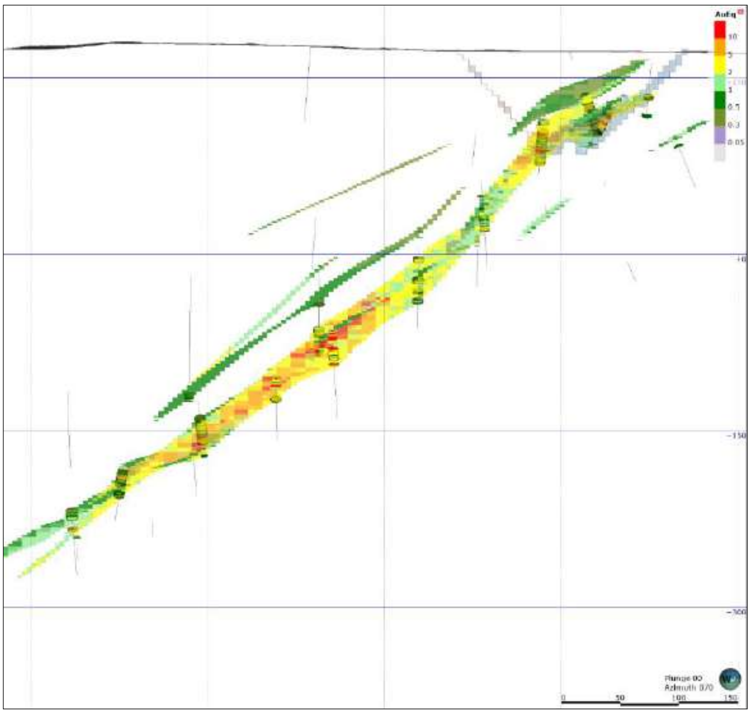


Figure 14-36: Raja prospect OC-UG model thick section (25 m) looking east-northeast (070°) parallel to main high-grade trend (34°/340°)

Note the main lens carries >90% of the tonnes and grade at Raja. Central area on the section includes drill hole PAL0093 – the best hole on the property in terms of grade times width (and photo in Figure 10-2).

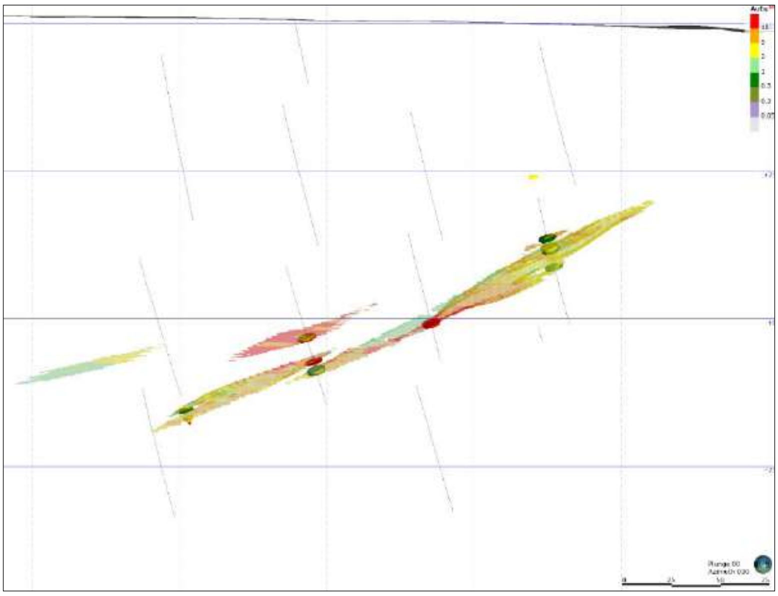


Figure 14-37: Joki prospect OC-UG model thick section (25 m) looking northeast (030°)

Note the main lens has internal waste lenses present, but these are only partly visible in this vertical section. Also note the very high grade in the thin upper lens.

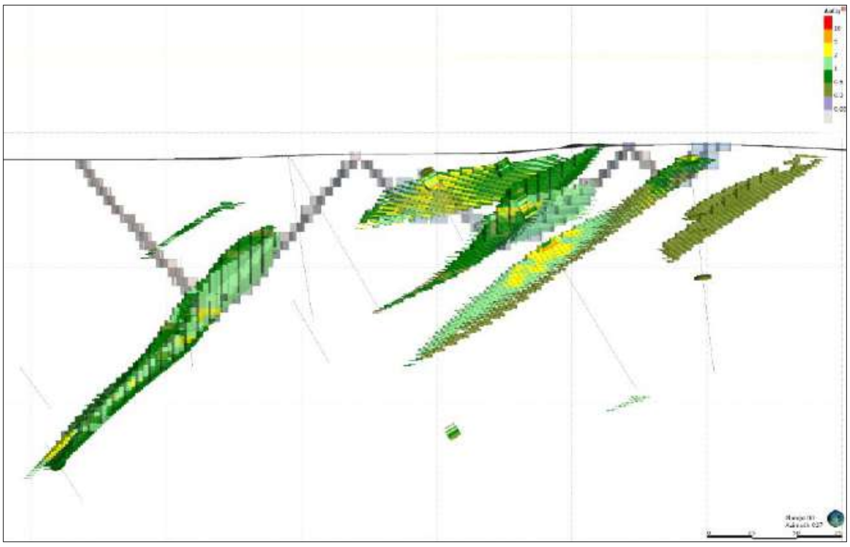


Figure 14-38: Rumajärvi prospect OC-UG model thick section (25 m) looking north-northeast (027°)

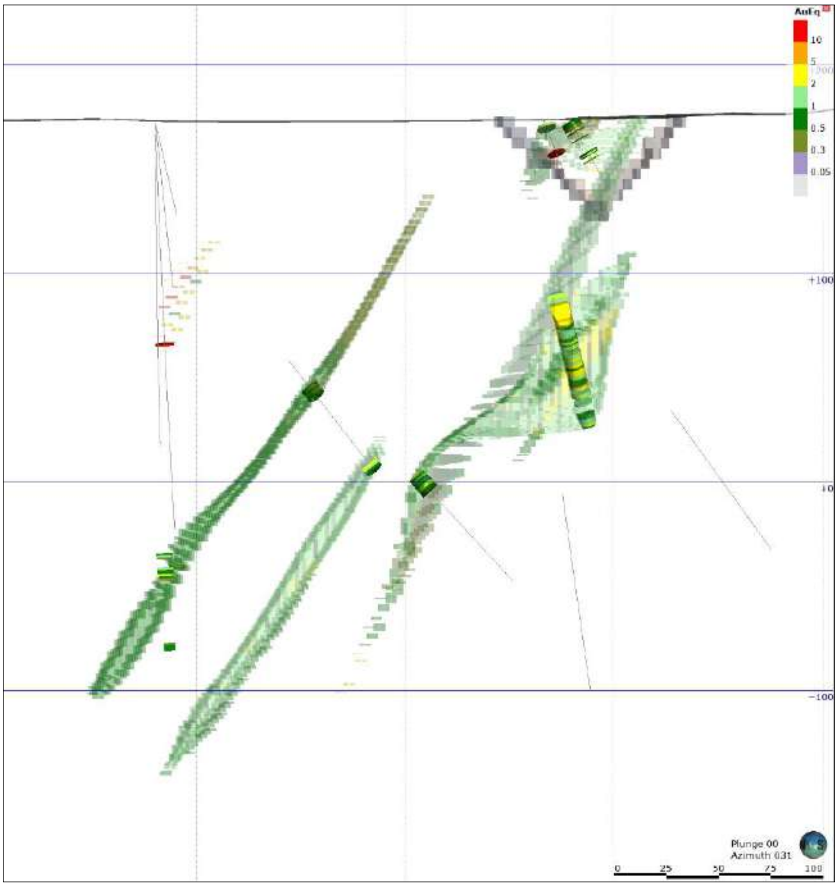


Figure 14-39: The Hut prospect OC-UG model thick section (25 m) looking north-northeast (031°)

The thick intersection in PAL0259 is a complex as the gold and cobalt wireframes are separated. This is the reverse of nearly every other mineralized zone on the property where gold and cobalt have a better spatial correlation. The overall effect of this separation in this view is that the AuEq wireframes are thicker than might be expected.

14.9 Mineral Resource Estimation Methodology

14.9.1 Introduction

The Mineral Resource estimate, prepared under the CIM Definition Standards 2014, is presented here for the underground-only base case scenario, discussed in Section 14.7, which represents the most reasonable prospect for eventual economic extraction (RPEEE). For the underground-only scenario, a depth of 20 m below the base of solid rock was regarded as the near-surface limit of potential mining (UG only).

The previous Inferred Mineral Resource estimate outlined in the technical report titled "Mineral Resource Estimate NI 43-101 Technical Report – Rajapalot Property" dated 26 August 2021 (Previous MRE), is the basis for the updated Mineral Resource statement and all estimations remains the same, only calculated gold equivalent has been updated.

The original gold equivalent ($AuEq^1$) stated in Previous MRE was calculated on each block using the then-projected long term projected prices of USD1,590 per troy ounce and USD23.07 per pound for gold and cobalt, respectively. This results in $AuEq^1$ = Au (g/t) + Co/1,005 (ppm).

The updated gold equivalent ($AuEq^2$) and ounces stated in this PEA report are based on the updated metal prices of USD1,700/oz Au and USD60,000/t Co and recovery assumptions of 95% Au and 87.6% Co. ($AuEq^2$ = Au (g/t) x 95% + Co x 87.6% / 911 (ppm)).

14.9.2 Mineral Resource classification

The Mineral Resource estimates are presented here are considered to be of Inferred category based on the following:

- Drillhole spacing is not yet sufficient for full variogram analysis.
- Although down-hole grade continuity is good, major and intermediate axes of the variograms are commonly limited in quality. In order of quality from best to worst, the estimation quality of the wireframes within the block models are as follows:
  - Palokas prospect less than 200 m depth;
  - core of Raja main lens;
  - South Palokas main lens;
  - Joki East main lens;
  - deeper Palokas main lens;
  - remainder of lenses in block models; and
  - many thin lenses contain limited sample assay data.

Recommendations on upgrading from Inferred to Indicated category are presented in Section 14.10 (Validation).

14.9.3 Gold Equivalent calculations ($AuEq^1$ & $AuEq^2$)

$AuEq^1$ was used in original Inferred Mineral Resource to determine cut-off grade of 1.1 g/t $AuEq^1$ and to show RPEEE.

Based on long term projects of gold and cobalt prices in USD (CIBC, June 2021), the gold equivalent ($AuEq^1$) formula is calculated to be:

- $AuEq^1$ (g/t) = Au + (22.0462 x Co x 23.07/10000)/(1590/31.10348).
- Gold price USD1,590 per troy ounce.
- Cobalt price USD23.07 per pound.
- Conversion of cobalt pounds to tonnes, factor is 22.0462.
- Conversion of gold from grams to ounces, factor is 31.10348.

This results in $AuEq^1$ = Au (g/t) + Co/1,005 (ppm)

$AuEq^2$ is used in reporting the updated mineral resource to reflect the change in long-term commodity prices and include recovery factors.

Based on long term projects of gold and cobalt prices in USD, the gold equivalent ($AuEq^2$) formula is calculated to be:

- $AuEq^2$ (g/t) = (Au x 95%) + (22.0462 x Co x 27.22 x 87.6%/10000)/(1700/31.10348).
- Gold price USD1,700 per troy ounce.
- Cobalt price USD27.22 per pound.
- Conversion of cobalt pounds to tonnes, factor is 22.0462.
- Conversion of gold from grams to ounces, factor is 31.10348.
- Gold recovery is 95%.
- Cobalt recovery is 87.6%.

This results in $AuEq^2$ = Au (g/t) x 95% + Co x 87.6% / 911 (ppm)

14.9.4 Mineral Resource Estimate Summary

The Inferred MRE for the Rajapalot Project, with an effective date of 26 August 2021, is summarised in Table 14-9 based on the underground-only option. The gold equivalent formula is $AuEq^2$=Au x 95% + Co x 87.6% / 911, based on assumed commodity prices of Co USD27.22/lb and Au USD1,700/oz and recovery factors for Au (95%) and Co (87.6%). Rounding of grades and tonnes may introduce apparent errors in averages and contained metals. Drilling results to 20 June 2021. These are Mineral Resources that are not Mineral Reserves and do not have demonstrated economic viability.

Table 14-9: Rajapalot Inferred Mineral Resources Effective 26 August 2021

Zone	Cut-off (AuEq¹)	Tonnes (kt)	Au (g/t)	Co (ppm)	AuEq² (g/t)	Au (koz)	Co (tonnes)	AuEq² (koz)
Palokas	1.1	5,612	2.8	475	3.1	501	2,664	562
Raja	1.1	2,702	3.1	385	3.3	271	1,040	288
East Joki	1.1	299	4.5	363	4.6	43	109	44
The Hut	1.1	831	1.3	428	1.6	36	355	44
Rumajärvi	1.1	336	1.4	424	1.7	15	142	19
Total Inferred Resources		**9,780**	**2.8**	**441**	**3.0**	**867**	**4,311**	**958**

- The independent geologist and Qualified Person as defined in NI 43-101 for the mineral resource estimates is Mr. Ove Klavér (EurGeol). The effective date of the MRE remains unchanged to the Previous MRE (August 26, 2021, available on SEDAR), and will be restated in the PEA technical report when it is filed.
- The mineral estimate is reported for a potential underground only scenario. Inferred resources were reported at a cut-off grade of 1.1 g/t (AuEq[1] Au g/t + Co ppm /1005) with a depth of 20 meters below the base of solid rock regarded as the near-surface limit of potential mining. Refer to the Previous MRE for details on the cut-off grade calculation used in calculating the Inferred Mineral Resource.
- Resource gold equivalent grades (AuEq[2]) and ounces stated here are based on the updated PEA metal prices of $1,700/oz Au and $60,000/t Co and recovery assumptions of 95% Au and 87.6% Co. (AuEq2 = Au g/t x 95% + Co ppm x 87.6% / 911).
- Wireframe models were generated using gold and cobalt shells separately. Forty-eight separate gold and cobalt wireframes were constructed in Leapfrog Geo and grade distributions independently estimated using Ordinary Kriging in Leapfrog Edge. A gold top cut of 50 g/t Au was used for the gold domains. A cobalt top cut was not applied.
- A parent block size of 12 x 12 x 4 m (>20% of the drillhole spacing) was determined as suitable. Sub-blocking down to 4 m x 4 m x 0.5 m was used for geologic control on volumes, thinner and moderately dipping wireframes
- Rounding of grades and tonnes may introduce apparent errors in averages and contained metals.
- Drilling results to 20 June 2021.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

14.10 Validation

The first stage of the validation of the Inferred Mineral Resource estimate involved a visual inspection of the relationship between the wireframe boundaries and the gold and cobalt drillhole assay data. It was confirmed that the edges of the wireframes and the internal waste boundaries (where included, that is, using a “Refined Geological Model”) are relatively good given the drillhole spacing.

Sections down dip and across strike in each prospect were checked against grade blocks and drillhole assay data to check for a reasonable correlation of drillhole grades with expected block grades. It is clear that where data support is good (close spaced drillholes, as shown in Figure 14-40 and Figure 14-43), the block models return the best results in terms of Slope of Regression (SoR) (Figure 14-41 and Figure 14-44) and a relatively low kriging variance (Figure 14-42 and Figure 14-45).

Swath plots in these areas of higher data support have a much better correlation. A selection of swath plots is presented in Figure 14-46 to Figure 14-57.

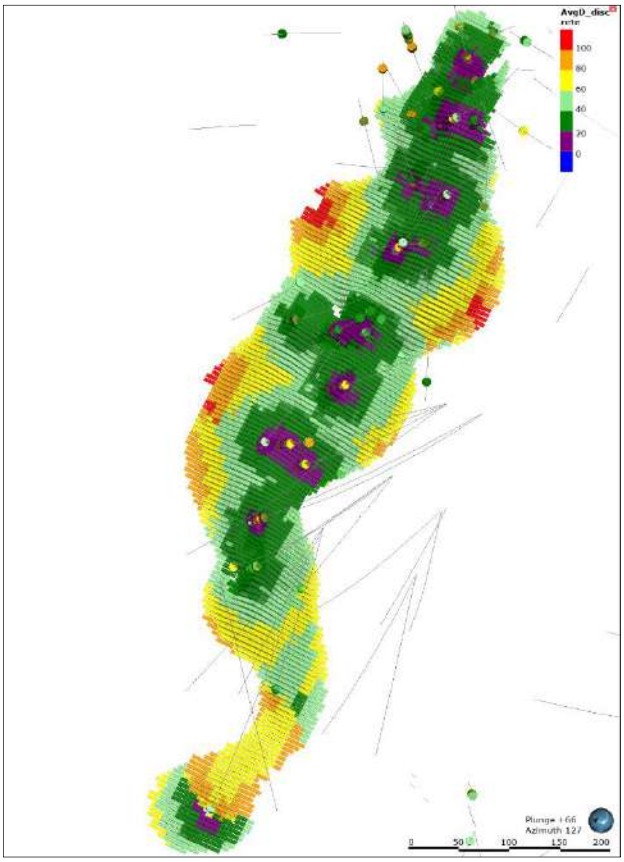


Figure 14-40: Plot of average distance between drillholes showing core of high-grade zone with greatest density of drillholes, view looking down on Raja wireframe steeply towards southeast (66/127)

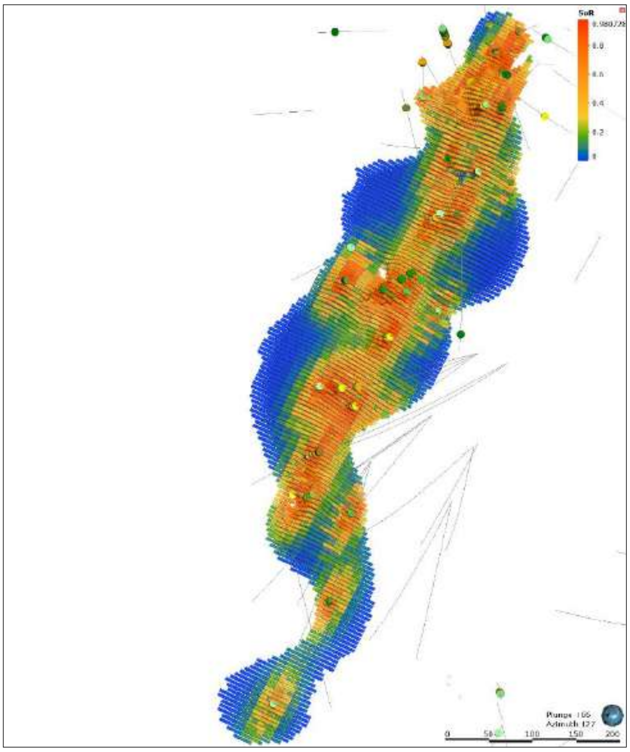


Figure 14-41: Slope of regression analysis (ideal value is 1.0 for Ordinary Kriging), view looking down on Raja wireframe steeply towards southeast (66/127)

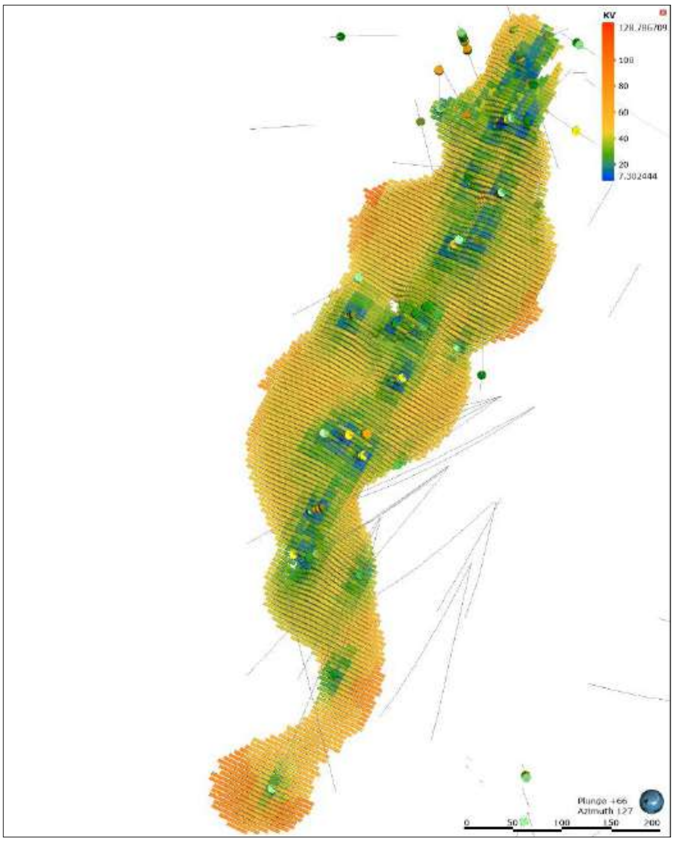


Figure 14-42: Kriging variance at Raja (lower the variance the better), view looking down on Raja wireframe steeply towards southeast (66/127).

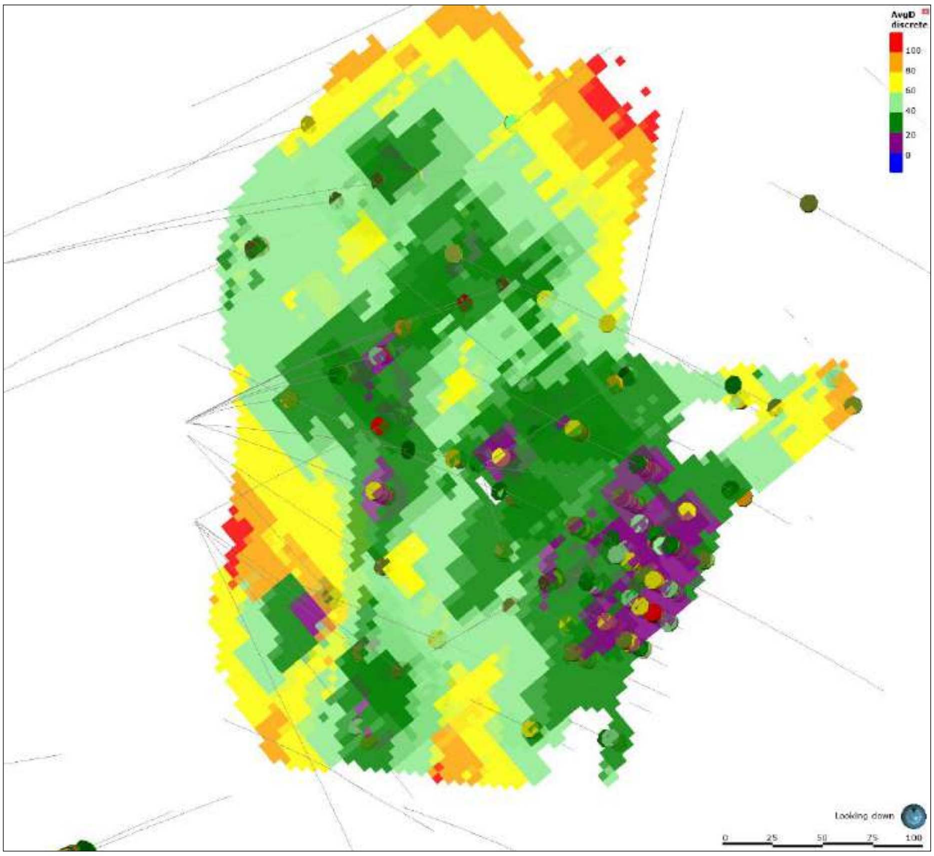


Figure 14-43: Drillhole spacing at map of blocks at Palokas, plan view

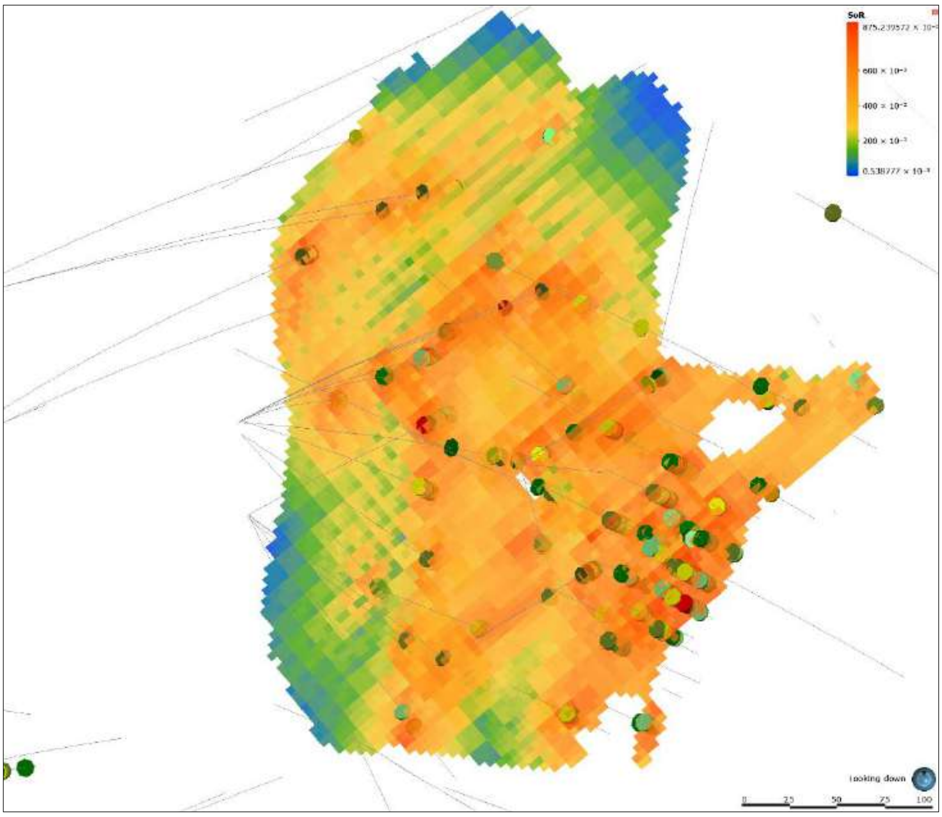


Figure 14-44: Slope of regression analysis of Palokas prospect block model, plan view

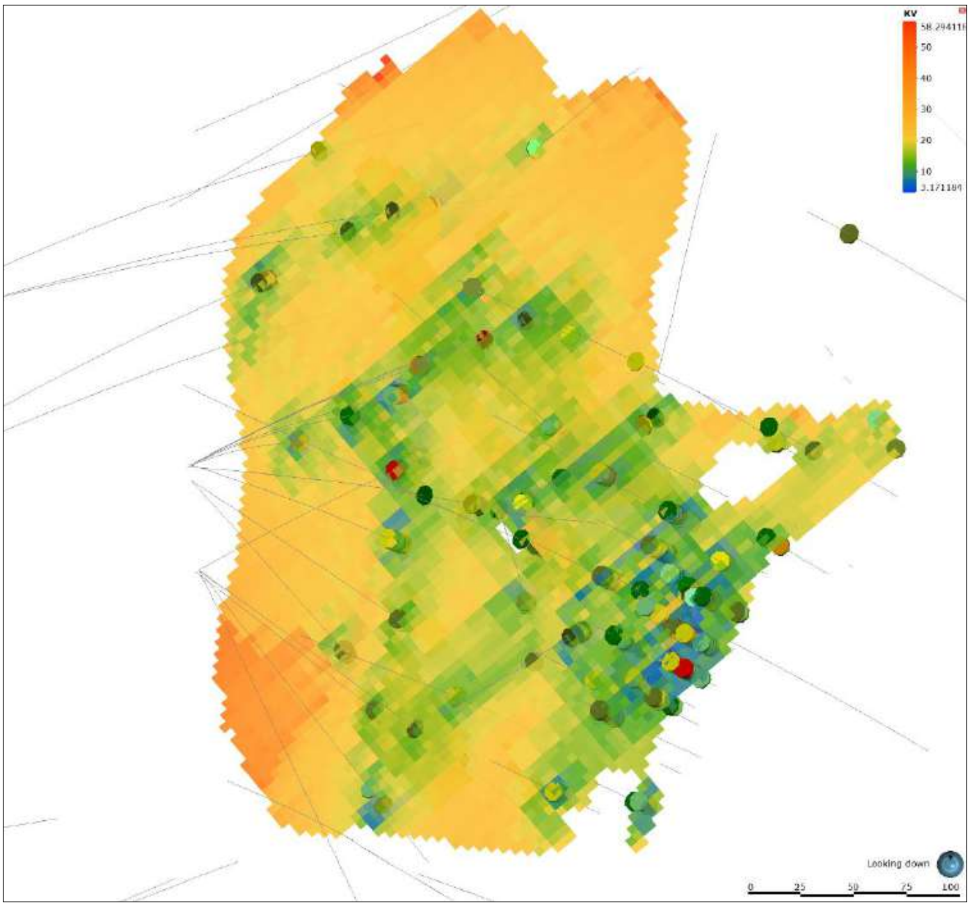


Figure 14-45: Kriging variance of Palokas block model at Palokas prospect, plan view

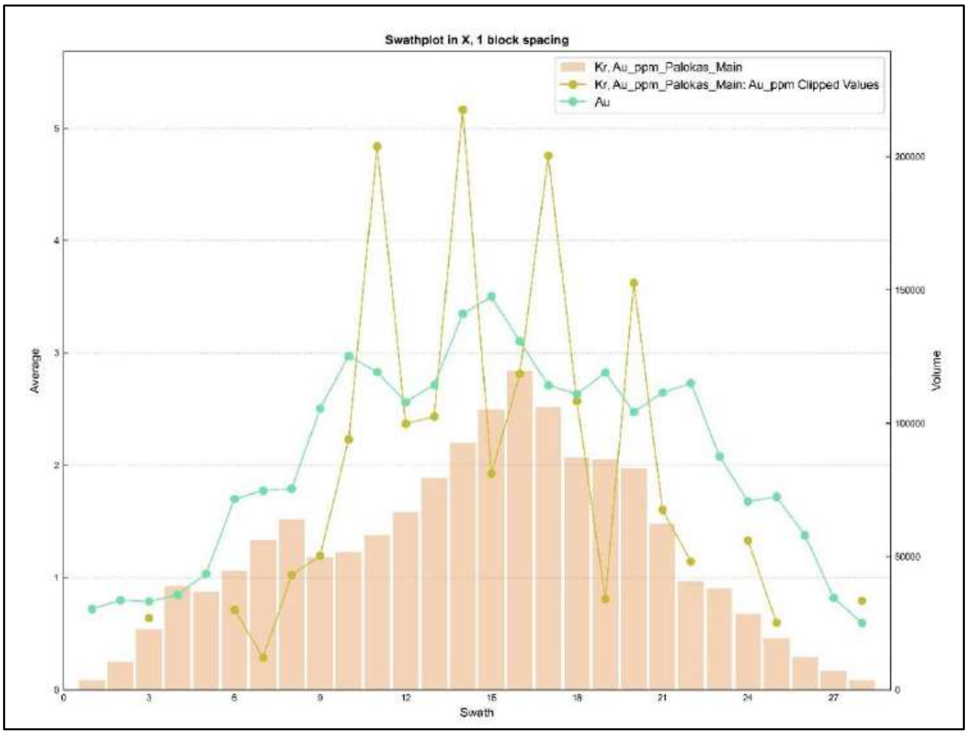


Figure 14-46: Palokas swath plot in X for the block model

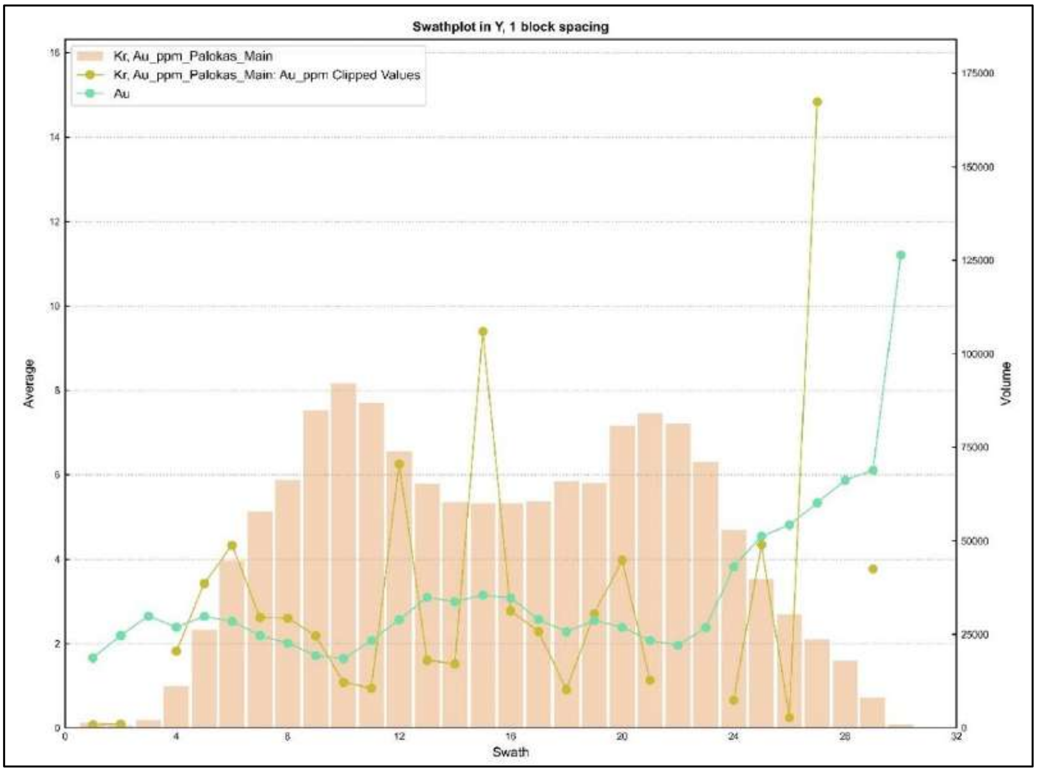


Figure 14-47: Palokas swath plot in Y for the block model

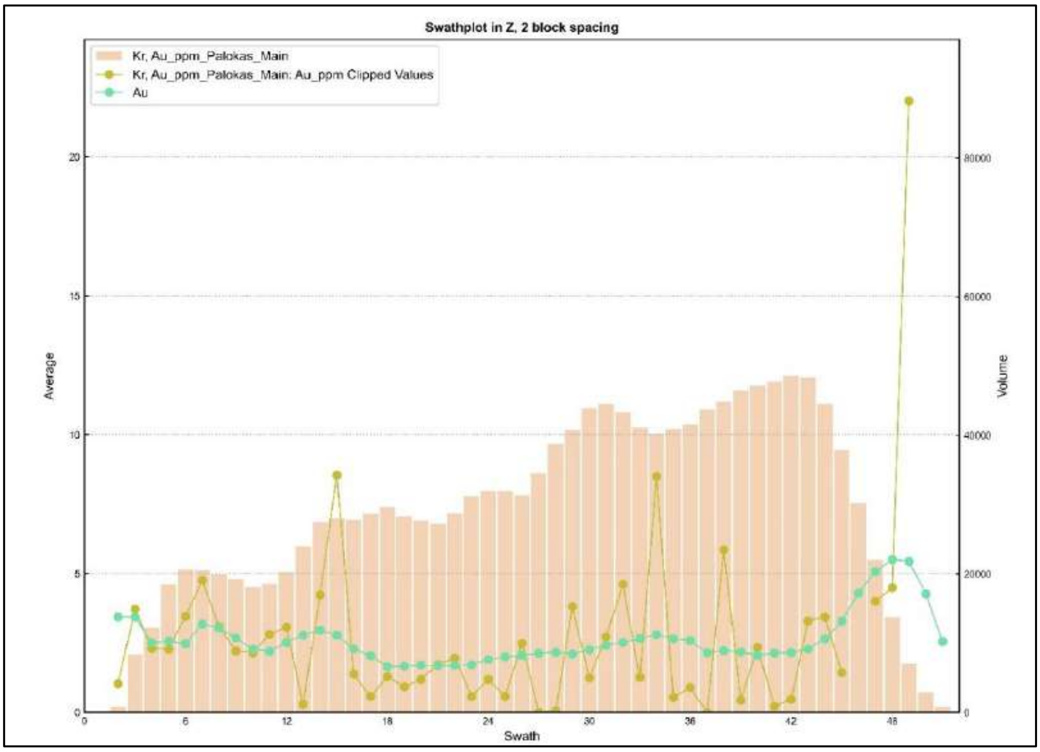


Figure 14-48: Palokas swath plot in Z for the block model

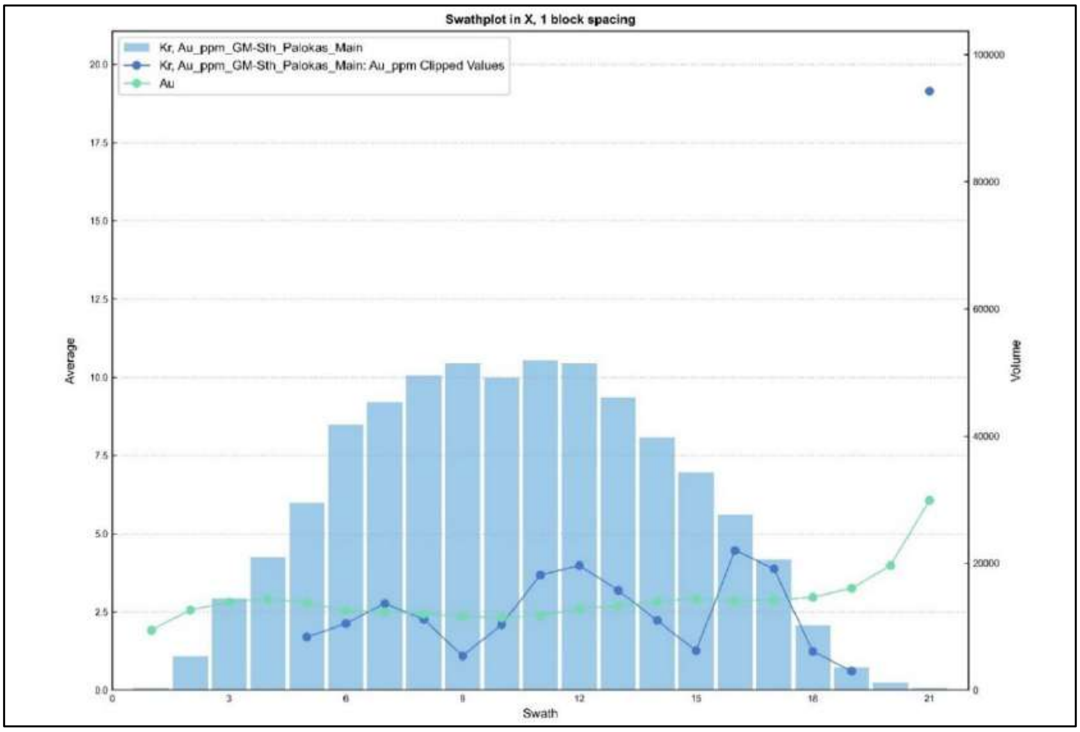


Figure 14-49: South Palokas swath plot in X for the block model (Palokas main gold wireframe)

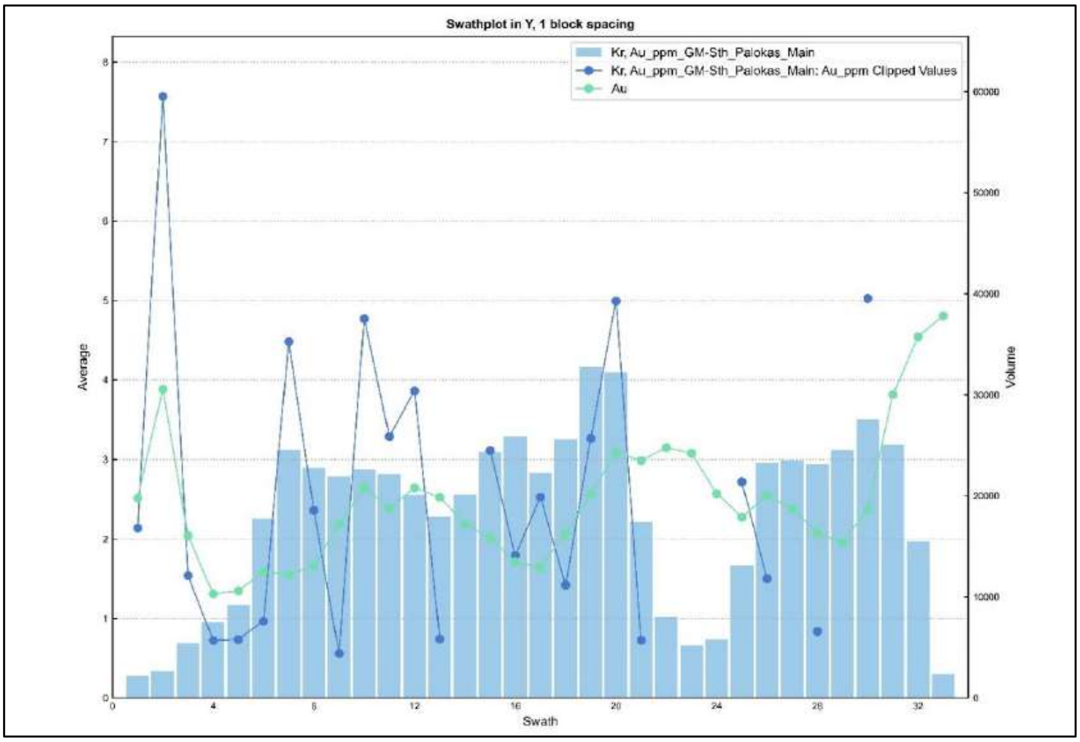


Figure 14-50: South Palokas swath plot in Y for the block model (Palokas main gold wireframe)

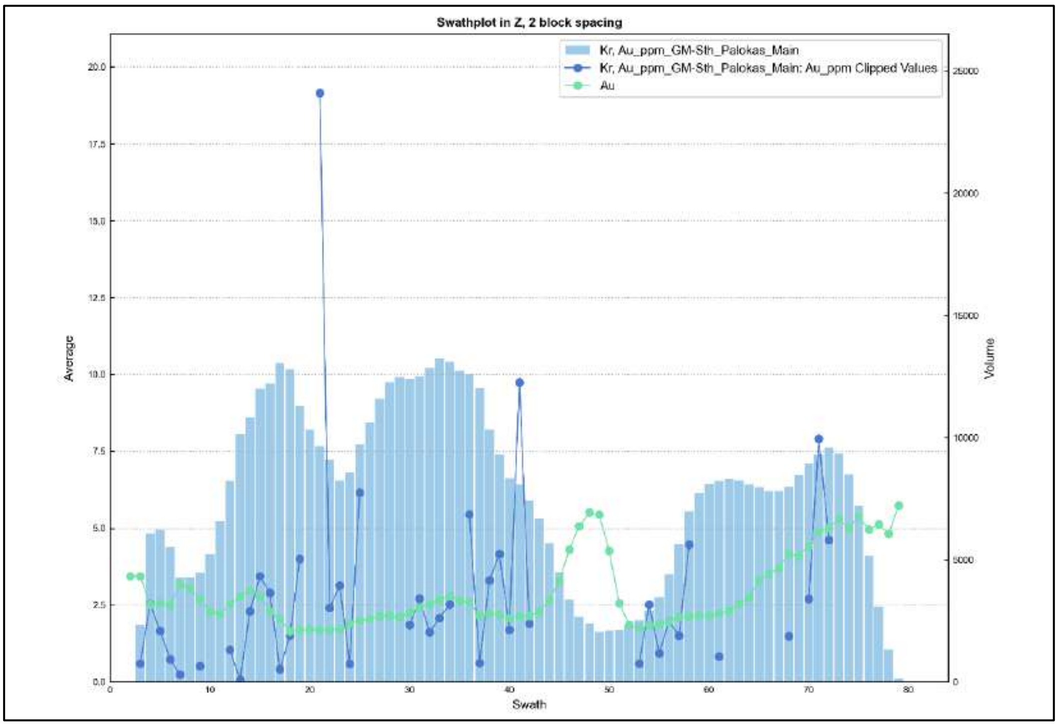


Figure 14-51: South Palokas swath plot in Z for the block model (Palokas main gold wireframe)

Note the higher average block grade region around swath 45 to 51 relates to high grades projecting across the gap in depth data owing to wide drill section widths. More drilling is required in this region.

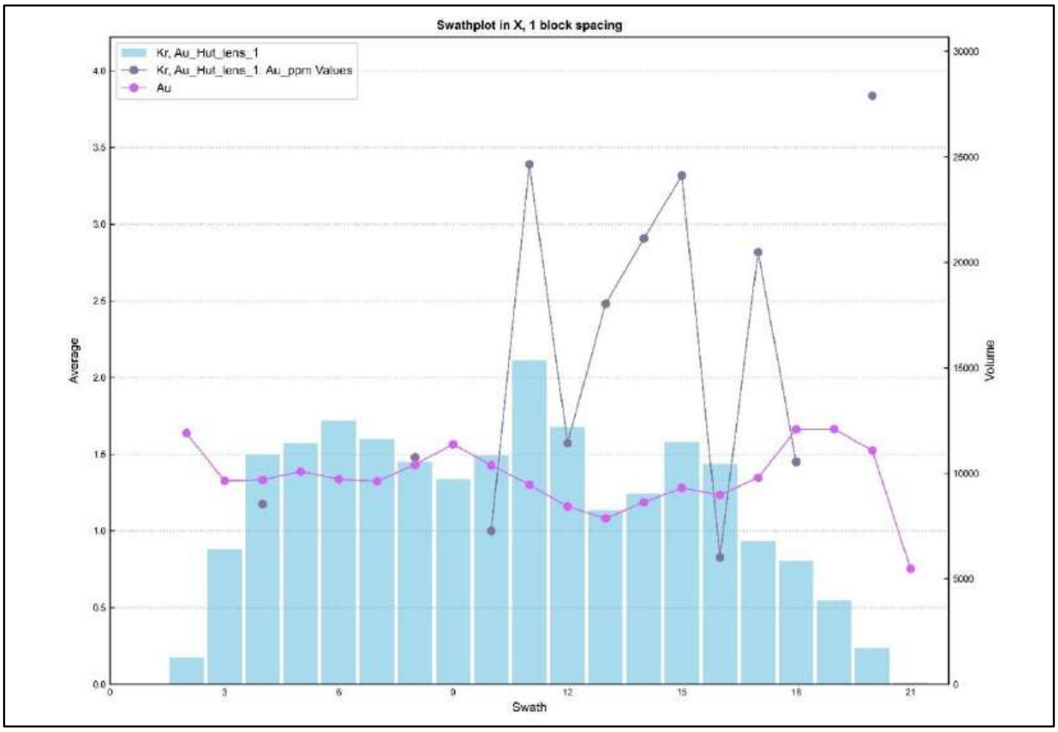


Figure 14-52: The Hut swath plot in X for the block model

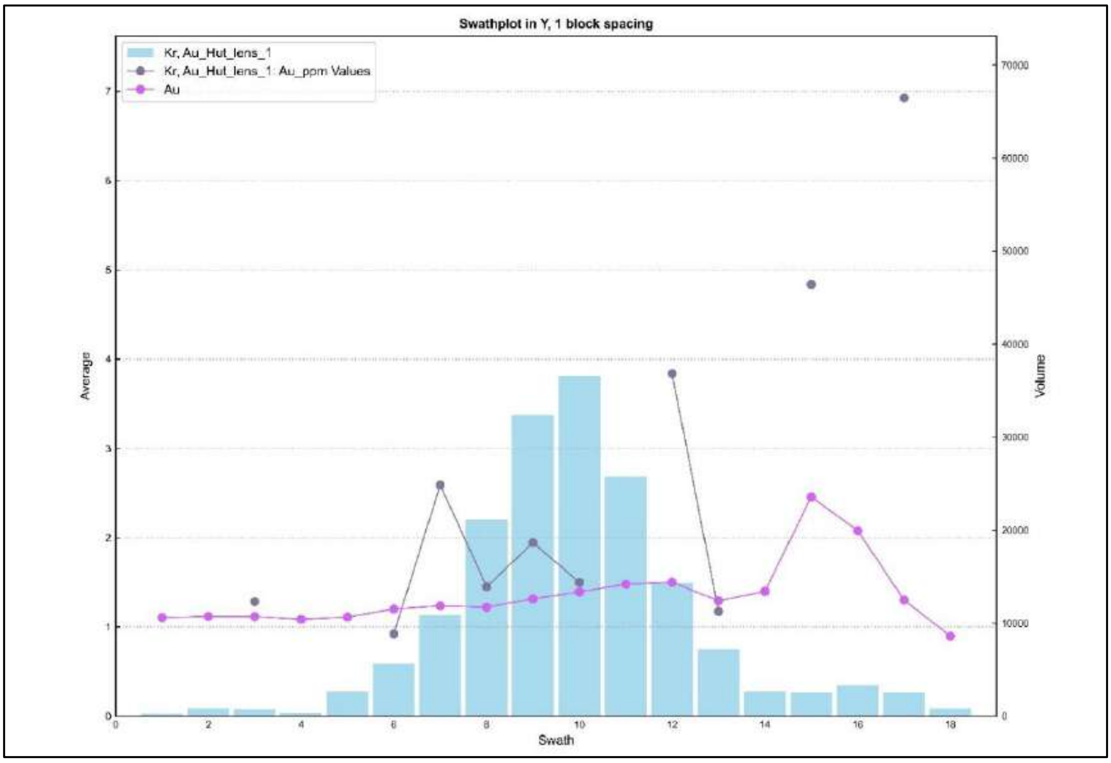


Figure 14-53: The Hut swath plot in Y for the block model

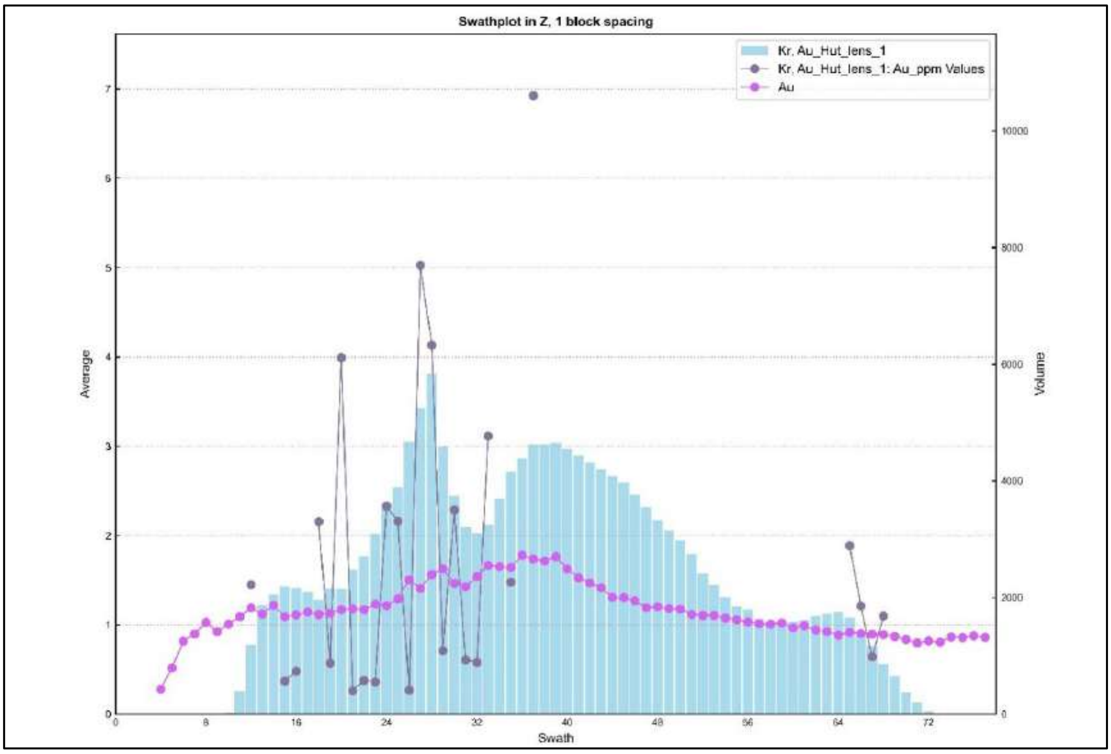


Figure 14-54: The Hut swath plot in Z for the block model

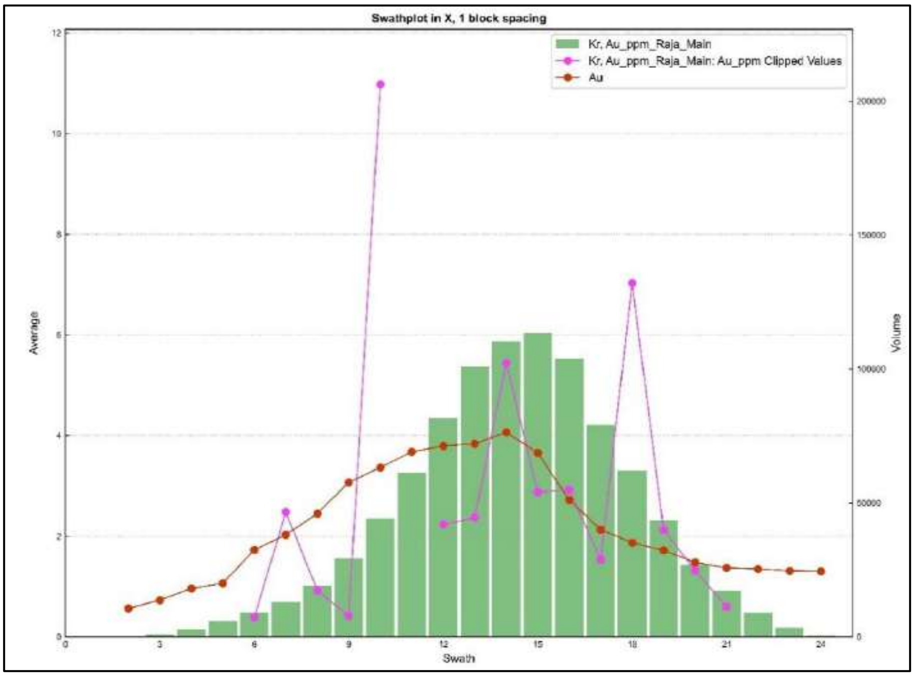


Figure 14-55: Raja swath plot in X for the block model

Note that the central zone correlation matches with the down-plunge direction, such that the core of the high-grade zone corresponds to swaths 13 to 17 in this view. More holes drilled on sections will tighten the edge of this swath plot on the X axis.

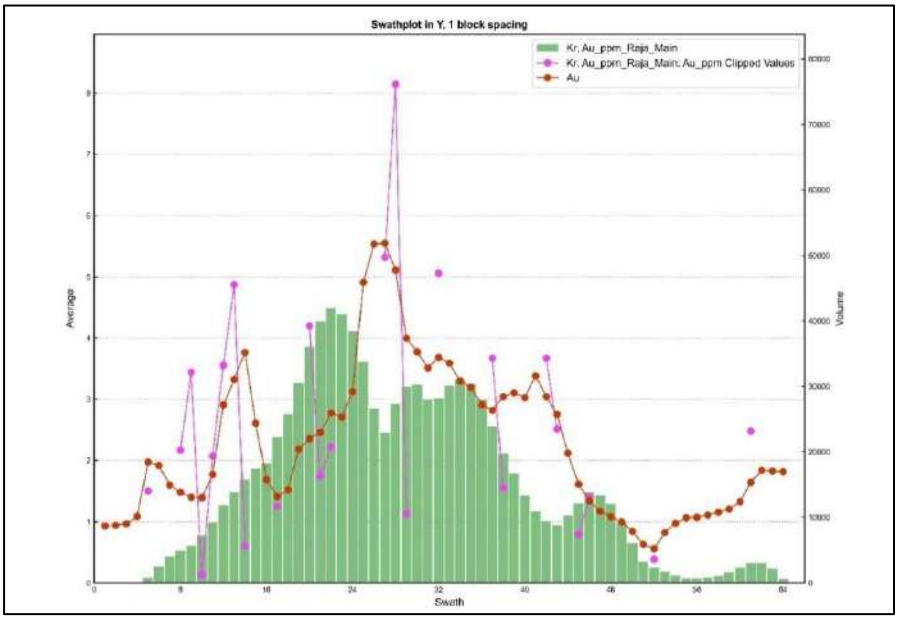


Figure 14-56: Raja swath plot in Y for the block model

Note that the correlation is moderately good for the swath slices proceeding down plunge. The gaps in data represent the zones between drill sections. New drill sections will significantly improve the swath plots on the Y axis, removing the gaps in clipped values, improving the correlation (and thereby improving the slope of regression and kriging variance data (shown in Figure 14-41 and Figure 14-42 respectively).

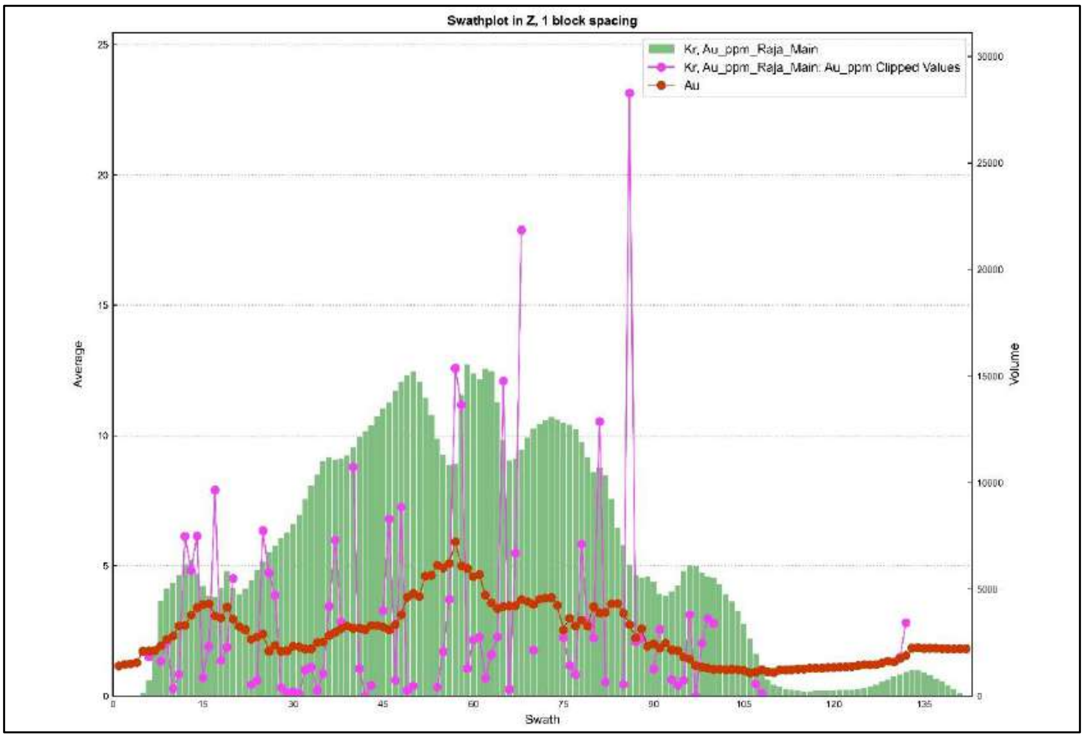


Figure 14-57: Raja swath plot in Z for the block model

Note there is a strong correlation of the clipped values and the block values is evident with good support. The gap of continuous data around swath 70 should be investigated to improve grade continuity at Raja.

The support evident in the data is appropriate for an Inferred Mineral Resource estimation. Palokas, South Palokas and Raja, which comprise approximately 88% of the contained AuEq ounces in the estimated Inferred Mineral Resource, will likely require an intermediate drill section to be considered as Indicated. Consideration may need to be given to drilling slightly closer on section to better determine grade continuity; however, early analysis shows that lateral constraints on the margins of wireframes may increase average grade.

Thus, 20 m drill spacing on section and 30 m between sections may give the support required for some resources to be classified at Indicated status.

14.11 Grade Tonnage Data

Grade tonnage graphs were created using the AuEq[1] block estimations in Leapfrog for each of the opencut and underground models for the OC-UG optimised resource reported in the Previous MRE. Grade tonnage data tables for each prospect were also created with ranges of 0.3 to 4.5 g/t AuEq and 1.1 to 4.5 g/t AuEq for opencut and underground models, respectively.

Grade-tonnage relationships were calculated for the Rajapalot All-UG model using AuEq[1], shown in Figure 14-58.

Grade – tonnage

all Rajapalot for the "All Underground" model

All UG							
					Rounded metal contents		
Cutoff	Tonnes (kt)	Au (g/t)	Co (ppm)	AuEq (g/t)	Au(koz)	Co (t)	AuEq(koz)
1.1	**9,780**	**2.8**	**441**	**3.2**	**867.6**	**4313**	**1014.1**
1.3	8,725	3.0	457	3.5	839.4	3984	970.6
1.5	7,967	3.2	460	3.7	816.5	3665	935.7
1.7	7,147	3.4	461	3.9	786.3	3298	892.8
1.9	6,460	3.7	474	4.1	758.7	3063	842.8
2.1	5,765	3.8	496	4.4	706.3	2862	810.0
2.3	5,209	4.0	511	4.6	675.9	2664	769.3
2.5	4,761	4.3	525	4.7	655.0	2499	725.8
3.0	3,749	4.8	552	5.3	575.4	2068	641.9
3.5	3,095	5.2	575	5.8	518.6	1781	573.4
4.0	2,595	5.6	594	6.1	463.2	1541	509.6
4.5	2,162	5.9	628	6.6	411.0	1357	456.0

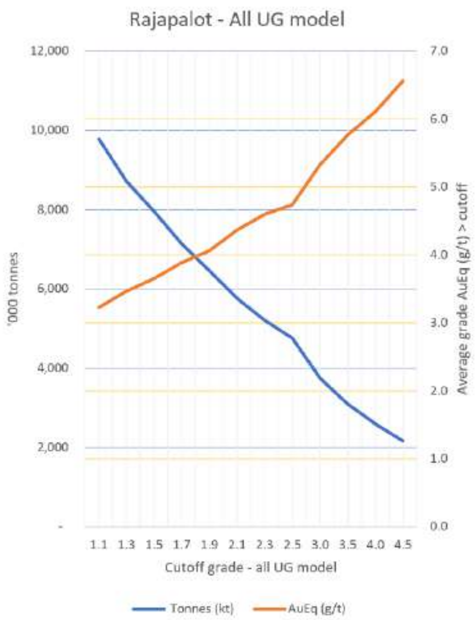


Figure 14-58: Combined table and graphic data for All-UG model at Rajapalot

15 MINERAL RESERVE ESTIMATES

There are no Mineral Reserve estimates for the Rajapalot property.

16 MINING METHODS

16.1 Introduction

The Rajapalot gold and cobalt project comprises five orebodies (Palokas, Raja, Joki, The Hut and Rumaj) within an area of approximately 3 km from west to east and 2 km from south to north, which commence from outcrops to 100 m below the surface, to a maximum depth of around 600 m.

The PEA mine plan considers a greenfield underground operation targeting a run-of-mine (RoM) production rate of 1.2 Mtpa through combined mining of three deposits at any one time to meet the target annual production. Each of the near surface deposits are planned to be individually accessed through decline box cuts with truck haulage to the RoM stockpile located at the process facility. RoM material is assessed against an economic cut-off for cobalt extraction, to be separately stockpiled, and campaign processed. All feed will be processed for gold recovery but only a proportion, on a feed campaign basis, for cobalt recovery.

The primary mining method selected for the Project is retreat longhole open stoping (LHOS) with 20 m level spacing and applied to the Palokas, Raja, The Hut and Rumaj deposits. Paste backfill is used to maximize mining extraction and reduce the tailings storage requirements on surface. The mining method selected for the Joki deposit is overhand Cut and Fill (C&F) due to its shallower dip angle with Cemented Rock Fill (CRF).

The PEA mine plan assumes an owner-operator approach, as is typical in Finland, with mine equipment purchased through a lease-to-own strategy on typical industry terms.

16.2 Mining Methods

16.2.1 Overview

The mining approach at Rajapalot considered underground and open cut methods determined by the assessment of:

- Geotechnically stable stope spans throughout the deposit and ground support requirements.
- Variable orebody width, dip and strike length.
- Impact of mining recovery and dilution.
- Practical level intervals to achieve a balance of access development for mining activities (drilling, blasting, ventilation, excavation and backfill).
- Working area requirements to achieve sustainable schedule targets (production rate and grades).

The Open cut mining assessment was estimated at only 1 year of combined mining potential at Palokas and Raja (less than 1 Mt) with an additional 10 Mt of waste to be stored on surface at the end of the mine life. The project economics were similar when comparing a combined OC-UG approach to an UG only approach. Following this assessment, the base case for the PEA was planned for UG mining only.

16.2.2 Longhole Open Stoping (LHOS)

The mining method selected for the Palokas, Raja, The Hut and Rumaj deposit, is overhand LHOS which requires ore development along or across the orebody strike (longitudinal or transverse respectively) for a level spacing of 20 m. Level waste development is required to provide a means of access from the decline access (typically in the footwall) for mining mineralized areas identified as economically mineable.

Using the LHOS approach, the orebody is developed to the strike extent on each working level and following this, each stope along strike is mined and paste filled on retreat. In general, stope production is overhand LHOS, working on top of or next to filled stopes and between 20 m high sill pillars (spaced every 7 levels). Stopes are mined in the transverse direction to deposit strike in wider sections of the deposit (see Figure 16-1), and as longitudinal stopes in narrow sections (Figure 16-2). Sill pillars are mined on retreat.

Due to the variability of narrow and bulk stopes along strike a common access drive profile of 4.5 mW x 4.5 mH was applied which is sufficient in dimensions for the range of equipment types required for the narrow and wide stope mining. The drive profile reduces the maximum vertical drill height to 15.5 m for all stoping areas.

Slot raises are initially drilled followed by rings of blast holes which can be either up or down holes (depending on access). Slot raises are blasted initially followed by rings into the void created by the slot.

The blasted stope ore can be removed on the lower level by tele-remote methods or limited to the stope brow if the loader is being operated manually. Once the individual stopes have been excavated to their open stope limits, a barricade is installed at the stope access and the stope void is filled with paste backfill and/or unconsolidated waste, depending on the location and sequence of mining.

After the paste backfill has cured sufficiently, then adjacent stopes, along strike or above/below the previously mined stope, can be mined using a similar sequence. The LHOS approach can be applied as overhand (bottom-up), working on top of fill or underhand (top-down) working under sufficiently strong, consolidated fill.

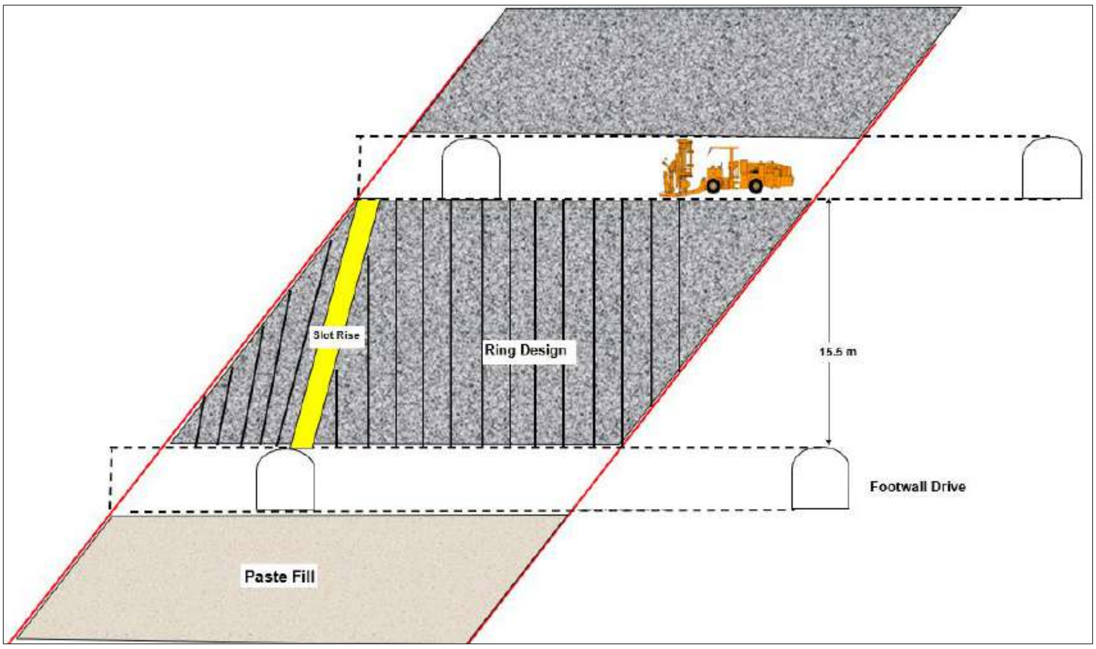


Figure 16-1: Schematic cross-section of Transverse LHOS method

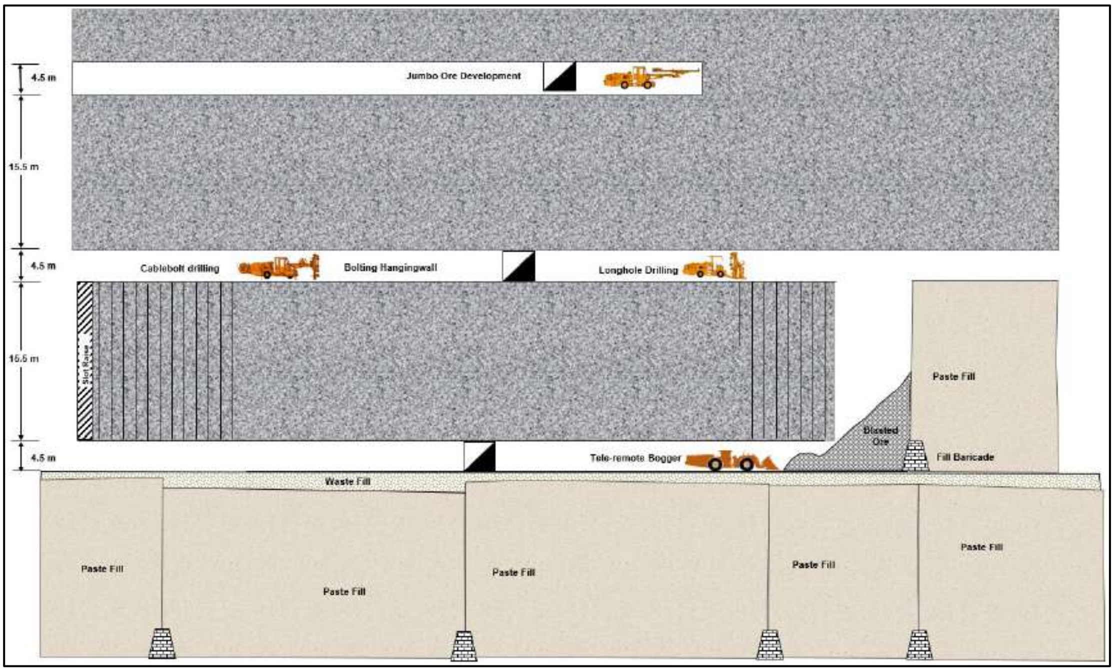


Figure 16-2: Schematic long view of Longitudinal LHOS method

16.2.3 Cut and Fill (C&F)

The C&F mining method was selected for the Joki deposit which commences once the decline reaches the footwall (FW) drive or Level access elevation of the orebody, usually midway along its strike length (see representative sketch in Figure 16-3). The stope sequence begins with the lowest 4.0 m high lift and each subsequent lift requires the back of the level access to be slashed down to reach the next lift. There are six lifts between levels for a total rise of 24 m from each access.

Multi-pass C&F is generally for orebody widths greater than 10 m from FW to hanging wall (HW). The mining begins by driving the Level access to the FW contact of Lift 1 (Figure 16-4). Then the drift is extended to the HW contact of the orebody. The Primary drift will be mined (note: generally, all multi-pass C&F drives are planned to be 4.0 mH x 4.0 mW) on the FW contact to the extremities of the orebody. The drift will then be filled mainly with CRF. After the fill has cured, the Secondary drift will be developed parallel to the Primary drift, with fill on one side and ore on the other side of the drift. After filling and curing is complete for the Secondary, a Tertiary drift is driven beside the Secondary drift, followed by filling and curing. This process is repeated until the HW is reached, which is the Quaternary drift. Once the fill has cured for the Quaternary drift, the Level Access will be slashed down to reach the next lift and the process will be repeated for the remaining lifts of multi-pass C&F.

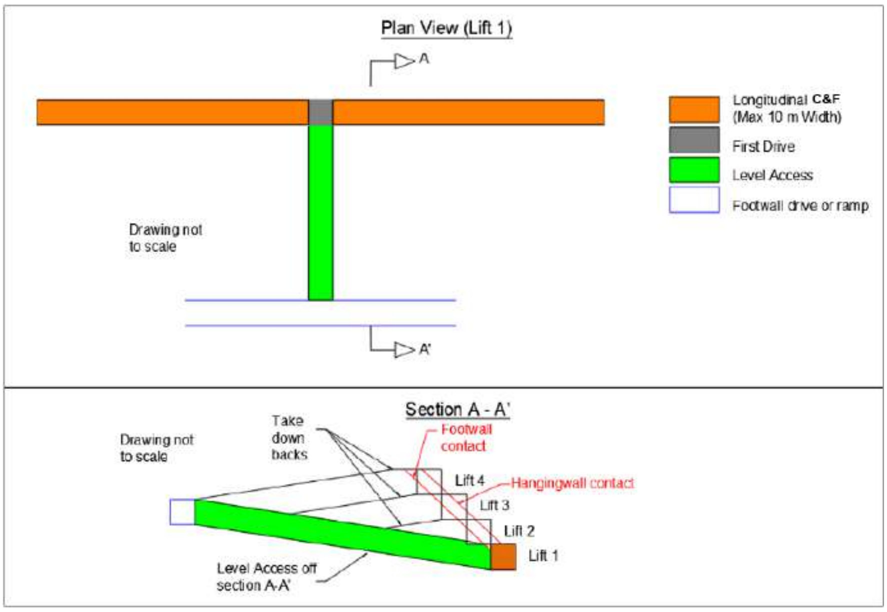


Figure 16-3: Plan and section view of Longitudinal C&F (Source: Eldorado, 2019)

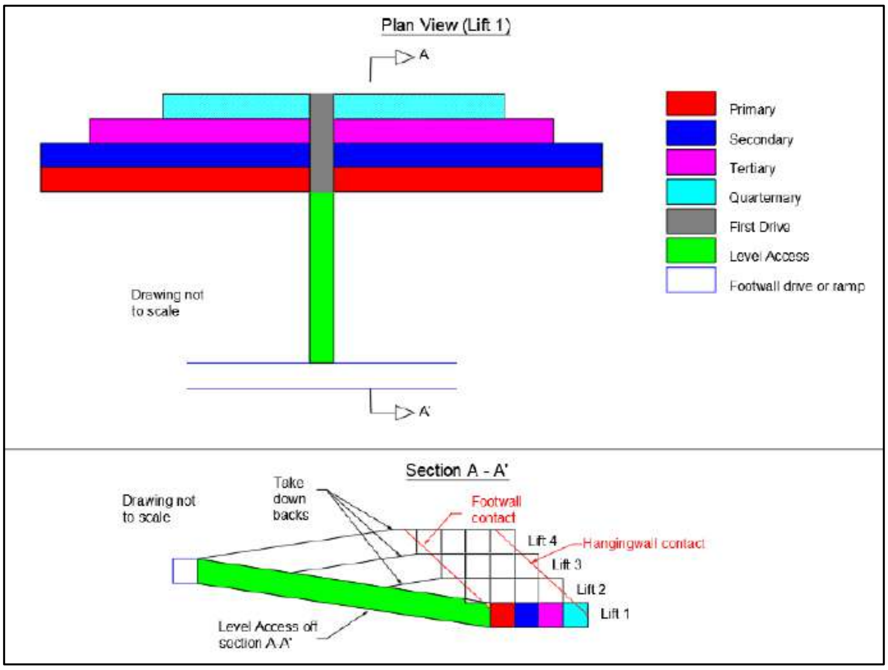


Figure 16-4: Plan and section view of Multi-Pass C&F (Source: Eldorado, 2019)

16.3 Preliminary Geotechnical Assessment

For the PEA level assessment, the geotechnical appraisal is based on limited core inspection and supplemented by the viewing of core photos. Geological models were produced by Mawson and provided to SRK. No new characterisation or geotechnical modelling was completed during this study by SRK.

16.3.1 Deposit Geology

Detailed geological descriptions of the deposit areas are provided in Sections 7 and 8. A typical rock type reference board at the Mawson site is shown in Figure 16-5. The main lithological domains proximal to the mineralization and underground mine access referred to for the geotechnical appraisal are summarised as:

- Palokas has the access located in the Albitic-calcsilicate rock types. This also hosts the mineralization and therefore will be the rock type bounding the stope volume. The outer hanging wall is remote from stoping and in the Muscovite-quartzite (see Figure 16-6).
- The Raja deposit mineralization is within the hanging wall Albitic-calcsilicates and an immediate stope footwall of Biotite Schist. The mine access in in a footwall of Muscovite-quartzite. Parallel lenses of ore are between 10 to 50 m spaced (see Figure 16-7).
- The Hut mineralization and footwall access is also within the Albitic-calcsilicate rock types. The distant hanging wall from the stopes is Mafic volcanics (see Figure 16-8).
- Ruma mineralization is spread across Mafic volcanics and Metasediments. The footwall access is within the same rock types as well. There are several parallel lenses of mineralization up to 80 m spacing (see Figure 16-9).
- The Joki deposit is flatter dipping and within the variations of the calcsilicate rock types. Footwall access will be within calcsilicates and volcanics (see Figure 16-10).

Figure 16-5: Representative core of the main lithology types (Mawson Site)

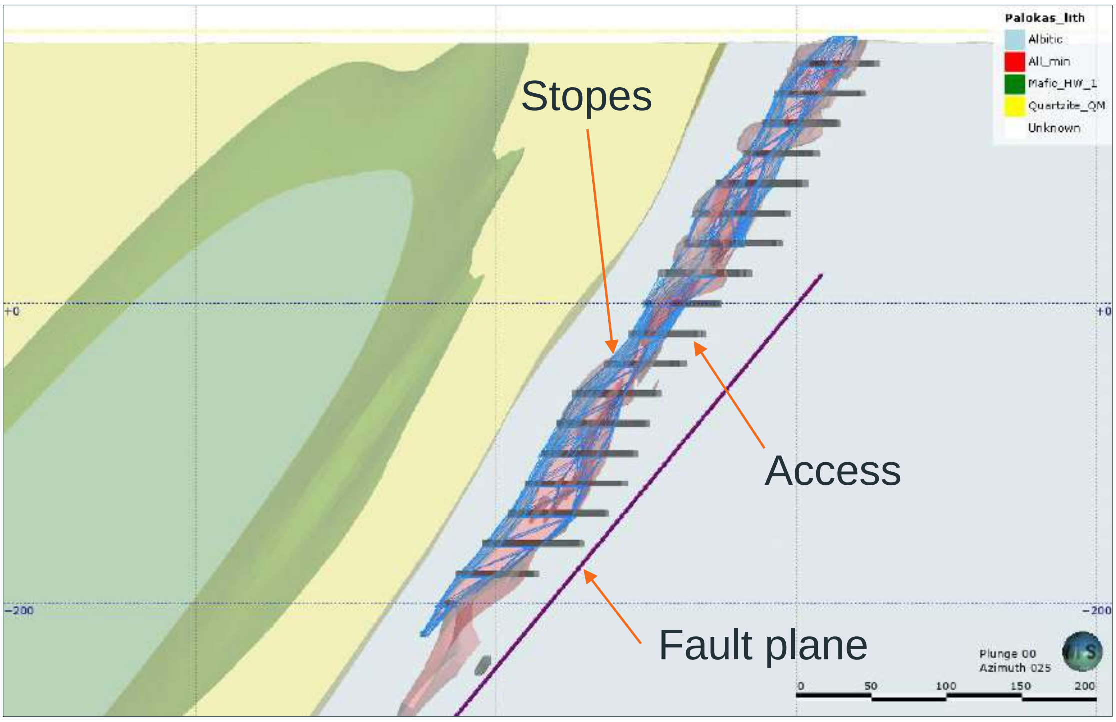


Figure 16-6: Typical section for the Palokas deposit showing major rock types, access and stoping layout

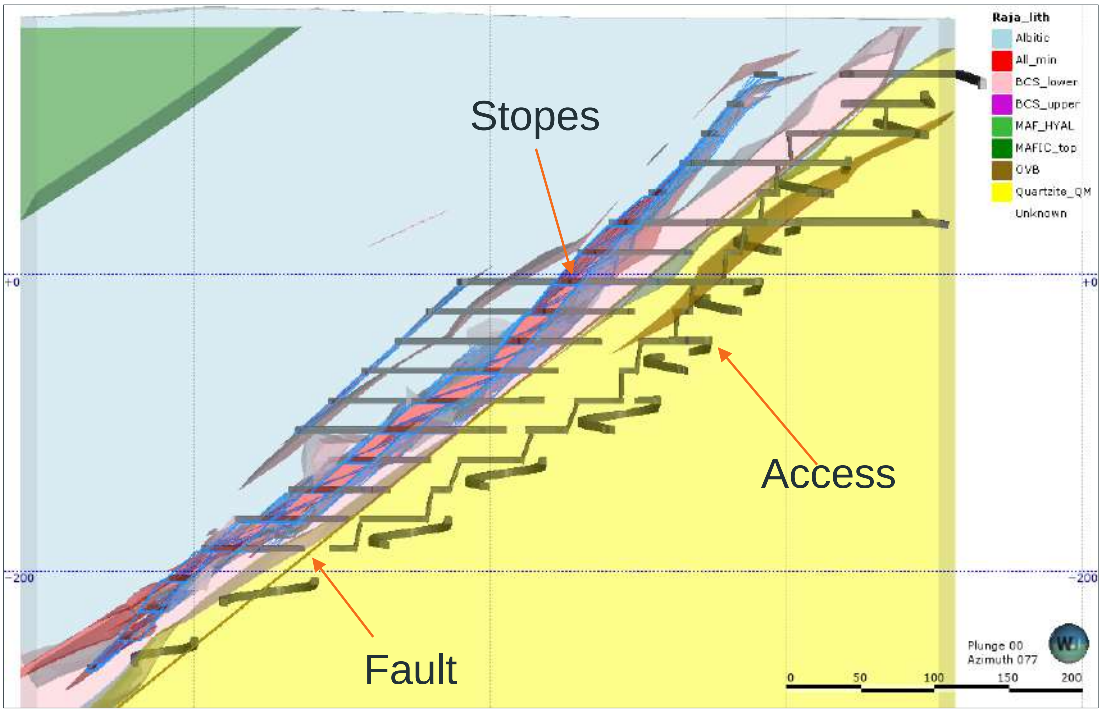


Figure 16-7: Typical section for the Raja deposit showing major rock types, access and stoping layout

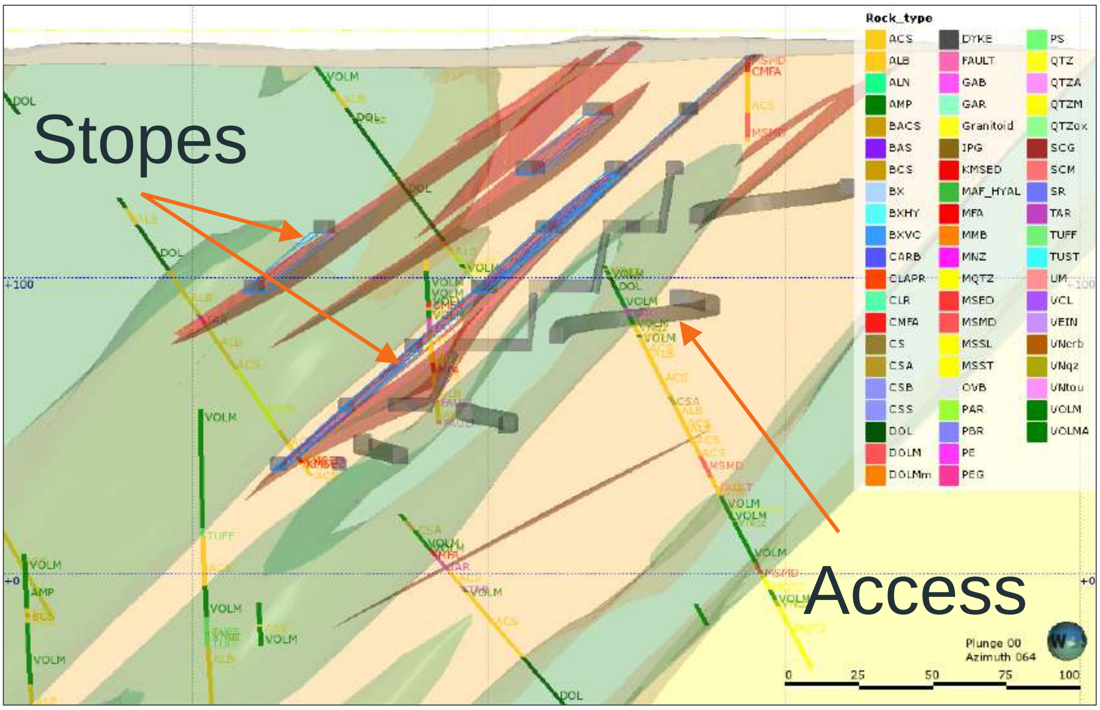


Figure 16-8: Typical section for The Hut deposit showing major rock types, access and stoping layout

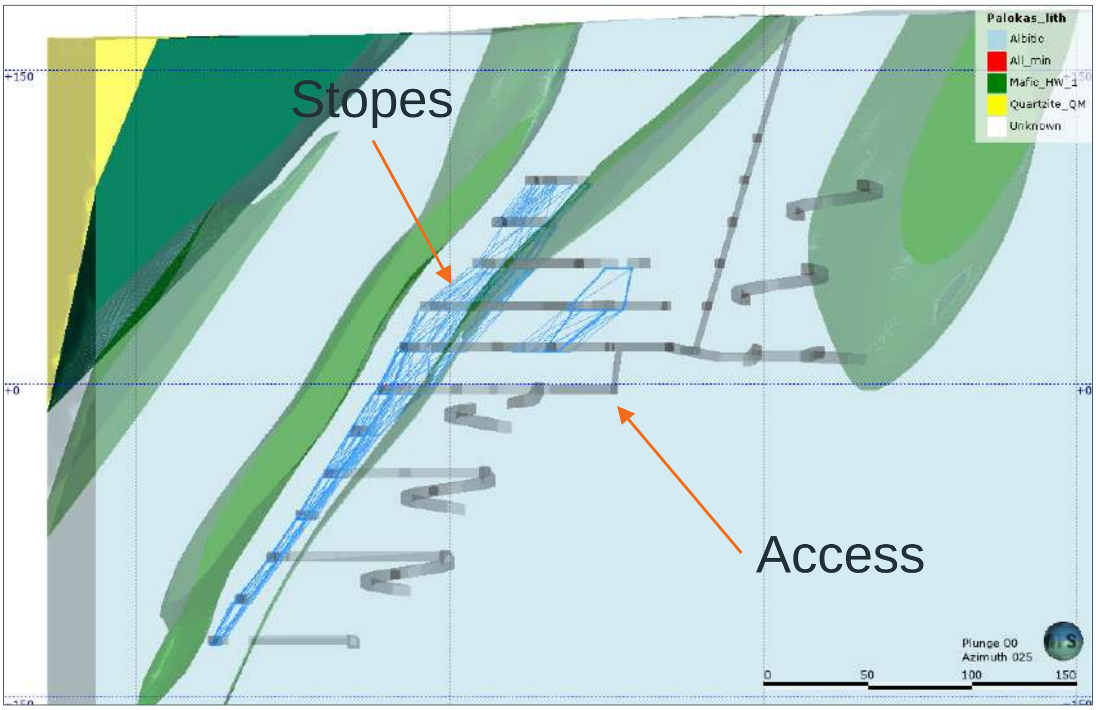


Figure 16-9: Typical section for the Rumaj deposit showing major rock types, access and stoping layout

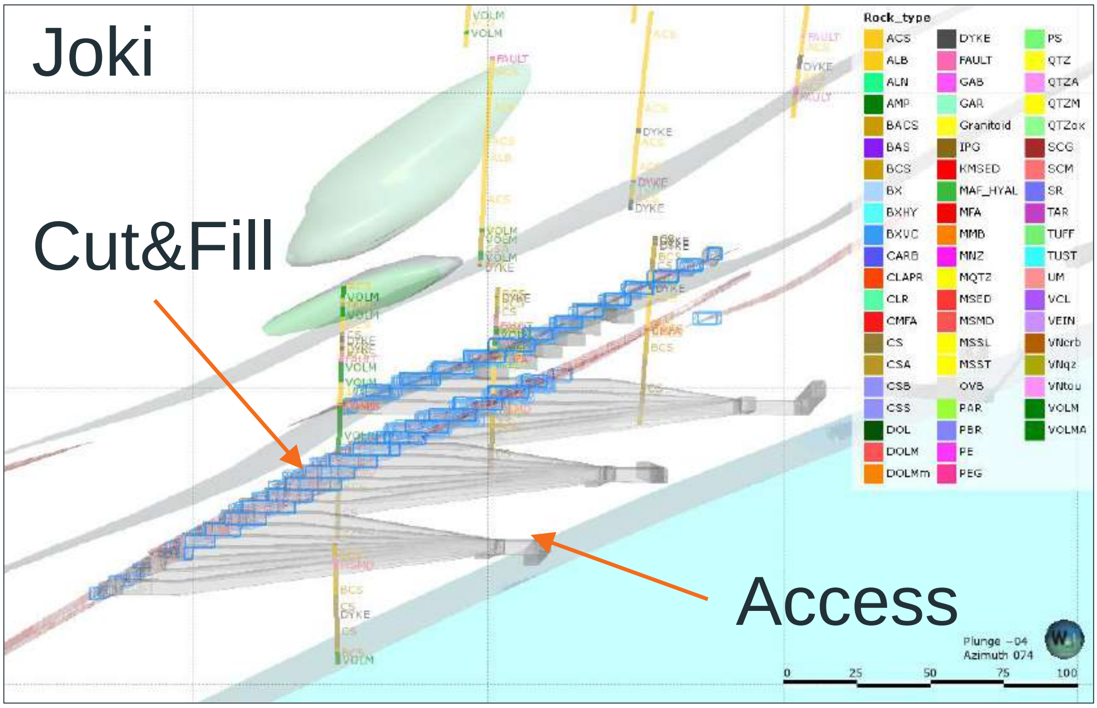


Figure 16-10: Typical section for Joki deposit showing major rock types, access and stoping layout

16.3.2 Rock Mass Characteristics

In general, all deposits have 'Fair' to 'Good' rock ratings from observation. The rock strength is estimated to be >150 MPa and likely >200 MPa in some areas. This includes the ore zone which is silicified Albitic calcilicate or mafic volcanic rock types. Therefore, footwall access development and stoping will be within high strength rock mass and rare zones of weaker rock strength influencing stability.

Jointing has been viewed from limited field inspection of core and mainly from core photographs. Typically, by observation there are 2 to 3 joint sets with one of these being variably healed foliation fabric parallel to mineralization that is open in some of the rock types. The spacing ranges from 0.5 to 2 m and is rarely below 10 cm spaced fractured zones; which represents a mildly blocky rock mass expected.

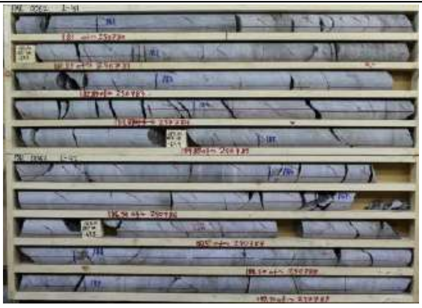
	Palokas	Rajapalot
Footwall Albitic Domain		
Ore- Foliated		
Ore - Silicified		
Hanging wall - Albitic		

Figure 16-11: Typical example core intercepts through mining domains

There are specific intercepts of weaker ore contact zones (shearing or alteration) on the FW or HW that will adversely influence stope stability and dilution; however, interrogation of the lithology logging has indicated “Fault” intercepts of typically <50 cm which has logging descriptions of high fracturing and also core loss associated with chlorite and biotite alteration. These are seen on the footwall side of the Palokas and also Raja deposits which will be intersected by footwall access. Visual inspection and Leapfrog show alignment of these zones which indicates a persistent feature. SRK has created highly simplified planes from these features (Figure 16-12).

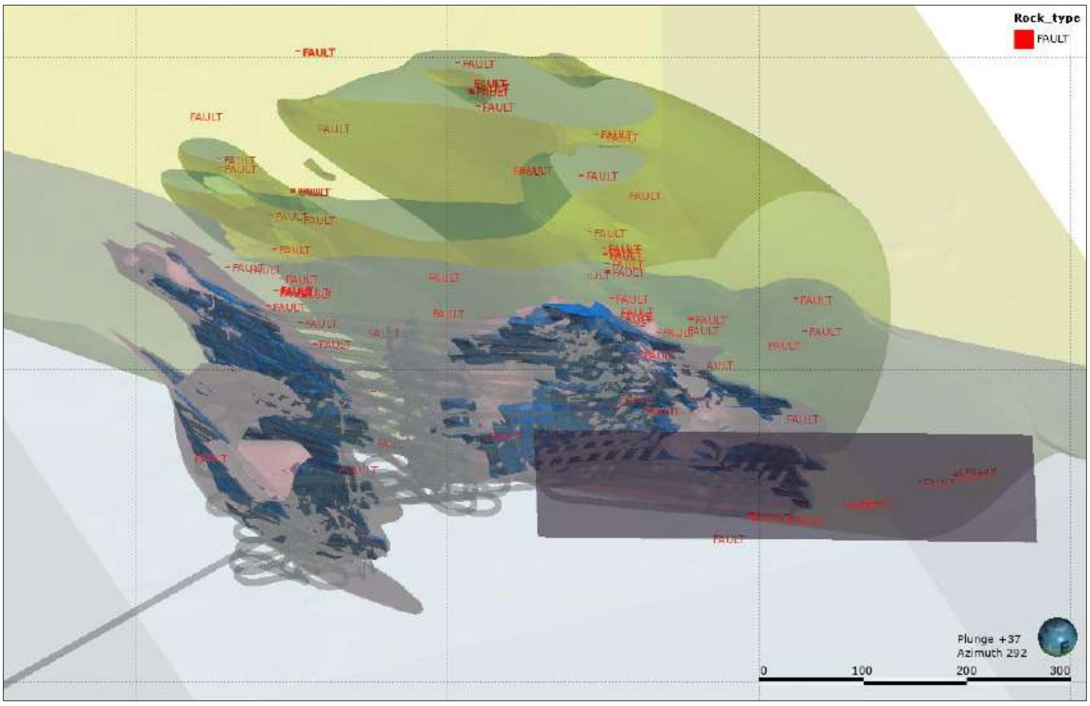


Figure 16-12: Logged "Fault" intercepts aligning in plane within the Palokas Footwall

No specific geotechnical logging is completed yet for any of the deposits. The data collection is limited to logging core for RQD logging as well as fracture count per metre (Breaks per metre in logging). Point load test (PLT) is collected routinely in logging as an estimate of intact rock strength. Structural measurements are made mainly for geological data collection with some of this data useful for the geotechnical characterisation, but not yet collected for this purpose.

The method of logging is fixed to the RQD and Fracture Frequency (FF) collected in 1 m intervals irrespective of the lithology, strength or any other geotechnical domain boundaries. This is considered simplified and not cognisant of the changes in fracturing or strength determined by geological differences. In particular, the 1 m interval does not consider highly fractured zones (although rare) in the fixed metre interval, which can bias the data and miss a significant weak structure or fracture zone relevant to underground excavation design.

As the distributions of logging metres in Figure 16-13 shows there is dominantly high RQD (70 to 100) and low FF (2 to 5/m) in the majority of the logging and this does not vary greatly by the major rock types or in the mining domains (FW/ORE/HW). This representation does not separately show the intercepts of high fracturing as these are not abundant; however, several fault zones and highly foliated zones are captured by low RQD and the fracture count (>10/m).

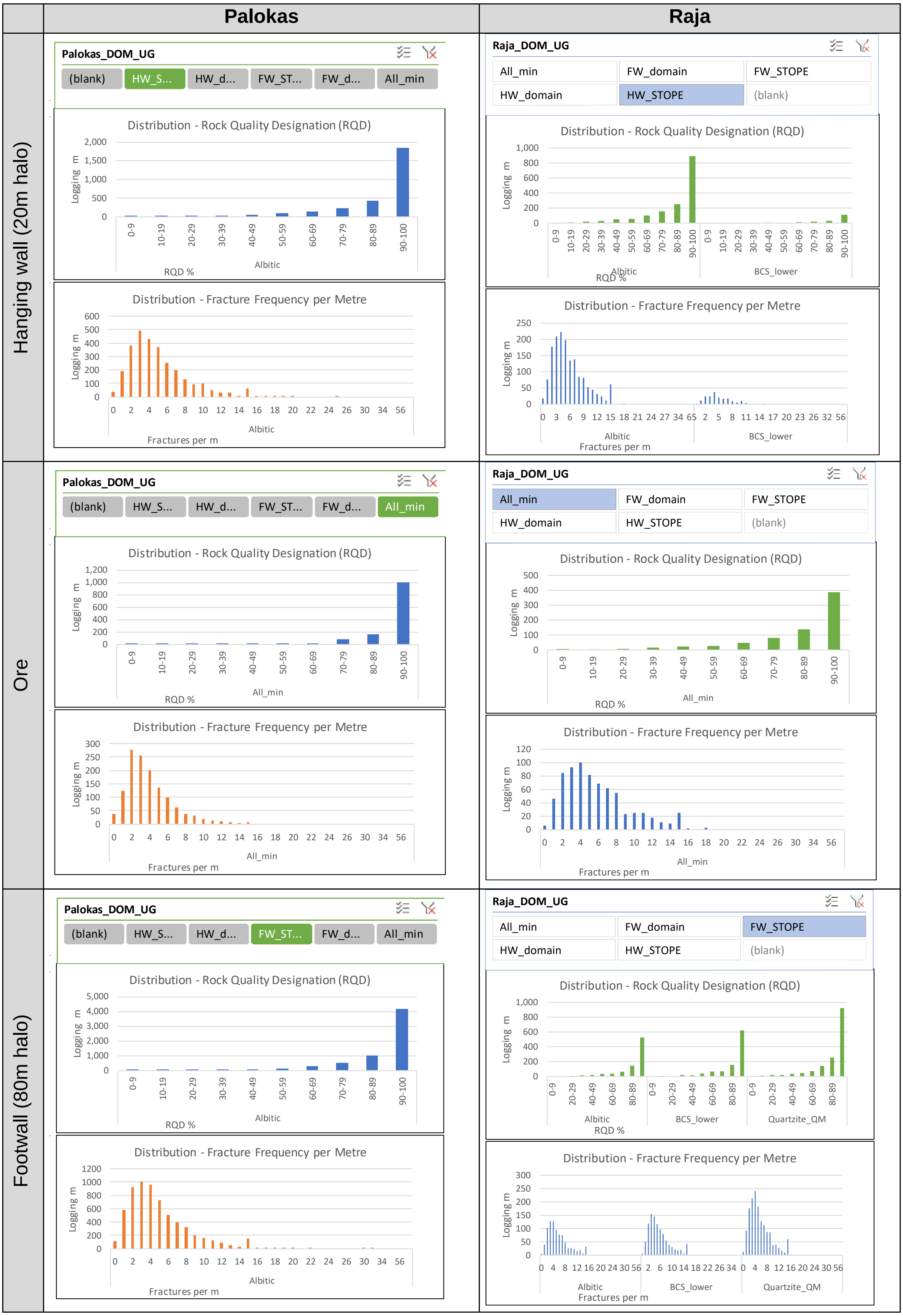


Figure 16-13: Logged RQD and FF distributions for Palokas and Raja, split by domain

16.3.3 Point Load Testing

Extensive PLT data are collected but there are no laboratory samples yet tested to establish rock type specific conversion factors. Therefore, only an assumed conversion factor of 15 is applied to the entire data set to estimate the Uniaxial Compressive Strength (UCS). This is a conservative estimate and overly generic for weaker rock types and also the very high strength rock types. The distribution by all rock types and the dominant footwall rock types are shown in Figure 16-14.

With an assumed average conversion factor of 15 the distribution of all rock types indicates a median strength of 120 MPa. The footwall Muscovite-Quartzites for Palokas and Raja are generally 120 MPa and also have a population >250 MPa.

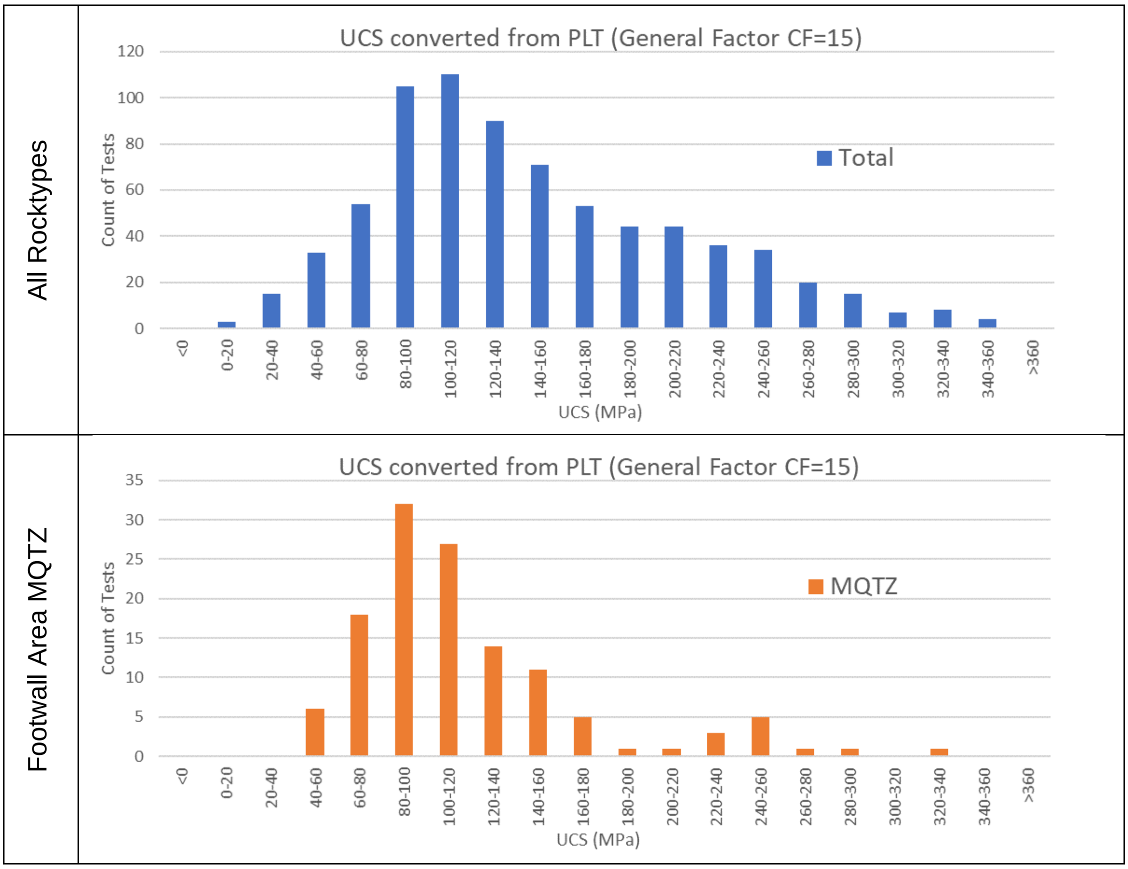


Figure 16-14: PLT converted to UCS strength by general conversion factor; all rock types (upper), dominant footwall rock types (lower)

16.3.4 Rock Quality Comments

Geotechnical logging of parameters and to the intervals that display changes in the geotechnical quality is required (i.e., not by fixing to regular 1 m intervals). For underground design the general rule of a maximum of 3 m logging intervals is usual for geotechnical logging. Significant fracturing, and core loss must be separated as a unique interval for classification. This is required to calculate the ratings in rock classification systems and a gap that has to be filled before more advanced levels of study.

Any stability analysis or empirical analysis for stoping, access, and ground support design requires calculated geotechnical ratings in the industry accepted systems. Without this, broad assumptions are made from limited inspection and further study stages require greater confidence in auditable raw data

16.3.5 Stress Regime

The deposits are considered shallow to intermediate depth and by comparison to other Nordic underground mines, the stress regime will likely be intermediate at the extremes of currently planned extraction. A generic stress regime of high horizontal stress to vertical ratios of 2:1 (known as the k ratio) is expected. In Finland, the maximum principal stress (σ1) orientation is most commonly horizontal and perpendicular to the deposit strike. This regime is assumed for all the Rajapalot deposits. Only with stress testing and downhole televiewer assessment, the stress magnitudes and direction can be updated from the assumptions.

16.3.6 Stope Stability and Stope Dimensioning

Stope extraction (LHOS) is for transverse and longitudinal orientations based on ore width with each having separate controls on stope face stability. C&F is proposed for the Joki deposit only and stability control is different to long hole stoping. Other than Joki, the deposits have a moderate to shallow dip angle which controls the hanging wall stope face stability and dilution potential in the longitudinal orientation.

The estimated Q rating is used with stress assumptions (described above) and the dominant ore parallel foliation as a controlling structure.

Stope Empirical Stability Assessment

An estimate of stable and unstable stope span is accomplished by use of the Modified Stability Chart Empirical Method through the use of the Modified Stability Number, N', as described in Hutchinson D.J. et al, 1996. N' is based initially on Q' (Q Prime) components of the Q system calculation which is:

$$Q' = \frac{RQD}{Jn} \times \frac{Jr}{Ja}$$

where:

$\frac{RQD}{Jn}$ is a measure of block size for a jointed rock mass

$\frac{Jr}{Ja}$ is a measure of joint surface strength and stiffness

The Modified Stability Number N' is calculated from:

$$N' = Q' \times A \times B \times C$$

Where:

A - is a measure of the ratio of intact rock strength to induced stress at the stope face.

B - is a measure of the relative orientation of dominant jointing with respect to the stope faces.

C - is a measure of the influence of gravity on the stability of the face being considered. Overhanging stope faces (backs) or structural weaknesses which are oriented unfavourably with respect to gravity sliding have a maximum detrimental influence on stability.

Figure 16-15 shows the charts for determining these factors (adapted from Hutchinson D.J. et al, 1996). The resulting empirical chart for the modified stability method relating N' to HR is adapted from Potvin, 1988 and Nickson 1992 documenting unsupported open stopes.

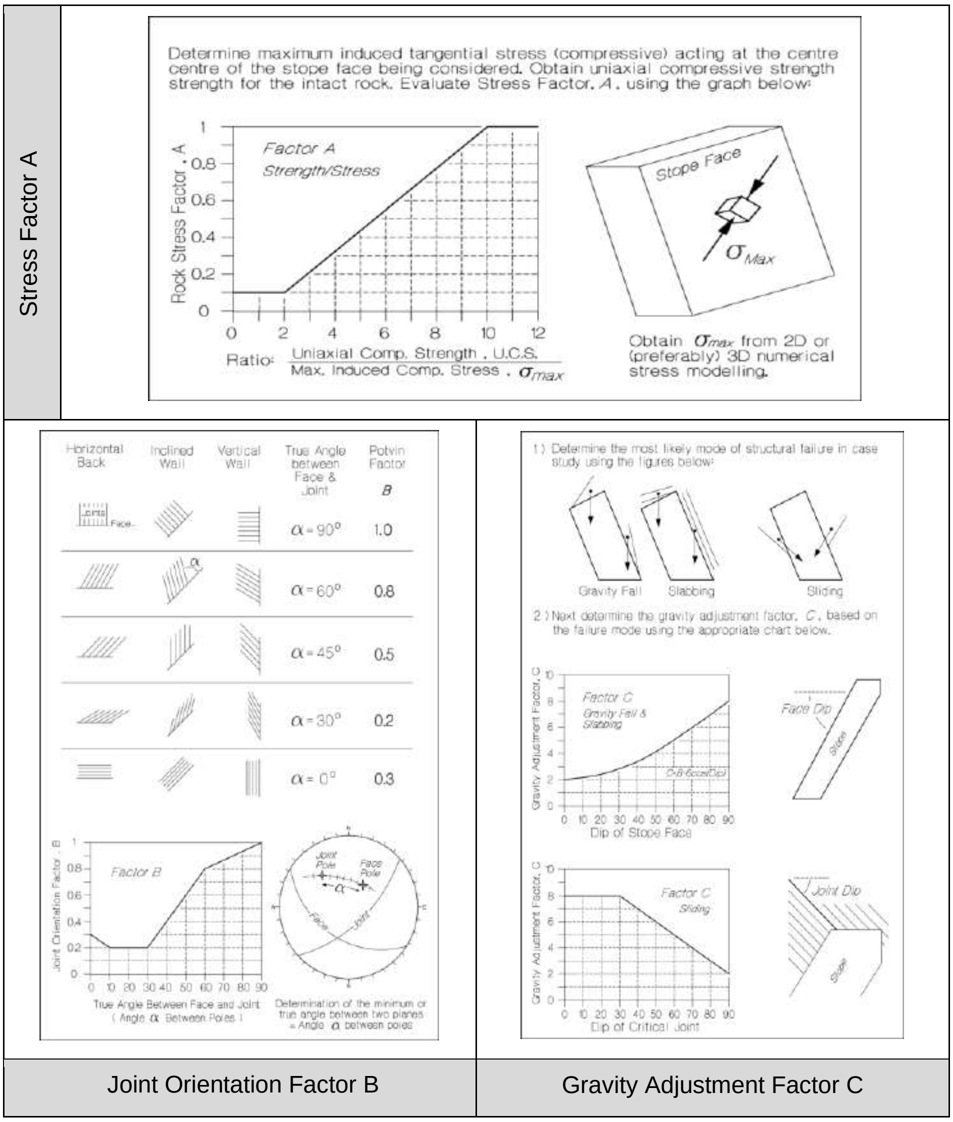


Figure 16-15: Stope Stability Chart adjustment factors to derive N'

Extraction sequencing for the LHOS with paste is working from bottom-up in 140 m high panels separated by sill pillars. The transverse areas are primary and secondary sequence with paste wall forming stope faces at secondary extraction. The longitudinal stopes will be mined in up against pastefill of the adjacent stope, negating the need for rock rib pillars. C&F at Joki is to be 4 to 10 m wide drifts with CRF the as working floor.

The preliminary summary of the maximum stope dimensions based on stable HR (estimated rock quality and jointing) is presented in Table 16-1.

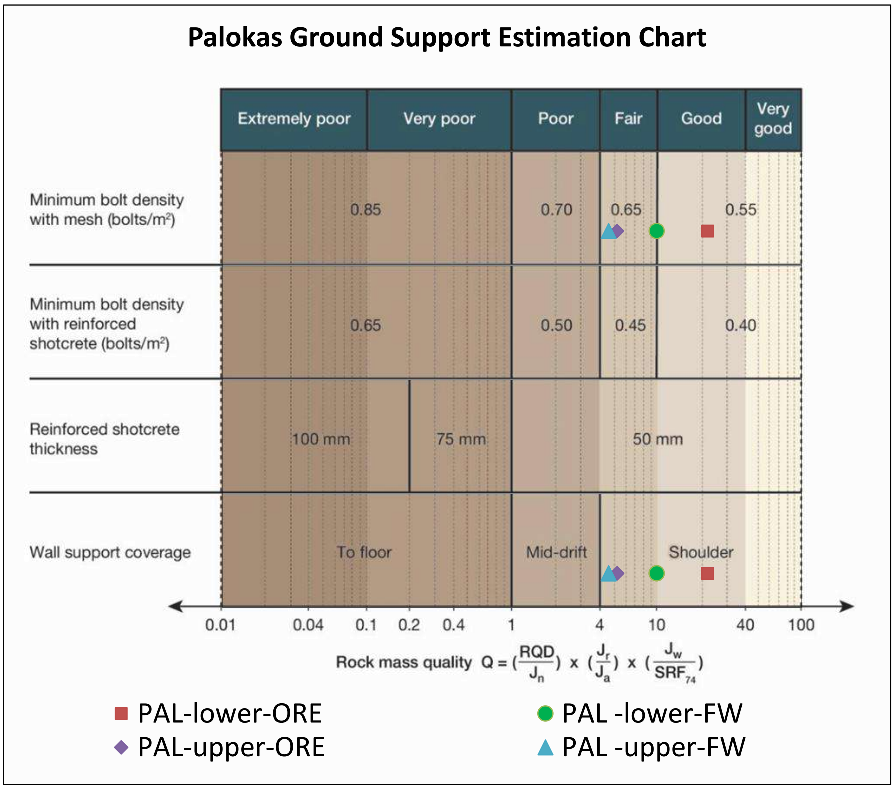
Table 16-1: Summary of Stope Stability chart unsupported HR and stable spans

Deposit	Estimated Rock Quality Q	N'	Unsupported HR	Fixed Stope Height (H,m)	Max Stable Stope Length (L,m))
Palokas Shallow	10 (Fair/Good)	10	5.7	20	30
Palokas Deep	10 (Fair/Good)	10	6.2	20	20
Raja Shallow	12 (Good)	7.7	5.2	20	30
Raja Deep	12 (Good)	5.5	4.6	20	20
Hut	8 (Fair)	9.1	5.5	20	30
Rumaj	10 (Fair/Good)	7.5	5.1	20	35
Joki	10 (Fair/Good)	8.4	5.4	10	35

16.3.7 Ground Support Estimation

Initial ground support estimations are based on the best estimate of rock mass conditions to be encountered by access and production development, as well as the expected demand of the ground control regime. The level of assessment for this study aimed at identifying the ground support requirements per excavation type and service life in order to provide input into the overall mining cost modelling process. The geotechnical information leading to this estimation is limited (from core photos) and assumptions have been made on similar mining and rock mass conditions, mainly in the Nordic region.

SRK has adopted an approach using an updated empirical system documented for ground support estimation from mining case histories. This system, published by Potvin Y. et al, 2016, utilises a refined approach based on the Q Support chart method last published by Barton in 2012 for tunnelling. The methodology is based on extensive review of 145 mines in Australia and Canada and is specifically suited to mining widths of 4 to 6 m.



Note: bolt numbers required expressed as units per square metre of coverage

Figure 16-16: Ground support estimation chart (Potvin Y. et al, 2016) for mesh and bolts (based on estimated rock quality)

The PEA study provides is a preliminary estimation for ground support quantity calculations used for cost estimation as provided in Table 16-2. These are based on international industry standards which may be chosen to be increased or decreased based on the Mawson approach to mining and meeting Finnish mining regulations. The next study phase will require geotechnical rock mass classification from logging and refined stress estimation to develop the inputs for ground support demand assessment and design.

Table 16-2: Ground support estimation for the planned development activity types at all the Rajapalot Deposits (normalised to quantity per linear metre)

Rock Quality Rating (Q)		Rare Conditions /Unexpected						Typical Range of Rock Quality							
		VERY POOR			POOR			FAIR			GOOD			V GOOD	
Activity Type	Profile	Bolts per linear m	Mesh m² per linear m	FRS Shotcrete m³ per linear m	Bolts per linear m	Mesh m² per linear m	FRS Shotcrete m³ per linear m	Bolts per linear m	Mesh m² per linear m	FRS Shotcrete m³ per linear m	Bolts per linear m	Mesh m² per linear m	FRS Shotcrete m³ per linear m	Bolts per linear m	Mesh m² per linear m
DECLINE	5.5mW x 5.5mH	11.7	15.8	1.36	8.3	12.8	0.69	5.7	8.8	0.44	4.8	8.8	0.44	4.8	8.8
WASTE DEV	5.0mW x 5.0mH	10.5	14.3	1.25	7.9	11.3	0.57	4.8	7.3	0.37	4.0	7.3	0.37	4.0	7.3
WASTE DEV	4.5mW x 4.5mH	8.5	12.0	1.1	6.3	9.0	0.5	3.3	5.0	0.2	2.8	5.0	0.2	2.8	5.0
ORE DRIVE/Xcuts	4.5mW x 4.5mH	8.5	12.0	1.08	6.3	9.0	0.45	3.3	5.0	0.20	2.8	5.0	0.20	2.8	5.0

The preference for the Rajapaolot deposits is for temporary and permanent reinforcement (2.4 m long bolts) and mesh. This will be mostly friction bolts (Split Sets, or Swellex) and permanent openings like the main access declines may require resin or grout anchored rebar rock bolts (probably limited to Raja and Palokas access due to the longer mine life). Localised (and limited) areas of poor ground conditions or large spans with high personnel exposure will require shotcrete for additional surface support.

Deep reinforcement is likely to be required at intersections formed between excavations. The wide spans created will expose wedges that are unstable and not anchored with primary reinforcement of 2.4 m long rock bolts. In particular, the intersections with the decline and the level access drives will form brows that may unconfine the rock mass from two sides. As well as this, the overall stress change from stope extraction will eventuate in the unconfinement of the rock mass allowing large blocks (unreinforced by rockbolt length) to mobilise into wide intersections.

A preliminary set of cable bolt designs are provided based on empirical approaches, engineering judgement of similar openings in similar rock mass conditions and stress regimes. This does not constitute “Design” at this stage of study as there are other variables that require a more detailed assessment to assess the instability potential including:

- likely structure orientations and conditions relevant to the location of intersections;
- the stress path likely to determine the loading that may clamp, or release wedges based on geometry; or
- the potential for the intersection to lose stress confinement and allow wedge release.

General cable bolt layouts for intersections between development profile sizes are shown in Figure 16-17.

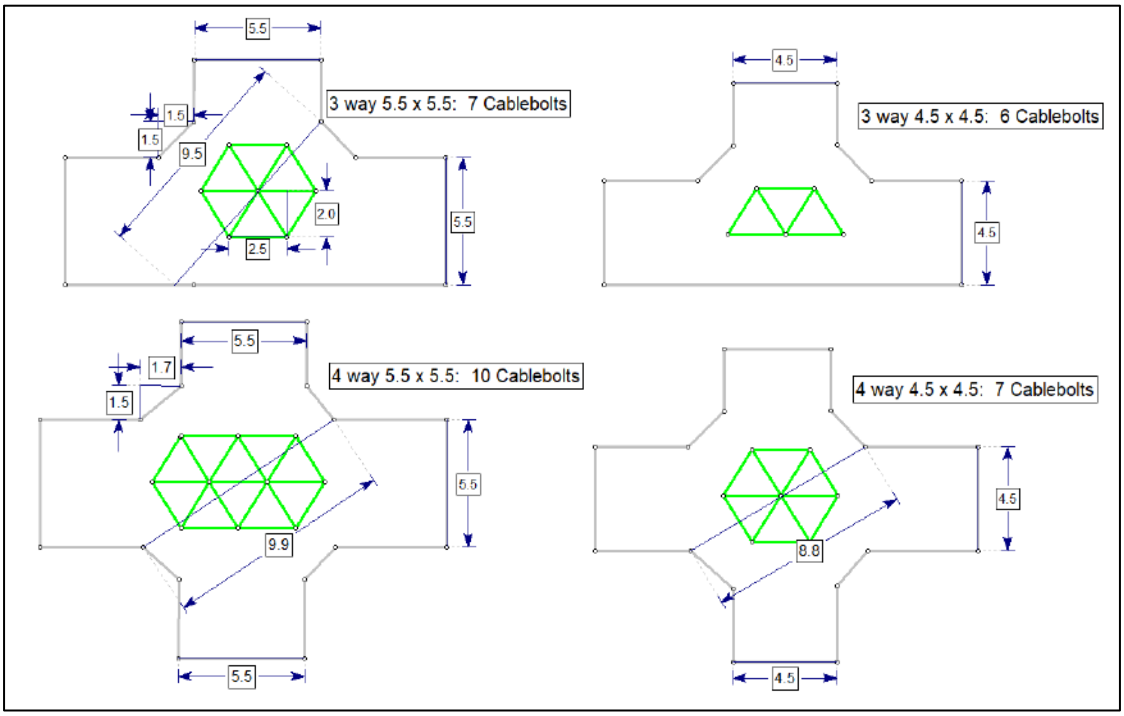


Figure 16-17: Cablebolt layout and numbers estimated for various intersection spans

16.4 Net Smelter Return and Cut-off

The initial Net Smelter Return (NSR) cut-off value (CoV) for the Rajapalot stope optimisation was selected at USD50/t using the preliminary cost and recovery parameters summarised in Table 16-3. The NSR values were estimated within each of the block models for gold and cobalt. The cobalt contribution to the NSR value is based on the ratio of sulphur to cobalt in a linear relationship to the At-Mine-Value (AMV) Factor and shown in Figure 16-18. There were no allowances or reductions in the NSR values for deleterious or penalty elements due to the preliminary stage of the Project.

In addition, RoM material is assessed against an NSR cut-off for cobalt extraction (USD2/t), to be separately stockpiled, and campaign processed. All feed will be processed for gold recovery but only a proportion, on a feed campaign basis, for cobalt recovery.

Table 16-3: Summary of NSR and CoV parameters

Item	Units	Value
Metal Prices		
Gold (Au) Price	USD/oz	1,700
	USD/g	54.7
Cobalt (Co) Price	USD/t	60,000
	USD/lb	27.2
Process Recovery Parameters		
Gold Recovery	%	95.0%
Co Recovery	%	80.0%
S recovery	%	85.0%
Product - Gold Doré		
Doré % Gold	% Au	75.0%
Shipment Escort	USD per kg doré	5.0
Shipment Fixed Cost	USD per shipment	2,000
Escort transport Cost	USD per shipment	1,500
Shipments per year	Number	50
Gold Payability	%	99.85%
Mineral Royalty Tax (MRT)	%	0.15%
Product - Cobalt Concentrate		
Co Concentrate Grade	%	0.706%
Moisture Content	%	10.0%
Payability	%	89.4%
Mineral Royalty Tax (MRT)	%	0.15%
Treatment Cost (TC)	EUR/dmt Conc.	100
Refining Charge (RC)	EUR/kg Co	0.20
Land Freight	EUR/wmt	50
Operating Cost Breakdown		
Underground Mining Cost	USD/t	35.0
Processing Cost	USD/t	12.3
G&A Cost	USD/t	5.0

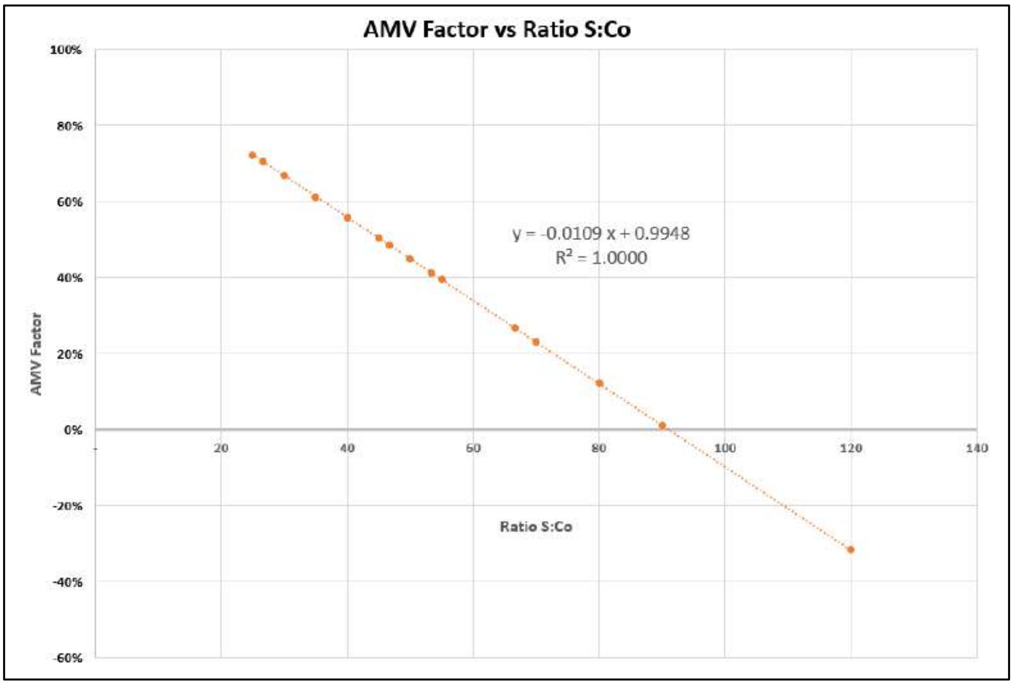


Figure 16-18: Ratio of sulphur to cobalt relationship to At-Mine-Value Factor (Source: Mawson)

16.5 Stope Optimisation and ROM Inventory

SRK used the Deswik Stope Optimiser (Deswik.SO) module to generate mineable shapes and quantify the diluted tonnes and grades available for the RoM inventory and schedule. For Palokas, Raja, The Hut and Rumaj, which are mined as LHOS, minimum stope shapes of 20 m height and 15 m length were considered. For Joki, which is mined as C&F, minimum stope shapes of 4 m height and 10 m length were considered as mining targets. A minimum mining width (MMW) of 3.0 m was applied for all mining shapes. The NSR and cut-off value used in the optimisation process are summarised in Section 16.4.

Figure 16-19 and Figure 16-20 provide respective plan and longitudinal views of the mining stopes (green) and designated sill (yellow) and crown (orange) pillars from the stope optimisation runs.

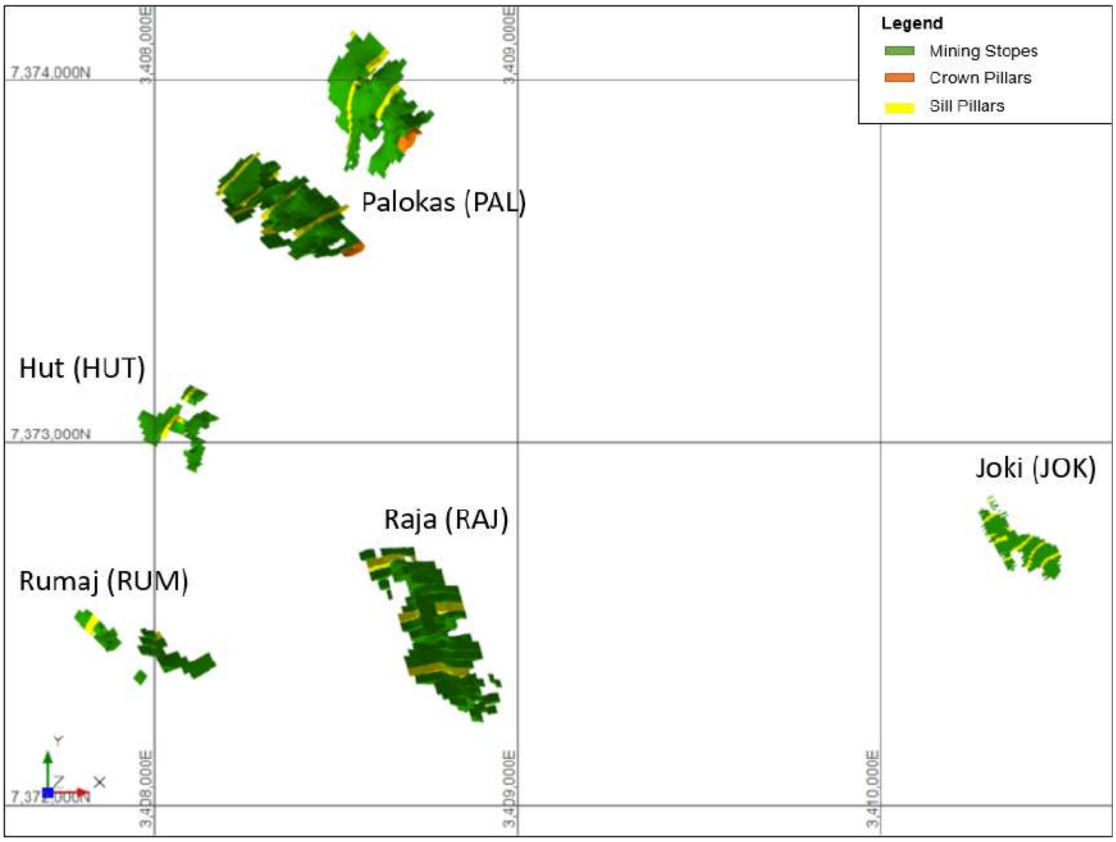


Figure 16-19: Plan view of the Rajapalot stope optimisation shapes

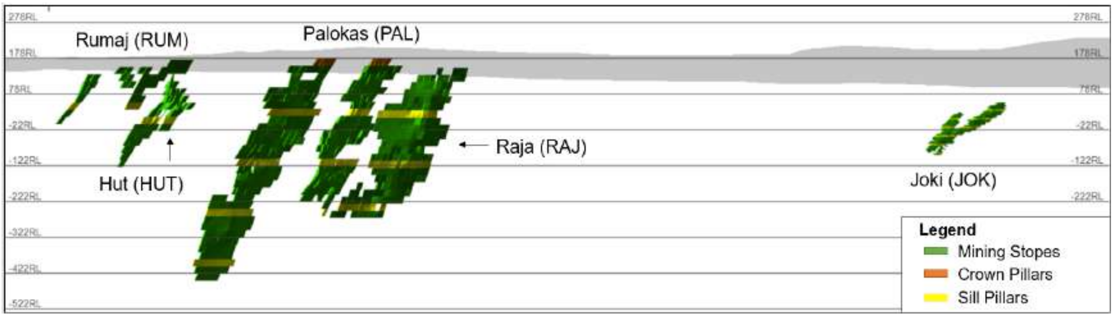


Figure 16-20: Long view of the Rajapalot stope optimisation shapes and topography

16.5.1 Modifying Factors

Mine external modifying factors (dilution and losses) were assessed at a high level and applied to the stope optimiser shapes (tonnes and grade) by designated mining method for each of the deposits as summarised in Table 16-4. All development is assumed to have 0% dilution and ore development is assumed to have 0% unplanned mining losses.

Table 16-4: Rajapalot Modifying Factors

Mining Method	Mining Dilution %	Mining Losses %
LHOS - Transverse	5	5
LHOS - Longitudinal	8	8
Cut and Fill (C&F)	2	2
Sill and Crown Pillars	10	10

16.5.2 ROM Inventory

The RoM mining inventory presented in Table 16-5 is inclusive of dilution and losses and totals LoM production of 10.1 Mt. The split of RoM inventory tonnage by mining method (inclusive of development) for all deposits (Table 16-6) resulted as follows:

- LHOS Transverse = 34.5%.
- LHOS Longitudinal = 49.2%.
- Cut & Fill = 3.3%.
- Pillar Recovery = 13%.

RoM Development (excluding C&F) contributes 17.4% to the RoM Inventory and the RoM Inventory by deposit (Development + Production), as follows:

- Palokas = 60%.
- Raja = 27%.
- Joki = 4%.
- The Hut = 6%.
- Rumaj = 3%.

Table 16-5: Rajapalot RoM Inventory

Mining Inventory	Unit	Totals	Palokas (PAL)	Raja (RAJ)	Joki (JOK)	Hut (HUT)	Rumaj (RUM)
ROM Development	t	**2,090,428**	973,369	491,866	398,647	151,476	75,069
ROM Production	t	**8,030,442**	5,100,009	2,282,963	-	417,737	229,733
Total ROM Tonnes	**t**	**10,120,870**	**6,073,379**	**2,774,829**	**398,647**	**569,213**	**304,802**
ROM Grade	ppm Co	**347**	379	305	225	267	388
	g/t Au	**2.26**	2.24	2.58	2.87	1.19	0.98
	ppm As	**206**	246	188	20	43	130
	ppm Cu	**157**	187	93	114	151	190
	% FeO	**8.88**	9.97	6.72	7.12	9.88	7.35
	ppm Ni	**88**	116	36	26	60	159
	% S	**1.82**	2.20	1.16	1.00	1.67	1.57
	ppm U	**27.5**	34.9	17.5	15.5	10.2	18.7
	ppm W	**83.3**	101.7	56.0	33.8	50.4	90.8
ROM Metal Content	t Co	**3,509**	2,303	846	90	152	118
	t Au	**22.9**	13.6	7.2	1.1	0.7	0.3
	t As	**2,090**	1,495	523	8	25	40
	t Cu	**1,585**	1,137	259	45	86	58
	t FeO	**899,160**	605,755	186,381	28,398	56,235	22,391
	t Ni	**895**	703	99	10	34	49
	t S	**183,821**	133,468	32,088	3,973	9,516	4,776
	t U	**278**	212.0	48.7	6.2	5.8	5.7
	t W	**843**	618	155	13	29	28

Table 16-6: Rajapalot RoM Tonnes by Mining Method

Mining Inventory	Unit	Totals	Palokas (PAL)	Raja (RAJ)	Joki (JOK)	Hut (HUT)	Rumaj (RUM)
LHOS - Transverse	t	**3,487,188**	2,021,918	1,319,761	-	124,682	20,828
LHOS - Longitudinal	t	**4,984,201**	3,274,359	1,067,724	-	384,332	257,787
Cut & Fill	t	**329,182**	-	-	329,182	-	-
Sill and Crown pillars	t	**1,320,298**	777,102	387,344	69,465	60,200	26,187
Total ROM tonnes	**t**	**10,120,870**	**6,073,379**	**2,774,829**	**398,647**	**569,213**	**304,802**

16.6 Mine Design

16.6.1 Introduction

The Rajapalot mine design considers individual boxcuts and declines to access each of the five deposits comprising the RoM Inventory. LHOS open stopes are based on 20 m level spacing and mined longitudinal to strike for orebody widths up to 15 m and transverse for widths greater than 15 m. Transverse stopes are mined through transverse ore drives spaced 15 m and connected to a FW drive while longitudinal stopes are mined through longitudinal ore drives connected to the decline access through cross-cut drives.

C&F stopes (Joki mine only) are mined at vertical lift intervals of 4 m height from a cross-cut ramp from the decline access which is progressively stripped to access each subsequent lift over a vertical distance of 24 m (6 lifts).

Sill pillars are slices of 20 m height located every 140 m in the LHOS areas and every 24 m in the C&F area. These sill pillars will use backfill material with an increased amount of binder compared to regular production stopes.

The PEA mine development layout is designed to provide logical, timely and efficient access to the stoping blocks at minimum cost, with the following factors considered:

- **Profile:** The profiles determined for the various types of development are based on the operating equipment selected, plus an allowance for any statutory clearance, or alternatively, internationally acceptable clearances.
- **Gradient:** The gradient for level access, ore drives, and footwall drives for this conceptual level of design has been considered as 0. A gradient of 1:50 is recommended for access drives to ensure effective drainage, with gradients designed to direct water to dewatering sumps. The decline gradient (1:7) is based on a trade-off between the maximum steepness to reduce the distance required to be developed between levels, and the provision of suitable operating conditions for the mobile equipment.

For stoping areas, the stockpile location and size must consider both the loader (bogger) and truck productivities. The maximum tramming distance for a loader ranges from 150 to 300 m while still maintaining acceptable productivities.

Maintaining high truck productivity in high tonne-kilometre (tkm) operations is of primary importance. Truck productivities assume loading directly from the stockpile to minimise truck idle time. The production stockpiles have been located as close as possible to the centroid of the stoping panels wherever possible.

Escapeway raises are designed to have a 1.5 m cross sectional area and return air raises, a 3 m cross sectional area. Ventilation systems will be connected to the level access on each level.

Figure 16-21 to Figure 16-23 provides respective plan, oblique and long views of the five mines for the Rajapalot Project showing the position, mining method and depth below surface.

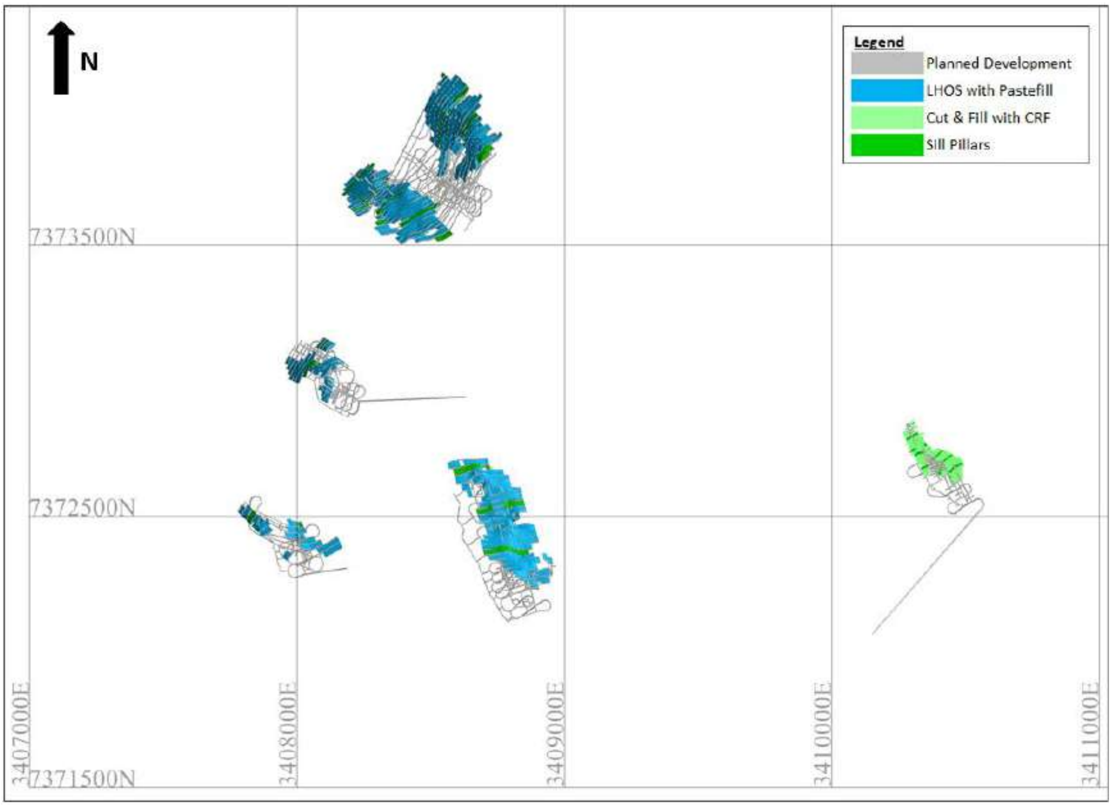


Figure 16-21: Plan view of the five mines for the Rajapalot by mining method

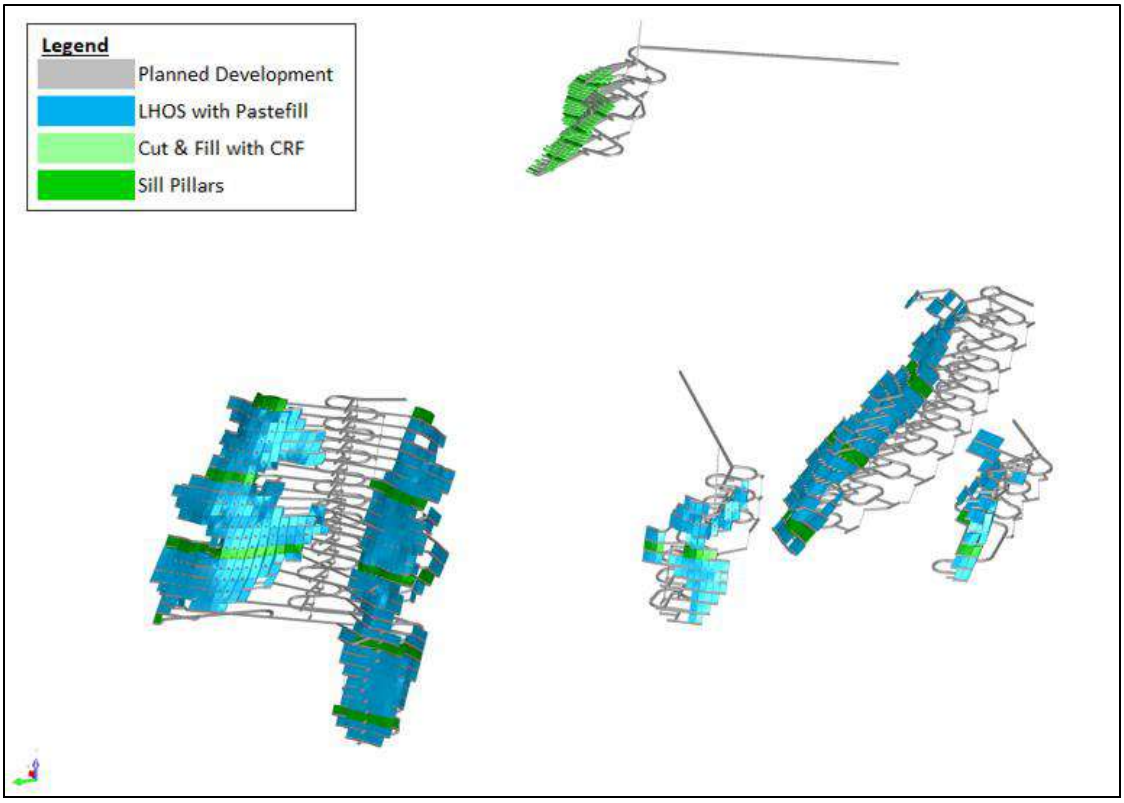


Figure 16-22: Oblique view of the five mines for the Rajapalot by mining method

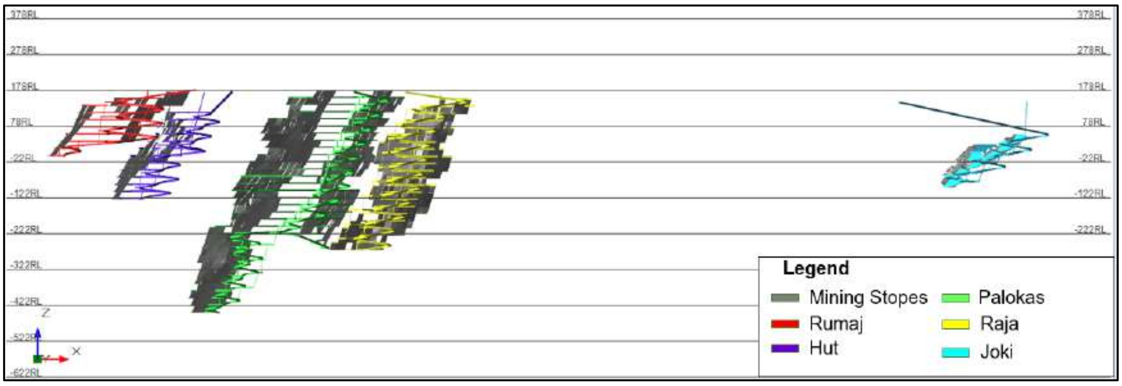


Figure 16-23: Long view of the five mines of the Rajapalot Project

16.6.2 Mine Development

The individual mine development designs for the Rajapalot deposits are shown as follows:

- Palokas mine (Figure 16-24).
- Raja mine (Figure 16-25).
- Joki mine (Figure 16-26).
- The Hut mine (Figure 16-27).
- Rumaj mine (Figure 16-28).

The development types and dimensions used for each mine design are summarised in Table 16-7 and also include ventilation raises and escapeways. Table 16-8 provides a summary of the lateral and vertical development metres for each mine design.

Table 16-7: Rajapalot Development (and Airway) Profiles

Development	Unit	Type	Width	Height	Diameter
Decline	m	Arch	5.5	5.5	
Level Access	m	Arch	5.0	5.0	
Transverse Ore Drive	m	Arch	4.5	4.5	
Longitudinal Ore Drive	m	Arch	4.5	4.5	
Footwall Drive	m	Arch	4.5	4.5	
Loading Stockpile	m	Arch	5.0	5.0	
Return Air Raise	m	Circular			3.0
Return Air Drive	m	Arch	4.5	4.5	
Escapeway Raise	m	Circular			1.5
Escapeway Drive	m	Arch	4.5	4.5	

Table 16-8: Summary of Lateral and Vertical Development Metres by Mine Design

Mine Development	Unit	Totals	Palokas (PAL)	Raja (RAJ)	Joki (JOK)	Hut (HUT)	Rumaj (RUM)
Lateral Development							
Decline	m	**12,915**	4,844	3,294	874	2,257	1,646
Level_X-Cut	m	**9,969**	1,883	1,719	5,039	914	414
FW Dev	m	**10,631**	7,474	1,298	-	588	1,271
Vent Dev	m	**3,627**	1,188	1,115	337	555	432
Ore Dev	m	**36,095**	16,807	8,493	6,883	2,615	1,296
Other Dev	m	**3,928**	1,249	1,265	408	612	394
Total Level Development	**m**	**77,165**	**33,445**	**17,184**	**13,542**	**7,541**	**5,453**
Vertical Development							
Level_Vent Raise	m	**2,063**	765	449	214	310	325
Level_Escapeway	m	**1,396**	568	376	126	208	119
Total Vertical Development	**m**	**3,459**	**1,333**	**824**	**340**	**518**	**444**

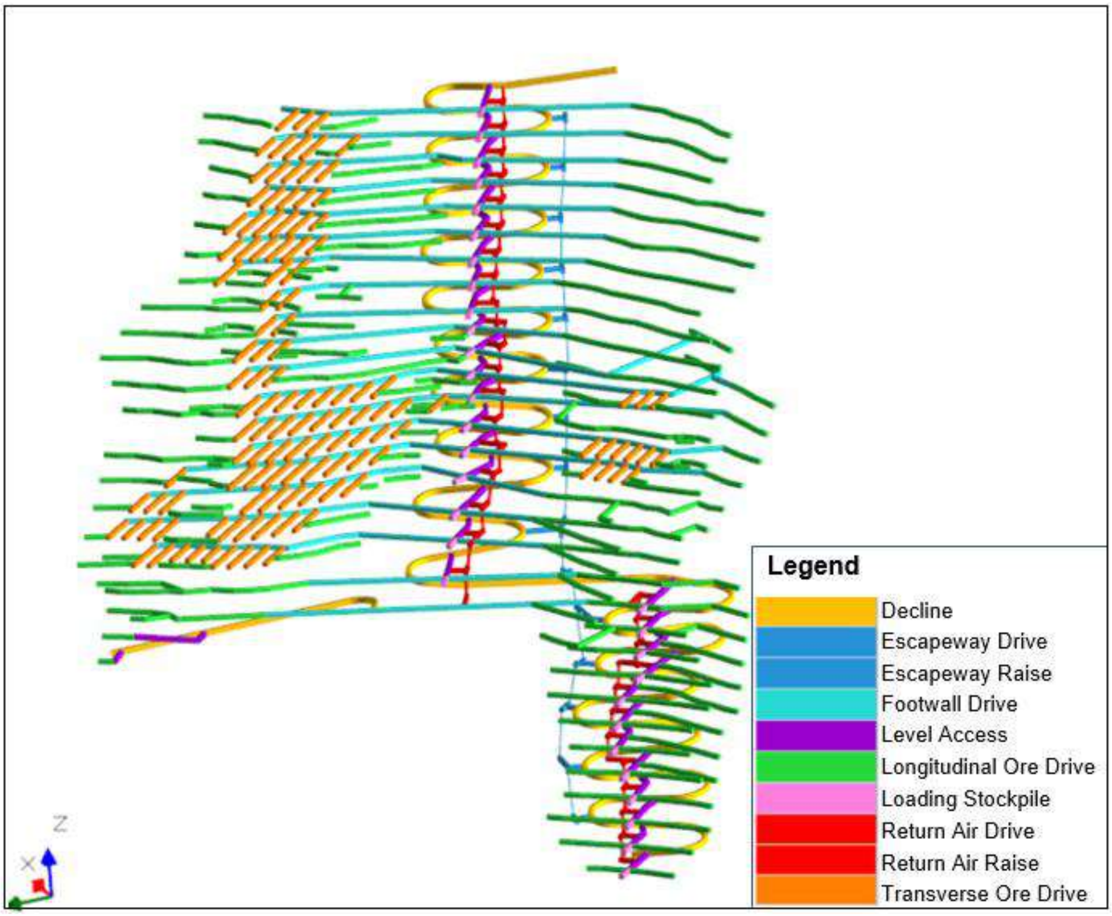


Figure 16-24: Oblique view of Palokas mine development design

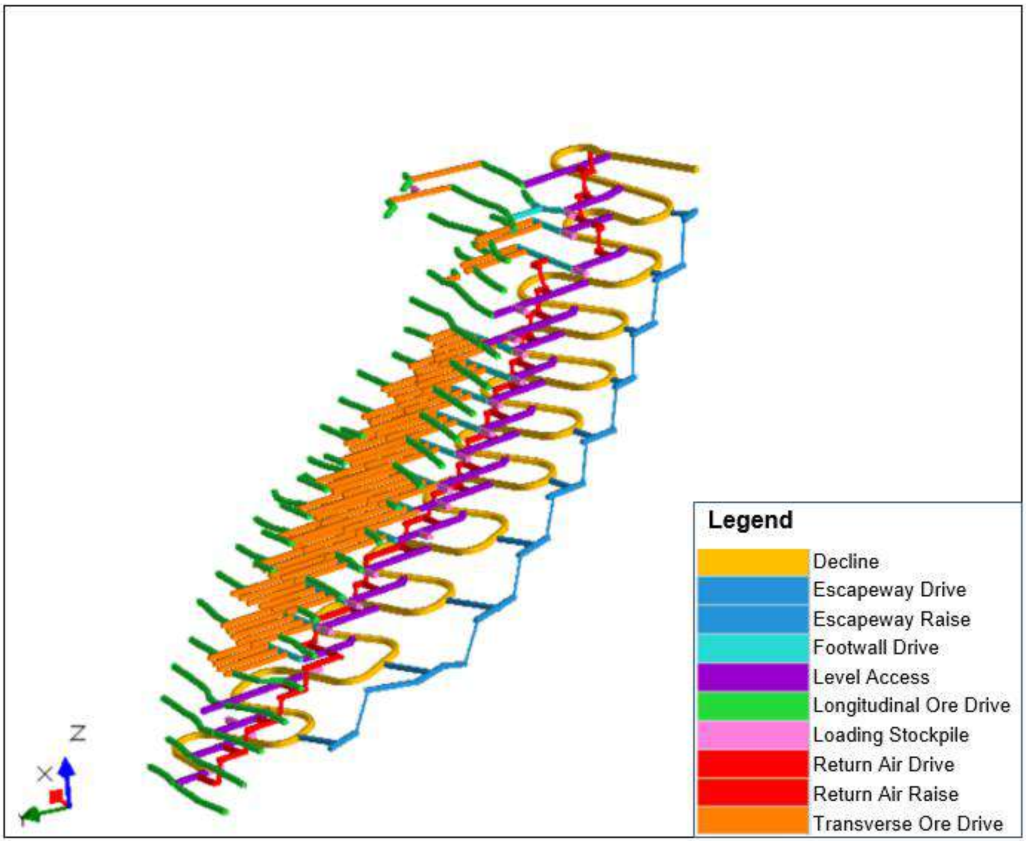


Figure 16-25: Oblique view of Raja mine development design

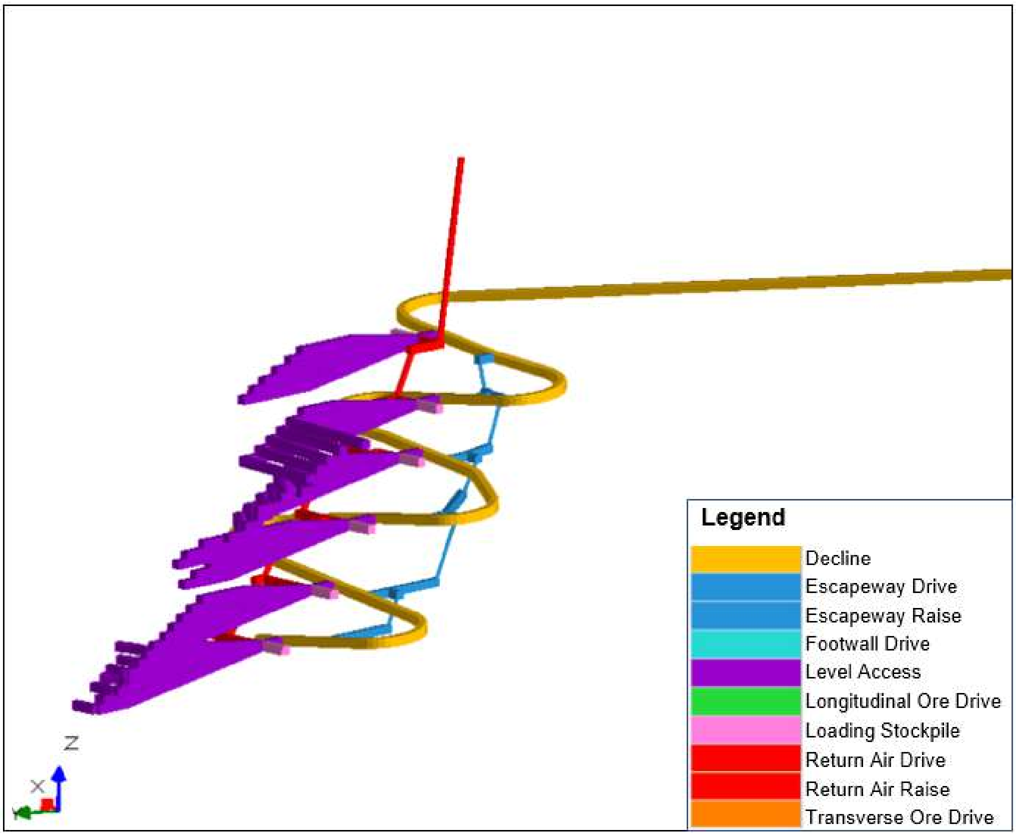


Figure 16-26: Oblique view of Joki mine development design

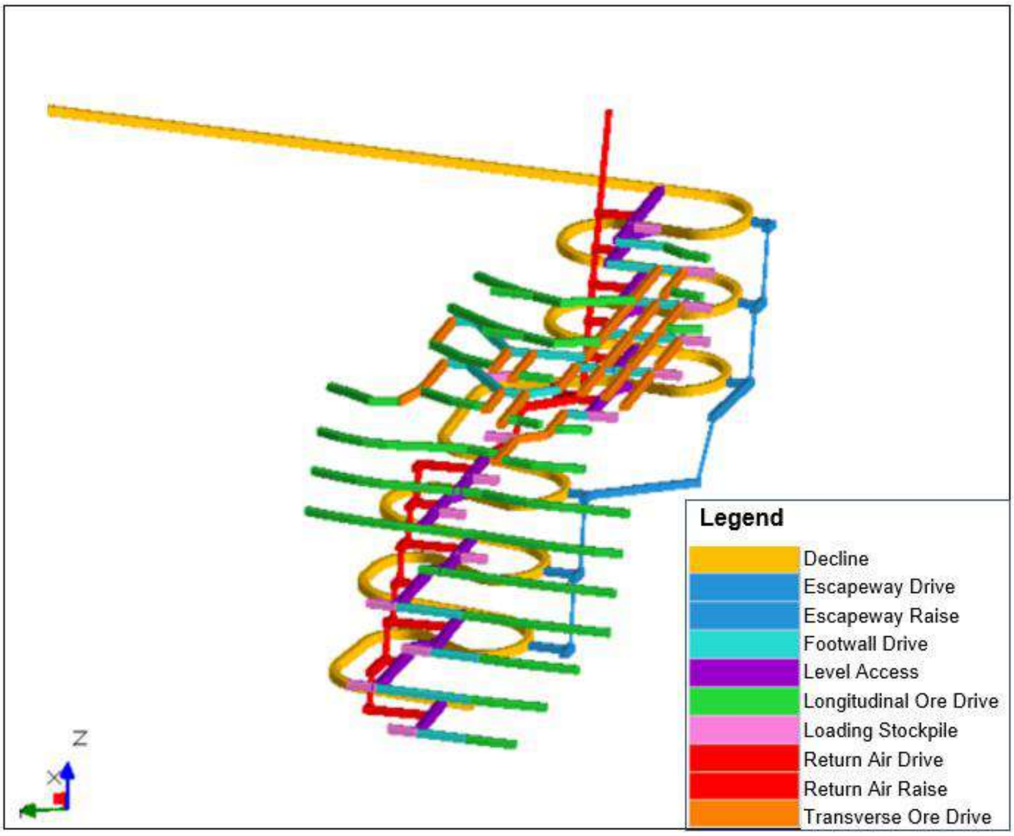


Figure 16-27: Oblique view of The Hut mine development design

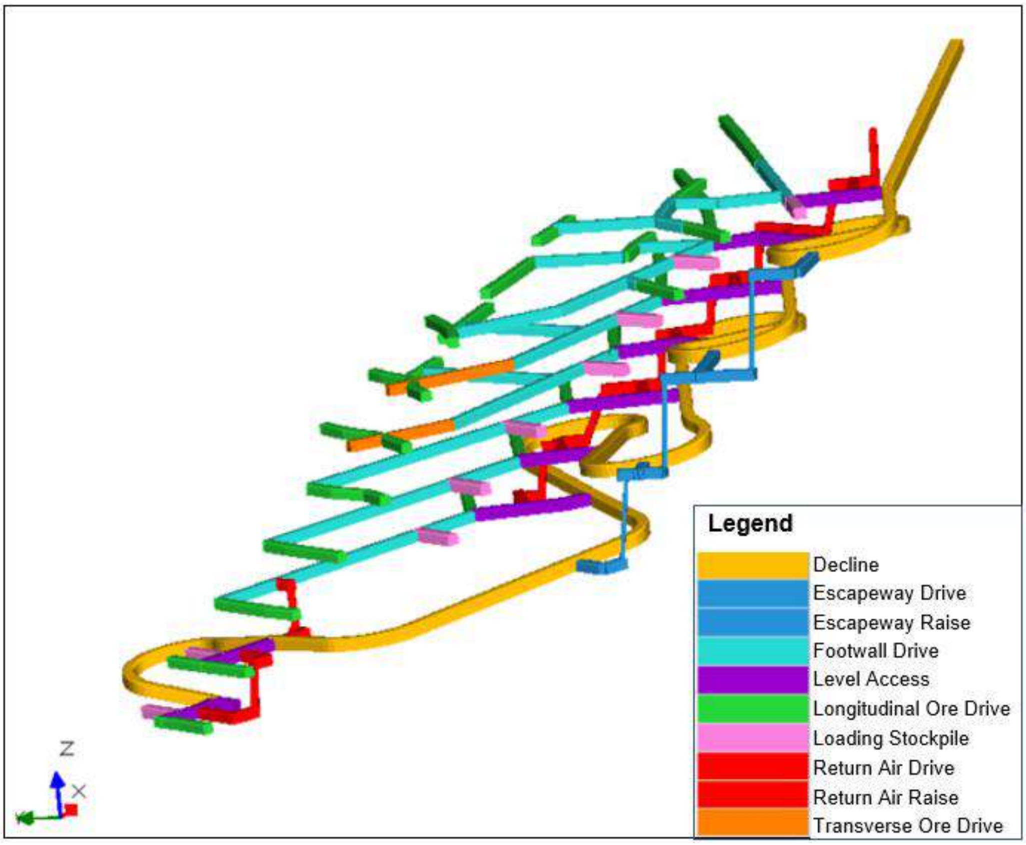


Figure 16-28: Oblique view of Rumaj mine development design

16.7 Mine Production

The production drill and blast design for Rajapalot project has been based on standard industry practice. Twin boom jumbo will be used for development and C&F production areas. Longhole drill rigs will be used in the LHOS production areas.

Traditional diesel-powered 17 t capacity loaders and 50 t capacity haul trucks will be used to move all mine waste to the respective designated surface and underground storage and RoM to surface stockpiling areas for mineral processing.

16.8 Backfill

16.8.1 Initial Filling Options

Based on the geometry of the orebodies, likely location of the decline and proximity to the processing plant, Table 16-9 presents the most credible backfilling options for each of the deposits, with the contribution to mining inventory presented to indicate the economic significance of each orebody.

Table 16-9: Preliminary Backfilling Options. Contribution to Mining Inventory

Mine:	Palokas	Raja	The Hut	Rumaj	Joki
Contribution to mining inventory:	**60%**	**27%**	**6%**	**3%**	**4%**
General approach	Pastefill sill. Pastefill above. Tight fill below sill				CRF & rock fill, bottom up with sill pillars
Fill delivery	Surface pipeline from backfill plant. Borehole from surface and cascade backfill into mine via reticulation		Remote deposit; pastefill with remote borehole (agitator truck delivery on surface)		Rockfill or CRF only. In keeping with low production rate & deposit size
Opportunity	Rock fill pads onto paste (for vehicular access) will reduce volume of waste rock brought to surface		CRF instead of pastefill for sill Rock fill pads onto paste (for vehicular access) will reduce volume of waste rock brought to surface		Be smart with primary/secondary approach to save cement
Comments	CRF unlikely to be viable - Delivery of cement slurry likely to be constrained		Additional fleet requirements for rockfill - and especially so for CRF		Low production rate, so keep it small

To satisfy the backfill requirements of the deposit, pastefill, rock fill and CRF will be required. The proportion and design of each type is estimated in the following sections. P&C notes that the disparate locations and geometries of the orebodies will influence the manufacture and placement options for the fill.

The following section discusses the backfilling options and makes recommendations based on the information available. The descriptions and design assumption have been developed for costing purposes only and are subject to confirmation through backfilling testwork.

16.8.2 Backfilling Methods

Pastefill, CRF and rock fill are all considered as suitable options for backfilling the Rajapalot deposit. The following section provides an overview of each method.

16.8.3 Pastefill

Pastefill may be appropriate on sites where a processing plant or tailings supply is available for paste production. Paste backfill generally comprises the whole stream of tailings and can be engineered for a range of horizontal or vertical exposures.

For pastefill, a full stream of tails is generally used to produce the backfill. As a rule of thumb, 15 to 20% of the tailings should pass 20 µm. Based on the information available (Figure 16-29), the whole-stream Particle Size Distribution (PSD) for Rajapalot appears suitable for pastefill; however, should be confirmed with testwork in the following stages of study.

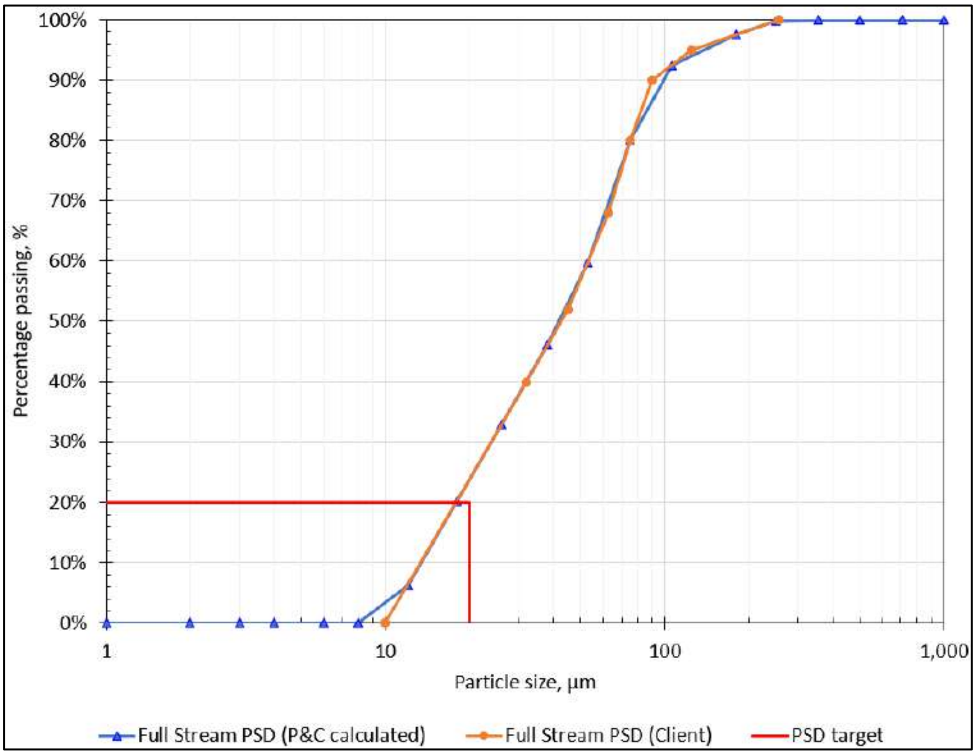


Figure 16-29: Initial Tailings PSD for Rajapalot

For vertical exposure applications, a 20 m high stope with a 15 m strike length requires 250 kPa to maintain stability when exposed (Figure 16-30). The backfill recipe is subject to the performance of the tailings, so would need to be tested to provide a reliable indication of the fill; however, based on P&C's experience and for PEA costing purposes, the following recipe is assumed:

- Tailings: 96%m;
- Binder: 4%m;
- Backfill mass concentration: 75%m; and
- Backfill density (wet): 1.96 t/m^3.

UCS requirement in kPa (incl. SF)		Stope strike length (m)						
		10	15	20	25	30	35	40
Stope height (m)	5.0	96	108	115	120	124	126	128
	7.5	124	144	157	166	173	178	182
	10.0	144	173	192	206	216	224	231
	12.5	160	197	222	240	254	266	275
	15.0	173	216	247	270	288	303	315
	17.5	184	233	269	297	319	336	351
	20.0	192	**247**	288	320	346	367	384
$UCS_{required} = \frac{\gamma \times H}{1+\frac{H}{L}} \times FS$	22.5	200	260	305	341	371	395	415
	25.0	206	270	320	360	393	421	444
	27.5	211	280	334	378	414	444	470
	30.0	216	288	346	393	433	466	494

Figure 16-30: Empirical Backfill Strength Requirement (Source: Belem T. et al, 2007)

For the purposes of this PEA, a benchmarked approach is taken to the underhand pastefill design. As with the CRF, the strength requirement is heavily influenced by mining conditions and exposures, but the benchmark will provide order of magnitude strength requirements from which to base the cost estimation. Based on work undertaken for other projects and to provide guidance for cost estimation purposes, a 3 MPa strength requirement is assumed, comprising:

- Tailings: 92%m;
- Binder: 8%m;
- Backfill mass concentration: 75%m; and
- Backfill density (wet): 1.96 t/m^3.

The backfill plant will be located close to the processing facility, so a surface pipeline is required to deliver pastefill to the Palokas and Raja deposits. Based on assumed parameters:

- a 90 Bar pi sufficient backpressure.

The pumping, pipe specification and pipe routing requirements are sufficient for a PEA cost estimation, but testwork and study are required to further define the system.

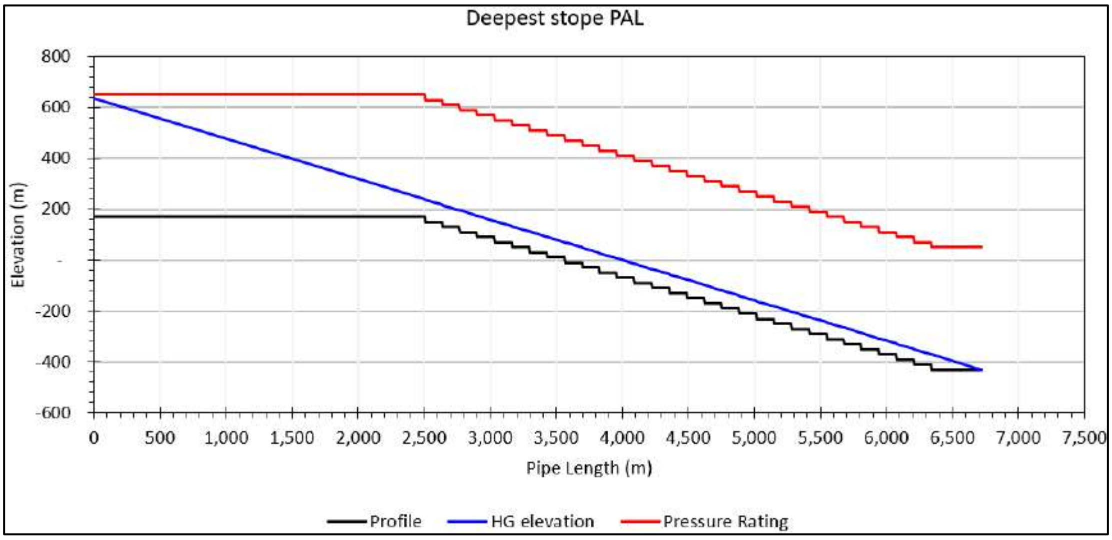


Figure 16-31: Preliminary Hydraulic Analysis assuming a level spacing of 20 m

Flowsheet

Figure 16-32 presents a typical paste plant flowsheet. The full stream of tailings is received into the thickener, where the underflow is directed to the backfill plant or the tailings storage facility.

For backfill use, the thickened tailings are further conditioned with a disc filter. The disc filter cake is combined with binder, water and a filter bypass slurry to produce a pastefill suitable for transportation and placement into the deposit.

The primary route for pastefill is transportation via the pump and reticulation to the Palokas and Raja deposits. Where remote boreholes are used (Rumaj and The Hut), the backfill will be discharged into agitator trucks and hauled to remote boreholes and onward distribution. Table 16-10 presents a preliminary, nominal mass balance for the backfill plant.

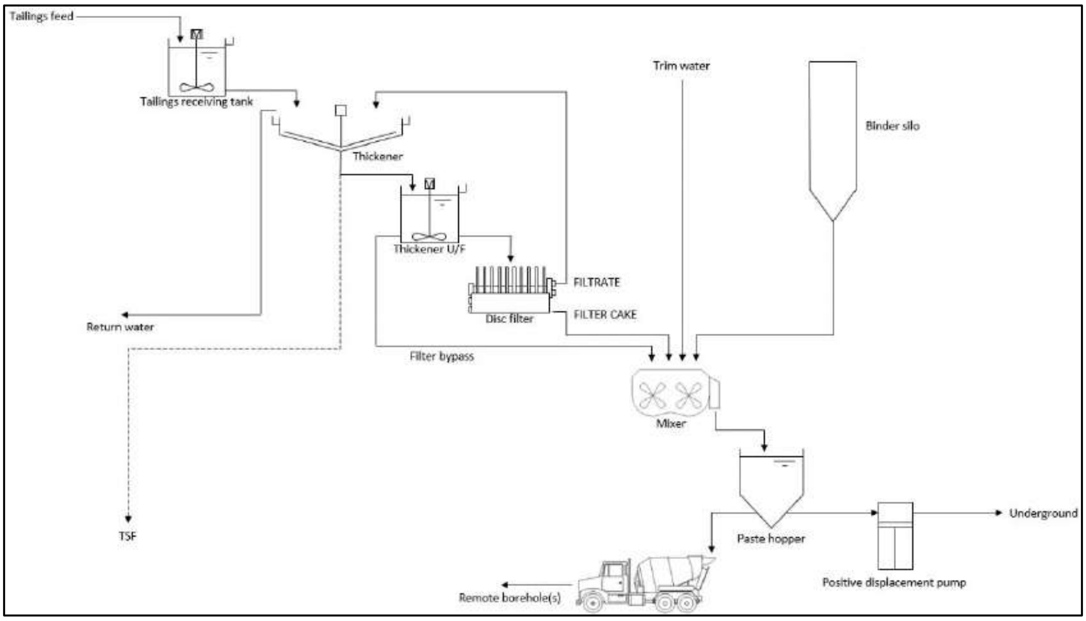


Figure 16-32: Paste Plant Schematic Sketch

Table 16-10: Mass Balance (Nominal)

Stream	Unit	Tailings feed	Thickener feed	Thickener underflow	Thickener overflow	Filter feed	Disc filter cake	Filtrate	Filter bypass feed	Trim water feed	Binder feed	Pastefill
Solids	t/h	137	137	137	-	117	117	-	20	-	7	144
Water	t/h	319	385	112	273	95	29	66	16	2	-	48
Total	t/h	455	522	248	273	212	146	66	36	2	7	191
Solids	m^3/h	48	48	48	-	41	41	-	7	-	2	50
Water	m^3/h	319	386	112	274	96	29	66	16	2	-	48
Total	m^3/h	367	433	160	274	136	70	66	23	2	2	98
Mixture density	t/m^3	1.24	1.20	1.56	1.00	1.56	2.08	1.00	1.56	1.00	3.15	1.96
Volume concentration	%v	13%	11%	30%	0%	30%	58%	0%	30%	100%	100%	51%

Stream	Unit	Tailings feed	Thickener feed	Thickener underflow	Thickener overflow	Filter feed	Disc filter cake	Filtrate	Filter bypass feed	Trim water feed	Binder feed	Pastefill
Mass concentration	%m	30%	26%	55%	0%	55%	80%	0%	55%	0%	100%	75%
Solids density	t/m^3	2.86	2.86	2.86	2.86	2.86	2.86	2.86	2.86	2.86	3.15	2.88

To place a pastefill, barricades are erected at the stope openings to contain the fill. Reticulation is run to the stope and the backfill is mixed in the plant and delivered to the stope via the reticulation. Normal practice is to pour a 'plug run' and then cure to the liquefaction limit to isolate the barricade from hydrostatic head (Figure 16-33). Once cured to a satisfactory strength, the mass pour is placed and allowed to cure. Once sufficient backfill strength has been validated by QAQC testing, the stope can be exposed, and mining continues.

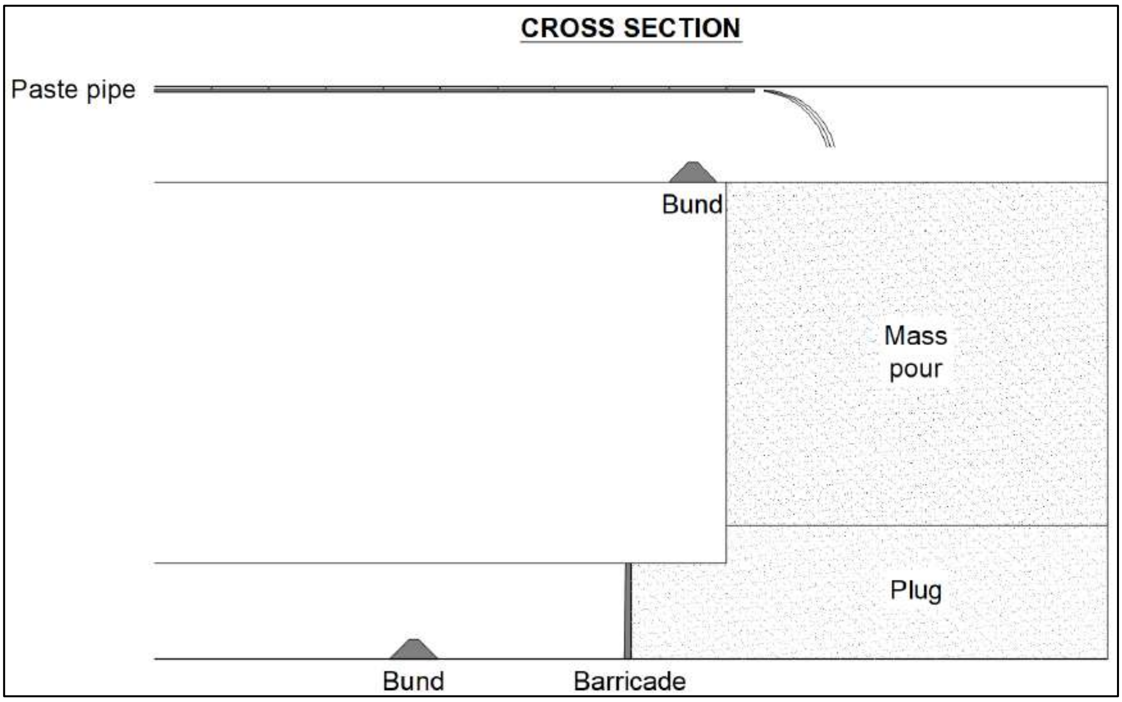


Figure 16-33: Section view of Pastefill Placement (Sketch)

Pastefill offers the potential to dispose of the whole tailings stream to the underground environment and can produce a stable backfill column which can be exposed vertically where adjoining other stopes. Pastefill is recommended where surface disposal of tailings need to be reduced and where rock fill or CRF is not available in sufficient quantity or viable for placement.

16.8.4 Cemented Rock Fill

CRF is produced by combining waste rock with a cementitious slurry. At the most primitive level, cement slurry is imported from surface by agitator truck and mixed into the waste rock with an LHD. The CRF is then hauled by LHD or truck to the tip head and placed into the stope.

CRF strength is generally estimated as equal to the lithostatic stress; Backfill Density x Gravity x Height (ρgh). This approach is conservative, but given the variables involved in CRF production is justifiable. Over and above this strength estimation, a safety factor is applied to increase confidence in the design. For a 20 m vertical exposure, 560 kPa strength is required.

P&C experience has shown that for vertical exposure and in practice, a minimum of 4% cement (by volume) is required to adequately coat the particles during mixing to ensure adequate bonding.

The determination of a CRF strength for underhand mining is more challenging and is typically undertaken during the latter stages of project development as a combination of numerical modelling and laboratory-scale testwork to validate the recipe. Based on benchmarking results and depending on the span to be opened, the strength requirement will be in the region of 6,000 to 9,000 kPa with a binder content in the region of 8% to 10% by mass.

The cement slurry pre-mix is mixed with the waste rock underground (typically at sumps) located within close tramming distance to the stope requiring backfilling. Loaders are used to place the CRF in C&F drifts, pushing up the fill as high as possible in the mine level (typically within 1 m of the backs).

Cemented rock fill is a method which offers flexibility in filling and depending on the process selection and enables the placement of fill independently of processing constraints. From an operational perspective, CRF may be more expensive per tonne than pastefill but capital costs are significantly lower than paste.

16.8.5 Rock Fill

Rock fill is the process of stowing loose rock, typically development or stripping waste, into a mined-out void. No binder is added, so rock stability is dictated by packing density and the confinement provided by the surrounding rockmass. Since this method is centred around disposing waste rock without the addition of any other components, it generally represents the lowest-cost option for void filling.

Development waste (nominally <0.3 m diameter) is generally fragmented and with adequate grading to pack and consolidate within the stope. Bridging is not normally an issue within this type of backfill.

To place rock fill, there needs to be access to the top of the stope from which the material is tipped. Fill rates are governed by stockpiling, fleet availability and the number of headings (or heading occupancy) available for tipping waste.

Rock fill is the cheapest option of backfilling within the mine and presents an opportunity to dispose of development waste in the underground setting.

16.8.6 Production Rate

The mining rate vs backfilling rate for Rajapalot is presented in Figure 16-34. The graph demonstrates that the production rate assumptions for Rajapalot fall within the expected range for a project of this size.

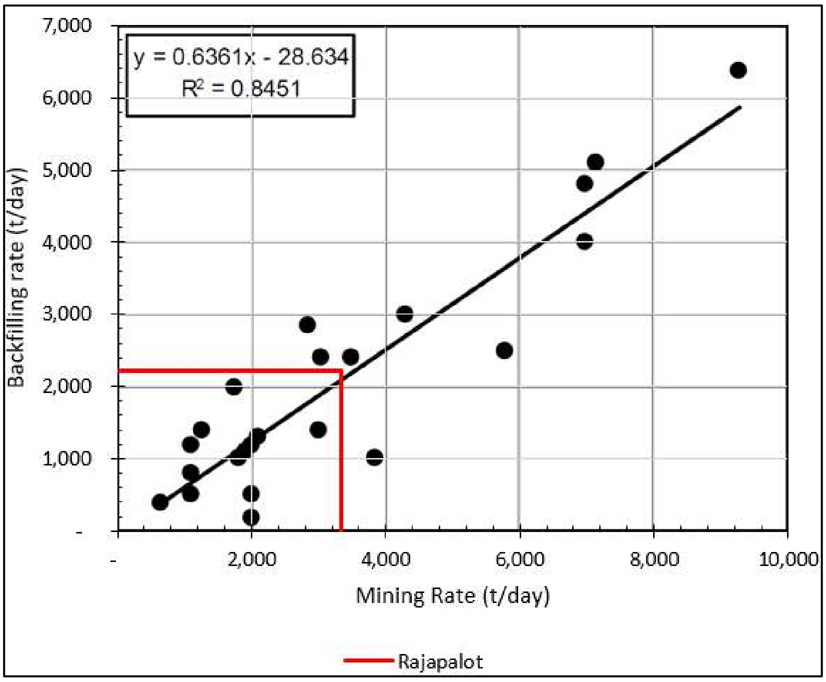


Figure 16-34: Production Benchmark (with reference to De Souza et al, 2003)

16.8.7 Conclusions

P&C has undertaken an initial assessment of the backfilling requirements for the five orebodies which make up the Rajapalot deposit. The Palokas, Raja, Rumaj and The Hut mines are suitable for a pastefill approach, whereas the Joki mine is suited to a CRF/rock filling approach.

Two pastefill strengths are considered, with one to provide general ground support and another high strength fill to act as a sill pillar when exposed from below.

The suitability of pastefill is based on assumptions from the information provided by the Client but no testing has been undertaken to confirm the suitability of the tailings of the manufacture of pastefill. P&C recommends that for the next phase of study, testwork is undertaken to validate the assumptions made in this report.

16.9 Mining Equipment

The equipment required to undertake mining activities at the Rajapalot mine was selected based on practical experience of working in similar mining environments including working mines in the Nordic region.

Table 16-11 provides a list of the primary and secondary support equipment considered in the mine plan and unit productivities used to determine equipment requirements over the LoM. The equipment operating factors used to estimate operating costs throughout the LoM are shown in Table 16-12.Table 16-13 shows the truck productivity parameters applied over the LoM. The trucking requirements (50 t capacity) have been assessed based on estimates of the haul distances by level and material type, provided in Table 16-14. It is assumed that development waste is stored in the surface mine waste storage facility which is designed for a capacity up to 3 Mt or underground in the newly created mining voids as CRF or loose fill.

Table 16-11: Mine Equipment and Productivity Assumptions

Fleet	Units	Productivity

		Per annum	Per month	Notes
Lateral Development				
Twin Boom Jumbo	dev m adv	3,360	280	Based on Twin Boom Jumbo development metres
Development Loader - 17t	tph	389,664	32,472	Based on Loader tonnes
Production LHOS				
Production Loader - 17t	tph	490,977	40,915	Based on Loader tonnes
Longhole Drill	drill m	75,000	6,250	Based on LH drill metres
Chargeup				
Chargeup wagon	tonnes charged	600,000	50,000	Based on Production Rate
Auxiliary Equipment				
Grader	tpa	1,650,000	137,500	Production rate based
Service (Fuel/Lube) Truck	Drills	5	5	1 x Service Truck for every 5 Drills
Integrated Toolcarrier	tpa	350,000	20,833	Production rate based
Grade Control/Probe Drill	drill m	15,000	1,250	Based on grade control metres (production rate based)
Backfill				
Agitator Truck	each	2	2	2 machines - physicals model required to refine the estimate
Shotcrete Sprayer	each	1	1	1 machine - physicals model required to refine the estimate

Table 16-12: Mine Equipment Operating Factors

Fleet	Availability (%)	Use of Availability (%)	Operator Efficiency (%)	Effective Utilisation (%)	Direct Operating Hours (DOH)	
					per year	per shift
Lateral Development						
Twin Boom Jumbo	83%	65%	100%	54%	4,661	6.5
Development Loader - 17t	82%	55%	100%	45%	3,897	5.4
Production LHOS						
Production Loader - 17t	82%	77%	100%	63%	5,455	7.6
Longhole Drill	85%	49%	100%	42%	3,599	5.0
Haulage						
Truck - 50t capacity	85%	53%	100%	45%	3,892	5.4
Chargeup						
Chargeup wagon	83%	50%	100%	42%	3,586	5.0
Auxiliary Equipment						
Grader	82%	55%	100%	45%	3,897	5.4
Service (Fuel/Lube) Truck	80%	50%	100%	40%	3,456	4.8
Integrated Toolcarrier	80%	50%	100%	40%	3,456	4.8
Grade Control/Probe Drill	80%	50%	100%	40%	3,456	4.8
Light Vehicle	80%	20%	100%	16%	1,382	1.9
Personnel carrier	80%	30%	100%	24%	2,074	2.9

Table 16-13: Truck Productivity Parameters

Trucking TKM Cycle	Units	Value
Truck Capacity	m^3	27
Loader Capacity	m^3	7
Speed up Ramp	km/hr	10
Speed down Ramp	km/hr	12
Loading time 7m^3 LHD	hrs	0.1
Dumping time	hrs	0.1
Capacity @ 90% Tray Fill	m^3	24.3
SG loose	t/m^3	2.1
Tonnage Capacity - Maximum Rated	t	50

16.10 Mine Personnel

The professional staff (including management), workforce, and maintenance personnel for the underground mine is estimated based on the typical levels for this size of operation, operating 2 x 12-hour shifts, 24 hours per day, and 7 days per week. The maintenance, underground operator, and labour estimates are based on the annual equipment estimates.

The majority of underground positions are based on three rostered crews working a 2-shift, 6-day rotation. A majority of the management and staff work only day shift. For the purposes of the PEA, all mining activities related to the underground mines will be carried out by owner-operator personnel. A summary of the estimate of mine personnel over the LoM schedule is provided in Table 16-19.

Table 16-14: Average truck haulage distances for each mine

Joki		Palokas		Raja		Hut		Rumaj	
TKMs Average Haul	Average Haul Ore/Waste (km)	TKMs Average Haul	Average Haul Ore/Waste (km)	TKMs Average Haul	Average Haul Ore/Waste (km)	TKMs Average Haul	Average Haul Ore/Waste (km)	TKMs Average Haul	Average Haul Ore/Waste (km)
Portal to Plant/WRD	1.00	Portal to Plant/WRD	1.00	Portal to Plant/WRD	1.00	Portal to Plant/WRD	1.00	Portal to Plant/WRD	1.00
Decline to Portal	0.70	Decline to Portal	2.14	Decline to Portal	1.12	Decline to Portal	2.11	Decline to Portal	1.85
Level 52	1.75	Level 158	4.82	Level 133	2.82	Level 115	3.84	Level 155	3.61
Level 48	1.73	Level 138	4.68	Level 113	2.68	Level 95	3.70	Level 135	3.47
Level 44	1.71	Level 118	4.54	Level 93	2.54	Level 75	3.56	Level 115	3.33
Level 40	1.73	Level 98	4.40	Level 73	2.40	Level 55	3.42	Level 95	3.19
Level 36	1.76	Level 78	4.26	Level 53	2.26	Level 35	3.28	Level 75	3.05
Level 32	1.79	Level 58	4.12	Level 33	2.12	Level 15	3.14	Level 55	2.91
Level 28	1.82	Level 38	3.98	Level 13	2.26	Level -5	3.21	Level 35	2.93
Level 24	1.85	Level 18	3.84	Level -7	2.40	Level -25	3.35	Level 15	3.07
Level 20	1.87	Level -2	3.70	Level -27	2.54	Level -45	3.49	Level -5	3.21
Level 16	1.90	Level -22	3.56	Level -47	2.68	Level -65	3.63		
Level 12	1.93	Level -42	3.42	Level -67	2.82	Level -85	3.77		
Level 8	1.96	Level -62	3.28	Level -87	2.96	Level -105	3.91		
Level 4	1.99	Level -82	3.14	Level -107	3.10	Level -125	4.05		
Level -4	2.04	Level -102	3.28	Level -127	3.24				
Level -8	2.07	Level -122	3.42	Level -147	3.38				
Level -12	2.10	Level -142	3.56	Level -167	3.52				
Level -16	2.13	Level -162	3.70	Level -187	3.66				
Level -20	2.15	Level -182	3.84	Level -207	3.80				
Level -24	2.18	Level -202	3.98	Level -227	3.94				
Level -28	2.21	Level -222	4.12	Level -247	4.08				
Level -32	2.24	Level -242	4.26	Level -267	4.22				
Level -36	2.27	Level -262	4.40						
Level -40	2.29	Level -282	4.54						
Level -44	2.32	Level -302	4.68						
Level -48	2.35	Level -322	4.82						
Level -52	2.38	Level -342	4.96						
Level -56	2.41	Level -362	5.10						
Level -60	2.43	Level -382	5.24						
Level -64	2.46	Level -402	5.38						
Level -68	2.49	Level -422	5.52						
Level -72	2.52	Level -442	5.66						
Level -76	2.55								
Level -80	2.57								
Level -84	2.60								
Level -88	2.63								
Level -92	2.66								

16.11 Mine Schedule

16.11.1 Introduction

The RoM Inventory available for scheduling is presented in Table 16-5 and a summary of the required lateral and vertical development for each mine is provided in Table 16-8.

16.11.2 Production Rate Assessment and Schedule Strategy

The production rate potential for each of the deposits was determined through an assessment of the tonnes per vertical metre and typical annual decline advance rates. An overall production target was set at 1.2 Mtpa, based on the RoM Inventory (Table 16-5), and it was estimated that this could be sustained over a 9-year period through mining three of the deposits at any time. The production strategy considers continuous mining of the larger two deposits (Palokas and Raja) over the LoM and mining the smaller three deposits (Joki, The Hut and Rumaj) sequentially with the order determined by higher gold grades, summarised as follows:

- Initial mining commences at Palokas (675 ktpa) + Raja (375 ktpa) + Joki (150 ktpa).
- When Joki is exhausted then production commences at The Hut (150 ktpa).
- When The Hut is exhausted then production commences at Rumaj (150 ktpa).

The first year of production considers a ramp up at 80% of the target production rate. The general sequence of mining is top-down, mining in an overhand approach between sill (and crown) pillars. Pillars are extracted on retreat with backfill support (paste or CRF). All of the deposits are located close to surface with minimal development required to access and commence production. A provision for boxcut access is included in the year prior to production at each of the deposits.

The exception to the overall development and production strategy is the Raja deposit which has higher gold grades at around 200 vertical metres below surface. The decline for Raja is advanced 1.5 km in the first year of pre-production (Year -1) to improve the gold grade in the earlier years of the production schedule.

The mine production schedule ramps down to 0.8 Mtpa in the final year of production (Year 9).

In addition to the overall mining schedule, RoM material (on a stope-by-stope basis) is assessed against an NSR cut-off for cobalt extraction (USD2/t), to be separately stockpiled, and campaign processed. All feed will be processed for gold recovery but only a proportion, on a feed campaign basis, for cobalt recovery.

16.11.3 Schedule methodology

The development and production scheduling are based on the RoM inventory as derived in Section 16.5 and summarised in Table 16-5. SRK prepared a simplified semi-automated spreadsheet approach for scheduling the required production and development for each level of each orebody. The mine inventory was scheduled for each level in an ordered sequence based on development access to achieve the production target rate of 1.2 Mtpa.

The annual production schedule is used to derive an equipment fleet schedule including commissioning and replacement periods for the duration of the operation. Labour requirements for each period are also estimated based on the development, production and equipment estimates.

16.11.4 Schedule results

Figure 16-35 shows the annual combined development and production RoM tonnes and grade schedule achieving a sustainable production rate of around 1.2 Mtpa over a 7-year period excluding the first and last year of production. The annual mine schedule physicals and key performance indicator (KPI) for Rajapalot project are presented as follows:

- Overall RoM production and grades in Table 16-15.
- Selective RoM Cobalt Schedule in Table 16-16.
- Lateral and vertical development in Table 16-17.
- Primary and secondary mine equipment including ventilation in Table 16-18.
- Mine personnel requirements for the underground operation in Table 16-19.
- Mine water management equipment for primary pumping stations and secondary pumps in Table 16-20.

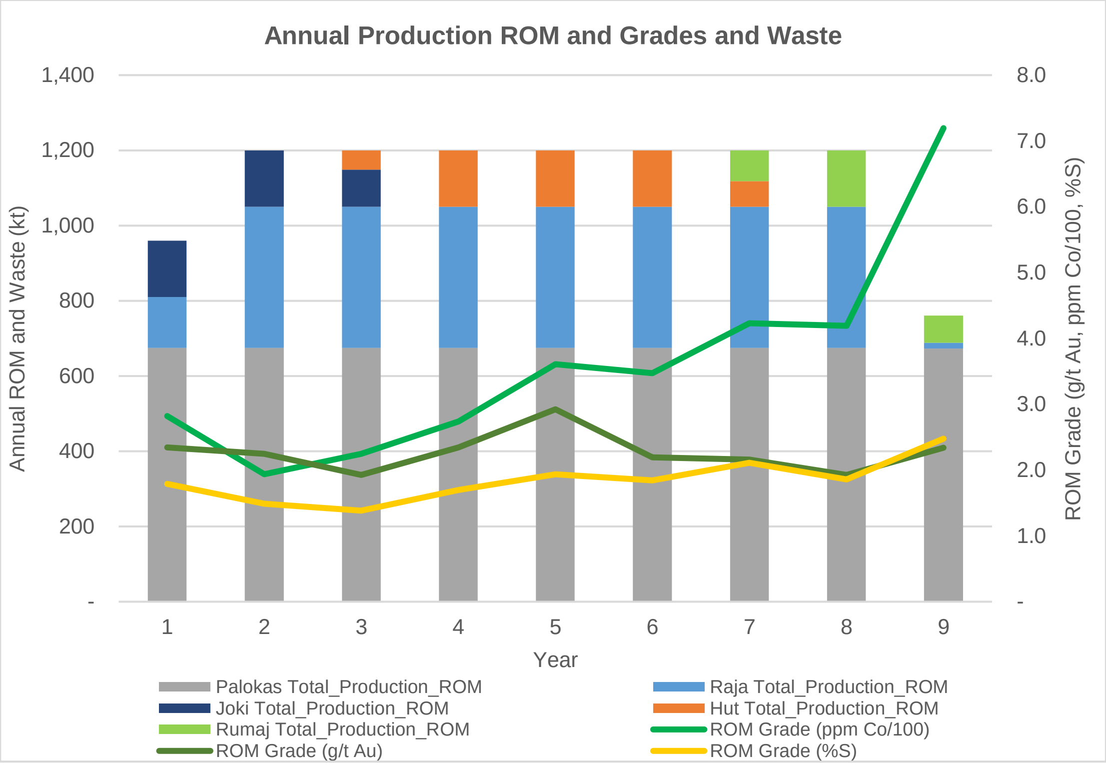


Figure 16-35: Annual Development and Production RoM and Grade for Rajapalot

Table 16-15: Overall RoM Schedule Production and Grades

Overall ROM Schedule Physicals	Units	Total	Year -01	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
Total Waste	**kt**	**2,982**	**140**	**339**	**331**	**438**	**224**	**209**	**204**	**350**	**466**	**282**
Development ROM												
Palokas	kt	**973**		125	93	106	103	109	105	94	147	91
Raja	kt	**492**		21	68	62	52	65	72	63	85	3
Joki	kt	**399**		150	150	99	-	-	-	-	-	-
Hut	kt	**151**		-	-	27	46	38	27	14	-	-
Rumaj	kt	**75**		-	-	-	-	-	-	35	26	14
Production ROM												
Palokas	kt	**5,100**		550	582	569	572	566	570	581	528	582
Raja	kt	**2,283**		114	307	313	323	310	303	312	290	12
Joki	kt	**-**		-	-	-	-	-	-	-	-	-
Hut	kt	**418**		-	-	25	104	112	123	53	-	-
Rumaj	kt	**230**		-	-	-	-	-	-	48	124	58
Total ROM	**kt**	**10,121**	**-**	**960**	**1,200**	**1,200**	**1,200**	**1,200**	**1,200**	**1,200**	**1,200**	**761**
ROM Grade												
Cobalt	ppm Co	**347**		281	217	333	351	353	224	365	400	719
Gold	g/t Au	**2.26**		2.51	2.73	2.30	2.32	2.47	1.97	1.95	1.86	2.34
Arsenic	ppm As	**206**		87	122	171	185	165	154	271	234	585
Copper	ppm Cu	**157**		192	106	135	152	138	87	185	223	225
Ferrous oxide	% FeO	**9**		11	9	8	8	9	8	8	8	11
Nickel	ppm Ni	**88**		112	87	85	80	81	68	91	88	121
Sulphur	% S	**1.82**		1.77	1.59	1.78	1.96	1.94	1.42	1.85	1.80	2.48
Uranium	ppm U	**28**		14	22	34	16	18	14	27	22	107
Tungsten	ppm W	**83**		84	105	112	75	69	53	59	57	167

Table 16-16: Selective RoM Cobalt Schedule

Selective ROM Cobalt Schedule Physicals	Units	Total	Year -01	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
ROM Stockpiled for Cobalt Processing	**kt**	**6,054**		**609**	**380**	**589**	**523**	**591**	**688**	**1,183**	**831**	**660**
ROM Grade												
Cobalt	% Co	**0.053%**		0.037%	0.038%	0.040%	0.047%	0.049%	0.048%	0.057%	0.062%	0.081%
Gold	g/t Au	**2.42**		2.55	2.44	1.97	1.87	2.36	3.34	2.76	1.71	2.47
Sulphur	% S	**2.07%**		2.03%	1.84%	1.79%	1.98%	1.88%	1.80%	2.12%	2.14%	2.79%

Table 16-17: Mine Physicals Schedule Development Metres

Mining Schedule Physicals	Units	Total	Year -01	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
Lateral Development												
Decline	m	**12,915**	**1,476**	969	919	1,803	860	606	649	1,460	2,342	1,832
Level X-Cut	m	**9,969**		1,816	1,993	2,340	349	366	433	918	1,102	651
FW Dev	m	**10,631**		1,460	1,038	1,146	1,422	1,204	1,100	1,538	1,449	274
Vent Dev	m	**3,627**		258	345	361	215	355	381	532	722	459
Ore Dev	m	**36,095**		5,114	5,364	5,058	3,480	3,656	3,526	3,553	4,470	1,874
Other Dev	m	**3,928**	**215**	245	354	445	291	462	250	416	798	451
Total Level Development	**m**	**77,165**	**1,691**	**9,861**	**10,012**	**11,153**	**6,616**	**6,650**	**6,339**	**8,417**	**10,885**	**5,540**
Vertical Development												
Level Vent Raise	m	**2,265**	**202**	281	274	186	75	101	273	221	264	388
Level Escapeway	m	**1,523**	**125**	24	198	154	126	127	121	166	244	238
Total Vertical Development	**m**	**3,787**	**326**	**305**	**472**	**340**	**201**	**228**	**394**	**387**	**508**	**626**

Table 16-18: Mine Equipment Schedule

Mine Equipment	Unit	Max LOM	Year -01	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
Twin Boom Jumbo	each	4	1	3	3	4	2	2	2	3	4	2
Development Loader - 17t	each	3	1	3	3	3	2	2	2	3	3	2
Production Loader - 17t	each	4	1	3	3	3	4	4	4	4	3	2
Longhole Drill	each	3	1	2	2	2	3	3	2	3	3	2
Truck - 50t capacity	each	10	1	8	9	9	8	8	8	9	10	8
Chargeup wagon	each	3	1	3	3	3	3	3	3	3	3	2
Grader	each	2	1	1	1	1	1	1	1	1	2	1
Service (Fuel/Lube) Truck	each	2	1	1	1	2	1	1	1	2	2	1
Integrated Toolcarrier	each	5	1	4	5	5	5	5	5	5	5	3
Grade Control/Probe Drill	each	2	1	2	2	2	2	2	2	2	2	2
Light Vehicle	each	32	13	27	29	31	29	29	28	31	32	24
Personnel carrier	each	11	5	9	10	11	10	10	10	11	11	8
Agitator Truck	each	2	2	2	2	2	2	2	2	2	2	2
Shotcrete Sprayer	each	1	1	1	1	1	1	1	1	1	1	1
Wheel Loaders (ROM & Backfill)	each	2	2	2	2	2	2	2	2	2	2	2

Table 16-19: Mine Personnel Schedule

Mine Personnel	Units	Year -01	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
Management	each	6	6	6	6	6	6	6	6	6	6
Technical Support	each	9	17	17	17	17	17	17	17	17	17
Mine Operations	each	46	100	103	115	100	100	97	115	122	85
Maintenance	each	28	62	69	74	72	72	69	77	77	57
Administration	each	2	3	3	3	3	3	3	3	3	3
Total Mine Personnel	**each**	**91**	**188**	**198**	**215**	**198**	**198**	**192**	**218**	**225**	**168**

Table 16-20: Mine Water Management

Mine Water Inflow estimate	Unit	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
Palokas (PAL)	m^3/day	1,147	1,147	1,147	1,145	1,145	1,145	1,148	1,148	1,148
Raja (RAJ)	m^3/day	734	734	734	719	719	719	682	682	682
Joki (JOK)	m^3/day	449	449	449	-	-	-	-	-	-
Hut (HUT)	m^3/day	-	-	-	606	606	606	-	-	-
Rumaj (RUM)	m^3/day	-	-	-	-	-	-	923	923	923
Total Mine Water Inflow	**m^3/day**	**2,330**	**2,330**	**2,330**	**2,470**	**2,470**	**2,470**	**2,753**	**2,753**	**2,753**
	litre/sec	**27**	**27**	**27**	**29**	**29**	**29**	**32**	**32**	**32**
Mine Dewatering Pumps and Equipment										
Primary Pump Stations										
Palokas (PAL)	each	-	1	1	1	2	2	2	2	3
Raja (RAJ)	each	2	2	2	2	2	2	2	2	3
Joki (JOK)	each	-	1	1	2	-	-	-	-	-
Hut (HUT)	each	-	-	-	1	1	1	2	2	-
Rumaj (RUM)	each	-	-	-	-	-	-	-	1	1
Total	**each**	**2**	**4**	**4**	**6**	**5**	**5**	**6**	**7**	**7**
Secondary Pumps										
Palokas (PAL)	each	-	10	10	10	10	10	10	10	10
Raja (RAJ)	each	10	10	10	10	10	10	10	10	10
Joki (JOK)	each	-	10	10	10	-	-	-	-	-
Hut (HUT)	each	-	-	-	10	10	10	10	10	-
Rumaj (RUM)	each	-	-	-	-	-	-	-	10	10
Total	**each**	**10**	**30**	**30**	**40**	**30**	**30**	**30**	**40**	**30**

16.11.5 Material Balance and Backfill Schedule

The material balance considers the filling strategy presented in Table 16-9, whereby Joki is backfilled using rock fill and CRF, and the remaining deposits are backfilled with a pastefill. Where pastefill is used, a pad of 0.5 m thickness is allowed for to enable traffic to pass on top of the fill.

Figure 16-36 presents the waste rock balance for the Rajapalot project. There is sufficient waste rock to fill the Joki mine and to construct waste rock pads on top of the pastefill.

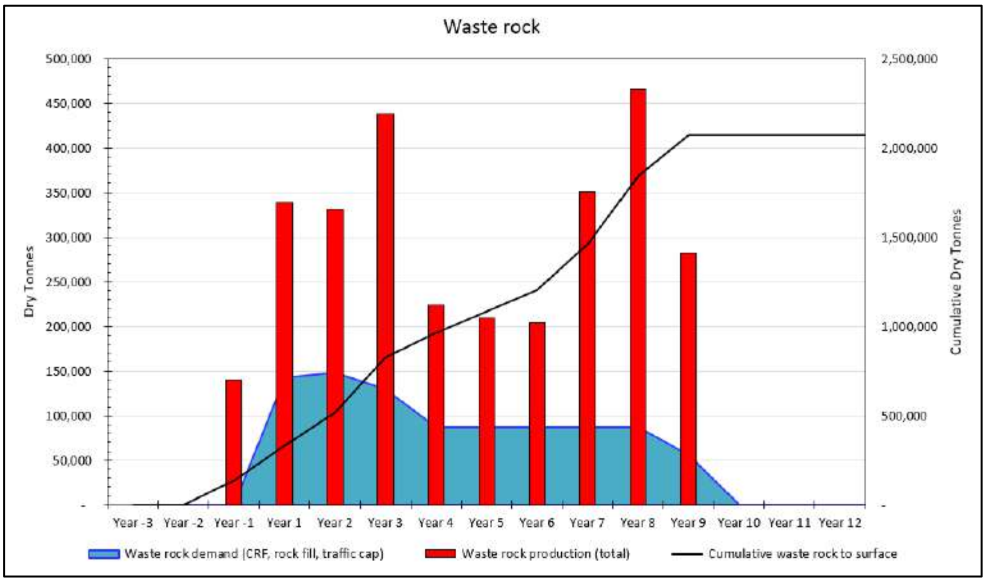


Figure 16-36: Waste Rock Balance

Pastefill comprises tailings, water and a cement binder. For the purposes of cost estimation, two strength cases are considered with 4% binder used for 'standard' fill and 8% binder applied in the sill pillars. The assessment of Figure 16-37 shows there are sufficient tailings to make a pastefill and that approximately 6 Mt (dry) of tailings will report to the surface storage facility during the life of the operation.

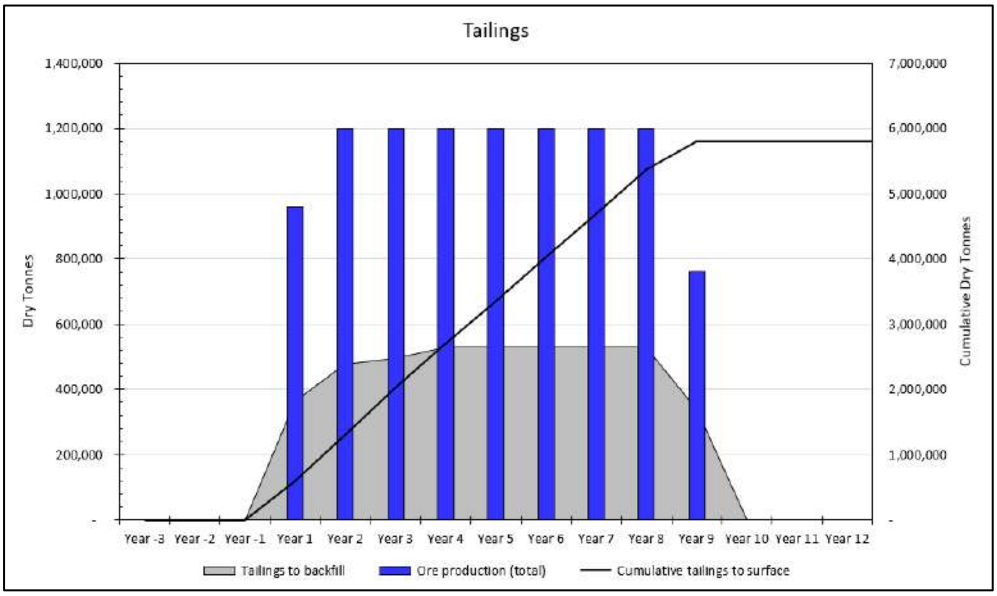


Figure 16-37: Tailings Material Balance

16.12 Mine Ventilation

The main ventilation layout for each underground mine at the Rajapalot Project has a similar approach which considers a decline access ramp which will work as the fresh intake airway which will be extended from the surface to the base of the orebody. A raise extending also from the surface to the base of the orebody will be used as an exhaust airway and will incorporate an escapeway which connects every production level through the access ramp.

A forcing fan will be installed on the top of the exhaust raise. This fan will produce fresh air moving from the surface through the decline to the lower available production level and then, the contaminated air will travel up through the exhaust raises to the surface. Once a level is finished, a Bulkhead can be installed to limit the leakage and recirculation in the level mining zone. An escapeway door must be installed in each escapeway access to avoid recirculation and only permit personnel access. The individual ventilation and escapeway raise designs are as follows:

- Palokas mine (Figure 16-38).
- Raja mine (Figure 16-39).
- Joki mine (Figure 16-40).
- The Hut mine (Figure 16-41).
- Rumaj mine (Figure 16-42).

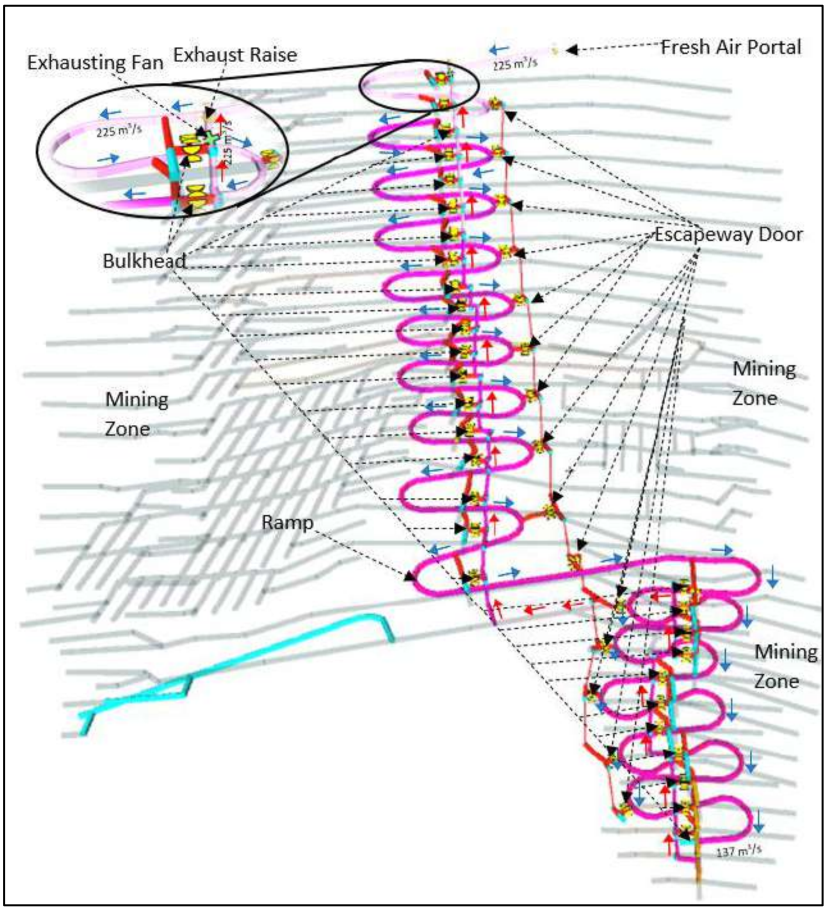


Figure 16-38: Overall Palokas Ventilation System Layout

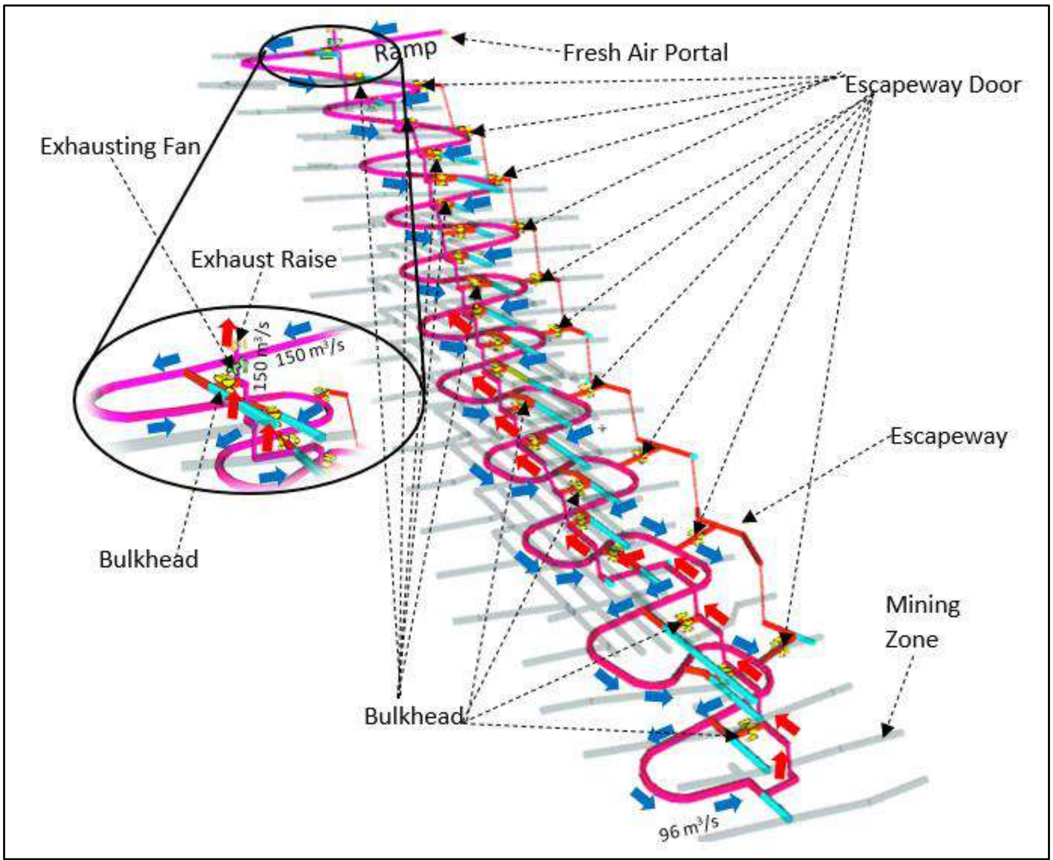


Figure 16-39: Overall Raja Ventilation System Layout

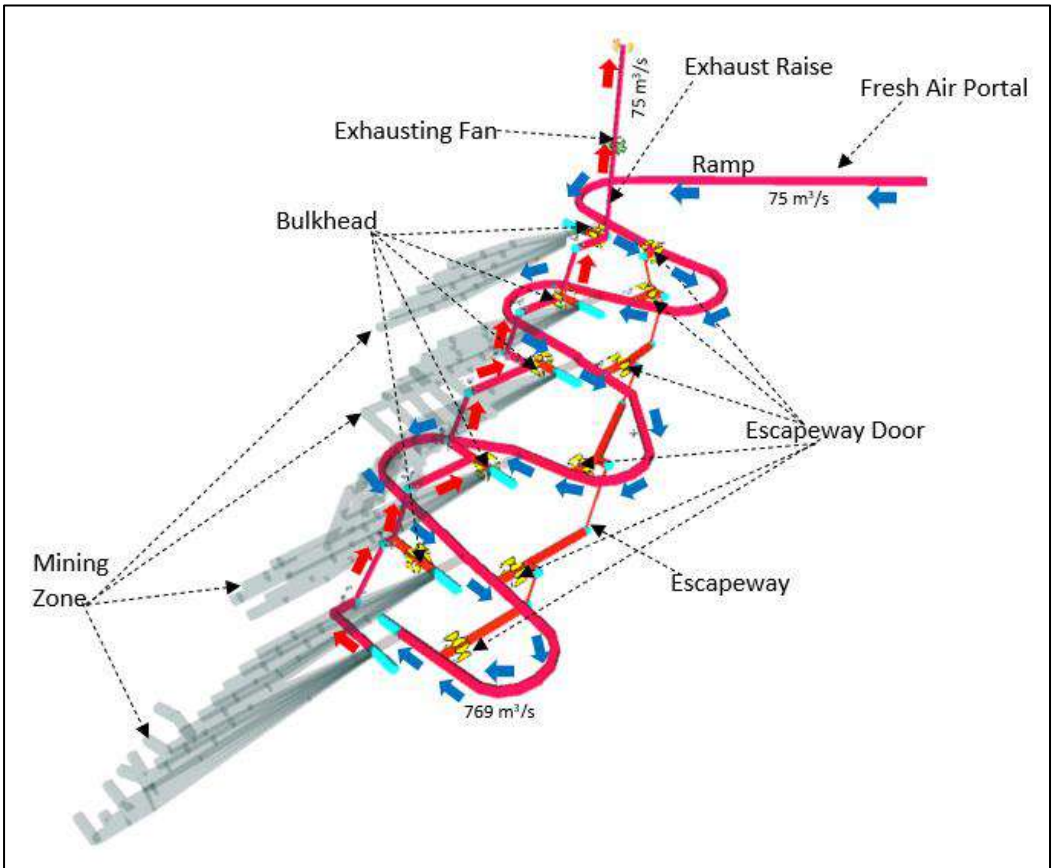


Figure 16-40: Overall Joki Ventilation System Layout

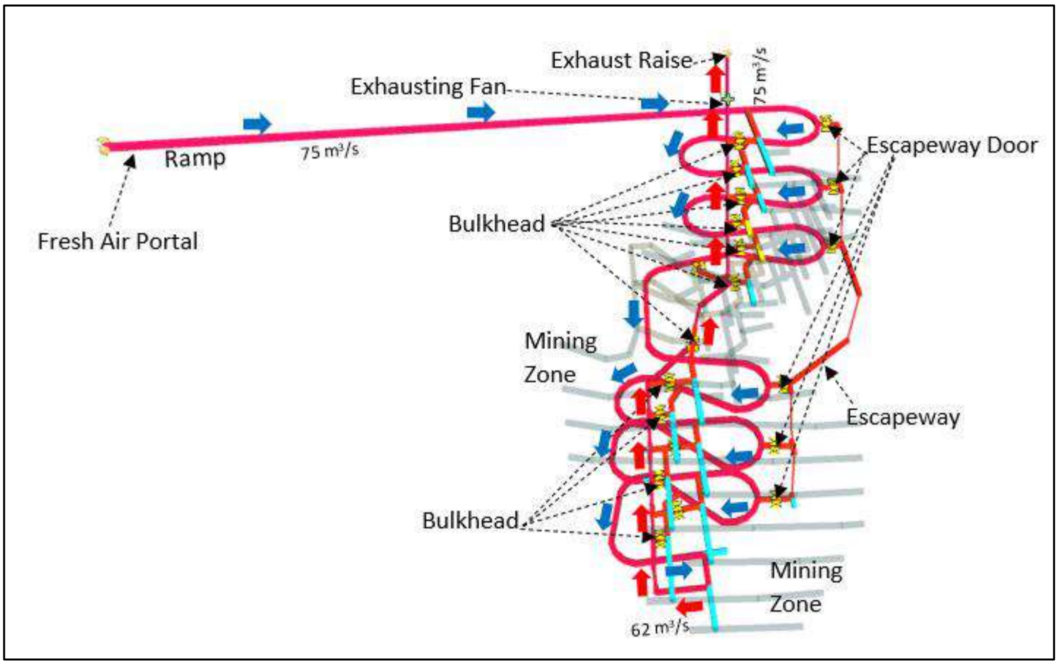


Figure 16-41: Overall The Hut Ventilation System Layout

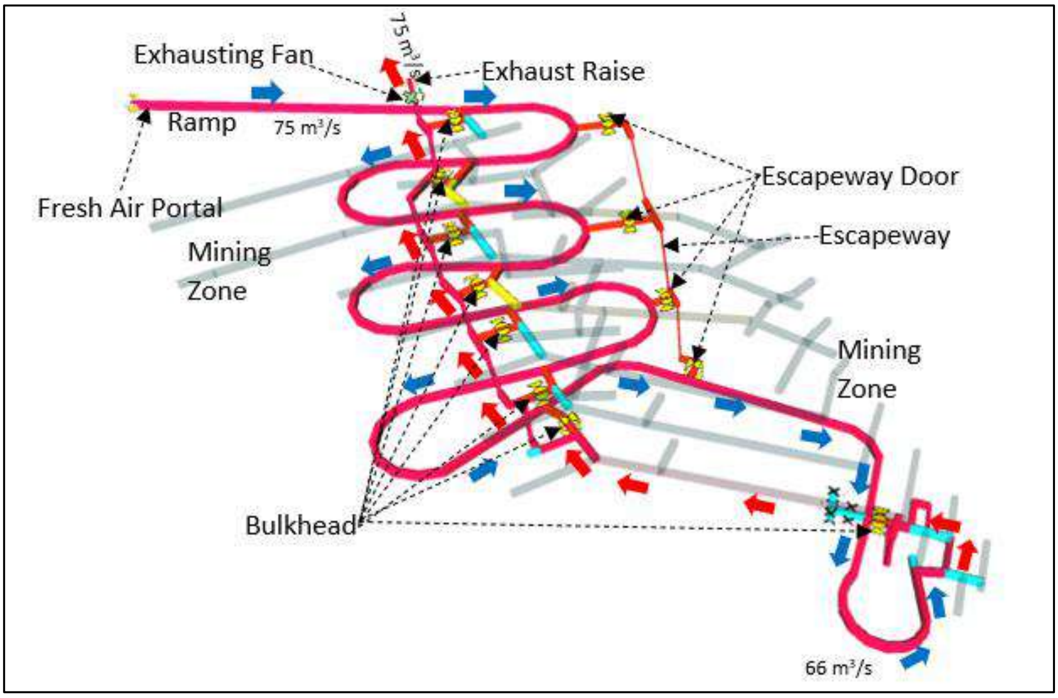


Figure 16-42: Overall Rumaj Ventilation System Layout

16.12.1 Airflow Requirement

The maximum airflow requirement in all Rajapalot's mines is estimated applying the following steps:

- The overall power required for all equipment over the LoM is summed based upon the project equipment fleet. The airflow requirement for each piece of equipment is calculated and then multiplied by the "equipment utilization factor" as shown in Table 16-21.

- Total production target (tonnes) per year is summed for each modelled year as shown in Table 16-22. The power produced per sector is obtained multiplying the kW/t by the partial tonnes production of each mining sector (Table 16-23), the results are shown for each year in Table 16-24.
- The required basic airflow for each mining zone is identified by multiplying the kW/year by the standard diesel dilution factor (0.06 m3/s/kW), selecting the higher result in the LoM, and applying a 10% leakage factor shows the area airflow requirement as identified in Table 16-25.

Table 16-21: Equipment Power Requirements

Diesel Equipment Input Criteria Fleet	Power (kW)	Utilization Factor %	Factored Power (kW)
Twin Boom Jumbo	198	10	20
Development Loader - 17t	336	75	252
Production Loader - 17t	336	75	252
Longhole Drill	158	10	16
Truck - 50t capacity	515	50	257
Chargeup wagon	158	50	79
Cemented Pastefill Carrier - Agitator	155	75	116
Grader	135	50	67
Service (Fuel/Lube) Truck	120	50	60
Integrated Toolcarrier	140	50	70
Grade Control/Probe Drill	110	10	11
Light Vehicle	150	10	15
Personnel carrier	120	25	30

Table 16-22: Total Power Requirement and kW/t Factor

Parameter	Units	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
Total Power Required (All Mines	kW	4,907	5,548	5,532	4,819	4,749	5,055	5,780	5,785	4,257
Total Produced	kW/t	0.0051	0.0046	0.0044	0.0040	0.0040	0.0042	0.0045	0.0048	0.0035

Table 16-23: Partial Production per Sector

Sector	Units	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
Palokas (PAL)	kt ore	675	675	675	675	675	675	675	675	675
Raja (RAJ)	kt ore	135	375	375	375	375	375	375	375	15
Joki (JOK)	kt ore	150	150	150						
Hut (HUT)	kt ore			51	150	150	150	68		
Rumaj (RUM)	kt ore							82	150	73
Total Production Target	**kt ore**	**960**	**1,200**	**1,200**	**1,200**	**1,200**	**1,200**	**1,200**	**1,200**	**761**

Table 16-24: Power Produced per Sector

Power Required per Mine	Unit	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
Palokas (PAL)	kW	3,450	3,121	2,984	2,710	2,671	2,843	3,043	3,254	2,394
Raja (RAJ)	kW	690	1,734	1,658	1,506	1,484	1,580	1,690	1,808	1,330
Joki (JOK)	kW	767	693	663						
Hut (HUT)	kW			227	602	594	632	676		
Rumaj (RUM)	kW							370	723	532

Table 16-25: Maximum Air Requirement per Sector

Mine	Airflow Quantity (m^3/s)									
	Max	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
Palokas (PAL)	**225**	207	187	179	163	160	171	183	195	144
Raja (RAJ)	**150**	41	104	99	90	89	95	101	108	80
Joki (JOK)	**75**	46	42	40						
Hut (HUT)	**75**			14	36	36	38	41		
Rumaj (RUM)	**75**							22	43	32

16.12.2 Ventilation Model

A basic ventilation model was developed for each sector to establish the operating duty of the main fans. The ventilation model was developed with the VentSIM™ simulation program. The resistance values for the branches in the ventilation model were calculated using standard friction factors and planned airway geometry. Shock losses at significant bends and transitions were incorporated into the model. The general development profiles and airway dimensions are identified in Table 16-7.

Fixed resistances (doors and bulkheads) are also required in specific areas of each mine sectors to isolates places as shown in Figure 16-43; The bulkheads ("Good Doors" or seals in VentSim) are required between the exhaust raises system and production levels to avoid recirculation and let the fresh air travel down to the lowest level available. A resistance value of 250 Ns^2/m^8 was used for each bulkhead; The escapeway access doors (personnel) are required to separate the access ramp to the escapeway route in order to avoid, in emergency cases, the contaminated air going into the escape raises. A resistance value of 5 Ns^2/m^8 was used for each door; The level doors are used in specific places when there are cross tunnels and when access is required into a production level while it is incorporated as an exhaust. A resistance value of 20 Ns^2/m^8 was used for each of these doors.

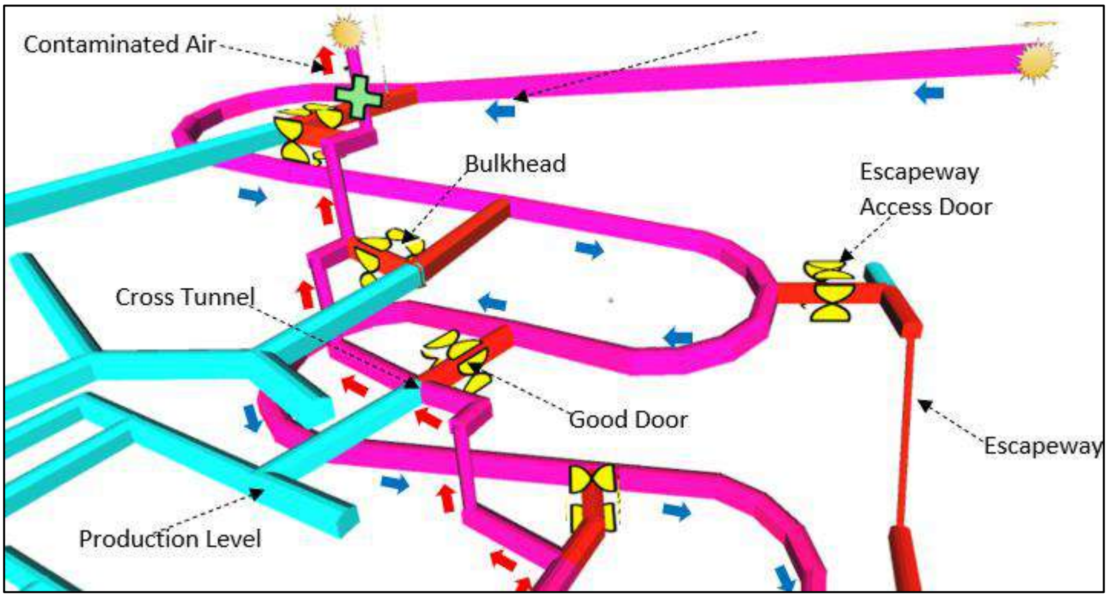


Figure 16-43: Main Disposition of Resistances

Based on the ventilation model, one exhausting fan would be required in each mine to supply the airflow requirement per zone (Table 16-25). It is assumed that primary fan systems would be installed at the top of the exhaust raises. The main fan parameters are shown in the Table 16-26.

Table 16-26: Main Fan Parameters

Mine Sector	Airflow (m^3/s)	Density (kg/m^3)	Total Pressure (kPa)	Motor Power (kW)
Palokas	225	1.14	4.20	1,259
Raja	150	1.14	2.55	511
Joki	75	1.16	1.34	134
Hut	75	1.15	1.17	117
Rumaj	75	1.15	1.09	109

16.12.3 General Auxiliary Ventilation

A ‘Production Fleet’ comprising one truck (50 t capacity) and one loader (LHD) (17 t capacity) is used to estimate an ‘Air Requirement per Production Fleet’ which considers a 50% of truck utilization and a 50% of LHD utilization. Applying these utilization factors to the power requirement of equipment shown in Table 16-21, using the standard diesel dilution factor, adding the results and considering a 10% of duct leakage, it determined each basic production level would require approximately 28 m^3/s to support the ‘Production Fleet’ as shown in Table 16-27.

Table 16-27: Air Requirements for Production Fleet

Equipment	Power (kW)	Utilization	Real Power Utilization (kW)	Air Req (m^3/s)
Production Loader - 17t	336	50%	168	10.1
Truck - 50t capacity	515	50%	257	15.5
TOTAL	**851**		**425**	**28.1**

The auxiliary ventilation generally is estimated for delivering the ‘Air Requirement per Production Fleet’ of 28 m^3/s. A 1.3 m diameter flexible duct system was assumed for the systems. It assumes a maximum of three drives been ventilated at the same time by one auxiliary fan.

Every level auxiliary ventilation system will draw airflow directly from the fresh air access ramps and working levels will be closed/sealed on completion to avoid the recirculation.

16.12.4 Air Heating

The project site in Finland typically experiences freezing temperatures during the months of November to March. Typical weather station values were used to identify the maximum heater duty point which is based on a 99% low temperature (-17.2°C) providing heating to +2°C, and an average power consumption based on the average temperatures during the colder months.

16.13 Hydrological Review

16.13.1 Introduction

Hydrological reviews are frequently loosely framed along the lines of the classical hydraulic cycle, describing firstly the local climate, then the surface water features, run-off and recharge, the groundwater environment and culminating in discharges to streams and rivers at the distal end of the catchment (Figure 16-44). The layout of this section is modelled on that format with a review of existing climatic, surface and groundwater data relevant to the project. This information is used to develop a preliminary conceptual hydrogeological model (CHM) and highlight any remaining gaps in understanding that relate to the water environment. Since these gaps at the project site are significant, there follows a final section dedicated to benchmarking similar mine sites in the Nordic region and globally to obtain parameters that can be deployed to build and calibrate the numerical groundwater model. The latter is described in Section 16.15 of this report and is used for estimating inflows to the Rajapalot mine plans and to establish the likely area of influence exerted by mine dewatering on the surface and near surface environment.

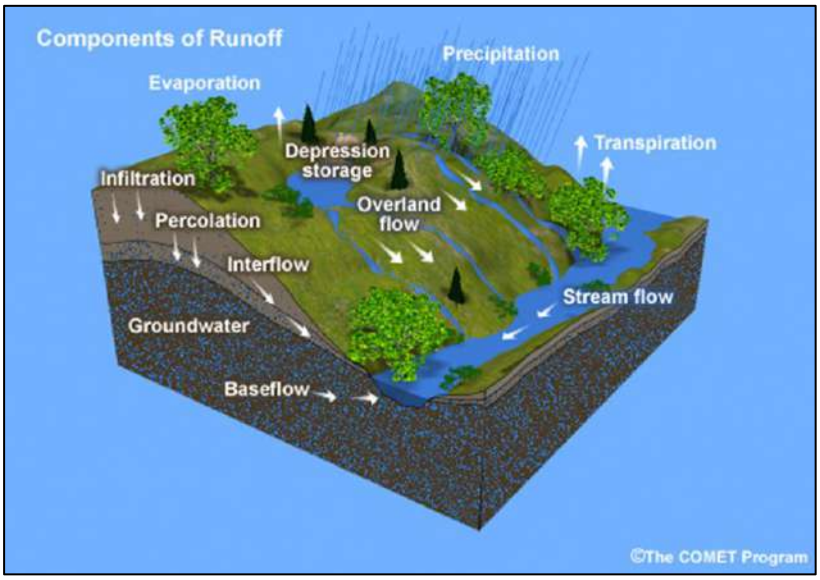


Figure 16-44: The Hydrological Cycle

16.13.2 Climate

Climate records for the Rajapalot project are comprehensive nationally and locally and are generally of good quality. There are several weather stations in the area, including one for the project. The station with the longest and most complete record is Meltosjarvi (1964 to 2022) some 16 km NW of the project and this has therefore been used to make observations about climate across the whole catchment.

Based on the Meltosjarvi station, mean annual precipitation in the Rajapalot area is 543 mm, with the driest period typically in February and March, during which minima can range between 1 and 10 mm, and the wettest in September and October when rainfall can peak between 100 and 160 mm (Figure 16-45).

Monthly average temperatures range between -12°C in January and 17°C in July. The spring thaw (freshet) is usually in early to mid-April but can sometimes extend into May. During this brief period, the bulk of precipitation that has been locked up as snow during the winter is released giving rise to peaks in stream and river flow and in recharge to the underlying sedimentary and hard-rock formations.

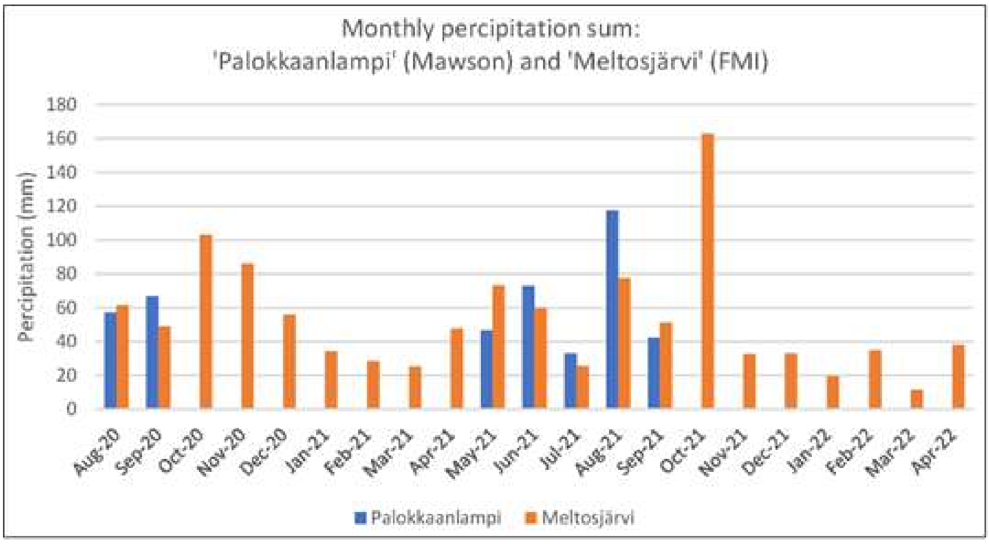


Figure 16-45: Monthly Average Precipitation for the Palokkaanlampi and Meltosjarvi Weather Stations

16.13.3 Surface Water

The Project resides near the top of the Kemijoki catchment, near the watershed divide and the adjoining Torniojoki basin to the west. The landscape is generally flat to gently undulating with more elevated ground covered in tree plantations, and lower ground by peat-filled depressions, aapa mires, lakes and streams. Run-off is to the SE in the direction of the Kemijoki River (Figure 16-46).

Data on the distribution of surface water features are good and there are some gauging station records on local stream and river flows. A flow gauging station exists on the nearby Palokkaanjoki stream where discharges range from less than 50 L/s up to approximately 800 L/s. These records started in August 2020 but are incomplete.

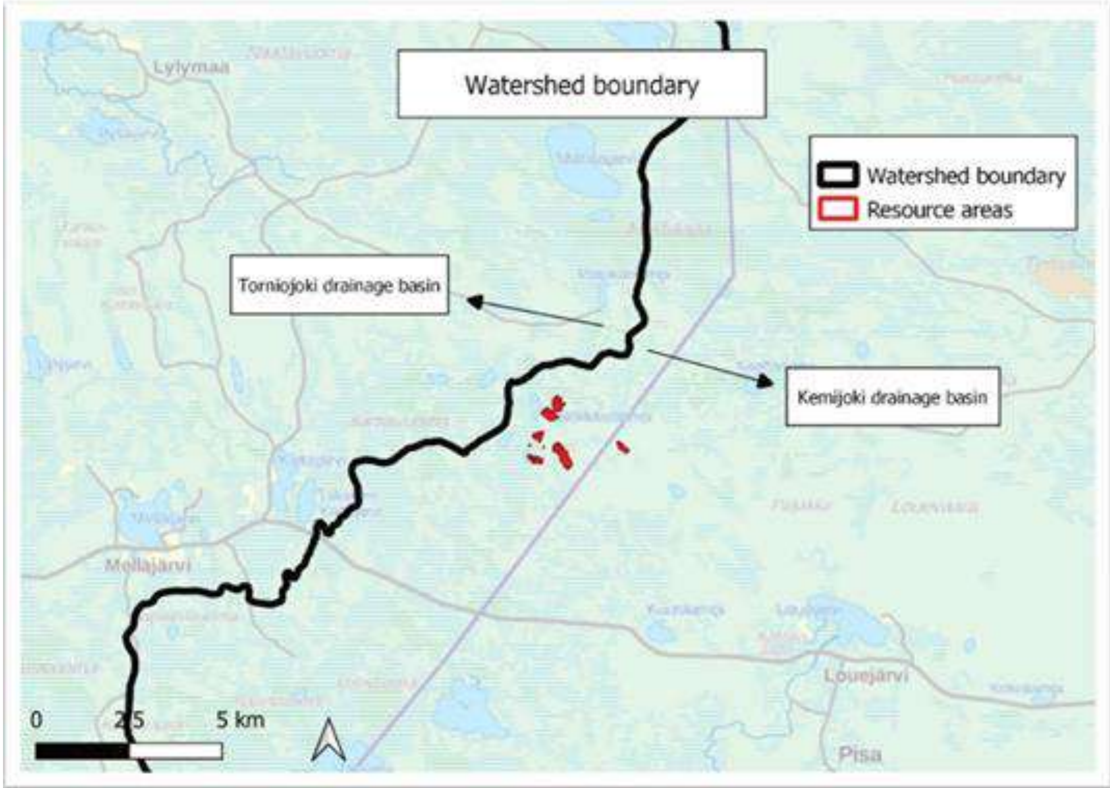


Figure 16-46: Position of project resource areas (shown in red) in relation to the watershed boundary and regionally important catchments

16.13.4 Geohydrology

Introduction

Site specific information on the groundwater regime at Rajapalot is very sparse, being limited for the most part to recent records of water levels in boreholes drilled by the Mawson contractors. The lithology and structure, particularly the hard rock, is much better understood and so something of the hydrogeological characteristics of the site can be inferred from the geology.

Peat and Glacial Sediments (Superficial Cover)

Based on the AFRY study (2021), most of the project area is covered by a veneer of glacial sediment and peat, with the latter being most developed in the topographic depressions and only the highest ground having bedrock exposure at surface. Overall, the sediment cover is quite thin, mostly less than 5 m, but attaining a maximum thickness of about 8 m (Figure 16-47). Finer grained silts and sands predominate in what has been described as a diamicton, with only the occasional outcrop of esker moraine containing gravels and coarse sands.

Based on these observations, it is likely that the cover sediments have a moderate to low hydraulic conductivity (K) typical of a heterogeneous till with a silt, fine sand and clay mix, except where moraine is present, in which case the K could be one or two orders of magnitude higher. According to Kruseman and de Ridder (1994), the K of glacial till ranges between 10^{-8} m/s and 10^{-6} m/s and the K of fine sand to gravel mixes ranges between 10^{-5} m/s and 10^{-4} m/s.

In terms of recharge potential to the underlying rock mass, it would be expected that this would be highest in areas of bedrock outcrop and moraine and lowest where the accumulations of peat and glacial till are at their thickest.

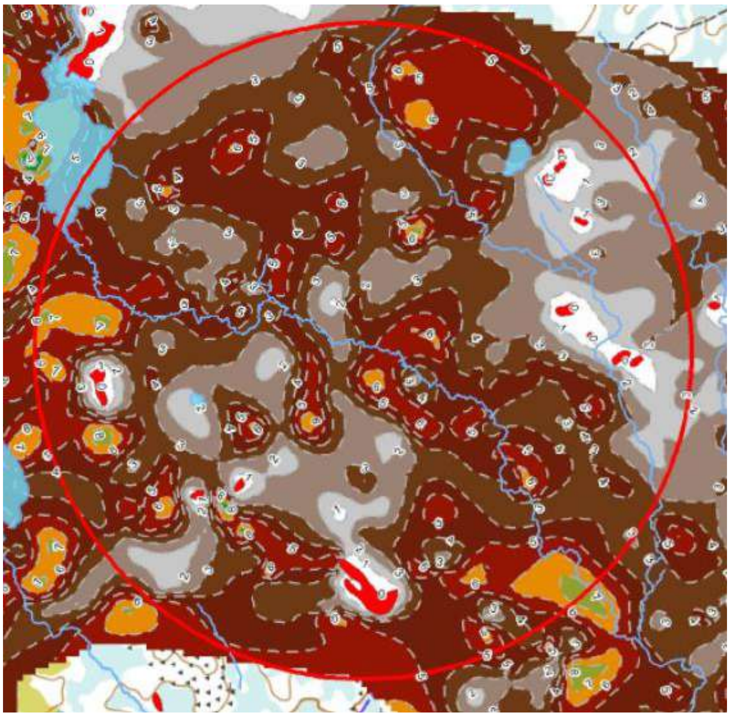


Figure 16-47: Soil cover thickness (in metres) across project area (Source: AFRY, 2021)

Bedrock

The Rajapalot deposit is in the Perapohja Belt, one of two such belts in the Lapland region, the other being the Central Lapland Greenstone Belt (CLGB) to the north. The bedrock comprises a mix of siliciclastic metasediments, carbonates and mafic volcanics, with quartz muscovite present in both the hanging wall and footwall of the deposit. The area has undergone amphibolite grade metamorphism with strong folding and thrusting. The main fold hinges and cigar-shaped ore lenses plunge about 40° to the NW.

According to Mawson geologists, the core recovered from the exploration programme reveals only a thin weathering profile at the surface and that the rock mass is otherwise quite fresh and compact. SRK also inspected a selection of core during the visit to Rovaniemi and noted there was little sign of heavy jointing or faulting. Although this still needs to be confirmed through in-situ hydrogeological testing at the project site, the general tightness of the rock suggests that it has low storage (S) and K properties.

Piezometry of the Superficial Cover and Bedrock

Groundwater monitoring in exploration holes has been in place since 2020 and there are water level sensors in a few of such holes that have provided a continuous hydrographic record.

Figure 16-48 shows results from one of Mawson's monitoring stations at the Raja site. The construction of the Raja well is not known, but it is assumed to be open (non-discrete) like the other exploration holes. Nevertheless, the hydrograph reveals a pattern of water level change that is common to many sites in high latitude, sub-arctic and arctic settings with peak levels during the spring freshet and again towards the end of the summer. The peak level in spring is in response to the unlocking of precipitation that has accumulated during the winter, and which is released through melting over a brief 3 to 4 week period in April and May. The second peak occurs in the months when rainfall is at its highest in Lapland. The absence of rainfall during the winter means that recharge is negligible and so, in response the hydrograph declines steadily until the following spring.

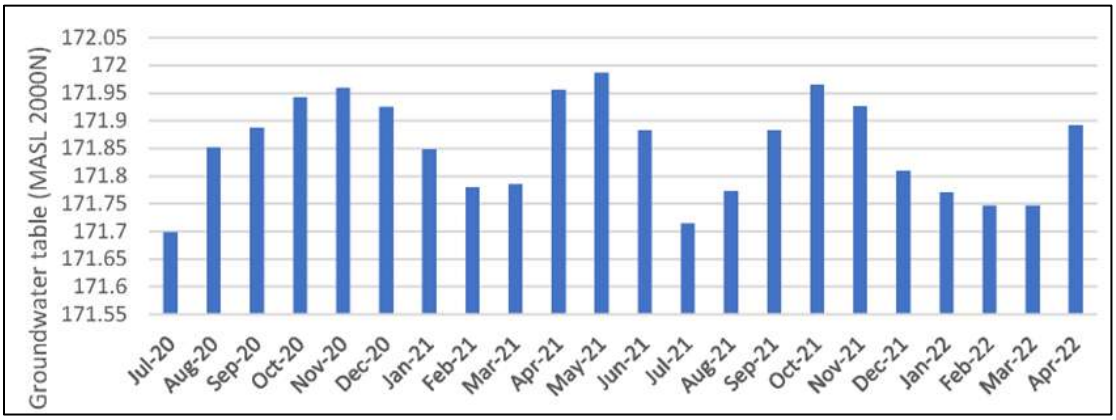


Figure 16-48: Monthly average groundwater table for the monitoring station at Raja

SRK notes that water levels in the Raja hole and more generally across the catchment are close to ground surface and have limited fluctuation. This would suggest, in keeping with observations of the core, that the rock has low K and a poor hydraulic connectivity.

16.13.5 Conceptual Hydrogeological Model

Introduction

Groundwater and surface water are the primary pathways between potential sources and receptors, and therefore it is important to develop a CHM of the local regime, including the interaction between groundwater and surface water. This model considers the 'ambient' pre-mining condition and then assesses what changes are a likely consequence of the mining, both during operation and after closure.

Summary of Hydrostratigraphic Units and Ambient Pre-mining Condition

The hydrological review has revealed that there are several hydrogeologically-distinct formations, referred to as hydrostratigraphic units at the Rajapalot site, the properties of which are important in governing the way in which the local surface and groundwater regime behaves. These units are broadly defined on lithological grounds as the peat (Unit 1), the glacial sediment (Unit 2), the weathered and fractured bedrock (Unit 3) and the fresh bedrock (Unit 4).

From experience, the peat hydrostratigraphic unit tends to have physical properties and behaviour that mean it is at least partially isolated from the underlying glacial sediments. This is particularly so in thicker sequences of peat where the high levels of humification towards the base of the unit, also referred to as the catotelm, mean that it has a very low K. Year-round ponding of water in aapa mires and local groundwater perching above the peat are symptomatic of this condition.

The glacial deposits that underlie the mires and outcrop at the surface beyond their fringes are spatially heterogeneous with till and moraine, although the till (commonly also referred to as diamicton) is the predominant material. As a unit these deposits will tend to drain quite poorly except where stringers of moraine (esker) allow rapid, free draining to occur.

The bedrock beneath the sediments comprises both weathered and fresh bedrock hydrostratigraphic units. The former is, according to the Mawson geologists quite thin in the project area. It is likely to be only weakly connected in hydrogeological terms to the surface through the superficial sediments, but directly in areas of higher ground that are free of superficial cover. It is probable that much of the recharge to the bedrock will occur at these locations where there is little or no sediment present. The fresh bedrock is largely intact, so water only has the potential to be conveyed via open faults that exist in the rock mass. Since well-developed structures appear to be uncommon at the project site, it is likely that the rock mass is poorly connected with low permeability and storage characteristics.

In terms of current groundwater conditions, it is very probable that the bulk of groundwater flow will be occurring in the sediment cover and the top few metres of bedrock where the weathering and fracturing are most intense. The groundwater gradient is shallow, in keeping with the local topography and flow will be to the SE in the direction of the Kemijoki River. The fresh bedrock in which the ore bodies mostly reside is much tighter and therefore the passage of groundwater will be imperceptible except where the occasional, better connected and more permeable (K) fracture exists.

Operational, Closure and Post-closure Conditions

Inflows to the mines are expected to be most elevated in the early stages of mine life during the initial excavation of the spiral declines when they pass through the more permeable weathered zone and the storage of groundwater in the rock mass is at its highest.

When mining the ore bodies, stopes, internal declines and headings may occasionally cross more open brittle faults. The permeability of these structures is likely to be low, but their very heterogeneous characteristics will mean that occasional high, short-lived inflows can be expected.

Dewatering of the mine will create a halo of lower piezometric pressure around the excavated void with the more pronounced drawdown propagated along faults that intersect the workings; however, the water management design adopted for the operation should be one that aims to prevent, as far as is possible, any substantial inflows to the mines, since it is imperative that drawdown produced by the operation does not project in any significant way to the surface. Hence, the conceptual design for this study includes advance grouting of all geological structures and potentially the upper-most workings where they approach the base of the weathered zone. This groundwater will eventually collect in the sump of the main pumping stations for the underground operations and be removed to surface for treatment and use in the mine water circuit.

After closure and once the pumps have been switched-off, the water table in the mine is expected to rebound. The decommissioning process will likely entail the backfilling of remaining voids in the orebody and the installation of structures that limit decant at surface. Given that the mine workings will be wholly within the fresh bedrock, it is probable that the rebound will be slow. Since the portal and other entry points to the mine will be positioned above the prevailing water table, it is expected that rebound water will not flow at the surface.

16.13.6 Summary

Monitoring of the surface water environment is already reasonably good and therefore knowledge of the shallow water dynamics is considerably more advanced than for the underlying sediments and the bedrock that hosts the orebody.

Recommendations on how to address the gaps in understanding regarding the groundwater environment are supplied at the end of this study; however, for the purposes of this PEA, it is still important to define likely inflows to the future operation and this necessitates an alternative strategy, one that uses analogues based on SRK's direct experience of hydrogeological testing in Lapland, coupled with the development of a benchmarking study to develop a database of mine inflows to underground operations both within the Nordic region and across the globe. These will be compared to the Rajapalot plans and used to help define and constrain a likely range of inflows to the future operation. The benchmarking study is described in the following section.

16.13.7 Benchmarking Studies

Hydrogeological Parameters

The fractured crystalline igneous and metamorphic lithologies present at Rajapalot have many physical traits in common with other greenstone belt sites in Northern Finland and it can therefore be argued that the hydrogeological characteristics are likely to be similar. Hence, in the absence of site specific data on the bedrock at the project site, SRK has chosen to draw on its experience of testing at other Lapland sites and its [1]database of hydrogeological results from such projects to fashion a synthetic data set appropriate to the geology at Rajapalot. This is summarised in its most relatable form in a chart of K versus depth (Figure 16-49).

Mine Inflows

SRK has undertaken some additional research to benchmark inflows to underground mining operations around the world. This involved researching public records and reports, the internet and [2]SRK's own database of mining projects. Since mines vary considerably in terms of the scale and physical extent of the operation, it is essential to associate inflow rate in some way with the size of the mine. As such, SRK decided to use ore production rate as an analogue for size. The chart in Figure 16-50 presents the results of SRK's research, comparing groundwater inflow rate in m^3/day with the ore extraction rate in t/day. With one exception, the Nordic mines

[1] Test results from other project sites are frequently confidential, so the dataset provided for this study has been anonymised to respect data protection requirements.

[2] Most records of mine inflows used in this study were obtained from the public domain through company annual reports, ESIAs, NI 43-101 etc., but the occasional source was confidential, in which the case the mine concerned was un-named.

recorded by this study populate the left hand side of the x-axis, although ore production rates plot fairly centrally within the collection of mines presented in the chart. This would suggest that inflows to Nordic mines, which are predominantly in crystalline bedrock, would generally be expected to be comparatively low for any given size of operation. In the case of Rajapalot, the ‘base case’ ore production rate will be 1.4 Mtpa and given the consensus view regarding the compactness and the rarity of jointing and fracturing within the bedrock at the project site, SRK has concluded that the design inflow to the future underground mines should plot centrally within the assemblage of Nordic operations, with a water to ore ratio just below 1:1. This would mean a most likely steady state (P50) inflow rate of 2,500 m^3/day (29 L/s), although given remaining uncertainties about the local formation hydraulic properties, the range of possible inflows considered for this study extends from 1,700 m^3/day (20 L/s) up to 4,200 m^3/day (49 L/s).

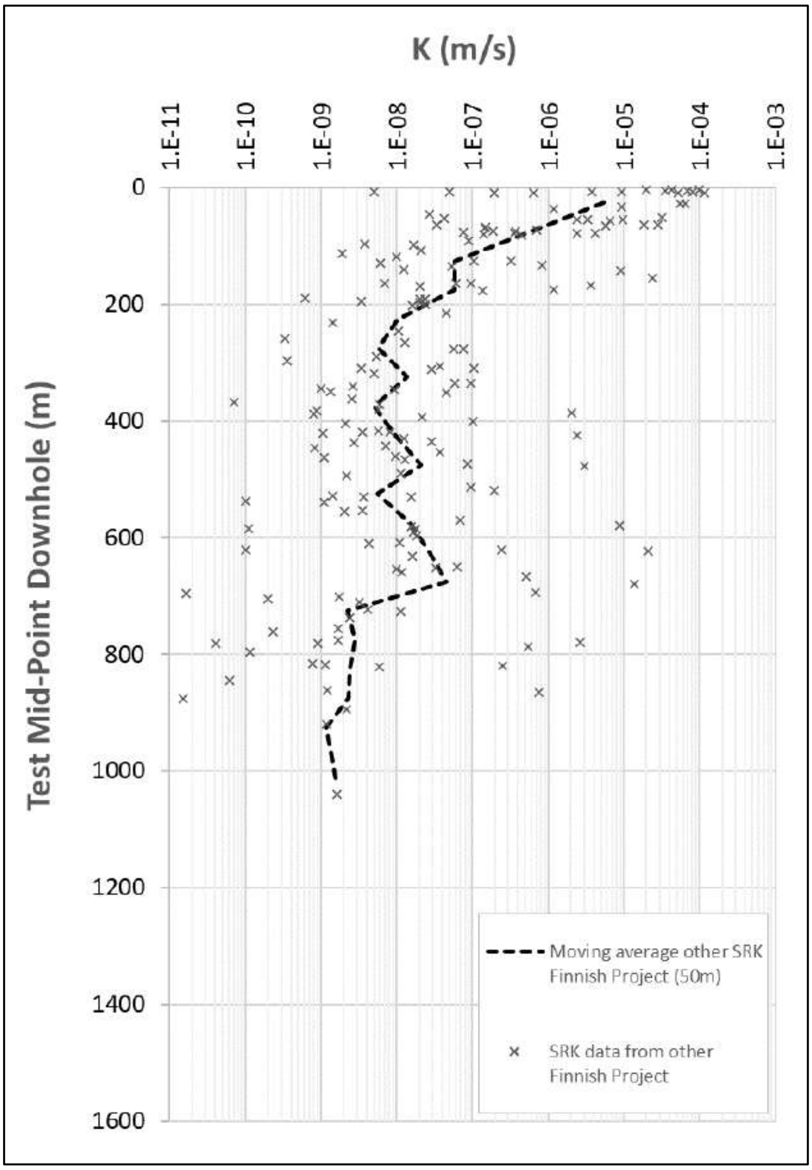


Figure 16-49: Chart of K (m/s) versus Depth (m) below ground

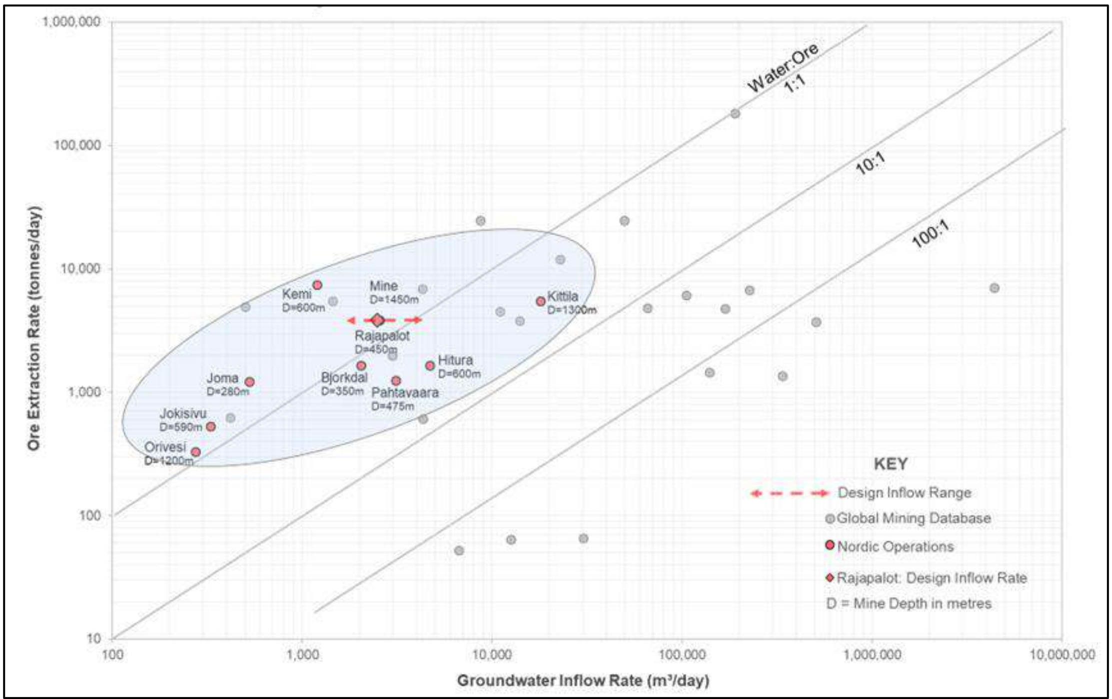


Figure 16-50: Groundwater inflow versus ore extraction rate from SRK database of current and previously operational underground mines

16.14 Groundwater Inflow Assessment

16.14.1 Introduction

Effective management of surface and groundwater in the context of a mine at the Rajapalot site is essential to facilitate access to the workings, to control and, where possible prevent ingress of water to limit pumping and treatment costs and associated impacts on the surface environment caused by derogation of water features that fall within the orbit of the mine and the cone of depression created by the dewatering operation.

The objectives of the water management section of the PEA study are to understand the likely effects of the local surface and groundwater regime on each of the five satellite operations at Rajapalot (Palokas, The Hut, Rumaärvij, Raja, Joki), to predict inflows to the operation and use the results to help develop concept-level water management infrastructure suitable for controlling such effects.

This section describes the development of a numerical groundwater model from construction and calibration through to predictive simulations with the mine infrastructure incorporated into the model mesh. The final part describes the infrastructure and practices best suited for managing water at the site using the model results and other knowledge about the local setting derived from the data review.

16.14.2 Groundwater Modelling Context and Approach

The decision was taken to use a numerical modelling code rather than a simpler analytical solution because of the extra complexity presented by having five satellites of different size, overlapping commencement dates, different periods of operation and, through their proximity,

[3]interference effects on near neighbours.

Given the stage of the project and the general paucity of hydrogeological information, SRK decided to adopt certain simplifying assumptions for the modelling. These are as follows:

- The model is steady state. The absence of transient (time variant) effects that reflect changes to the storage of water in the aquifers and aquitards caused by, for example seasonal changes in recharge, are more important in later technical stages once greater granularity is required for the mining schedule, the water balance and treatment and disposal requirements.
- The saturated formations are confined.
- Recharge to ground is the same across the whole model surface, regardless of elevation and cover material.
- The entire model perimeter has a 'no flow' boundary condition.
- The model internal borders, in this instance surface water features (streams, lakes) have a constant head equal to the elevation of the topography at their respective locations.
- All open pit walls and underground developments have constant head equal to the elevation of the proposed mines and associated with a discharge constraint that means all flow is out of and not into the model (i.e., the nodes can only behave as an exit point for water).

The software package used is the DHI finite element code called FEFLOW™, which is widely available and in common use for mining projects.

16.14.3 Model Construction

The Rajapalot mines are located approximately centrally within the model with the boundaries deliberately offset from the project by at least 10 km, mostly considerably further, in all directions. This is to ensure there are no artificial boundary effects impacting the behaviour of the groundwater regime in the environs of the mining operation. The boundaries have been located along river courses, or catchment divides both of which normally satisfy the 'no flow' assumption (Figure 16-51).

[3] Interference due to dewatering causes inflows in the affected mines to be lower because the impacted operations are drawing from the same store of water in the intervening rock mass, which results in more rapid depletion and a larger drawdown.

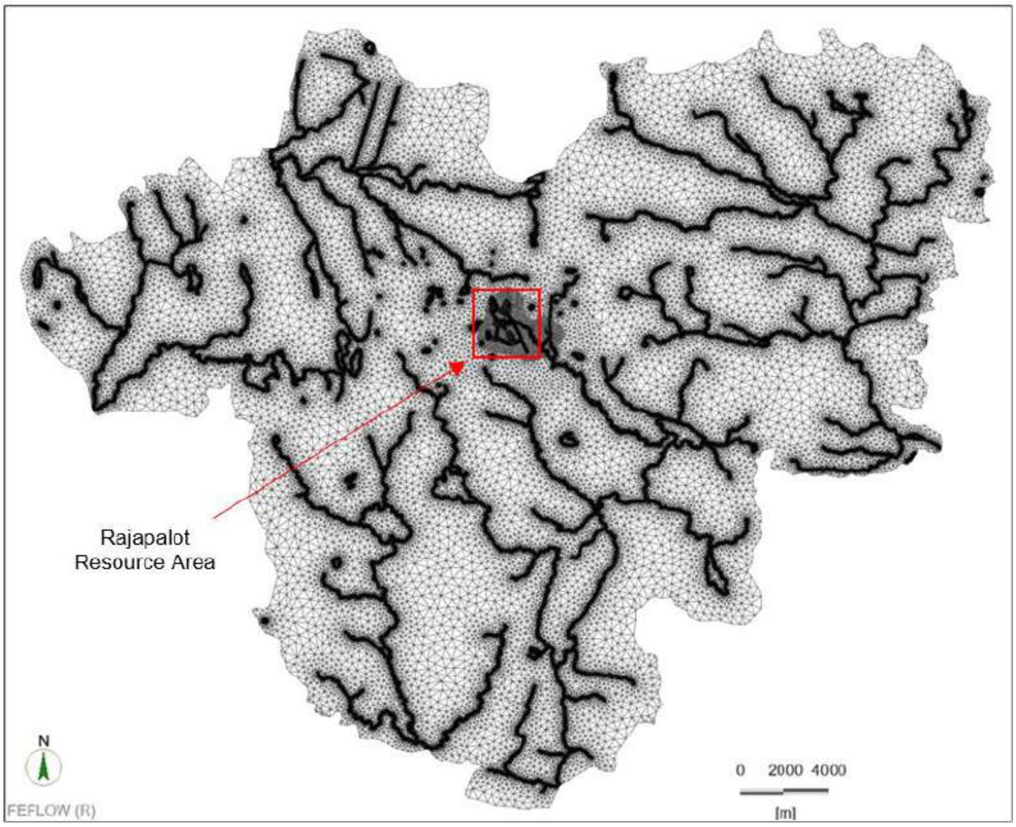


Figure 16-51: FEFLOW model extent, mesh arrangement and water courses

The model has 31 layers, the top 10 m layer representing the fluvio-glacial deposits, the next 7 layers representing the shallow bedrock zone to a depth of 175 m, in which jointing and weathering is more prevalent, and the remaining 23 layers the deep bedrock zone representing fresh, largely in-tact bedrock with sparse jointing. The model extends to a total depth of 1,500 m, which is considered more than adequate to accommodate the underground mines without incurring bottom boundary effects. Model input parameters include recharge, K and storage (S), although the latter does not influence the output when the model is run in steady state mode. Given the uncertainty over these inputs, those that affect the steady state model, namely recharge and K have been provided with a range of possible values with a most likely, 'base case' value at or close to the centre of the range. These values are summarised in Table 16-28.

Table 16-28: FEFLOW model inputs

						Model A7	Model A8	Model A9				
slice	slc dist	thck	elev	lyr	Unit	K base case R13% MAP (m/s)	K svy min R10% MAP (m/s)	K svy max R20% MAP (m/s)	Ss (1/m)	Sy (-)	OP BC nodes	UG BC nodes
1	10	10	177	1	Fluvio-glacial deposits	8.0E-06	6.2E-06	1.2E-05	5.0E-04	0.15	Yes	-
2	15	25	162	2	Shallow Bedrock Zone	9.0E-07	6.9E-07	1.4E-06	1.7E-05	0.05	Yes	-
3	25	50	137	3	Shallow Bedrock Zone	2.0E-07	1.5E-07	3.1E-07	1.7E-05	0.05	Yes	Yes
4	25	75	112	4	Shallow Bedrock Zone	6.0E-08	4.6E-08	9.2E-08	1.7E-05	0.05	Yes	Yes
5	25	100	87	5	Shallow Bedrock Zone	3.0E-08	2.3E-08	4.6E-08	1.7E-05	0.05	Yes	Yes
6	25	125	62	6	Shallow Bedrock Zone	2.0E-08	1.5E-08	3.0E-08	1.7E-05	0.05	Yes	Yes
7	25	150	37	7	Shallow Bedrock Zone	1.3E-08	9.6E-09	1.9E-08	1.7E-05	0.05	-	Yes
8	25	175	12	8	Shallow Bedrock Zone	9.5E-09	7.3E-09	1.5E-08	1.7E-05	0.05	-	Yes
9	25	200	-13	9	Deep Bedrock Zone	7.0E-09	5.4E-09	1.1E-08	1.7E-05	0.010	-	Yes
10	25	225	-38	10	Deep Bedrock Zone	6.0E-09	4.6E-09	9.2E-09	1.7E-05	0.010	-	Yes
11	25	250	-63	11	Deep Bedrock Zone	4.8E-09	3.7E-09	7.3E-09	1.7E-05	0.010	-	Yes
12	25	275	-88	12	Deep Bedrock Zone	4.0E-09	3.1E-09	6.2E-09	1.7E-05	0.010	-	Yes
13	25	300	-113	13	Deep Bedrock Zone	3.3E-09	2.5E-09	5.0E-09	1.7E-05	0.010	-	Yes
14	25	325	-138	14	Deep Bedrock Zone	2.8E-09	2.1E-09	4.2E-09	1.7E-05	0.010	-	Yes
15	25	350	-163	15	Deep Bedrock Zone	2.5E-09	1.9E-09	3.8E-09	1.7E-05	0.010	-	Yes
16	25	375	-188	16	Deep Bedrock Zone	2.3E-09	1.7E-09	3.5E-09	1.7E-05	0.010	-	Yes
17	25	400	-213	17	Deep Bedrock Zone	2.0E-09	1.5E-09	3.1E-09	1.7E-05	0.010	-	Yes
18	25	425	-238	18	Deep Bedrock Zone	1.8E-09	1.3E-09	2.7E-09	1.7E-05	0.010	-	Yes
19	25	450	-263	19	Deep Bedrock Zone	1.5E-09	1.2E-09	2.3E-09	1.7E-05	0.010	-	Yes
20	25	475	-288	20	Deep Bedrock Zone	1.4E-09	1.1E-09	2.1E-09	1.7E-05	0.010	-	Yes
21	25	500	-313	21	Deep Bedrock Zone	1.3E-09	9.6E-10	1.9E-09	1.7E-05	0.010	-	Yes
22	25	525	-338	22	Deep Bedrock Zone	1.2E-09	8.8E-10	1.8E-09	1.7E-05	0.010	-	Yes
23	25	550	-363	23	Deep Bedrock Zone	1.1E-09	8.3E-10	1.7E-09	1.7E-05	0.010	-	Yes
24	25	575	-388	24	Deep Bedrock Zone	1.0E-09	7.7E-10	1.5E-09	1.7E-05	0.010	-	Yes
25	25	600	-413	25	Deep Bedrock Zone	9.5E-10	7.3E-10	1.5E-09	1.7E-05	0.010	-	Yes
26	25	625	-438	26	Deep Bedrock Zone	8.8E-10	6.7E-10	1.3E-09	1.7E-05	0.010	-	Yes
27	25	650	-463	27	Deep Bedrock Zone	8.3E-10	6.3E-10	1.3E-09	1.7E-05	0.010	-	Yes
28	50	700	-513	28	Deep Bedrock Zone	7.5E-10	5.8E-10	1.2E-09	1.7E-05	0.010	-	-
29	100	800	-613	29	Deep Bedrock Zone	6.8E-10	5.2E-10	1.0E-09	1.7E-05	0.010	-	-
30	200	1000	-813	30	Deep Bedrock Zone	5.3E-10	4.0E-10	8.1E-10	1.7E-05	0.010	-	-
31	500	1500	-1313	31	Deep Bedrock Zone	5.0E-10	3.8E-10	7.7E-10	1.7E-05	0.010	-	-

The reduction in K with increasing depth shown in Figure 16-49 has also been replicated, although the maximum, base case and minimum trend lines have been displaced to the left of the moving average for the analogue results to reflect the tighter (lower K) nature of the rock mass which is present at the project site (Figure 16-52).

Once the model had been adjusted to make sure the groundwater levels broadly reflect those observed at the site, the various open pit and underground 3D mine designs were incorporated into the mesh (Figure 16-53). The model was then run according to the 9-year LoM schedule shown in Figure 16-54. The Palokas and Raja mines are active for the entire LoM, with overlap from the Joki mine in the first three years, The Hut mine in the next three years and Rumajärvi in the final 3.7 years.

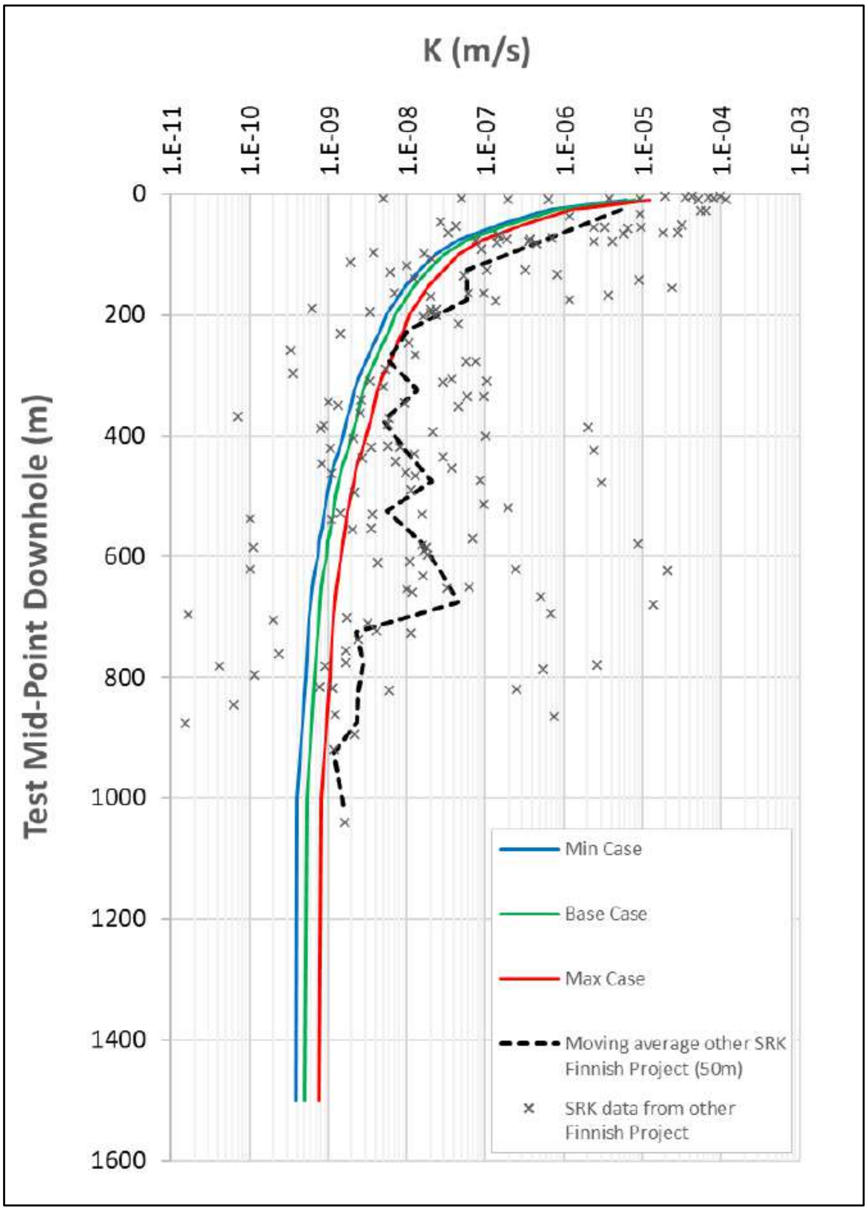


Figure 16-52: K versus Depth chart with Maximum, Base Case and Minimum K trend lines

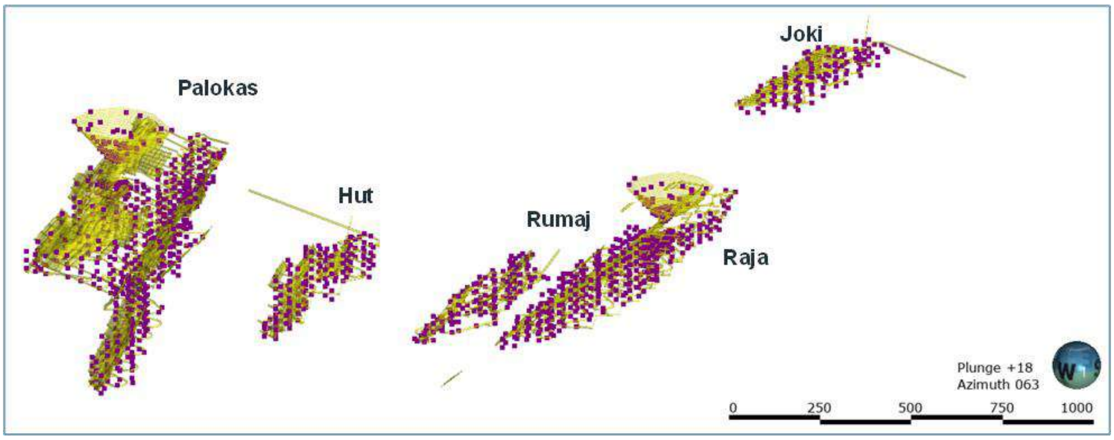


Figure 16-53: Incorporation of Mine Design into model mesh

Note: 3D tilted view of model nodes (purple square) superimposed with 3D mine designs

Mine	Years in operation	
	from (yr)	to (yr)
Palokas (PAL)	0	9.7
Raja (RAJ)	0	9.7
Joki (JOK)	0	3
Hut (HUT)	3	6
Rumaj (RUM)	6	9.7
Full LoM	**0**	**9.7**

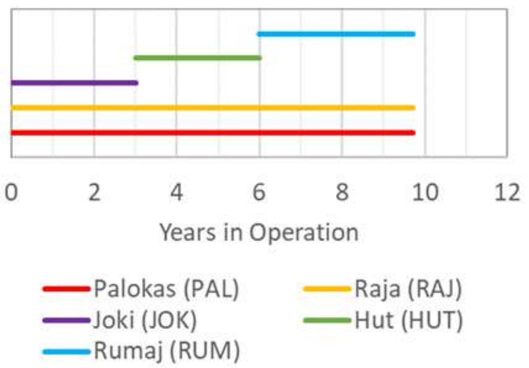


Figure 16-54: FEFLOW model schedule

16.14.4 Model Results

The ‘base case’ inflow prediction (recharge = 13%[4] MAP) for all the mine complex ranges between 2,330 m^3/day (27 L/s) and 2,753 m^3/day (32 L/s) over the course of the LoM (Table 16-29), with a minimum likely flow in the first phase when Joki is mined of 1,793 m^3/day (21 L/s) and a maximum flow of 4,236 m^3/day (49 L/s) in the last phase when Rumajärvi is mined.

Table 16-29: Mine inflow predictions for the satellite operations at Rajapalot

Years of operation	Mine Feature	Inflow Predictions (m^3/d)		
		svy min (R10%)	base case (R13%)	svy max (R20%)
0 to 3yr (PAL, RAJ, JOK)	Total	1,793	2,330	3,585
	Palokas	883	1,147	1,765
	Raja	564	734	1,129
	Joki	346	449	691
3 to 6yr (PAL, RAJ, HUT)	Total	1,900	2,470	3,800
	Palokas	881	1,145	1,762
	Raja	553	719	1,106
	Hut	466	606	933
6 to 9.7yr (PAL, RAJ, RUM)	Total	2,118	2,753	4,236
	Palokas	883	1,148	1,766
	Raja	525	682	1,050
	Rumaj	710	923	1,420

[4] Mean Annual Precipitation (“MAP”) for Rajapalot = 543 mm

Figure 16-55 shows the drawdown created by the operations at Rajapalot, in plan view (A) and in cross-section (B), respectively. The cones of drawdown are very steep, reflecting the low K character of the rock mass and the region of influence exerted by the dewatering, most evident in plan view by drawdowns of 5 to 10 m in the bedrock extending up to 400 or 500 m from the centre of each satellite operation.

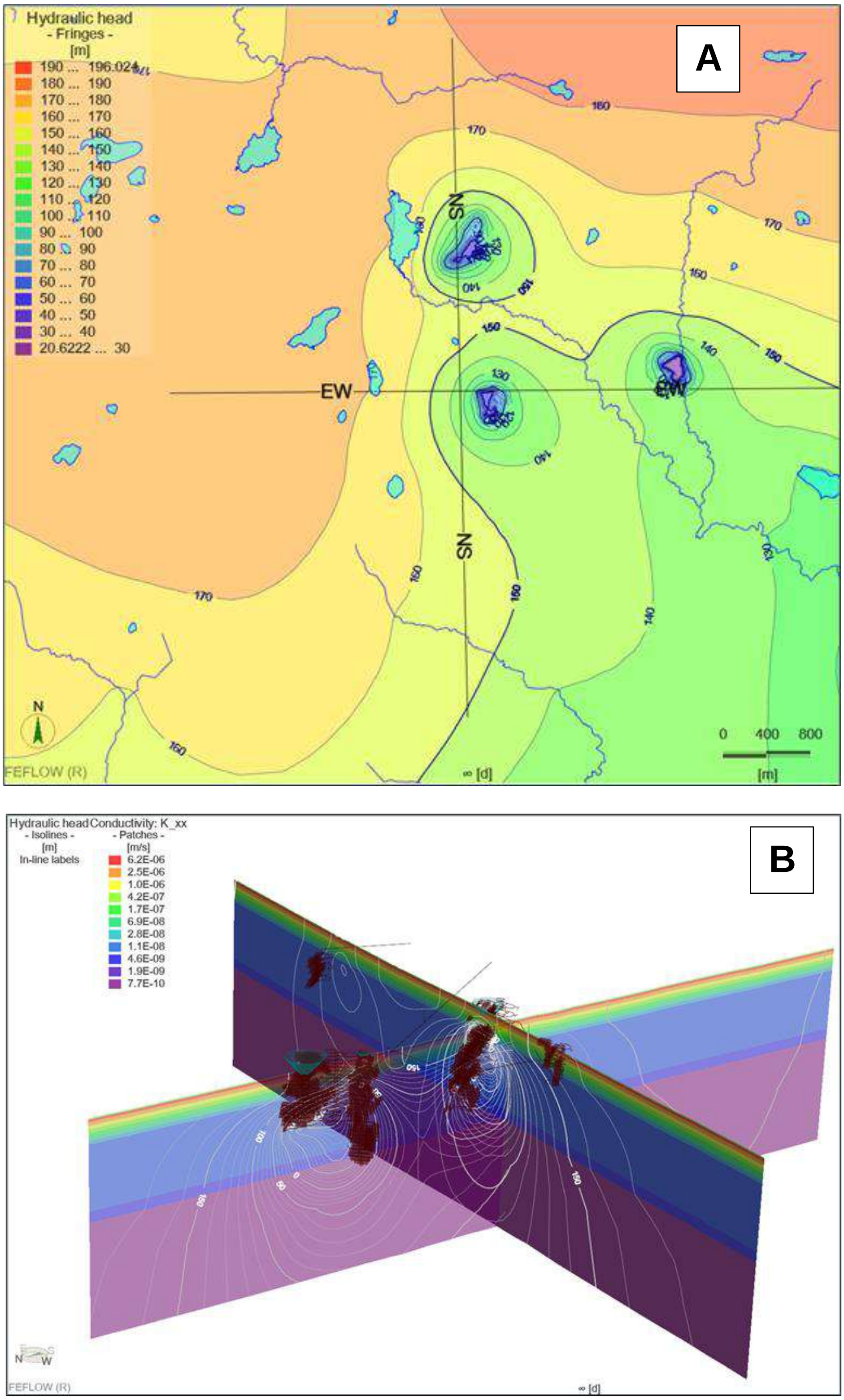


Figure 16-55: Example of modelled head predictions in plan view [A] and in cross-sections [B] along north south and east west axes

16.15 Water Management Approach

Based on the assumptions made in this study regarding the surface and groundwater regime at Rajapalot, it is likely that there will not need to be any active intervention pre-mining to advance dewater the rock mass around the underground operations. Whilst the low K and S properties of the formations that host the deposits still need to be confirmed through in situ testing, if they exist as expected, then conventional sump and pump arrangements should be adequate to dewater the underground mines. Additionally, the impact of run-off from the surrounding catchment, particularly during the spring freshet can be limited through the installation of berms and interception ditches around surface facilities. Although there appears to be little evidence of large-scale faulting at the site, the operators should allow some contingency for probe holing and advance grouting of declines and headings in case these structures are found to exist.

16.15.1 Prevention

Surface entry points for underground mines such as declines (including boxcuts), ventilation shafts and escapeways are prone to collect water. Surface flow will be mitigated.

16.15.2 Collection and Containment

Water that enters the mine will be collected and directed to central management locations using drains, boreholes, and piping which are arranged to prevent accumulation and limit fines contamination. The management facilities will have sufficient surge capacity in the event of a power outage or pump failure.

16.15.3 Mine Dewatering

Water will be directed to a settling sump(s) and the overflow of clear water to a 'clean water' sump for main line pumping to surface management facilities.

The dewatering approach has been assessed separately for the individual mines to provide an early-stage approach and preliminary estimate water inflow, primary pumping station requirements based on the depth of mining below surface over the LoM and a provision for secondary pumps (Table 16-30). Future exploration and investigation will need to collect additional geotechnical and hydrogeological data which will be used to refine the approach to dewatering and water management in future detailed studies.

Table 16-30: Mine inflow and pumping estimates for each mine

Mine Water Inflow estimate	Units	Year 01	Year 02	Year 03	Year 04	Year 05	Year 06	Year 07	Year 08	Year 09
Palokas (PAL)	m^3/day	1,147	1,147	1,147	1,145	1,145	1,145	1,148	1,148	1,148
Raja (RAJ)	m^3/day	734	734	734	719	719	719	682	682	682
Joki (JOK)	m^3/day	449	449	449						
Hut (HUT)	m^3/day				606	606	606			
Rumaj (RUM)	m^3/day							923	923	923
Total	**m^3/day**	**2,330**	**2,330**	**2,330**	**2,470**	**2,470**	**2,470**	**2,753**	**2,753**	**2,753**
	litre/sec	**27**	**27**	**27**	**29**	**29**	**29**	**32**	**32**	**32**
Mine Dewatering Pumps and Equipment										
Primary Pump Stations										
Palokas (PAL)	each	-	1	1	1	2	2	2	2	3
Raja (RAJ)	each	2	2	2	2	2	2	2	2	3
Joki (JOK)	each	-	1	1	2	-	-	-	-	-
Hut (HUT)	each	-	-	-	1	1	1	2	2	-
Rumaj (RUM)	each	-	-	-	-	-	-	-	1	1
Total	**each**	**2**	**4**	**4**	**6**	**5**	**5**	**6**	**7**	**7**
Secondary Pumps										
Palokas (PAL)	each	-	10	10	10	10	10	10	10	10
Raja (RAJ)	each	10	10	10	10	10	10	10	10	10
Joki (JOK)	each	-	10	10	10	-	-	-	-	-
Hut (HUT)	each	-	-	-	10	10	10	10	10	-
Rumaj (RUM)	each	-	-	-	-	-	-	-	10	10
Total	**each**	**10**	**30**	**30**	**40**	**30**	**30**	**30**	**40**	**30**

17 RECOVERY METHODS

17.1 Process Plant Overview

The process described in this chapter is based on metallurgical test work presented in Section 13 and typical engineering principles deployed widely in Finland and globally in the gold and base metals industry. A simplified flowsheet of the process is presented in Figure 17-1. The run-of-mine plant feed will be classified as ‘gold’ and ‘gold-cobalt’ types and processed on a campaign basis. Both feed types share the same process with the exception of an added flotation step for the gold-cobalt feed. The saleable products of the plant are gold doré and a cobalt concentrate.

The plant feed is crushed in a three-stage crushing process to 100% passing 12 mm. The crushed plant feed is ground in a single stage ball mill to a P80 of 70-75 µm. A split of the cyclone classification underflow will report to a gravity and intensive leach circuit before electrowinning.

The grinding circuit cyclone overflow is transferred to the gold leaching circuit. Following thickening, leaching is undertaken in a cyanide in leach (CIL) process with one leaching tank and six CIL tanks. The loaded carbon is transferred to acid wash and stripping before electrowinning. The stripped carbon is regenerated for the CIL process.

Electrowinning for the intensive leach solution and stripped carbon eluate is completed in separate electrowinning circuits. After electrowinning, the gold sludge is filtered, dried and smelted into gold doré bars.

The CIL circuit tailings is detoxified and used in the process and pumped to either flotation (gold-cobalt) or backfill feed tank (gold).

For gold-cobalt feed, the target of flotation is to produce a marketable cobalt concentrate. Flotation is completed in a rougher circuit and the rougher concentrate is cleaned twice to produce the final cobalt concentrate. The flotation product is thickened and filtered before it is transported from site. Flotation residue reports to the backfill plant. The backfill plant splits the material based on backfill feed requirements and pumps the balance to the wet residues facility where water can be returned to the process.

The concentrator plant water management plan is shown in Figure 17-2 and described in more detail in Section 17.3.14. Water circulation is maximised in the process design, making up to 80% of the process water requirement. Input water is majority sourced from underground dewatering, with a small amount of raw water sourced from a nearby groundwater source for chemical preparation and potable water post treatment.

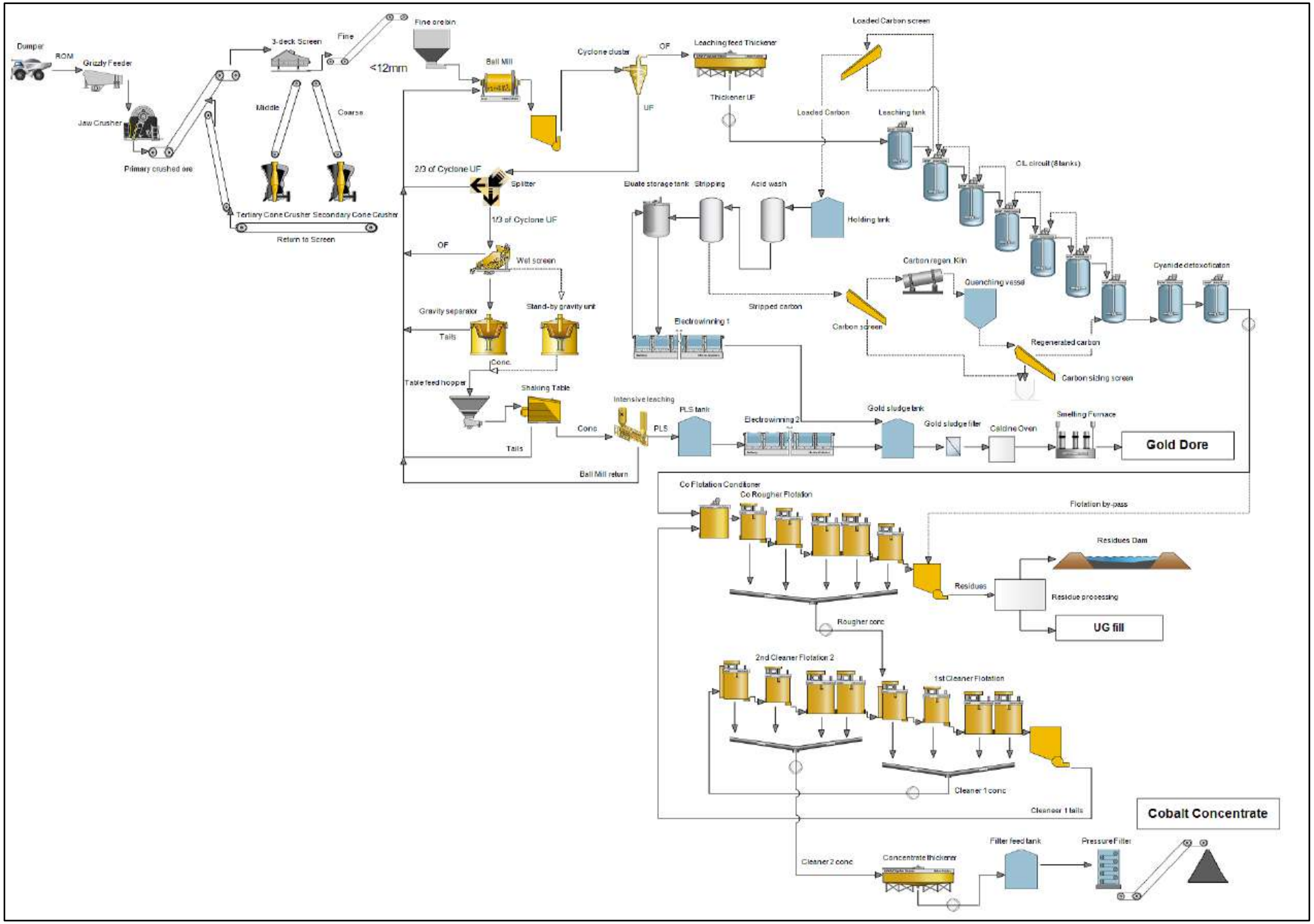


Figure 17-1: Proposed process flowsheet

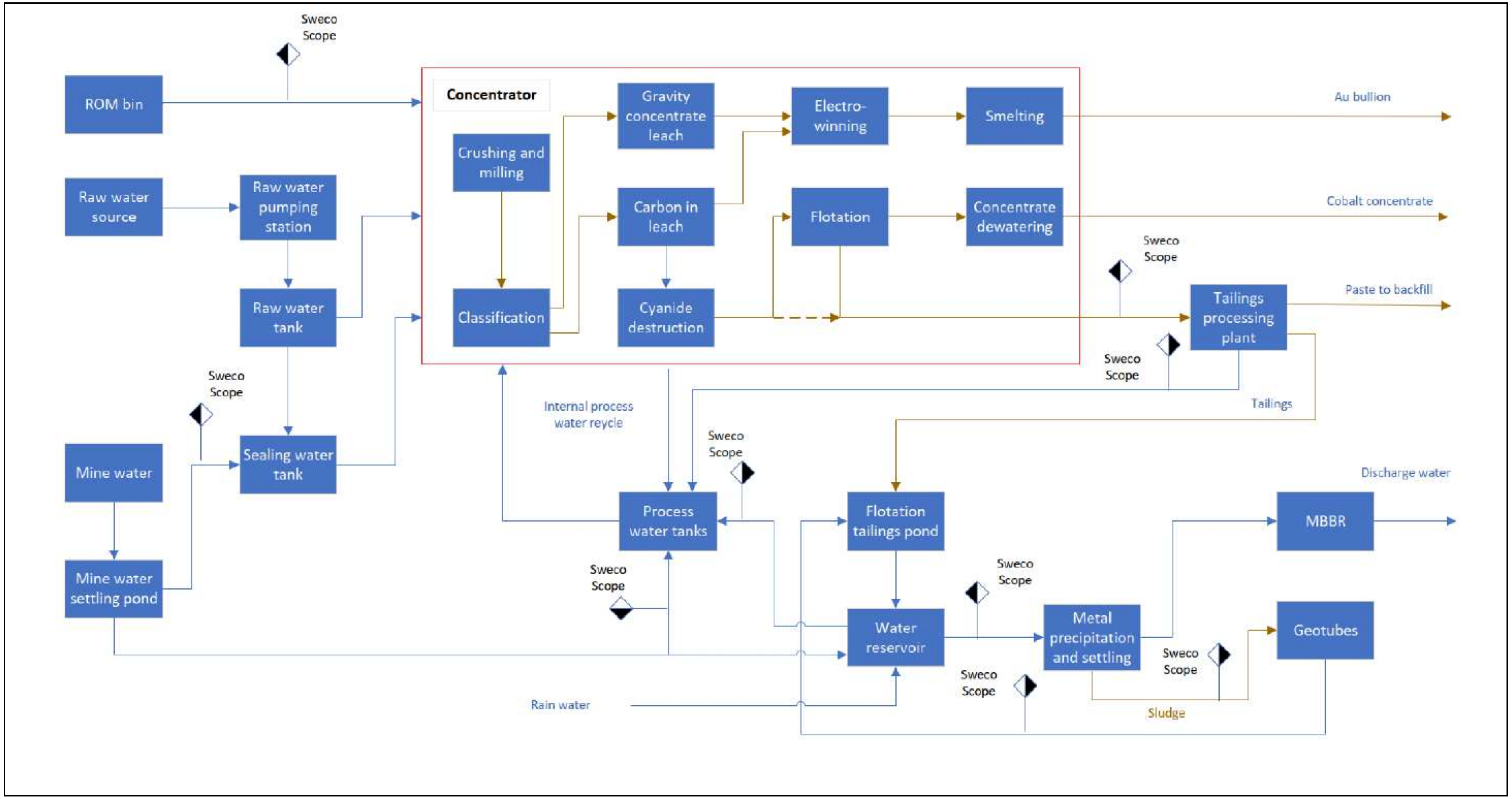


Figure 17-2: Block diagram of main water flows in the processing plant

17.2 Design Criteria

The process design is based on the preliminary process design criteria (PDC) shown in Table 17-1. The PDC values are based on test work, typical engineering values, and discussion with Mawson’s ’s metallurgical consultant (QP, Craig Brown). The design criteria will be updated accordingly in later project stages as more information is available.

The Rajapalot processing plant is designed to process 1,200,000 t of plant feed per annum with a rate of 3,600 t a day. The designed yearly operating hours are 8,000. The grinding and concentration plants are designed to be operated 365 days a year and 24 hours a day. The crushing plant is run 16 hours a day. This indicates a 91% availability for grinding and concentration and 61% availability for crushing.

Feed will be classified as ‘gold’, and ‘gold-cobalt’ and will be campaigned through the plant. The processes are the same, except gold-cobalt feed is subject to a flotation step for stopes with an estimated NSR value higher than the marginal cost of production. The estimated overall gold recovery of the process is 95% as doré, and the sulphur recovery in flotation is expected to be around 88% with a grade of 35% S in the concentrate. Recovered cobalt to the sulphide flotation concentrate is a function of the mineral form of the cobalt, (mix of cobaltite and fine-grained linnaeite) grade of cobalt in the plant feed, and the ratio of cobalt to sulphur.

Table 17-1: Preliminary major process design criteria

Designation	Unit	Value	Reference
Operating Days	d	365	Typical
Operating Hours	h/d	24	Typical
Annual Tonnage	ktpa	1,200	Assumed
Processing Capacity	tpd	3,600	Calculated
Design Feed Grades			
Au	g/t	4.0	Assumed
Co	ppm	660	Assumed
S	wt-%	3.30	Assumed
Availability			
Crushing	% (h/a)	60.9 (5,333)	Assumed, Typical
Grinding and Gravity Concentration	% (h/a)	91.3 (8,000)	Assumed, Typical
Leaching and Cobalt Beneficiation	% (h/a)	91.3 (8,000)	Assumed, Typical
Feed Rates			
Grinding (nominal)	tph	150	Calculation
Crushing (design)	tph	225	Assumed, Typical
Gravity concentration	tph	175	Calculation
Shaking table	tpd	12	Typical
Intensive leaching	tpd	1.2	Assumed
CIL (nominal)	tph	150	Calculation
Flotation (nominal)	tph	150	Calculation
Grinding Feed P100	mm	12	Assumed (vendor)
Leaching Feed P80	µm	70-75	Test work
Mill Outputs, Solids (nominal)			
Au (Doré)	troy oz/d	445	Calculation, test work
Flotation Concentrate	tpd	323	Calculation, test work
Residue	tpd	3,277	Calculation, test work
Residence times			
Leach Plant	h	30	Test work
Flotation Rougher and Scavenger	min	25	Test work, typical
Flotation Cleaner	min	20	Test work, typical
Effluent Treatment			
Precipitation and nitrogen removal feed (nominal)	m^3/h	150	Calculation
Disclaimer: multiple design criteria values have been estimated without laboratory or pilot scale testing. These have been marked with "assumption / assuming", or "typical",etc. All these require further confirmatory test work.			

17.3 Detailed Process Description

17.3.1 Crushing

The RoM is crushed in a 3-stage crushing circuit targeting a final crushed product size of approximately 12 mm. The crushing circuit is designed for a 225 tph feed rate with an availability of 61%.

Primary crushing is operated in an open circuit jaw crusher. The feed to the jaw crusher is with a vibrating grizzly feeder. The assumed blasted top size is 600 mm. Oversized material will be broken down with a hydraulic rock breaker. The feed opening of the jaw crusher is 1,200 x 830 mm and the closed side setting (CSS) 100 mm.

The primary crushed material is conveyed to a triple deck screen with 60 mm, 20 mm, and 10 mm apertures. The +20 mm particles are fed from the screen to a secondary crushing stage with a cone crusher. The secondary crusher CSS is 20 mm, and it operates in a closed cycle with the screen by returning the crushed ore to the belt conveyor from the primary crushing. The 10 to 20 mm particles are fed to a tertiary cone crusher operated in a closed circuit with the screen similarly as the secondary cone crusher. The CSS of the tertiary crusher is 9 mm. The crushed product P80 is approximately 8 mm. The -10 mm particles are screened and conveyed to the fine ore bin, which has a 2.8 day residence time allowing for crusher overhauls and weekend operations.

17.3.2 Grinding

The grinding circuit is operated with a single stage conventional overflow ball mill. The feed rate to grinding is 150 tph and availability 92%. The ball mill installed power is 2,750 kW and size 4.7 m (diameter) x 7.05 m (Effective Grinding Length, EGL). The ball mill is fed from the fine ore bin and water is added to reach target solids concentration in the mill. Lime (CaO) powder is fed to the mill by adding it directly from a CaO silo to the mill feed conveyor; CaO flow is controlled so, that slurry pH is 10.5 before addition of cyanide.

The ball mill discharge is pumped to a hydrocyclone cluster. The hydrocyclone overflow is removed from the grinding circuit to the CIL feed thickener and the underflow split in two parts. One third of the underflow is passed through a 2 mm wet screen, the undersize of which reports to gravity separation with a centrifugal gravity separator. The screen oversize and two-thirds of the hydrocyclone underflow will be combined with the gravimetric tailings and launder back to the grinding mill. The P80 of the grinding product is approximately 70 to 75 µm.

17.3.3 Gravity separation and upgrade

Gravity separation is done in two stages. Firstly, with a centrifugal gravity separator in the grinding circuit with the concentrate upgraded by shaking table in the gold room. Secondly, the gravity tailings are combined with the rest of the hydrocyclone underflow and wet screen overflow to be returned to the ball mill.

The shaking table concentrate is pumped to the intensive leaching unit and tailings back to the grinding circuit. Approximately 1.2 tpd of gravity concentrate is produced.

17.3.4 Gravity concentrate leaching

Shaking table concentrate is treated in batches in an intensive leach reactor. The intensive leach reactor is rotated and operated with a concentrated sodium cyanide (NaCN) solution together with oxidant and pH adjustment and air sparging. The pregnant leach solution (PLS) containing the leached gold is transferred to a holding tank before electrowinning. The intensive leaching tailings are pumped back to the grinding circuit.

17.3.5 Carbon In Leach (CIL)

The grinding circuit hydrocyclone overflow reports to a 14.4 m (diameter) pre-CIL thickener to increase solids content from 40 w/w% to >50 w/w% before leaching. Overflow from the thickener is recirculated to the process water tank, which is used in the grinding circuit. The thickener underflow goes through a trash screen before entering the leaching circuit.

The CIL circuit consists of a single leach-only tank and six CIL tanks. The leach-only tank allows for optimising leach conditions and significant leaching of gold into solution prior to carbon contact. The leaching and CIL tanks are sized 1,100 m^3 each at 11 m (diameter) x 11,05 m (height). CIL tank sizing is based on a 30-hour residence time.

Each tank is agitated with a mixer. Air is sparged into the tanks through a circular air pipe at the tank bottom. The tanks are arranged in a staggered formation. Tank bypassing is made possible for maintenance purposes.

The slurry flows through the tanks from the first tank towards the last tank in the formation and the activated carbon in the opposite direction. Slurry is pumped from one tank to another through an interstage screen that allows slurry to pass but retains carbon in the tank; pumping allows the tanks to be built on the same height throughout the CIL circuit. Carbon transfer is implemented with carbon transfer pumps.

The CIL tailings will be passed through a carbon safety screen to ensure that gold laden carbon is not lost if it has accidentally passed through the interstage screens. From the safety screen, the slurry will flow gravimetrically to cyanide detoxification.

The loaded carbon is pumped from the first CIL tank to the loaded carbon screen that separates the loaded carbon from the slurry. The loaded carbon is then transferred to acid wash and stripping.

17.3.6 Acid Wash and Stripping

Acid wash and stripping are conducted in two similarly sized columns. In acid wash the loaded carbon from the screen is washed with a HCl solution of typically 3%. The aim is to remove acid soluble scale build up from the loaded carbon.

Acid wash is completed in batches. The loaded carbon from a holding tank after the loaded carbon screen is flushed to the column. After the acid wash cycle is completed, the washed carbon is neutralized with sodium hydroxide (NaOH) solution before it is transferred to the elution stage.

During elution the adsorbed gold is stripped from the loaded carbon into solution. The preliminary process design for elution is a pressurized Zadra process.

After stripping the cooled down pregnant strip solution containing the gold will be pumped to the electrowinning circuit. As elution is completed in batches the stripped carbon will be transferred to the carbon reactivation circuit after each cycle.

17.3.7 Electrowinning and Gold room

In electrowinning the pregnant solutions from intensive leaching and elution will be treated in separate electrowinning circuits. Both circuits will be operated similarly. Treating the two solutions separately in electrowinning will ensure accurate metal accounting.

The electrowinning cells will be arranged in parallel and use stainless steel wool mesh cathodes. The loaded cathodes will be periodically removed from the electrowinning cells and washed to remove the gold bearing sludge. As there are two circuits this can be completed separately for each solution source. The sludge will be filtered, dried, and mixed with fluxes and then smelted in a gas-fired furnace to produce doré bars.

17.3.8 Carbon reactivation

The stripped carbon is transferred from the stripping circuit to the reactivation circuit. The stripped carbon is first screened to remove water and fine carbon. The fine carbon is temporarily stored in a tank and dewatered with a filter for disposal. The fine carbon will be replaced with new carbon for the CIL circuit.

The screen oversize will be transferred to the carbon reactivation kiln feed hopper and then to the reactivation kiln. The reactivated carbon will be passed through a carbon sizing screen to ensure that the fines have been removed and to allow addition of new carbon. After this, the reactivated carbon is transferred back to the last tank in the CIL circuit.

17.3.9 CIL Tailings detoxification

CIL tailings will flow gravimetrically to cyanide detoxification, after which the slurry is diluted to 35 w/w% with return water from tailings treatment through the process water tank. The detoxification is preliminarily designed to be done with the INCO process. In the process, cyanide is oxidized to cyanate (OCN-) by using oxygen and sulphur dioxide with the help of a soluble copper catalyst.

The process is usually operated in at pH 8.5-9.0. Sulphur dioxide is introduced to the reaction by using sodium metabisulphite ($Na_2S_2O_5$) in liquid form prepared on site. Copper sulphate ($CuSO_4$) solution is used for copper ion addition. Oxygen for the reaction is provided with air sparging. The design allows for two consecutive tanks with a volume of 220 m^3 and 5.9 m (diameter), each to provide 1 h total residence time.

The detoxified residues are pumped to either flotation for cobalt recovery (gold-cobalt), or directly to the residue processing plant.

17.3.10 Flotation

The CIL circuit residues of the selected feed sources will be processed by flotation for cobalt recovery. Cobalt is dominantly contained in two separate mineral forms: cobaltite (CoAsS) and fine-grained linnaeite intimately associated with pyrrhotite. Cobalt recovery is maximised by way of bulk sulphide flotation. Concentrate cobalt grade will be a function of the sulphur to cobalt ratio in the feed and the recoveries of the specific constituents. Anticipated concentrate sulphur grade is 35%, resulting in a concentrate cobalt grade ranging between 0.6% to 1.0%. Recovering a high proportion of the sulphides reduces the acid generating potential of the residue impoundment.

The flotation circuit consists of a rougher circuit and a cleaner flotation circuit to upgrade the cobalt content in the concentrate.

The rougher stage consists of a 30 m^3 conditioner and five 30 m^3 mechanical flotation cells. During conditioning the pH is adjusted with sulfuric acid and reagents are added. The main reagents are sodium ethyl xanthate (SEX) as collector, $CuSO_4$ as activator and methyl isobutyl carbinol (MIBC) as frother.

The rougher flotation residues are final residues of the concentrator plant and are pumped to the residue processing plant.

The cleaner flotation circuit consists of a first and second (re-cleaner) cleaner stage. The first cleaner stage is operated with four 5 m^3 mechanical flotation cells and the second with four 5 m^3 cells. The reagents used are the same as during rougher flotation. Upgrading is achieved largely by rejection of silicate diluents.

The second cleaner stage will be arranged in the layout before the first cleaner stage in a single line layout. The cobalt rougher concentrate is pumped to the first cleaner stage and the first cleaner stage concentrate is pumped to the second cleaner stage feed box. The second cleaner stage residues will gravitate to the first cleaner feed. The first cleaner residues will be pumped back to the rougher circuit to maximize recovery. The second cleaner concentrate will be pumped to cobalt concentrate thickening and filtration.

17.3.11 Concentrate thickening and filtering

The cobalt concentrate will be thickened and filtered after flotation. Based on typical thickening rates a 11 m (diameter) high-rate thickener is provided for thickening. The concentrate slurry is pumped from flotation to the thickener feed well where flocculant is added. The feed well enables slurry and flocculant mixing as well as a good flow distribution to the thickener bed. Dilution is also made possible in the feed well. The thickener overflow water will be returned to the flotation circuit to minimise reagent contamination of carbon in the CIL plant.

Thickener underflow is pumped to a filter feed tank and then to a pressure filter to produce the final dewatered concentrate. The filter cake is removed from the filter and conveyed to final product handling.

17.3.12 Plant residues management

Plant residues report to the backfill plant, further described in Section 16.8.

17.3.13 Reagents

The main reagents used in the process, and their application, are tabulated below (Table 17-2). The reagents marked as dry will be delivered to site dry in appropriate packaging. The dry reagents will be fed to an appropriate mixing tank in which they are prepared, then transferred to storage tanks to be dosed as required.

The reagents arriving as solution will be used as such with dosing pumps. If dilution is needed a mixer tank will be used and the ready to use reagent stored. Some reagents can also be diluted directly with a static mixer, if necessary.

Sodium cyanide preparation, handling, using and storage will be conducted according to Finnish and international law and standards (Cyanide Code). Sodium cyanide preparation is conducted in a separate reagent preparation room with its own ventilation system and hydrogen cyanide monitoring.

Table 17-2: Reagents used at the plant and in water treatment

Reagent	Use	Condition
Sodium Cyanide (NaCN)	CIL, Elution	Dry briquettes with wax coating, big bag in a wooden box.
Lime (CaO)	CIL pH control	Dry, bulk truck transportation
Sodium hydroxide (NaOH)	Acid Wash, Elution, water treatment	Solution, bulk truck transportation
Sodium Metabisulfite	Cyanide detoxification	Dry, big bags
Hydrochloric acid (HCl)	Acid wash	Solution, bulk truck transportation
Sulfuric acid (H_2SO_4)	Flotation pH adjustment	Solution, bulk truck transportation
Sodium ethyl xanthate (SEX)	Flotation collector	Dry, big bags
Methyl Isobutyl Carbinol (MIBC)	Flotation frother	Solution, 1000 l container
Flocculant	Thickening, water treatment, nitrogen removal	Dry, big bags
Copper Sulphate	Flotation activator, Cyanide detoxification catalyst	Dry, big bags
Air	CIL, Flotation, nitrogen removal, Cyanide oxidation	Supplied with blowers
Activated carbon	CIL	Dry, big bags
Grinding balls	Grinding	Truck delivery, packed in barrels
Methanol	Nitrogen removal	Solution, bulk truck transportation
Phosphoric acid	Nitrogen removal	Solution, 1000 l container
Coagulant	Water treatment, nitrogen removal	Solution, bulk truck transportation

17.3.14 Water Balance and Treatment

Water is mainly intended to be circulated within the processing plant. Approximately 80% of process water requirements are made up of dewatering within the process and return from the residue dam. Simplified water balance at the processing plant including the water tanks is presented in Figure 17-3.

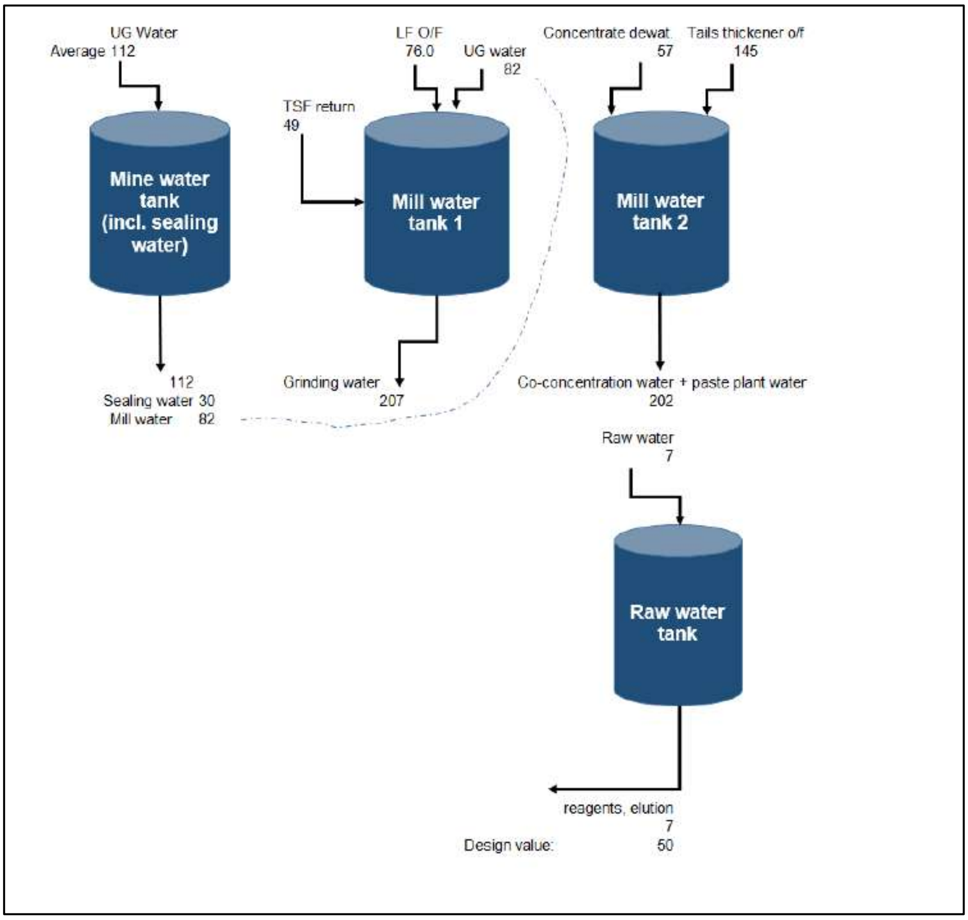


Figure 17-3: Simplified water balance at the processing plant

The remaining 21% of the required water is taken from the underground mine site. Approximately 112 m^3/h of water is pumped from the mine and rock storage runoff areas. From the settling pond, 112 m^3/h of mine water is pumped to the mill to be used as process water make-up and sealing water. At the mill, the mine water is stored in a tank with a volume of 220 m^3. Any excess mine water is pumped to the water reservoir, from which it is further pumped to nitrogen removal.

Raw water is required at a nominal rate of 7 m^3/h, pumped to the mine site from a ground water source with a design capacity of 50 m^3/h, into a 50 m^3 raw water tank at the mill. The ground water quality is sufficient to be used in all necessary process purposes (reagent preparation, elution and sealing water) without additional treatment; however, it is mostly intended for use in reagent preparation and as potable water (following basic treatment).

The residue (tailings) storage facility of the mine site consists of a residue pond, a water reservoir and a water treatment facility. Residue solids deposit in the pond, and water from the pond is decanted and pumped water treatment. The objective of water treatment is to ensure the discharge water quality meets, at a minimum, Finnish and EU standards, and also to a quality that is at or better than its ultimate point of discharge. Whilst insufficient information is available to determine the discharge quality of the water, allowances have been made for infrastructure to treat anticipated volumes of water for expected impurities prior to discharge, namely metals and nitrogen.

The first stage of treatment is a metals precipitation facility. Water flow from the pond to the precipitation facility is approximately 140 m^3/h. At the facility, the water pH is adjusted to precipitate metals out of solution. The solids are coagulated, flocculated, thickened with lamellas and the formed sludge is pumped into geotubes located next to the residue storage pond. Water from the geotubes flows into the tailings pond, while sludge is retained in the geotubes.

Water that is separated from the solids at the precipitation facility flows to the water reservoir, from which it is pumped to a moving bed biofilm reactor (MBBR) nitrogen removal plant with a capacity of 150 m^3/h. Removal of nitrogen, which will likely emanate from mine blasting emulsion, and may contaminate the site water balance, is now considered best practice in Finland. The MBBR system consists of an aeration tank (similar to an activated sludge tank) with special plastic carriers. The plastic carriers provide a surface where a biofilm, which breaks down the nitrogen, can grow. The carriers will be mixed in the tank by the aeration system and is planned to have good contact between the substrate in the influent wastewater and the biomass on the carriers.

Additionally, 50 m^3/h of water from the water reservoir is pumped to the mill water tank as recycle (tank volume 400 m^3). Excess mine water, that is not utilized at the mineral processing plant, is also pumped to the water reservoir and further to the MBBR plant. To avoid excessive concentrations of ions in the recirculating process water, 20% of the processing plant water demand is covered by mine water. Treated water is discharged from the mine site through a pipeline as described in Section 18.3. Considering precipitation, evaporation and runoff, a nominal flow of 140 m^3/h out from mine the site is expected to be discharged.

17.4 Plant Design General

The plant design and basis of cost estimate incorporates modern process plant design features including:

- Standby pumps are allowed for all key process pumping duties.
- Maintenance tooling and access, including cranes and hoists, rock breaker, mill relining machine, etc.
- A high degree of automation with appropriate instrumentation and control system provided for, including for example, an on-stream concentrate metal grade analyser.
- Equipment sourcing assumption is on the basis of European supply, almost exclusively Sandvik and Metso.

Crushing, milling and flotation are all housed inside buildings with heating and HVAC (heating, ventilation, and air conditioning) services allowances (refer to Section 18.6.1). CIL and major tankage are located outside, as is typical in Finland.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

Mining of Rajapalot is envisaged as an underground only operation with an overall production target of 1.2 Mtpa, sustained over a 9-year period through mining three of the deposits at any time. The production strategy considers continuous mining of the larger two deposits (Palokas and Raja) over the LoM and mining the smaller three deposits (Joki, The Hut and Rumaj) in sequence.

Figure 18-1 shows the conceptual site layout where the main surface infrastructure is located outside of the Natura 2000 area including:

- plant Infrastructure including process, heating and backfill plants, support buildings and facilities;
- plant residues (tailings) impoundment; and
- mine waste impoundment.

Access roads are required from the process plant (and RoM pad) to each of the underground mine decline portals, also considering the provision of mine services for mine power, dewatering, paste backfill reticulation and water supply. The access roads will be used for moving mine equipment, personnel and supplies and transporting RoM and waste out of the mine.

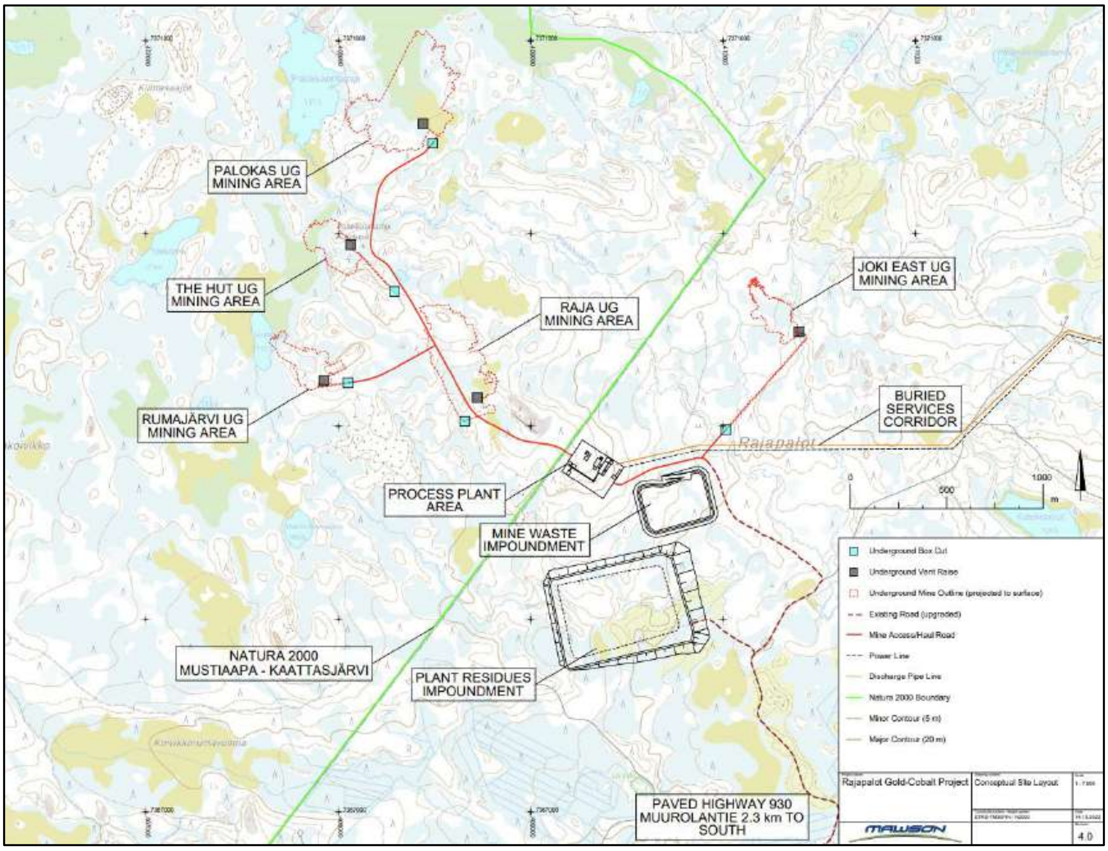


Figure 18-1: Plan view of the Rajapalot deposits and site layout

The Project is located 32 km from the capital of Lapland, Rovaniemi. Access to the Project is along an existing 3 km unsealed public road, which connects to a paved national highway (930). The Project benefits from good access and no significant logistics constraints or challenges exist for the proposed operation (Figure 18-2). The infrastructure included in the Project supports a standalone year-round mining operation at Rajapalot.

New connecting infrastructure consists of a power line and water discharge line. Site infrastructure includes surface heating plant, plant residues impoundment and ancillary supporting infrastructure.

Figure 18-2: Rajapalot Regional Plan

The following sections provide a summary of the planned infrastructure for the Project.

18.2 Power Supply

18.2.1 Introduction

Power for the Rajapalot Project site will be provided via a dedicated 110 kV power line connecting to an existing substation, 28 km from the Project.

The average operating load of the surface infrastructure (including backfill) is estimated at 5.2 MW, with the mine averaging 4.9 MW (LoM). In addition there is approximately 18 MW of installed heating capacity capable of handling the max plant and mine ventilation heating load in mid-winter although plant heating (10 MW installed) is currently envisaged as sourced from a biomass boiler.

18.2.2 110 kV Power Line

Power for the project will be supplied via a new 110 kV connection to the Finnish national power network maintained and administered by Fingrid Oy (Fingrid). The connection will be a dedicated line running west from the project area to an existing substation at the Valajoskis hydroelectric plant (see Figure 18-2 to Figure 18-4).

Fingrid is Finland's national transmission system operator. Fingrid has a legal obligation to connect regional and distribution networks and power plants to its main grid and develop the Finnish electricity power system. On request and against reasonable compensation Fingrid is obliged to provide access to the main grid for electricity consumption sites and power generating installations with technically approved connection solution.

Grid Connection Fees

Fingrid levies a fixed connection fee for new connections to the main grid. The fixed connection fees are based on the average costs borne by Fingrid due to corresponding connections at the same voltage levels. The connection fees are adjusted annually on the basis of the actual construction costs of the overall grid. Capital cost for the new grid connection to the project is assumed in the project capex.

Valajaskoski Power Station

Following consultation with Fingrid, it was confirmed Valajaskoski's power station has a free connection point for 110 kV power line available in 2022. The connection could be established without any additional cost for rebuilding the existing switchgear.

It is envisaged the 110 kV power line will start at the existing switchyard of the Valajaskoski power station, where it continues as an aerial cable west via existing power line corridors to Hirvas (B). The Kemijoki river is crossed twice as per the existing network. From Hirvas it continues west in a new corridor to Tulikoski (C). The remainder of the route will be via underground cable, buried alongside the water discharge line. Table 18-1, Figure 18-3 and Figure 18-4 show the proposed powerline information.

No detailed route planning has yet been undertaken.

Table 18-1: Power line sections

Power line sections	Cable	From - To	Length (km)
Hirvas - Valajaskoski	Existing corridor, aerial cable	A – B	7
Tulikoski - Hirvas	New corridor, aerial cable	B – C	6
Rajapalot - Tulikoski	Underground cable	C – D	15
Total			**28**

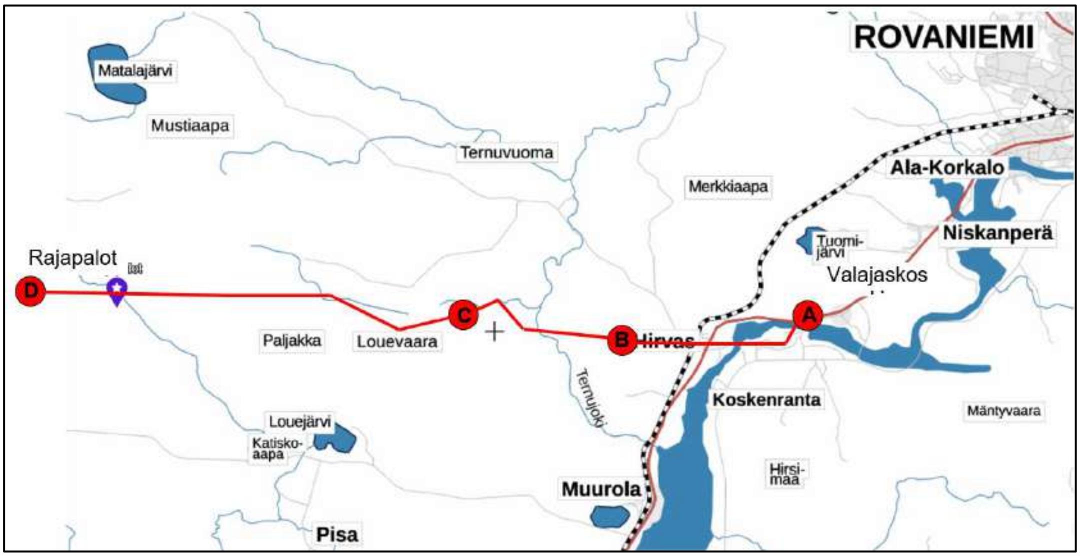


Figure 18-3: Route of 110 kV Power Line (Valajaskoski-Rajapalot)

Figure 18-4: Valajaskoski hydroelectric power station

18.2.3 Site Power Distribution

The site power distribution network will be 20 kV. A site substation will step down the power supply from 110 kV and to local sub-switch stations where the voltage is transformed to operating voltages (1,000, 690, 400 and 220 V). Three sub-stations are expected: concentrator plant, tailings and water treatment, and mining.

18.3 Treated Water Discharge Pipeline

Treated water is assumed to be pumped away from the project area to a discharge point assumed at the Ternujoki river. The future EIA will consider the impacts of surface water volumes on the local water balance, and, as such, the assumption of discharge into a river with sufficient existing flow to handle the new flows is the standard approach in Finland. There are alternative discharge locations available and further studies are required to confirm the most appropriate discharge point.

The treated water discharge pipeline will be buried to prevent freezing and will follow the same C-D route shown in Figure 18-3. The size of the pipe sizing was selected assuming that a volume of the discharge water would be approximately 300 m^3/h. This flow is approximately 2 times the nominal flow calculated in the water balance, to account for increased pumping during the spring snow thaw. A High Density Polyethylene (HDPE) 315 PN 10 SRD 17 pipe has been assumed (maximum flow velocity 1.5 m/s).

18.4 Surface Infrastructure Heating

18.4.1 10 MW Heat Plant

A 10 MW biomass boiler plant is planned for generating heating energy at the Rajapalot site. Main fuels are wood-based biomasses such as saw dust, bark, forest residuals and wood chips. Using waste wood-based biomass as fuel is a renewable resource is common in Finland and is typically viewed favourably by locals as it provides a use for by-products from local forestry activities.

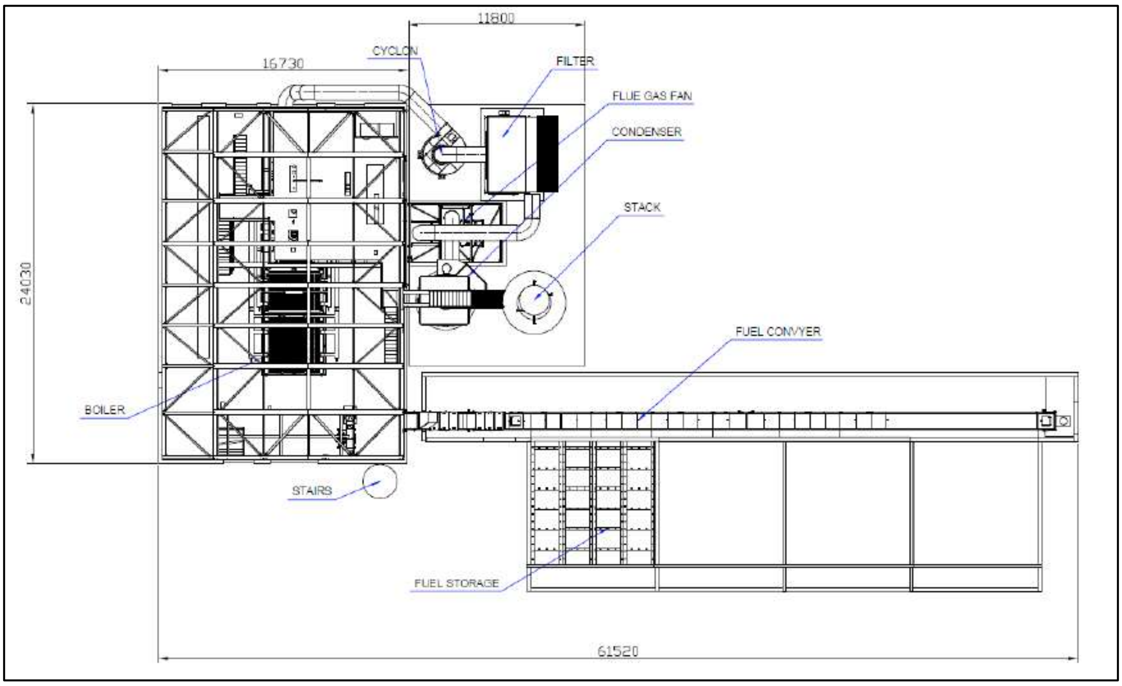


Figure 18-5: Heat Plant

The heat plant will be located close to the Concentrator plant with an estimated footprint of approximately 16 x 24 m and a fuel storage area of approximately 12 x 40 m.

Operating the heat plant requires one part-time operator. The fuel of the heat plant is delivered by trucks which unload the fuel directly to the moving floor fuel storage. The automated conveying system feeds a boiler where the fuel is combusted on the grate. After the boiler, flue gases are led to a bag house filter for removal of particle emissions.

18.4.2 Heat Distribution and Transfer

The delivery of heat network is made by insulated black steel pipes. Heating of the production spaces is executed by the ventilation units and the circulated air units. In the control room, offices and laboratory, conventional radiators are used to provide heat transfer. Alternatively, thermal radiators can be installed in the ceilings. Shower rooms, etc, are equipped with floor heating.

18.5 Roads

Access road to the project is via paved highway 930 at the Aavasaksantie 1770 marker, 30 km from south-west of Rovaniemi. The final 5 km is predominantly via an existing municipality gravel road (3.4 km), a forestry road (0.4 km), thereafter via an existing off-road track.

The site is expected to have moderate-to-heavy heavy traffic year around from personnel and goods. The minimum width of the roads has been established to be 5 m. The existing forest truck roads varies from 4 m to 5 m, so there is a need for minor upgrades to the existing access road.

Improvements of the existing roads will be made according to the guidelines shown in Tierakenteen suunnitteluohje 2004 by Liikennevirasto (Finnish Transport Agency). The chosen basic structure for the roads is Soratie 70 SR, which is a gravel road design and can be used for both public and private roads with heavy vehicle traffic. The design value for the carrying capacity of this structure is 70 MPa.

Based on this, some existing roads need both widening and strengthening of the current road structure. Proposed improvements by road type are shown in Table 18-2.

Table 18-2: Road upgrade specifications

Road type	*Method for improving the carrying capacity*	*Widening*
Public road, width 6,5 m	Base course 150 mm + Binder course 50 mm	-
Public road, width 5,0 m	Base course 150 mm + Binder course 50 mm	Meeting places
Forest truck road, width 4,5 m	Base course 300 mm + Binder course 50 mm	0,5 m + meeting places
Forest truck road, width 4,0 m	Base course 450 mm + Binder course 50 mm	1,0 m + meeting places
New road	Base course 450 mm + Binder course 50 mm	Meeting places, side ditch
New road with soil replacement	Soil replacement + Base course 450 mm + Binder course 50 mm	Meeting places, side ditch

In addition to the road access to concentrating plant site, an internal road network will be needed. Based on the conceptual site layout, the road infrastructure improvements consist of:

- 3.4 km of access road upgraded to 5 m width
- 0.4 km of access road which requires upgrade from a forest trail to gravel road
- 1.0 km new gravel road to access the plant site
- 0.6 km new internal roads

There are no known environmental or culturally important areas or places along the access road corridor that would affect the proposed road routes.

18.6 Plant Infrastructure

18.6.1 Buildings and Facilities

The buildings shown in Table 18-3 have been estimated. Permanent buildings are proposed for all site structures. Note that buildings costs are not reported in the Infrastructure sub-total of the Capital Cost, rather they are integrated into the civil and structures sub-total.

Due to the project location 35 km from Rovaniemi, a potential opportunity is to conduct administrative and major overhaul work in offsite facilities (contractor or lease) to reduce the capital cost of initial infrastructure. This will be evaluated in future phases of study.

Table 18-3: Summary building specifications

Building	Design Comments
Crushing 416 m^2 (16 x 26 m) 352 m^2 (16 x 22 m) 352 m^2 (16 x 22 m)	3 enclosed buildings. Slab foundation Insulated and heated. Architectural façade Cone silo with 35° drawdown
Concentrator 1564 m^2 (23 x 68 m)	Enclosed building. Slab foundation. Some equipment with independent foundations. Steel structure and overhead gantry crane included. Insulated and heated. Architectural façade. Brick internal walls where fireproof.
Control room 160 m^2 (8 x 20 m)	Pre-fabricated concrete partition walls (fire and noise). Slab foundation. Insulated and heated. Conventional finishes.
Electrical Room 276 m^2 (23 x 12 m)	Pre-fabricated concrete partition walls (fire and noise). Insulated and heated.
Tanks 1342 m^2 (22 x 61 m)	Slab only, no building. Steel tanks with separate foundation (1-1.5 m thick) Bunded, 1.5x volume of largest tank.
Water Treatment 350 m^2 (14 x 25 m)	3 separate buildings (chemicals, metals precipitation, MBBR) Slab foundation. Insulated and heated. Chemicals building bunded 1.5 x volume of largest tank.
Loading space 576 m^2 (24 x 24 m)	Slab and enclosed. Insulated but not heated.
Workshop 1,200 m^2 (16 x 75 m)	Combined mining/mobile and fixed plant workshop. Slab foundation. Crane, height as applicable to site vehicles. Insulated and heated. Architectural façade
Emergency Power 150 m^2 (10 x 15 m)	As per Electrical Room
Administration 700 m^2 (20 x 35 m)	Combined site admin and crew change and laboratory. Mining, process and administration salaried staff. 2 stories (1,200 m^2 total floor space) Permanent building, insulated and heated. Conventional finishes

Allowance has been made for permanent buildings according to the Finnish national construction standards. The specifications include:

- fire class buildings greater than two stories is P1, all other P2, (Finnish building code, part E2), including sprinkler extinguishing equipment will be provided where necessary.
- Structure design Consequence Class CC2 (EN1990).
- Corrosivity of the environment for steel structure C4 (high) (EN ISO 12944-22).

- all load bearing steel structures or structures which are part of the structure system have a minimum steel grade of S355 specified according to EN 10025-2. The execution class is generally EXC2.
- An example layout of the two largest structures, the concentrator and tank area, are shown in Figure 18-6 and Figure 18-7.

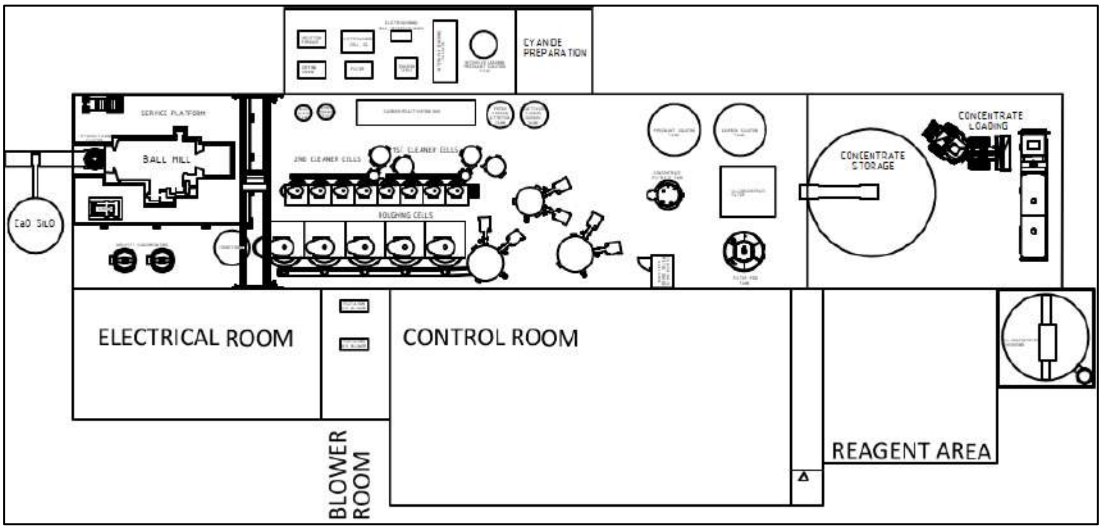


Figure 18-6: Concentrator Plan

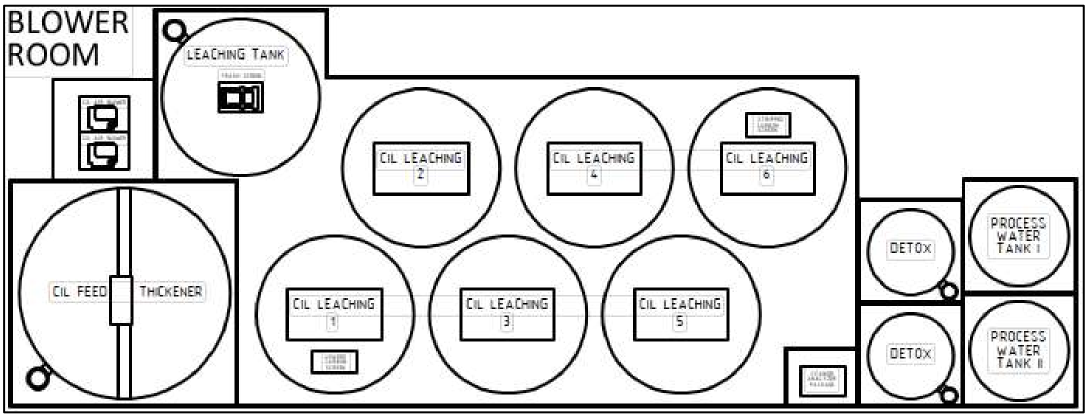


Figure 18-7: Tank Area Plan

18.6.2 Raw Water

The raw water is pumped from a bore-field assumed within 1,000 m of the plant site at a nominal rate of 7 m^3/h according to the conceptual water balance. No hydrological assessment of any particular location has been made, but regional pump tests taken by Mawson, access to raw water is not expected to be a limiting factor.

Ground water quality is known to be excellent and is expected to be suitable for reagent preparation. A biochemical water treatment plant is included for potable water production.

18.6.3 Fire Water Pumping Station

Fire water is pumped from the fire water tank. The pumping station needs to be semi-warm to prevent water from freezing in winter and contains two pumps and one diesel generator in reserve.

18.6.4 Other Infrastructure

Other infrastructure allowances in the estimate are as follows:

- **Distributed Control System:** Distributed control system (DCS) with centralized control room is assumed. It enables high-quality data and process management and can connect to package plant programmable logic controllers ("PLC").
- **Building Automation:** The building automation system is included in DCS. The automation substations are placed in the ventilation machine room and in the heat distribution room. The new substations are connected to the building automation system. The temperature and the quality of air in the production and office facilities are controlled by regulators which are connected to the building automation system.
- **Communications:** The mine and production sites will be equipped with sufficient internet connections. Mobile phones are used for personnel communications. Hand-held and base-station radios purchase costs are excluded.
- **Parking Area:** The total area of the parking lot is 1,000 m^2 allowed for. The parking area is paved and designed to have the same road structure layers as designed for the other roads in the area.
- **Domestic Water Supply and Sewage Network:** The offices will have stand-alone systems for domestic water and sewage.
- Laboratories and production facilities are equipped with emergency showers including eye-wash fountains and ordinary showers. Insulated and trace heated emergency showers (suitable for outdoor installation) are installed in spaces in which is an occasional danger of ice formation.
- **Emergency Power Station:** The emergency power station structure contains only a thick reinforced concrete pad without a building. A 3 m high perimeter fence is included.
- **Cooling Systems:** Cooling of the offices is arranged by a cooling coil at the air handling unit, either utilising ceiling radiators or air conditioning beams (active chilled beams) which are installed in the rooms. Those rooms where internal thermal loads are high (such as computer rooms) are equipped with fan coils.
- **Compressed Air Systems:** The compressed-air piping in the production building is executed as a ring network made of weldable PN16 steel pipe. The network is installed with a slope towards the drains.

18.6.5 Earth Works

The following outlines the general design and construction approach to earthworks, as generally employed for concentrating buildings and structures, tank area, water treatment area and heat plant.

No plant or infrastructure site specific geotechnical evaluation has been undertaken. However, Mawson has collected over 1,700 samples of the upper till horizons, and mapped till depth using 'base of till drilling in in over 100 locations in a 4 x 4 km area proximate to the processing and infrastructure area. Mawson has advised the findings from these investigations are that the till averages approximately 5 m to basement rock. The organic layer of the till is typically less than 10 cm deep. This information has informed the design and cost estimate. Future phases of study will include geotechnical investigation.

Whereas the subsoil in the production area is mostly moraine, on top of the moraine layer there is a thin humus layer. This topsoil layer must be removed before starting the actual construction work. Soil containing humus and peat will be stockpiled and used for rehabilitation.

Excavation is estimated to approximately 2.5 m depth from existing ground level.

Footings estimated to be on the crushed stone layer and ground supported (no piling). Below the crushed stone layer, there are fillings of frost resistant material consisting, for example, sand and gravel. The layers are isolated from each other by geotextile.

Frost insulation is applied under ground floor and outside next to the external wall. All foundations have a thick layer of crushed stone below them.

18.7 Off-site Logistics

No specific off-site logistical infrastructure is necessary for operations. The site is accessible year round via public highways. Rovaniemi has a major regional rail and road freight hub, and an international airport. Cobalt concentrate will be trucked from site in road vehicles as is typical in Finland.

18.8 Underground Mine Infrastructure

18.8.1 Introduction

The PEA mine plan considers a greenfield underground operation targeting a RoM production rate of 1.2 Mtpa over a mine life of 9 years. The production strategy considers continuous mining of the larger two deposits (Palokas and Raja) over the LoM and sequential mining of the three smaller deposits Joki (year 1 to 3), The Hut (year 3 to 7) and Rumaj (year 7 to 9). Each of the near surface deposits are planned to be individually accessed through decline box cuts with truck haulage to the RoM stockpile located at the process facility.

The mine infrastructure will need to align with the production strategy, with permanent infrastructure over the LoM at Palokas and Raja and a phased/share infrastructure approach for the smaller production at the Joki, The Hut and Rumajärvi mines. Consideration is also given to locate, where possible, mine infrastructure outside of designated Natura 2000 areas. The main mine infrastructure considerations include:

- mine backfill (summarised in Section 16.8);
- mine ventilation (summarised in Section 16.12) including intake air heating during the winter months;
- mine dewatering system (summarised in 16.15);
- electrical reticulation infrastructure;

- service and fresh water supply; and
- other working facilities for lunchrooms and storage of spares/consumables and magazines (explosives and detonators).

As described in Section 18.6, the surface plant infrastructure will include combined facilities to accommodate mine administration, technical offices and change house facilities for the underground mine and maintenance crews. This also includes provision for a central mining/mobile and fixed plant workshop with major overhaul work to be undertaken in offsite facilities.

18.8.2 Electrical Reticulation Infrastructure

Power will be supplied to each of the mine portal areas at a high voltage (HV) supply of 20 kV. From the portal, power will initially be delivered to support decline development at the low voltage (LV), typically at 1,000 V. When development has progressed far enough to reach the first substation location underground, a HV line will be installed.

Power will be reticulated by armoured HV cable suspended from the development backs to substations where it will be stepped down to LV and distributed to working areas for use by mining equipment. The maximum LV run is approximately 450 m and this determines the requirements for substation relocations.

18.8.3 Mine communications

Communications for the mine are assumed to be a radio-based communications system. This system is installed in stages and extends with progress of the main decline development and will provide all voice and data communications within the mine.

18.8.4 Fuel Bays

Surface fuelling facilities would be located outside of the Natura area and in the vicinity of the process facilities. Underground fuel bays may be considered for the larger mines of Palokas and Raja to increase productive working time of mobile equipment.

18.8.5 Explosive and Detonator Magazines

Separate and secured underground explosives magazines (one for detonators and the other for primers/bulk explosives) will be required. They will be sized to store a week’s supply of detonators, primers, bulk explosives and blasting accessories.

It is expected that emulsion explosives will be required for underground development and production activities. The explosives magazines will require an inventory log and the necessary suppression equipment in the event of a fire. The explosives magazines will typically be located away from main accesses and working areas and linked to the return airway circuit.

Magazine storage and handling of explosives will be in compliance with Finnish regulations.

18.9 Mine Waste Management

The mine plan for Rajapalot includes provision for a surface mine waste impoundment (see Figure 18-1) with a working storage capacity up to 3 Mt. Mine waste will be generated through the portal boxcut, decline and access development for each of the mining areas. A portion of th mine waste generated is likely to be suitable for construction (such as roads and civil works) and the CRF (Joki mine). There will also be opportunities to use a portion of development waste within the underground mine for unconsolidated rockfill and road sheeting on top of pastefilled stopes which will reduce the materials handling requirements and reduce costs.

18.10 Tailings Residue Management

The current mine plan estimates a total of 10.1 Mt of RoM feed, and a full processing rate of 1.2 Mtpa. Approximately 4 Mt of tailings will be utilised as mine backfill, leaving approximately 6 Mt of tailings produced that will require storage in an engineered tailings storage facility (TSF).

No previous studies on tailings management have been completed to date. SRK identified potential sites for the storage of tailings to inform this PEA. Several TSF options were modelled to accommodate the required volume.

18.10.1 Regulatory requirements

SRK has considered several guidance documents when preparing this study, including the following:

- International Guidelines:
  - Global Industry Standard on Tailings Management (GISTM, 2020).
- European Union Guidelines:
  - EU Best Available Technique Reference Document: European Commission Reference Document on Best Available Techniques for Management of Waste from the Extractive Industries, in accordance with Directive 2006/21/EC 2018.
- Canadian Dam Association:
  - Technical Bulletin: Dam Safety Reviews (CDA, 2016).
  - Dam safety guidelines (CDA, 2013).
- Mining Association Canada:
  - A Guide to the Management of Tailings Facilities. Version 3. (MAC, 2019).

18.10.2 Design Criteria

The principal objective of the design and operation of the TSF is to ensure secure containment for tailings solids and impounded process water. The tailings will be subject to a thickening process within the backfill plant where excess water will be recycled into the mill building. The thickened tailings will be pumped to the TSF via a slurry pipeline with a solids content of 55% by mass (W_{solid}/W_{total}).

The design criteria assumed for the purpose of this assessment are summarised in Table 18-4.

No geotechnical testing has been carried out on representative tailings samples. SRK has assumed an in situ tailings porosity at deposition and used the density of the mineralized material to determine an assumed in-situ density for the purpose of volumetric modelling.

Table 18-4: Rajapalot TSF Design Criteria

Criteria	Units	Value	Notes
Tailings Physicals			
Life of Mine	Years	9	SRK Mine Plan
Total Ore Processed	Mt	6	Total from 5 deposits
Percent solids by weight	%	55	P&C Mass balance
Tailings Specific Gravity	-	2.86	P&C Mass balance
Placed in situ tailings density	t/m^3	1.4	SRK Assumption
Target tailings storage volume	Mm^3	4.3	SRK Calculation
Main Embankment Geometry			
Embankment Crest Width	m	10	SRK Assumption
Upstream slope inclination (operational)	-	2.5V:1H	SRK Assumption
Downstream slope inclination (operational)	-	2.5V:1H	SRK Assumption
Design Storm Freeboard Allowance	m	1	SRK Assumption

18.10.3 TSF Site Selection

Minebridge software MuK3D was used to identify and model potential TSF alternatives utilising natural topography to store the required tailings volume. The area is relatively flat with no areas identified in which tailings can be contained within a valley using natural topography. Site constraints are presented visually on Figure 18-8 and discussed below.

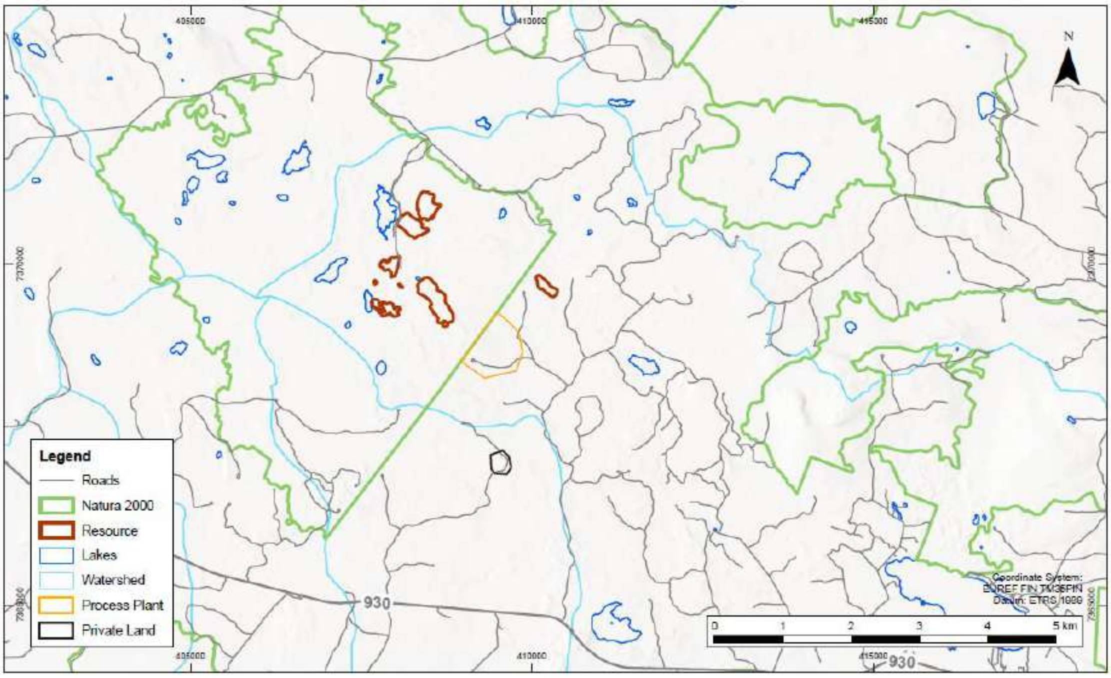


Figure 18-8: Site Constraints

The five underground deposits are located within Natura 2000 areas, while the proposed processing plant and surface infrastructure are located outside of Natura 2000 areas. Potential options for a TSF were studied within a 5 km radius of the processing plant to reduce the pumping distance of slurry tailings from the thickeners to the TSF. Additional site location considerations were:

- Presence of private protected areas.
- Proximity to roads to reduce the visual impact.
- Keeping the footprint of the TSF within one water catchment (FID-1166/Jako3Tunnu-65.156)
- Avoiding water bodies, including minor lakes.
- Preference for close proximity to the waste dumps, assuming waste rock material will be used for embankment construction.
- Consideration of dam height to reduce visual impact of the TSF on the landscape.

An overview of locations considered is presented in Figure 18-9. All five locations were modelled as thickened slurry tailings and key details of each option are presented in Table 18-5.

Table 18-5: Summary of TSF Options Modelled

TSF Option	Distance from process plant (km)	Embankment height (m)	Embankment crest elevation (mRL)	Embankment Volume (m^3)	Surface Area (m^2)	Storage Efficiency[1)] (%)
A	2	12	169.5	273,908	940,000	6
B	3	14	157.5	878,216	880,000	15
Ca	1.5	15.5	161	1,039,107	635,000	18
Cb	1.5	24.5	170	1,934,434	365,000	29
D	2	12	136	334,138	720,000	7
Ea	0.25	14.5	149	1,007,357	555,000	18
Eb	0.25	32	167	4,049,155	250,000	46
Ec	0.25	15	150	998,214	580,500	19
Ed	0.25	25	160	2,344,979	378,713	34

1) Storage efficiency is the volume of embankment material divided by the volume of tailings material storage as a percentage.

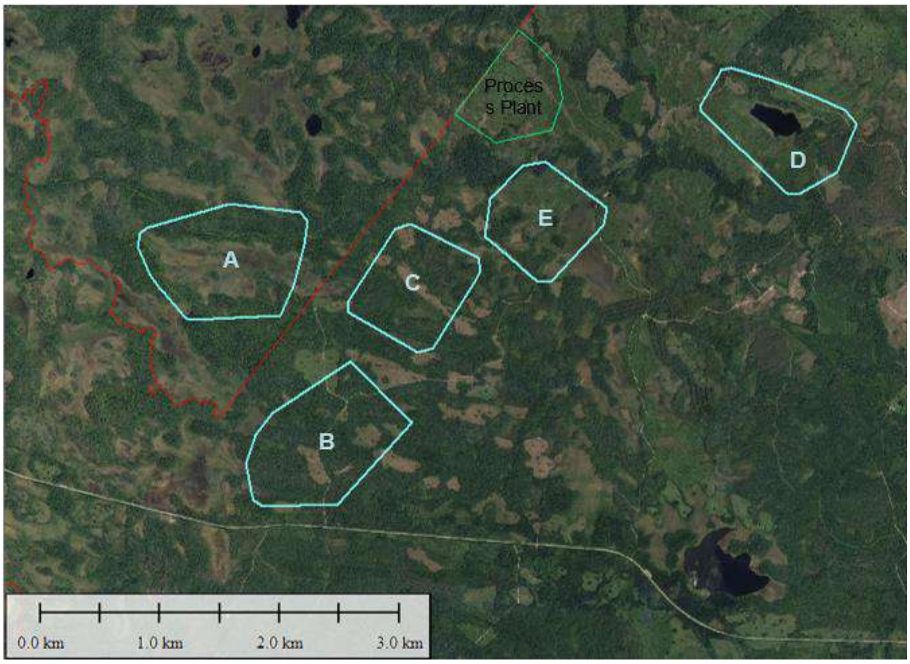


Figure 18-9: Generalised TSF Locations Considered

Following an option evaluation and scoring process, which included multidiscipline consultant and Mawson review, Option Ed was considered to be the preferred alternative (Figure 18-10). Future phases of study will evaluate site alternatives in more detail.

Figure 18-10: TSF Option Ed

18.10.4 TSF Conceptual Design

The tailings will be deposited sub-aqueously as thickened tailings. An embankment will be constructed around the perimeter of the facility creating a ring dyke. The embankment will reach a final height of 25 m, at an elevation of 160 mRL and provide for a LoM tailings storage capacity of 6 Mt.

The embankment will be raised in stages using the downstream method which is a higher cost and lower risk / higher safety factor method as opposed to centre line or upstream raise. A cross section of a typical embankment is presented in Figure 18-11.

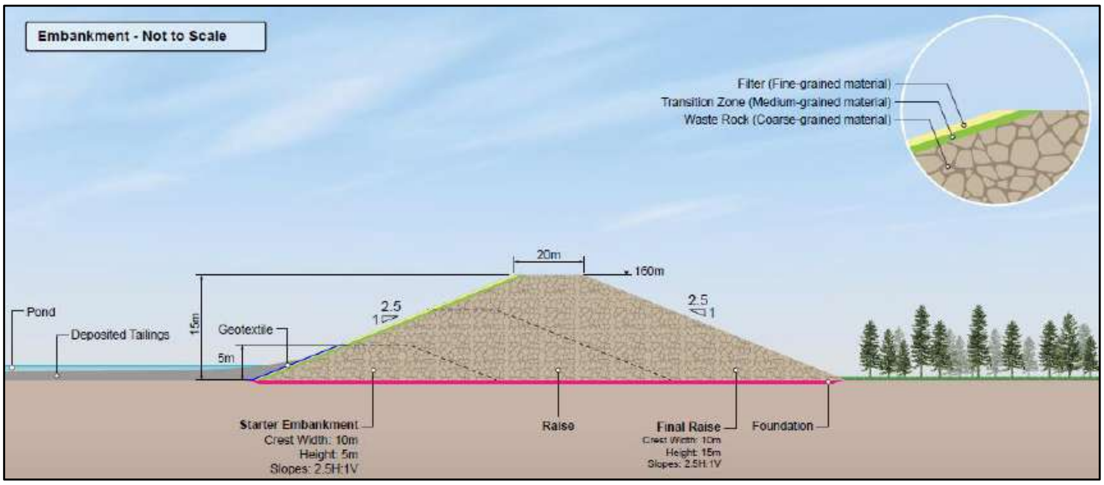


Figure 18-11: Conceptual TSF Embankment

Tailings are anticipated to be deposited into the facility through a delivery pipeline system which will be placed on the embankment crest with spigots placed at 50 m intervals. Deposition can then take place from the north/east/south/west in stages, to ensure the pond is kept in the centre of the facility, away from the embankments.

The proposed dam concept consists of a rockfill dam with a liner on the upstream side. The dam will include a filter zone on the tailings side for restricting the movement of fine particles and prevent piping. At this stage it is anticipated that material to construct the starter embankment will be sourced from a local borrow pit. The remaining material will be available from underground mine predevelopment and/or the waste rock dump facility.

There will be a surplus of water in and on the impounded tailings. Water within the TSF that does not seep through the permeable rockfill dam will be transported back to the mill using a floating barge and a series of pipes and pumps.

It is envisaged that the TSF will require lining to mitigate the risks associated with seepage from the base of the TSF. This is based on previous work with similar mining processing operations, with no geochemical evaluation of tailings having been completed to date. Additional analysis during subsequent design phases may rule out the requirement for a fully lined facility, but at this staged a fully lined facility has been costed for. The liner system will consist of a geosynthetic clay liner (GCL), overlain with a 1.5 mm HDPE geomembrane and a geotextile protector.

18.10.5 Conclusions

A conceptual TSF design has been presented and costed for the storage of tailings at Rajapalot mine. A topographical survey and several constraints were studied to identify potential locations for a TSF. Five general areas were presented, with various dam height options. Following discussions with Mawson and interdisciplinary consultants, Option Ed was taken forward for costing purposes and inclusion within the PEA as it scored the best from a cost and lowest permit risk perspective. A conceptual design was outlined which included a lined TSF and ring dyke structure to contain tailings expanded using the downstream raise method.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

No market studies have been undertaken, nor have any contracts been established yet.

19.2 Gold

Gold, especially in doré form, is a readily marketable metal with transparent pricing. Doré will be shipped from site most probably to a European gold refinery. An allowance has been made in the project economics for freight, insurance and refining totalling an average over the life of mine of USD3.60/oz Au.

The NI 43-101 PEA applies a long-term gold price assumption of USD1,700/oz, which is below the 3-year tailing average to 18 July 2022 of USD1,754/oz, and below average analyst consensus forecasts for the medium term (Figure 19-1).

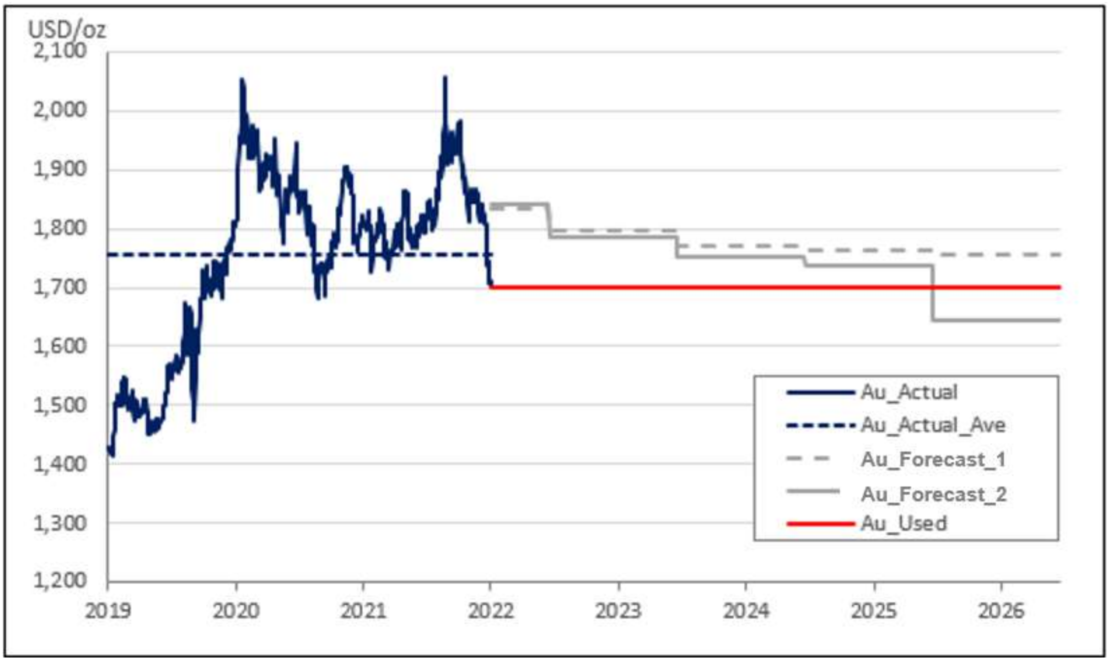


Figure 19-1: Gold price history and analyst consensus forecasts (June 2022, real terms)

Two forecast datasets are provided, reflecting the average of analyst predictions; one published by Group 1 (a market data provider) containing up to 32 analyst annual datapoints, and one published by Group 2 (an investment bank) containing up to 30 annual datapoints. Constituent estimates are not necessarily unique.

19.3 Cobalt Market

The cobalt market has, and continues to, undergo a significant transformation. Historically cobalt's main use has been as a metal hardening alloy, and as a catalyst in the petroleum and chemical industries. More recently cobalt has emerged as a key part of the transition economy, through its role in batteries of increasing life and energy density, and keeping the structure stable during continuous charge and discharge cycles. On the supply side, the market is characterised by a high level of geographic concentration (74% mine production from the Democratic Republic of Congo (DRC), 30% of which is from two single mines) and 98% of cobalt is produced as by-product of other base metal production, namely copper and nickel. Refining is dominated by China, but Finland is the next largest global refiner. Geopolitically the trend is towards decarbonisation, critical mineral self-sufficiency and improving traceability. The combination of these market forces presents a favourable outlook for cobalt supply and demand, especially outside of the DRC, and hence the attractiveness of a cobalt supply from the Rajapalot project.

19.3.1 Demand

Prior to the commercialisation of cobalt-based lithium-ion batteries in the 1990s, cobalt was 99% consumed in the industrial and chemical industries. Demand split has changed following the rapid uptake of lithium-ion batteries, thus the market is split into ‘new world’ and ‘old world’ demand drivers. Total cobalt demand has grown strongly at 9.8% compound annual growth rate (CAGR) since 2015, with 2021 demand totalling 175 kt. New world demand now dominates the cobalt market share (Figure 19-2), driven by electric vehicle (EV) uptake, with 2021 marking the first time EVs have been the largest consumer of cobalt globally. This trend is predicted to continue.

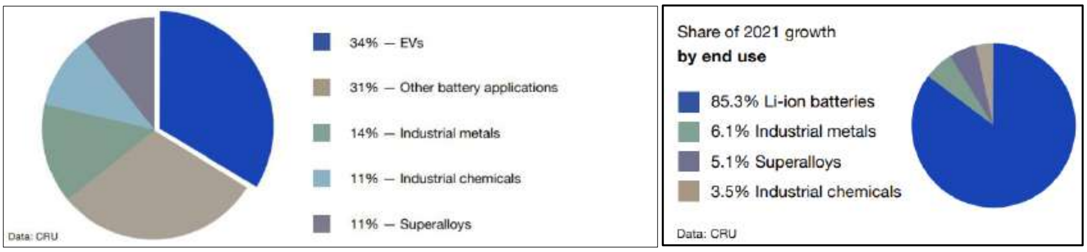


Figure 19-2: Cobalt demand and growth by sector (Source: Cobalt Institute)

The future growth outlook for the largest segment of demand (batteries) is strong. New world demand growth continues to outpace overall economic growth rates and will continue to dominate cobalt demand for the foreseeable future. Commodity Resource Unit (CRU) forecast 2030 demand to increase to 320 kt from 175 kt in 2021 (7% CAGR).

Lithium-ion Batteries

Lithium-ion batteries possess high specific energy (energy/weight), low rates of self-discharge and are generally maintenance free. Lithium-ion batteries are classified as either cobalt or non-cobalt based. Within cobalt-based batteries, there is a trend towards thrifting of cobalt in battery chemistry. In ‘high nickel’ (approximately 10% Co), however, despite the lower intensity compared to ‘medium nickel’ cathodes (20-40% Co), cobalt remains a critical material for the stability and safety of high nickel chemistries, which limits ultimate thrifting rates.

The main non-cobalt battery technology is lithium iron phosphate (LFP). LFP demand growth is strong in China due to its relative lower cost to nickel and cobalt-based chemistry, acceptance of short range ’mini-EVs’, as well as subsidies and policy; however, LFP still accounts for only 25% of EV batteries globally (Figure 19-3), and uptake is expected to be constrained in Europe and North America where nickel- and cobalt-based chemistries dominate due to consumer preferences for larger and longer-range vehicles.

The overall growth in demand of EVs is expected to result in strong cobalt demand growth that outpaces incremental thrifting and replacement headwinds.

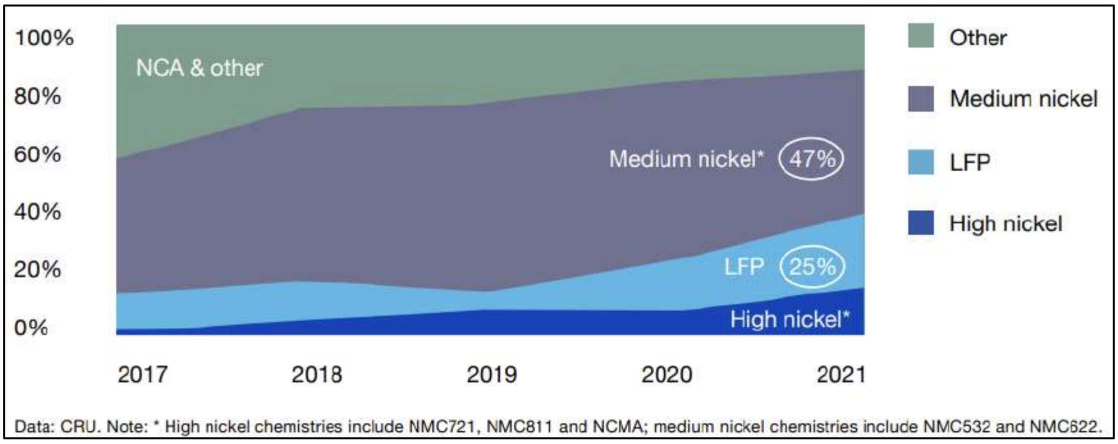


Figure 19-3: Global light duty EV cathode chemistry share (% GWh) (Source: Cobalt Institute)

19.3.2 Macro economic forces

There is a global push to decarbonise the economy, which has been growing in response to increased climate change awareness and consequent emissions targets set and agreed under international treaties such as the 2016 Paris Agreement and United Nations Framework Convention on Climate Change (UNFCCC). Globally transport contributes approximately 20% of global emissions, of which road vehicles account for approximately 80%. Electrification of road vehicles is therefore seen by governments as a key mechanism for lowering emissions. For example, Norway has announced a ban of the sale of petrol-powered cars by 2025, while the United Kingdom, Ireland, Germany and the Netherlands plan to do the same by 2030, and France by 2040.

Globally EV subsidies are occurring on an increasingly large scale, targeting both EV purchase assistance and COVID-19 linked bailouts for incumbent auto original equipment manufacturers. As shown in Figure 19-4, uptake in Europe and China has demonstrated the impact of policy and subsidies on EV growth. China was an early adopter of subsidies and showed earliest global demand growth. More recently, Europe and UK grants and subsidies total up to EUR9,000 and GBP8,000 per vehicle, respectively, with both moving to ban new internal combustion engine vehicles from 2035 support sustained uptake. The US has been slower to act; however, the scope of Biden's Clean Energy Plan is becoming clearer, with USD400Bn over 10 years for among other things 500,000 EV charging stations to support an estimated 25 million EVs (vs approximately 2.5 million today).

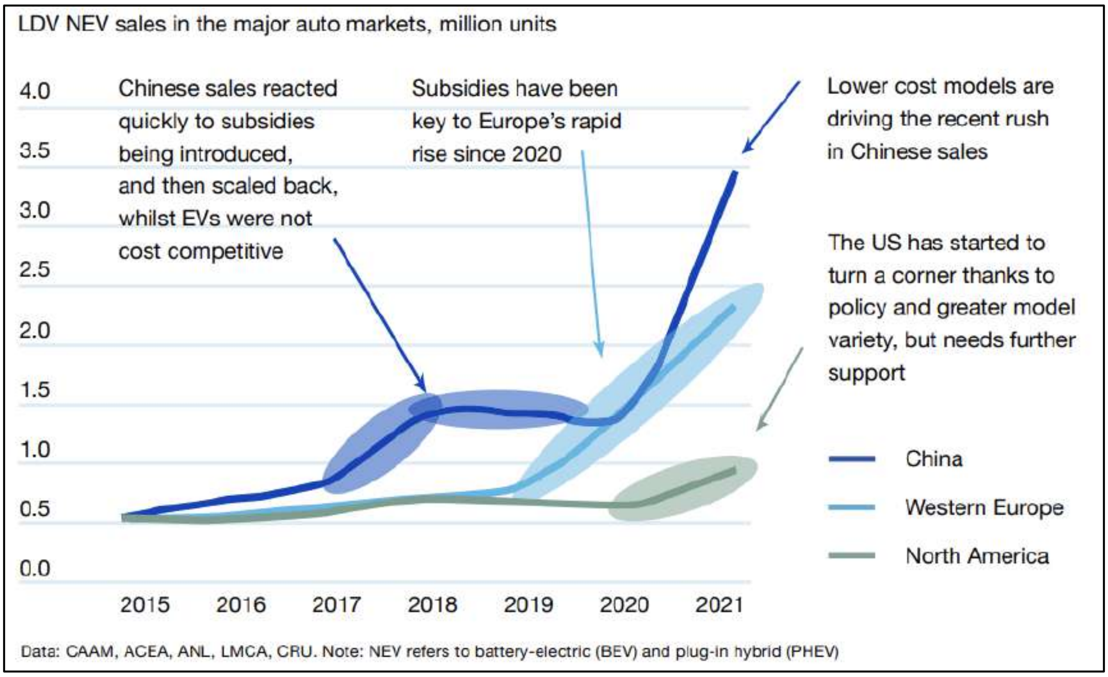


Figure 19-4: EV trends in three major automotive markets (Cobalt Institute)

19.3.3 Supply

Global mined supply of cobalt is approximately 160 ktpa, with the world's overall cobalt demand projected to grow to 270- 370 ktpa by 2030, and up to 500-800 ktpa by 2050. Cobalt supply is highly concentrated, with the DRC producing 74% of cobalt production in 2021; its highest level to date. With the exception of production in Morocco and artisanal cobalt mining in Congo, cobalt is exclusively mined as a by-product, predominately of copper or nickel. Slower predicted supply growth rates in these commodities heightens supply shortfall risk in the cobalt market. S&P Capital IQ reports that from a total of 53 studies published since 2019 where cobalt was listed as a commodity, only 4 were listed as primary cobalt projects (2 FS, 2 PFS, 0 PEA). Only 10 of the 53 projects reported an annual cobalt production level, indicating the cobalt was not recoverable economically or is insignificant in the majority of theoretical by-product cases.

Given the demand growth linkages to the green transition, there is an increasingly strong emphasis being put on ethical supply chain sourcing by key buyers of incremental cobalt. Automakers have begun adopting various standards aimed at improving the traceability and credentials of their feedstock used in EV batteries. A raft of new offtakes have been announced with operations globally, with an emphasis on developed nations, including more recently direct investment and financing in exchange for offtake, which is a marked shift in approach reflecting supply uncertainties moving forward.

19.3.4 Cobalt in Europe and Finland

Cobalt has been gazetted in every EU list of Critical Raw Materials (CRM), since the first list was published in 2011. The EU defines a CRM as those which are most important economically and have a high supply risk. In 2020 it ranked cobalt as its second most important CRM for its industrial ecosystem. Supply risk looks at the country-level concentration of global production of primary raw materials and sourcing to the EU, the governance of supplier countries (including environmental aspects), the contribution of recycling (i.e., secondary raw materials), substitution, EU import reliance and trade restrictions in third countries. With 86% of domestic consumption imported predominately from the DRC, Europe's cobalt supply risk is deemed high.

In its CRM strategy, the EU notes that achieving resource security requires action to diversify supply from both primary and secondary sources, reduce dependencies and improve resource efficiency and circularity, including sustainable product design. In September 2020, the EU announced the formation of the European Raw Materials Alliance (ERMA) alongside its Action Plan on CRM, which aims to:

- develop resilient value chains for EU industrial ecosystems;
- reduce dependency on primary critical raw materials through circular use of resources, sustainable products and innovation;
- strengthen the sustainable and responsible domestic sourcing and processing of raw materials in the European Union; and
- diversify supply with sustainable and responsible sourcing from third countries, strengthening rules-based open trade in raw materials and removing distortions to international trade.

As a result of these and similar EU directives, and in recognition of Mawson's significance to the future cobalt supply chain, Mawson has been the beneficiary of over CAD2M in EU and Finnish government grants to support the study of the cobalt resource and extraction potential. The initiatives behind this support extend across the Finnish cobalt ecosystem.

Table 19-1 lists Europe's top 16 cobalt certified resources as published by S&P Global Market Intelligence, noting the number of projects which are considered in-active, typically due to unfeasibly low grades (including lack of by-product credit) or some other major impediment to development. Rajapalot is the 5th largest active cobalt resource in Europe.

Table 19-1: Europe's top 16 cobalt certified resources (Source: S&P Global Market Intelligence)

	Asset	t Co		Asset	t Co
1	Terrafame	300,000	9	Hautalampi	3,600
2	Kevitsa	28,226	10	Lappvattnet	3,506
3	Sakatti	20,300	*11*	*Saramaki**	*3,060*
4	Kuusamo	11,305	*12*	*Kiskama**	*1,793*
5	*Ronnbacken**	*10,085*	*13*	*Kuhmo**	*1,616*
6	*Calatrava**	*5,700*	14	Vuonos	1,064
7	**Rajapalot**	**4,835**	15	Kylylahti	800
8	*Rautavaara**	*4,115*	16	Lainejaur	680

**Inactive*

Finland hosts Europe's only cobalt mine production and dominates Europe's cobalt resources. Terrafame's Talvivaara mine produced 1,215 t Co in 2020, and Boliden's Kevitsa produced 495 t Co. Talvivaara was first configured to produce the less-refined nickel-cobalt sulphate, but has more recently expanded to value-add by producing higher quality, specifically for the battery industry, and separated nickel and cobalt sulphate battery chemicals. Kevitsa produces a nickel-cobalt concentrate for refining at its Finnish Harjavalta smelter.

Finland also hosts Europe's two other major downstream processing facilities. The Kokkola refinery owned by Umicore and ASX-listed Jervois Global refines half of the world's non-Chinese cobalt metal with a capacity of 6,250 t Co. Jervois recently completed a feasibility study to double production capacity to 12,250 t per annum. The Harjavalta smelter produces bycobalt which feeds the Nornickel Co sulphate refinery and BASF's cathode plant.

Other Cobalt Sources

Within Finland there is significant cobalt endowments in mine residues that are poorly defined. Under historic metal price environments, cobalt was largely overlooked in the large nickel and copper mines, and thus a lot of the cobalt was deposited to tailings and sulphide concentrate/residues as a non-economic waste product. Non-public sources estimate total contained cobalt of 50 to 100 kt. The Finnish industry and academia consortia BatCircle have been evaluating the scale and geometallurgy of these cobalt resources with a view to support future economic extraction, possibly in a centralised processing facility, for which concentrate production from Rajapalot would also be a potential feed source.

Similar exercises are ongoing within Australia and elsewhere, with ASX listed Cobalt Blue Holdings announcing the "Cobalt in Waste Streams Project" in collaboration with the Queensland government, which is attracting direct and indirect government funding.

19.3.5 Product marketing

The flowsheet as defined in this PEA contemplates production of a sulphide concentrate, with cobalt being the primary metal of value. A concentrate of this form is amenable to a variety of downstream process routes which are employed in both Finland and abroad. Engagement with processors and end users has commenced, however no contracts are yet in place.

Given the by-product nature of the majority of cobalt production, cobalt concentrates are not widely traded and thus the market is relatively opaque. Mawson has had preliminary discussions with a variety of potential downstream processers. These discussions have confirmed that Rajapalot's European location makes its cobalt attractive from an ethical and sustainable sourcing perspective, and a variety of potential downstream processors Mawson has engaged with have indicated that they have corporate strategies and plant configurations to accept third party ores. Several have expressed interest in a concentrate sample.

Owning to the large Finnish cobalt endowment and increasingly strong EU self-sufficiency tones, Mawson also believes there is good potential for additional domestic/EU facilities to be constructed in the future which would provide additional processing optionality.

Acknowledging that each technology route will yield a different set of treatment costs, payabilities, recoveries and logistics cost, this study assumes a combined logistics and treatment charge of EUR100/dmt, a fixed deduction of 0.075% (resulting in LoM average payability of 91.6%) and a refining charge of EUR0.20/kg Co. For the projected concentrate grade and cobalt price used, this is equivalent to an AMV of 60% of the contained cobalt in concentrate value, which is considered reasonable at this stage of study.

No value has been ascribed to potential by-product credits arising from the arsenic, sulphur (acid, etc), nickel and tungsten contained within the concentrates.

Optimisation Opportunities

Test work has demonstrated the potential to produce a simple cobalt concentrate, and this forms the basis of the PEA. The test work also showed amenability to produce two separate cobalt products through the use of magnetic separation to extract a significant proportion of the pyrrhotite, creating a lower grade non-arsenic bearing cobalt concentrate, and a separate cobaltite dominant concentrate with a higher cobalt grade, which also contains most of the arsenic (which may be considered deleterious by some downstream processors). The Managem Bou-Azzer facility in Morocco processes concentrate with a high arsenic grade using a roaster and hydromet facility to create London Metals Exchange (LME) grade cobalt metal and arsenic trioxide, among other things, as saleable by-product.

A small modular hydromet plant should also be evaluated due to the typical high recoveries and higher NSR resulting from a reduction in transport and third-party toll treatment charges associated with relatively low concentrate grades. Pressure Oxidation (POX) and the Albion processes appear to be good candidates for processing the entire concentrate, or the cobaltite bearing portion thereof. Bio-leaching can be implemented at a small scale (in tanks) and has precedents in Finland (Terrafame, Mundo Minerals/Elementis).

Future metallurgical and marketing studies will focus on maximising the NSR of the potential cobalt products whilst limiting on-site capex and process complexity.

19.3.6 Cobalt Pricing

Mawson has selected USD60,000 per tonne (USD27.2/lb) as a fixed long-term price to evaluate the economics of the Rajapalot project. This pricing is lower than spot and in line with average consensus forecast pricing (Figure 19-5) and is considered reasonable against peer group studies published since 2019.

Cobalt metal is traded on the LME, but owing to the concentrated nature of the producers, traders, intermediate processes and end users, most cobalt is traded directly via contract and not via the LME. In 2021 only 0.57% (841 t) of global cobalt demand was traded on the LME, so as a result the LME is not considered an accurate proxy for the cobalt price. A more representative measure of prices is provided by S&P Platts and Fast Markets market economist firms, who survey market participants (both buyers and sellers) to gauge actual pricing in the market. Platts is therefore considered to be a more reasonable basis to evaluate real market pricing, with LME shown for comparative reference.

Two forecast datasets are provided, reflecting the average of analyst predictions; one published Group 1 (a market data provider) containing up to 18 analyst annual datapoints, and one published by Group 2 (an investment bank) containing up to 15 annual datapoints. Constituent estimates are not necessarily unique.

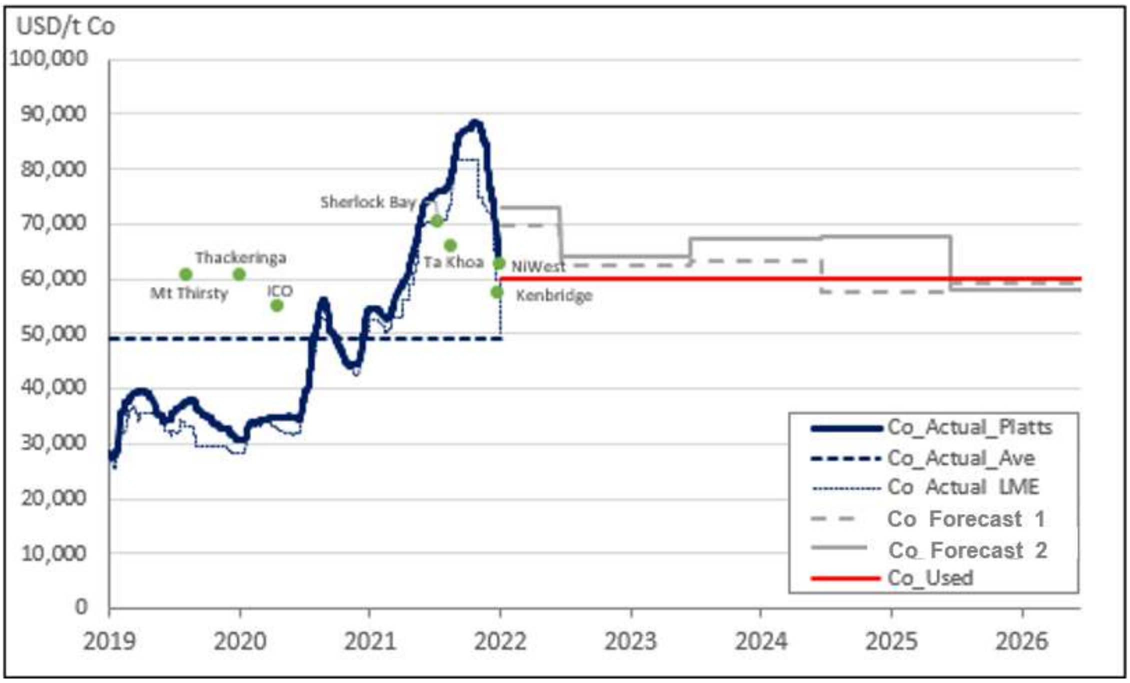


Figure 19-5: Cobalt price history, analyst consensus forecasts and sample published study prices (June 2022, real terms)

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

Mawson has completed a significant number of environmental studies and has been conducting baselining assessments across the relevant parts of its tenement package as well as surrounding areas, in support of its exploration activities and the evaluation of the impact of a future mining project. The studies have thus far confirmed that there are no such plant or animal species which would be unique to the project area or the larger vegetation zone area.

20.2 Environmental and Social Setting

20.2.1 Geographical setting

The property is located in the northern Finland region known as Lapland, close to the Arctic Circle and approximately 35 km west-southwest of the city of Rovaniemi. The project lies across the boundary of the local municipalities Ylitornio and Rovaniemi (Figure 20-1).

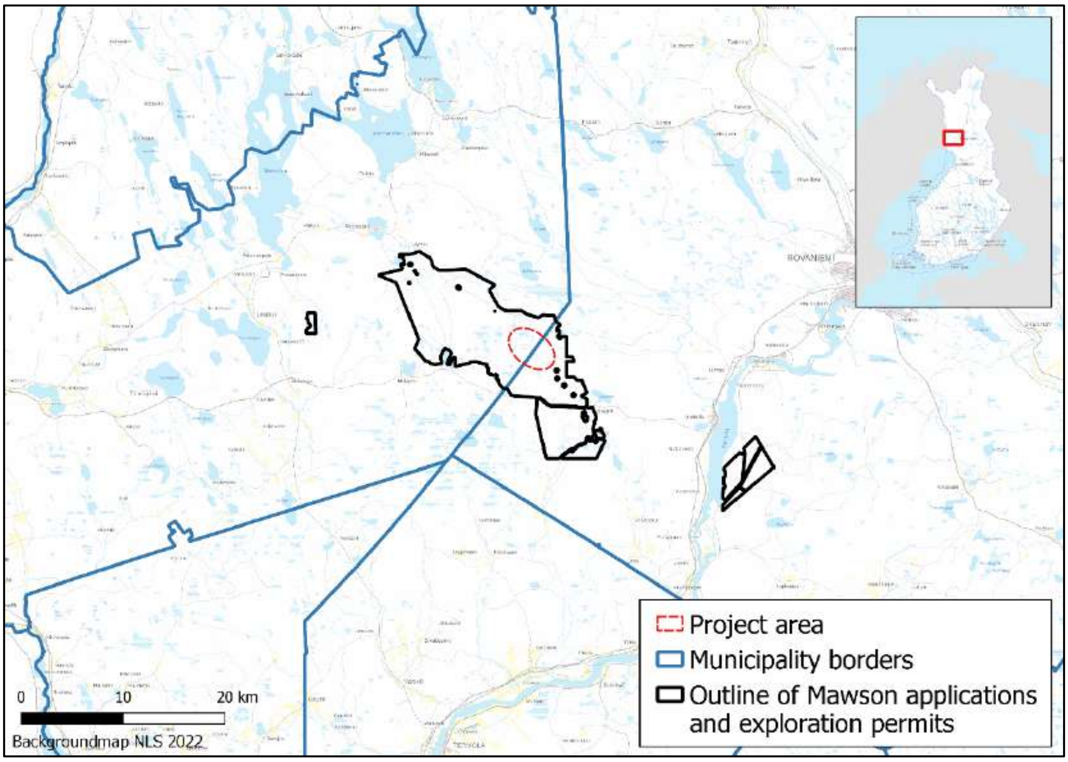


Figure 20-1: Location of the Rajapalot project and local municipality boundaries

20.2.2 Environmental Studies

Mawson has completed over ten years of flora, fauna and water base line studies and Natura impact assessments. Environmental studies have been conducted across the relevant parts of its tenement package as well as surrounding areas, in support of its exploration activities and the evaluation of the impact of a future mining project (Figure 20-2 to Figure 20-4, and Table 20-1).

The studies have thus far confirmed that there are no such plant or animal species which would be unique to the project area or the larger vegetation zone area.

Similar environmental studies will be carried out on planned routes of the water discharge pipeline and powerline route.

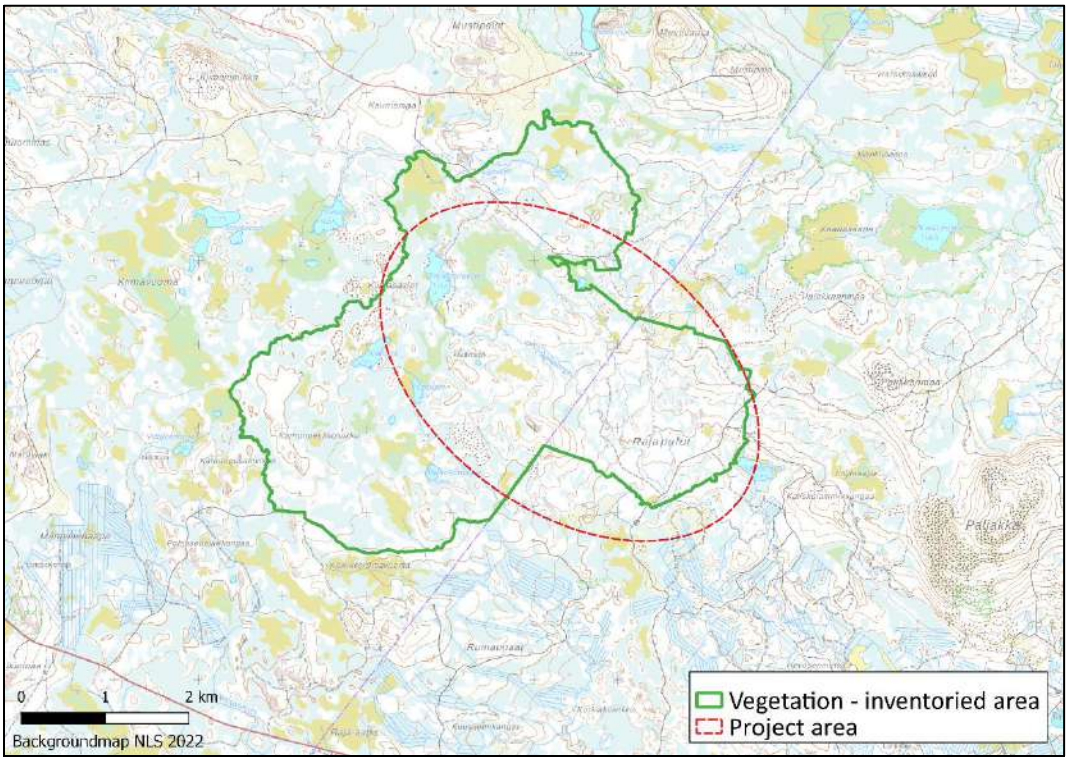


Figure 20-2: Vegetation inventoried area in relation to Project area

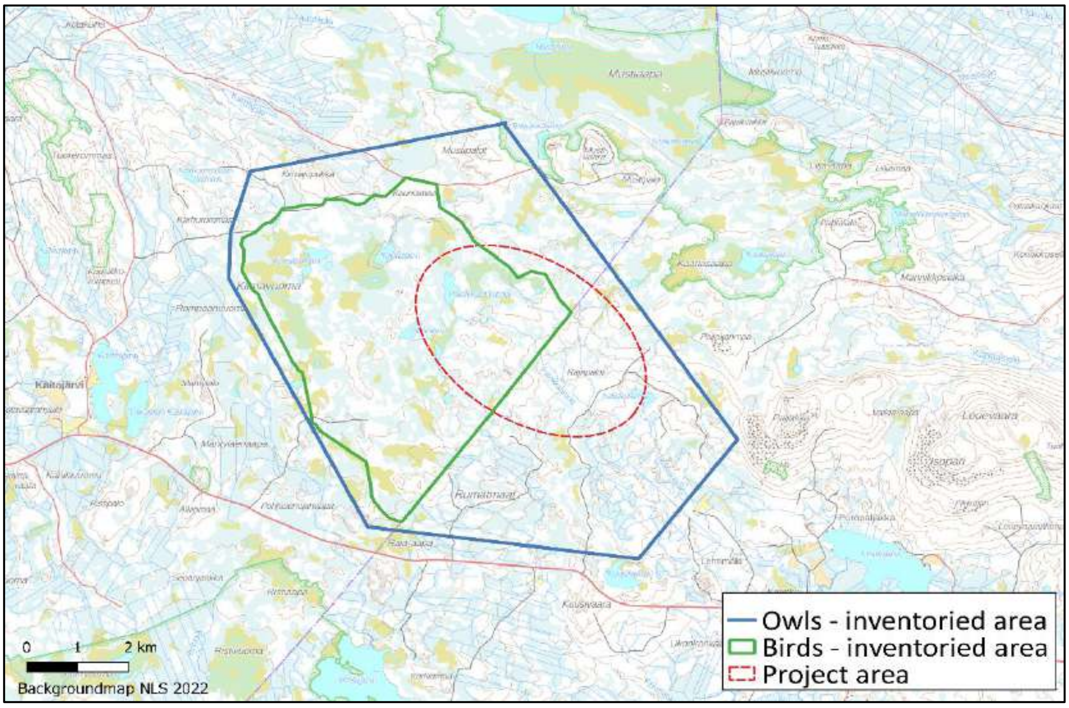


Figure 20-3: Owls and birds inventory area in relation to Project area

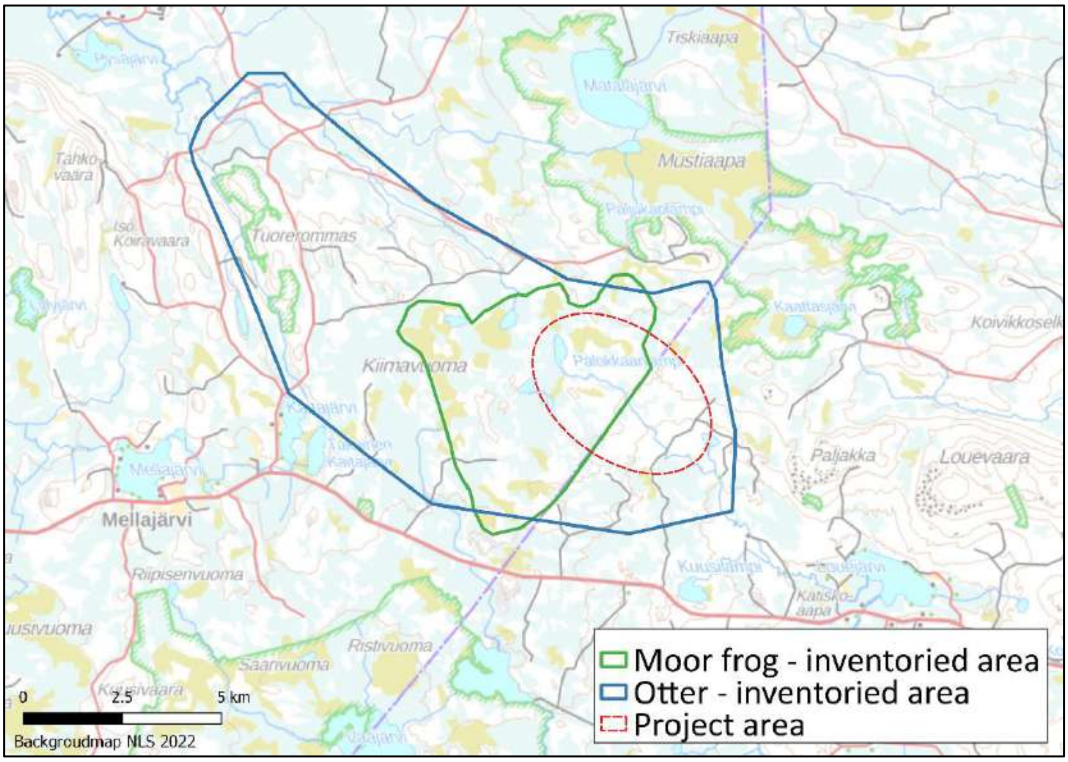


Figure 20-4: Moor frog and Otter inventoried area in relation to Project area

Table 20-1: Summary of most important nature studies implemented in project area and surroundings 2010 to 2022

Study	2010 to 2018	2019	2020	2021	2022
Water quality	x	x	x	x	x
Fish					x
Benthic and bottom fauna	x			x	x
Water flow			x	x	x
Water levels	x	x	x	x	x
Groundwater				x	
Habitat types	x	x	x	x	x
Vegetation	x	x	x	x	x
Polypores	x				
Birds (all)	x	x	x	x	x
Owls	x	x	x	x	x
Endangered species monitoring	x	x	x	x	x
Capercaillies	x				
Otter	x	x	x	x	x
Frogs	x	x	x	x	x
Insects	x	x			
Weather conditions (rain, wind)			x	x	x
Bats, river pearl mussel, flying squirrel	x				

20.2.3 Catchment areas

All the Inferred Mineral Resources discussed in this report lie within the catchment of the Kemijoki River. The starting point of River Kemijoki is in northwest of Finland, and it runs across Northern Finland to the Gulf of Bothnia. The River Kemijoki runs through cities of Kemijärvi and Rovaniemi before reaching the Gulf of Bothnia at the city of Kemi.

The Rajapalot project area is near the catchment area of Louejoki, which flows into River Kemijoki, which is the longest and biggest river in Finland (Figure 20-5).

The Rajapalot project area is near the catchment area of River Tornionjoki which flows into River Kemijoki and is the selected option for the location of the water discharge pipe.

The Rajapalot site is near the watershed of two streams, Palokkaanjoki and Kuusijoki, both flowing into River Louejoki. All the resource areas of Rajapalot are in the catchment area of the Palokkaanjoki stream which discharges into Lake Louejärvi while the Kuusijoki stream joins the River Louejoki after the lake (Figure 20-6). The River Louejoki is a tributary of the River Kemijoki.

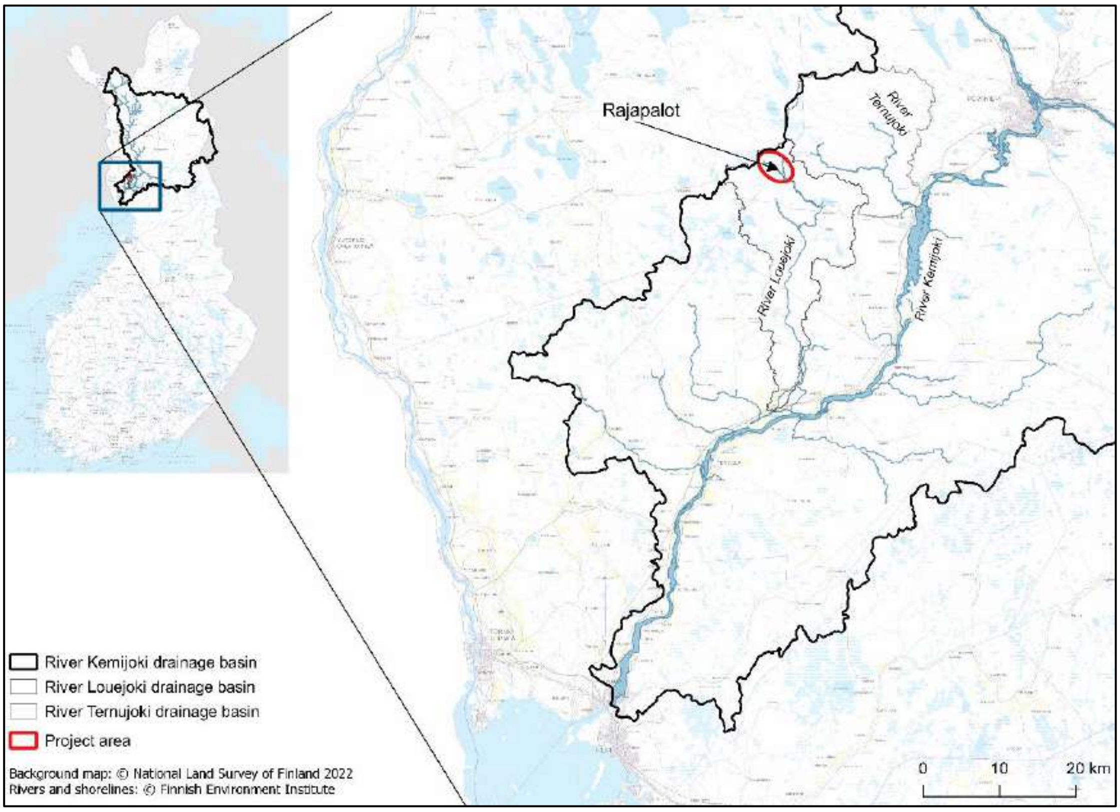


Figure 20-5: Catchment areas of Kemijoki, Louejoki and Ternujoki rivers

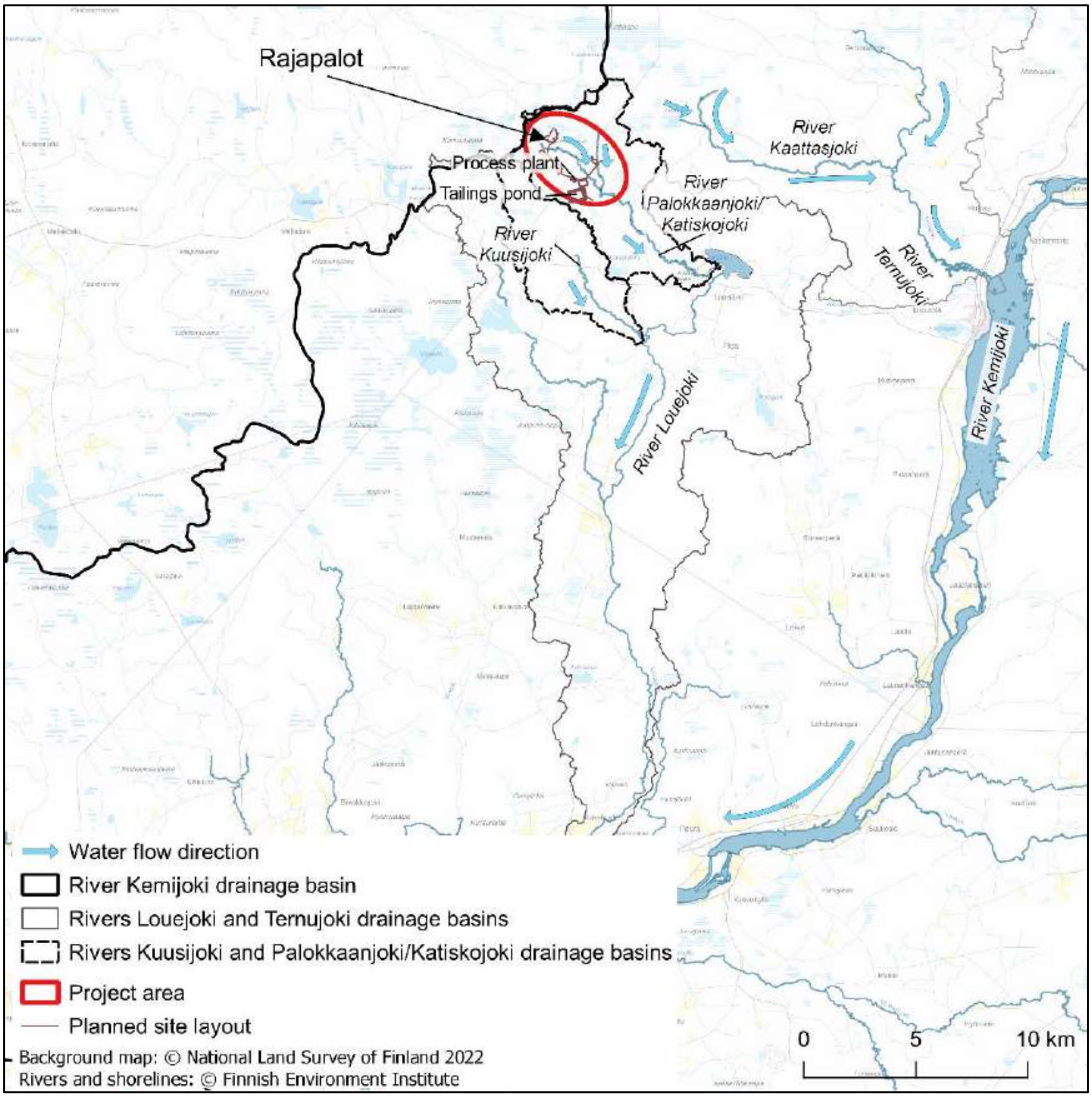


Figure 20-6: Catchment areas of Palokkaanjoki, Katiskojoki and Kuusijoki streams

20.2.4 Surface water status

In EU countries, The Water Framework Directive (WFD) prescribes environmental objectives and an adaptive water governance system. The environmental objectives in Article 4 of the WFD impose two main obligations on the EU Member States: to prevent deterioration of the status of all surface and groundwater bodies within the Union, and to protect, enhance and restore all water bodies in order to achieve 'good water status', originally by the end of 2015 and with full implementation by 2027. To have an environmental permit, ecological status of classified water bodies below the project area cannot deteriorate. This also applies to post-closure impacts. To minimize surface water impacts, discharge quantity and quality can be adjusted by water treatment which forms the basis of the PEA assumption. Different discharge place alternatives will be assessed in future phases of study.

The ecological condition of water bodies adjacent to Rajapalot are shown in Figure 20-7, and described as follows:

- River Kemijoki being such a long river is divided into shorter water bodies. Downstream from the Rajapalot project area is a water body called Ala-Kemijoki, which starts from Rovaniemi and continues to the Gulf of Bothnia. Ecological status of river Kemijoki (Ala-Kemijoki) is moderate, and heavily modified due to several hydropower plants blocking the passage of fish. There are 16 hydroelectric power plants constructed on the water way, 12 of which are upstream from the project area. The physio-chemical quality of river Kemijoki is high, but because of the hydroelectric power plants natural habitats of e.g. salmonid fish and typical bottom fauna of rapids are missing.
- Ecological status of both River Louejoki and Lake Louejärvi is good. The River Ternujoki is not downstream from the Rajapalot project area, but is the selected option for the location of discharge pipe. The ecological status of River Ternujoki is good.

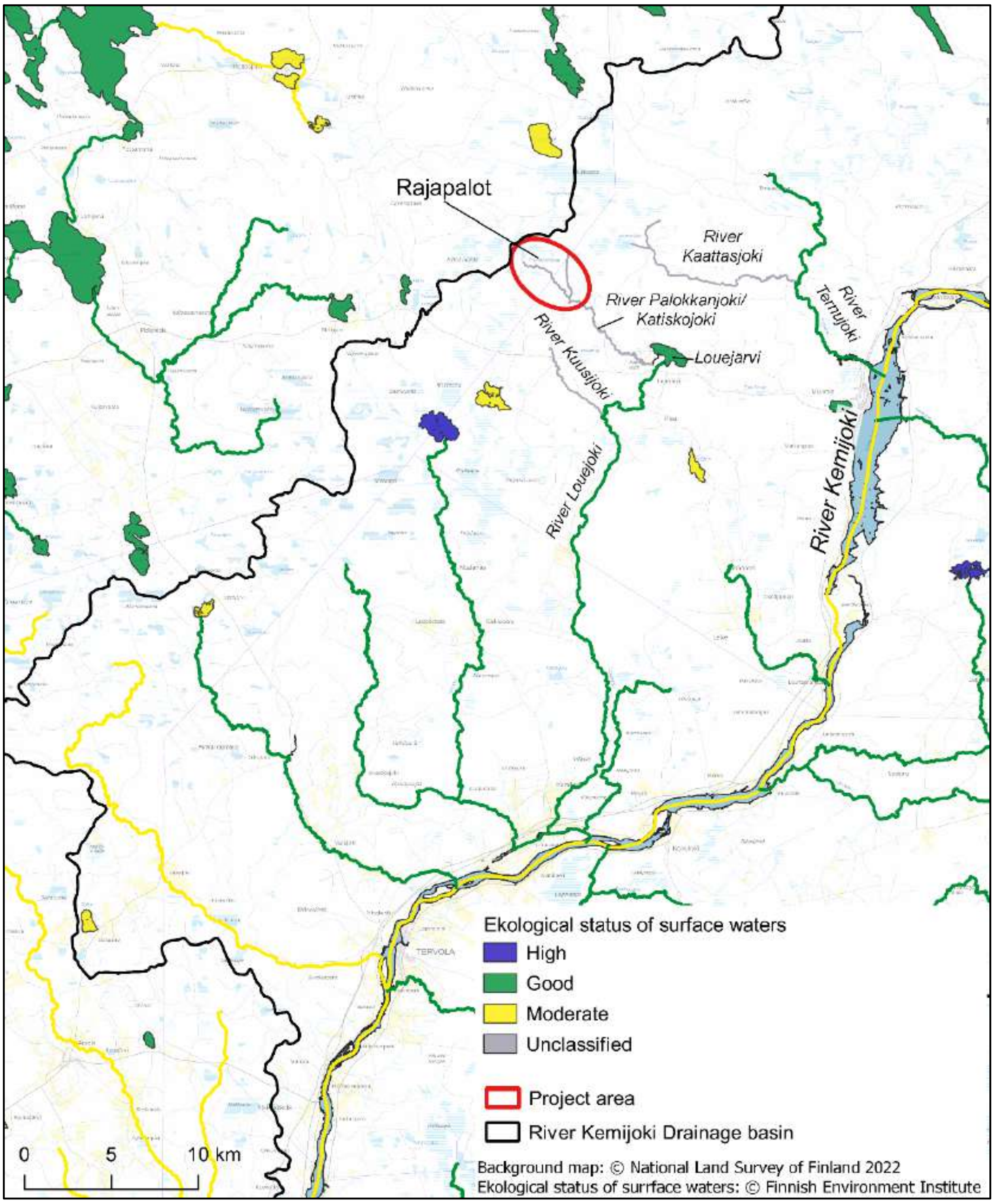


Figure 20-7: Ecological status of water bodies

20.2.5 Groundwater

In Finland the main aquifers were mapped and classified in the 1990s into so-called groundwater areas according to the needs of water supply management and protection requirements (previous classes I–III). The legislation on protecting groundwater was updated in February 2015 and groundwater dependent ecosystems (both terrestrial and surface water) were acknowledged. The groundwater area classification is following:

- Class-1 important for water supply.
- Class-2 suitable for water supply.
- Class-E groundwater dependent ecosystems (both terrestrial and surface water).

The nearest classified groundwater area (Kaunismaa, id 12976153, class 2) is located 1.5 km NWN from the project area (Figure 20-8). Kaunismaa lies in the Torniojoki drainage basin. The AFRY 2020 report estimated ground water flows to the southeast, so the project is unlikely to impact this body.

The nearest classified groundwater area in the Kemijoki drainage basin side is in Louejärvi (id 12699263, class 2E) which is 10 km SE from the project area. In addition, there are three other classified groundwater areas Kolvavaara-Louejärvi (id 12699106, class 1E), Ketunmaa (id 1284517, class 2E) and Loue (id 1284503, class 1) that lie in the drainage basins between the project area and Kemijoki. Two of these Kolvavaara-Louejärvi and Loue, have altogether four water intake stations.

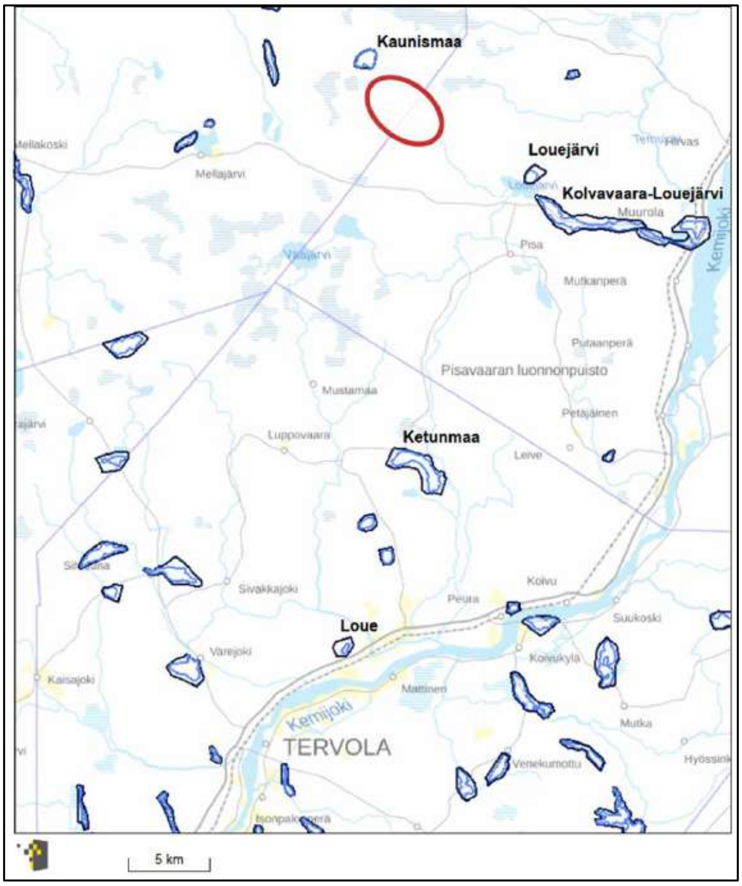


Figure 20-8: Classified groundwater areas and Project area

Other local groundwater users

There are no other groundwater users within the project area.

20.2.6 Hydrogeology

Hydrogeology information is provided in Section 16.13.4 of the PEA.

20.2.7 Soil

Soil of Rajapalot area (Figure 20-9) consists of mixed component soil and peat. Outside Mawson tenure areas, there are also smaller areas of sandy or gravelly soil types. There are a lot of peatlands, few lakes or ponds and rivers or riverbeds in the area. The abundance of peatlands suggests that in these areas the soil consists of low hydraulic conductivity soils such as clay, silt, or moraine. In the tenure area the topsoil is rather thin, mainly under 5 m, and at most only up to 8 m thick.

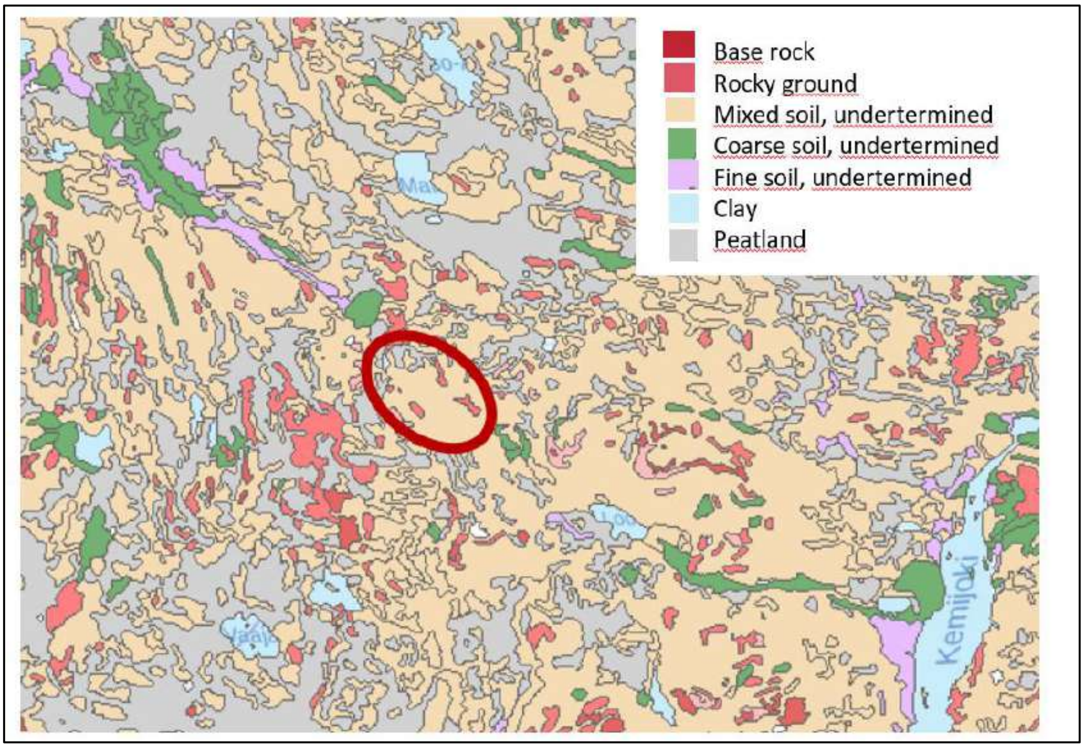


Figure 20-9: Soil map of Project area and its surroundings

20.2.8 Habitat, flora and fauna

The forests belong to the boreal coniferous forest zone. The dominant tree species in Northern Finland is Scots pine. Low-lying shrubs are common, including for example juniper, blueberry, lingonberry, cloudberry, lichens and sphagnum moss blankets the forest floor throughout. The project area outside the Natura area is forestry land and the forests are in commercial use. The forestry area and its surroundings have been ditched in many places.

20.2.9 Nature protection areas

The project area partially overlaps the Mustiaapa-Kaattasjärvi Natura 2000 area (Figure 20-10). Mustiaapa-Kaattasjärvi is a large area of 6,117 hectares. The protection grounds of Mustiaapa-Kaattasjärvi Natura 2000 area are based to the EU´s Habitats Directive and are listed in EU

Commission´s official Data form in:

https://natura2000.eea.europa.eu/Natura2000/SDF.aspx?site=FI1301301

Mustiaapa-Kaattasjärvi Natura 2000 area is important especially for the waterfowl while many species belonging to the listed species of EU´s Bird Directive nest there. One of the bird species living in the area is endangered. These bird species living or nesting in the area occur also elsewhere in the EU, Finland and Lapland broadly. One of the bird species is endangered and has a territory at outskirts of the project area.

The habitat types and vegetation of Mustiaapa-Kaattasjärvi Natura 2000 area are typical for northern Finland, especially South-Western Lapland. Two endangered plant species occur in the Mustiaapa-Kaattasjärvi Natura 2000 area, but the occurrence locations do not overlap with the resource areas or areas where mining infrastructure will be placed.

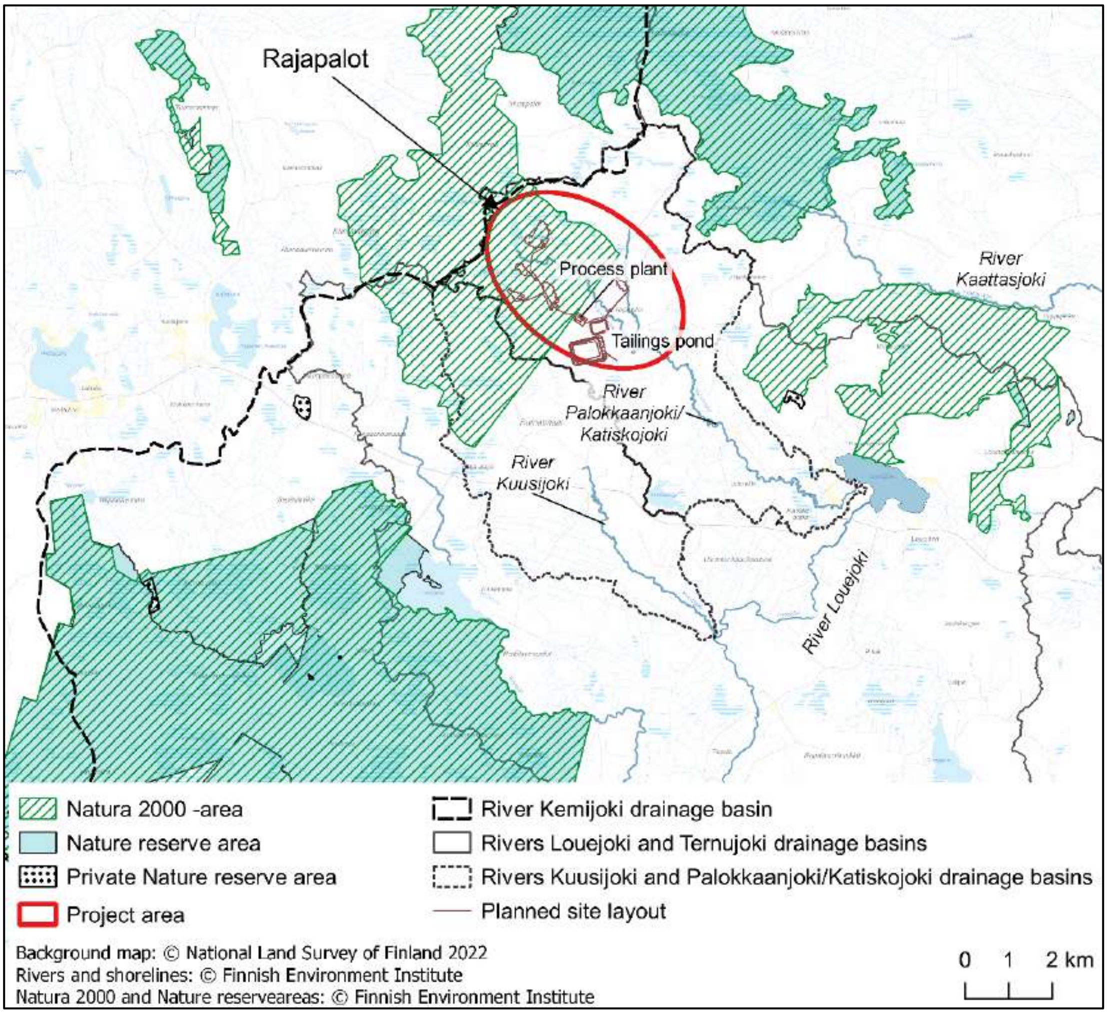


Figure 20-10: Natura 2000 area, private Nature reserve area and Project area

20.2.10 Landscape, cultural heritage and monuments of antiquity

There are no landscape areas of national or regional importance in the vicinity of the project area. There are either no known historical or cultural sites or monuments close to the project site. A thorough audit to identifying possible sites for any valuable cultural environment and/or monuments of antiquity will happen during EIA field studies.

20.2.11 Local communities

The nearest villages to the project are Meltosjärvi, Mellanjärvi and Muurola, which are shown in Figure 20-11.

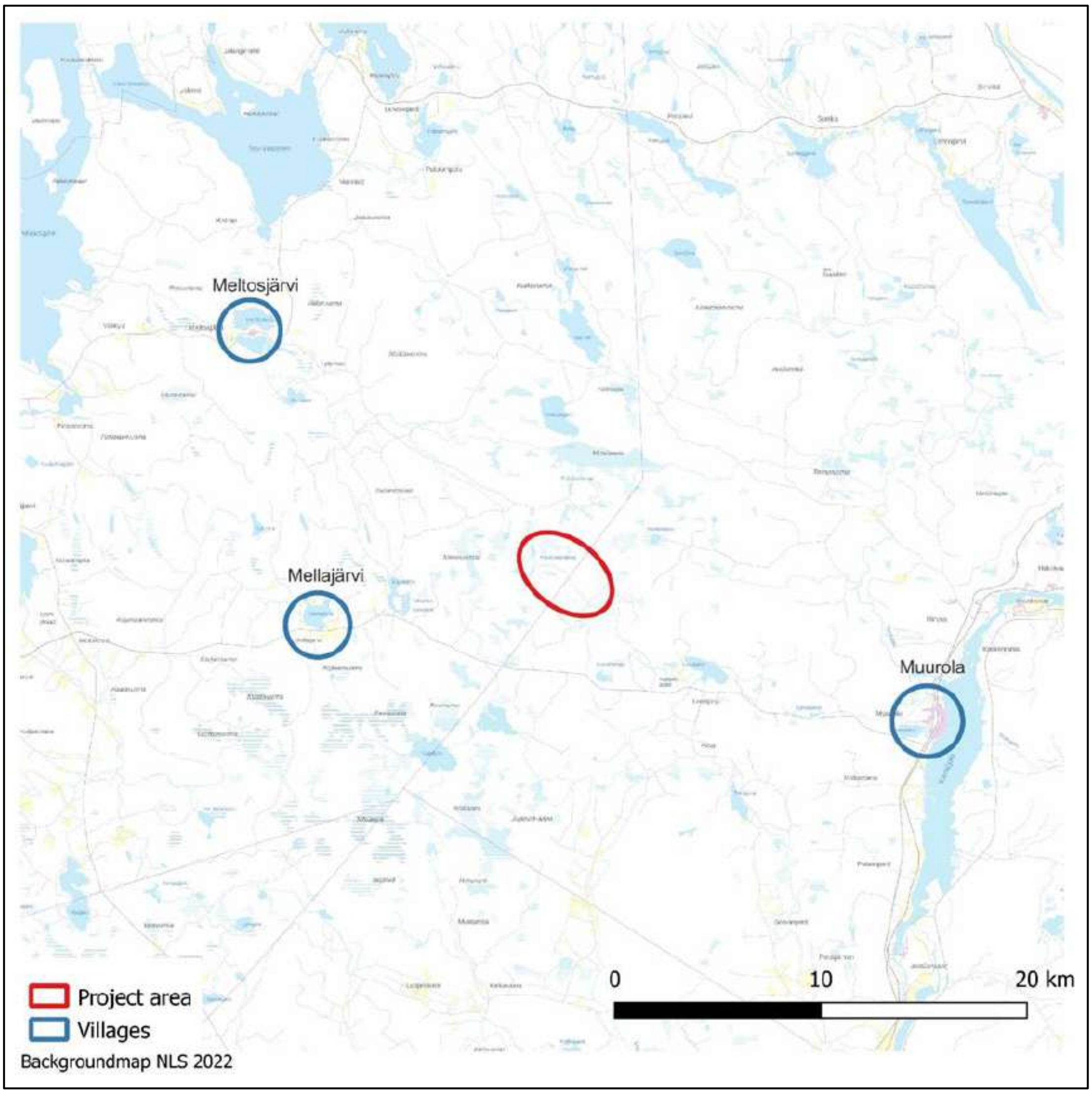


Figure 20-11: Rajapalot project area and nearby villages

20.2.12 Social structure

Homes and holiday homes in the region are shown in Figure 20-12. There are no homes or holiday homes at the project area. Only few holiday homes are near the project area.

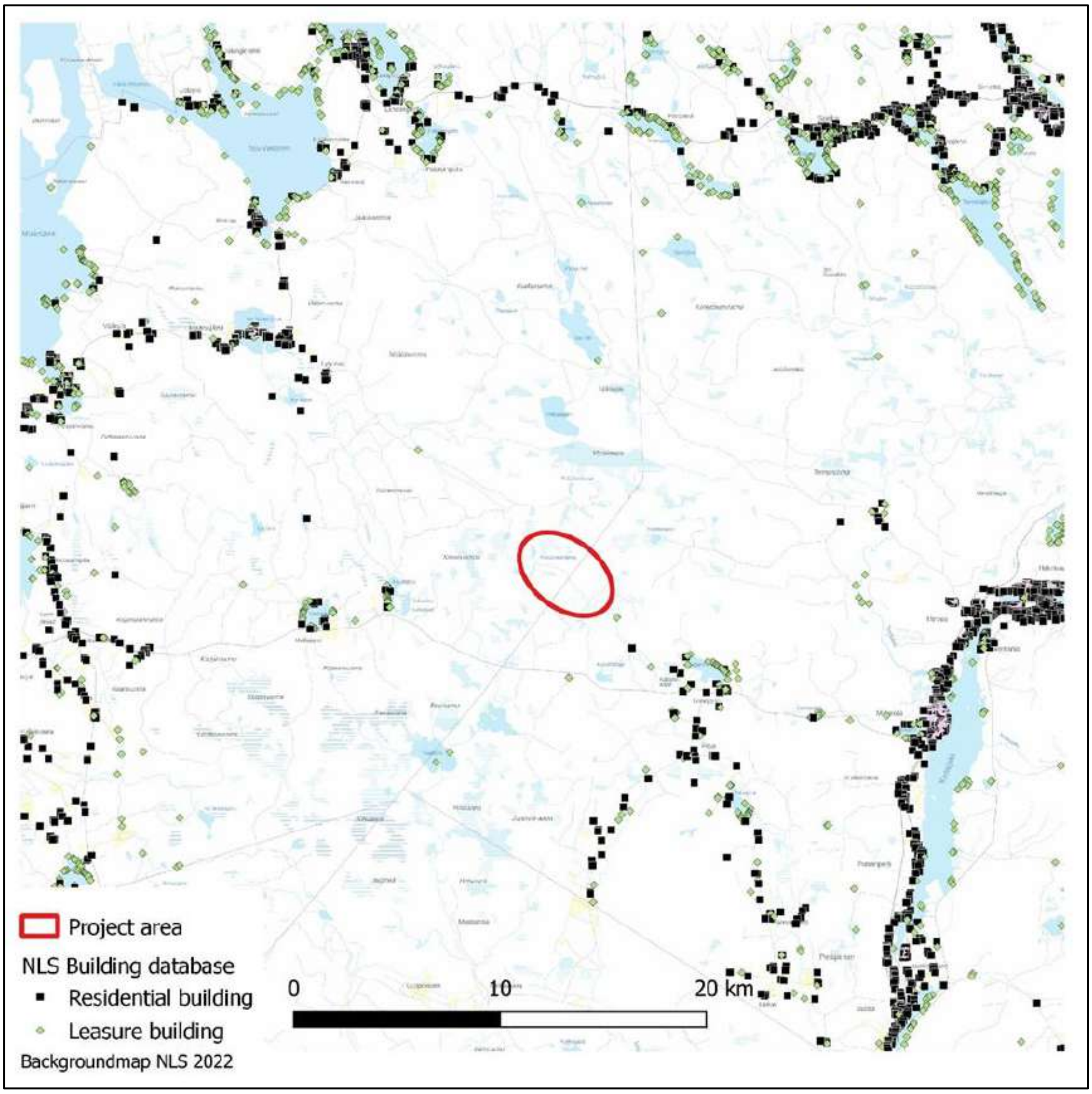


Figure 20-12: Population centers (black squares mark permanent residential houses and light green marks holiday homes)

20.2.13 Resettlement issues and Indigenous people

The project area is not in or close to Sami people homeland area which locates in the northernmost parts of Finland (Figure 20-13). There are no resettlement issues of Sami people related to the project.

There are no residential or holiday homes close to the project area, thus no known resettlement issues.

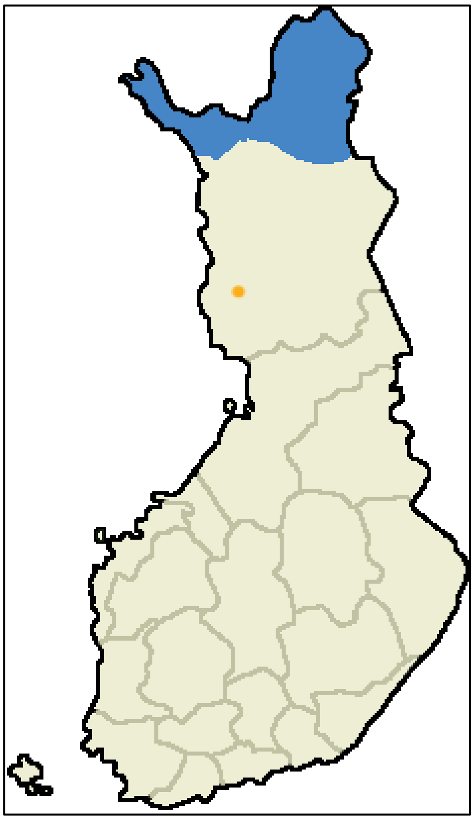


Figure 20-13: Sámi homeland location in Finland (shaded blue) and project area location (yellow point)

20.2.14 Reindeer herding

In addition to local residents and landowners, reindeer herders are an important stakeholder group that uses the project area. In Finland anyone living within the area of Finnish reindeer husbandry who is a citizen of the European Union has the right to own reindeer, in contrast to the situation in Norway and Sweden, where only Sámi are legally permitted to own reindeer. The reindeer husbandry area is demarcated for reindeer herding (Reindeer Husbandry Act 848/1990). The area covers 122,936 km^2, approximately 36% of Finland's total area. The reindeer herding area consists of all of Finnish Lapland and northern parts of Northern Ostrobothnia and Kainuu regions excluding some smaller urban areas.

Mawson has a balanced interaction relationship with the reindeer herders operating in the area. Reindeer herders in the area belong to a single cooperative called Palojärven paliskunta. Reindeer herders have representation in the official steering groups related to the EIA and land use planning processes. Impacts to reindeer herding as well as the mitigation measures determined in later stage of the project planning will be assessed in detail as part of the EIA.

20.3 Project alternatives

In the EIA phase project alternatives will be studied, such as waste rock locations and configuration, water discharge pipe route and discharge location, route alternatives for 110 kV power line, route alternatives for internal roads.

20.4 Regulatory Framework and Approvals

20.4.1 General Information Concerning the Regulatory Framework

A significant number of regulations are applicable to mining projects in Finland. The legislation regulates all aspects of the mine life cycle, from exploration, through permitting, construction, operations, closure and post-closure. The key legislation applicable to the project is listed in Table 20-2.

Table 20-2: Key legislation applicable to mining project in Finland

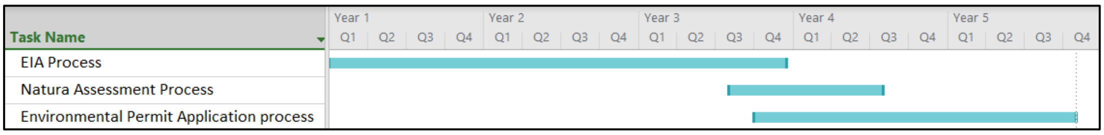
Legislation	Description
Act on Environmental Impact Assessment Procedure (252/2017) and Decree on EIA Procedure (277/2017)	Applies to all projects that may be expected to have considerable negative environmental impacts, of which mining is included.
Mining Act (621/2011) and Mining Decree (391/2012), Decree on Mine Safety (1571/2011)	Outlines the conditions for exploring and exploiting 'extractable minerals'; Governing mineral exploration and mining activities.
The Environmental Protection Act (527/2014)	Implements the European Union directive on Integrated Pollution Prevention and Control (IPPC). Contains the stipulations on environmental protection. Aims at the prevention and minimization of pollution and is applied to all activities that cause or may cause significant environmental impacts. Applies to activities that generate waste, and to the recovery and disposal of waste.
Environmental Protection Decree (713/2014)	Implements the objectives of the Act, including permit requirements and permitting procedure, BAT assessment.
Water Act (587/2011, 611/2017), Water Decree (1560/2011), Decree on dangerous and harmful substances on water bodies (1022/2006), Act on waterbody management (1299/2004), Decree on waterbody management (1040/2006).	Regulates protection, use and management of surface water and groundwater resources.
Land Use and Building Act (132/1999)	Aims to organize land use and building to create the basis for high quality living environments, to promote ecologically, economically, socially and culturally sustainable developments, to ensure participatory planning processes. Most importantly governs the land use planning procedures and construction permits. Renewal of this Act has been on-going and was recently taken into Parliament procedure but only regarding the Building Act content.
Nature Conservation Act (1096/1996) and Nature Conservation Decree (160/1997)	Outlines nature and landscape conservation and management; nature conservation areas, protected and threatened species and natural habitats, Natura 2000 areas.
Waste Act (646/2011), Waste Decree (179/2012), Mining Waste Decree (190/2013), Decree on landfills (331/2013)	The Waste Act is the overarching law governing waste management in Finland. It covers all types of waste also including mining waste but Mining Waste Act specifically regulates mining waste management.
Dam Safety Act (494/2009) Dam safety instructions, Decree on Dam Safety (319/2010)	Regulates dams and embankments including TSF dams.
Government Decree on air quality (79/2017).	Finnish air quality objectives include binding limit values and nonbinding national guideline values. The mandatory air quality limit values correspond to those of the European Union's Air Quality Directive 2008/50/EC.
the Act on Compensation for Environmental Damage (737/1994)	This act guarantees full compensation for environmental damage in cases where those liable for compensation are insolvent, or the liable party cannot be identified.
The Council of State Decision (993/1992) regarding noise	Provides outdoor and indoor noise level guidelines.

Chemicals Act (599/2013), and Decree on the Industrial Processing and Storage of Dangerous Chemicals 59/1999. Also REACH Decree (1907/2006) and CLP decree (1272/2008).	The purpose of this Act is to protect health and the environment from hazards and harm caused by chemicals. Chemical safety
Reindeer herding Act (848/1990), Reindeer herding Decree (883/1990)	Regulates the reindeer husbandry livelihood and other livelihoods possibly affecting it.
Cultural heritage Act (295/1963)	Regulates how cultural heritage shall be considered and preserved.
Electricity market Act (588/2013)	Regulates e.g. powerlines, construction and land access.
Act on redemption of fixed assets and special rights (603/1977)	Governs how redemption procedures shall take place and be compensated.
Act on appealing procedures (808/2019, 434/2003)	Defines how the appealing procedures take place

20.4.2 Finnish Permitting Process – Environmental and Water Permits

In Finland, environmental permits are required for all activities involving the risk of pollution of air and water or contamination of soil. An indicative permitting process timetable is presented in Table 20-3. Mawson has already commenced its EIA process, which is usually carried out simultaneously with other technical and/or financial studies.

Table 20-3: An indicative timetable of permitting process for Rajapalot Project



The permitting authority is Regional State Administrative Agency for Northern Finland. After filing a permit application, the authority will publish the application to allow the relevant other authorities, and anyone affected by the plans to comment and make proposals concerning the requirements for the permit. Permit decisions may be appealed to the Administrative Court of Vaasa and subsequently to the Supreme Administrative Court.

Additional deviation permits according to the Nature Protection Act (1096/1996) or Natura 2000 assessments may be required in case that the EIA determines there has been a significant impact on sensitive species or habitats. The Centre for Economic Development, Transport and the Environment (ELY-Centre) can grant an exemption under defined conditions described in Article 16(1) of the Directive.

20.4.3 EIA Process

An EIA (environmental impact assessment) procedure for mining projects in Finland is required prior to the permitting. In Finnish legislation for Environmental Impact Assessment are EIA Act (252/2017), EIA Decree (277/2017) and EU directive 2011/92/EU.

EIA and permitting are separate processes and before environmental permit decision, the EIA must be completed.

The purpose of the EIA process is to compare project alternatives and promote dialog with different stakeholder groups. Aim is to produce information to support technical design of the project and vice versa, technical design solutions and mine plans are needed in the EIA to establish most viable project alternative all viewpoints considered.

Factors that will be assessed in the EIA procedure include impacts on soil, water, air, climate and climate change, organisms and biological diversity, and on the wider utilisation of natural resources. Assessments of impacts on water typically feature most prominently in the EIA for mining projects. In addition, impacts on human health, living conditions and amenity as well as on spatial structure of communities, buildings, landscapes, townscapes, and cultural heritage will be assessed as part of the EIA procedure.

A guide to climate impact assessment has been issued by the Ministry of the Environment (Publications of the Ministry of the Environment 2021:18, Climate Impact Assessment, assessment in the EIA -Identification of impacts and consistent treatment). The report presents checklists for different types of projects that will help the authorities in the guidance and supervision of climate impact assessments. The lists address climate impacts from the perspective of both the mitigation of and adaptation to climate change.

Appropriate Natura 2000 assessment is carried out in accordance with the Nature Conservation Act (1096/1996). Natura assessment is carried out as a separate process of the EIA. The preparation of Natura 2000 assessment can be started during the EIA assessment but will only be finalised once the results of the EIA assessment are known. Once the results of the EIA assessment are known, a decision can be taken to prepare an application for an environmental permit. Mawson carries out Natura assessment for project alternative for which environmental permit will be applied for.

Based on the results of the Natura assessment, it can be determined whether exemption from Natura 2000 network is eventually needed or not. Exemption is not needed if harmful impacts on the protection grounds and nature values of the Natura area can be prevented either completely or they can be mitigated to the extent that the impacts do not significantly weaken the protection values of the area. So if the appropriate assessment concludes that there is an adverse effect on integrity of the site, it will be necessary to examine whether preventive or mitigation measures can be introduced to remove these effects.

These mitigation measures must be directly linked to the likely impacts that have been identified in the Natura assessment and can only be defined once these impacts have been fully assessed and described. The identification of mitigation measures, like the impact assessment itself, must be based on a sound understanding of the species and habitats concerned. The mitigation measures may, for instance, involve a change or restriction on the dates and timetable for implementation of some activities (for example, avoiding certain works during the breeding season of a particular species). If these mitigation measures can successfully remove or prevent the adverse effects identified, then the project can be approved.

20.4.4 Mineral Rights and Mine Safety Permitting

Mineral rights, including decisions concerning mining permits, are regulated under the Mining Act (621/2011). TUKES, the Finnish Safety and Chemicals Agency, is the responsible authority.

The Mining Act also requires completion of an EIA before the mining permit can be granted. The EIA report and reasoned conclusion from the competent authority must be attached to the mining permit application. A mining permit is required before an environmental and water permit can be granted. In Finland, the mining permit covers the whole operation area (not just the deposit) and requires therefore quite advanced project plans.

Ownership of the mining area land is not required by law, but ownership may simplify many issues related to compensations and liabilities.

To build a mine and start up the actual mining operations, a mine safety permit is required. According to the Mining Act 621/2011, this is called a mine safety permit (section 12 of the Mining Act) and is mostly related to work safety items. The application requires, for example, a mine general plan and an internal rescue plan. This permit governs all safety-related issues and is granted by TUKES.

20.4.5 Land Use Plans

The Land Use and Building Act (132/1999) structures the land-use planning system and contains provisions to ensure the environmental, economic, social and cultural sustainability of planning. Together with the Local Government Act (410/2015), which outlines the responsibilities of municipalities, it forms the framework legislation for land-use planning.

Regional land use plans (phase provincial plans) are the highest-level plans. They set out principles for land use and community structure, and designate areas that are needed for regional development. The phased provincial land use plan is a long-term plan and a guideline for the municipalities when drawing up and amending local master plans and local detailed plans.

Municipalities prepare two types of plans. Local Master Plans contain a description of the urban structure of the municipality and contain general objectives for community development. Local Detailed Plans are drawn up to guide development in particularly important or sensitive areas.

Two municipal areas where the Rajapalot gold-cobalt project is located, the City of Rovaniemi and Municipality of Ylitornio, at the request of Mawson, formally decided to start the sub-area Local Master land use planning processes. Both municipalities made decisions to propose to the Regional Council of Lapland (Lapin Liitto) to start the phased provincial land use plan for the Rajapalot gold-cobalt project. The board of Regional Council of Lapland agreed the proposal and decided to start the land use planning process as well.

Mawson started the preparation of the EIA and land use processes in late 2020. The initiation of EIA and land use planning is demonstrative of the long term, strong local stakeholder support for one of Finland’s strategic gold-cobalt projects and also to de-risk the project, as resource-expansion drilling will continue.

If ecological compensation and/or offsets will need to be done, their locations and new land use statuses will be presented in the land use plan.

20.4.6 Upcoming and recent changes in Regulation

Several mine-related legislative reforms are currently underway. The most relevant legislative updates concern the Mining Act (621/2011), the Nature Conservation Act (1096/1996), the Environmental Protection Act (527/2014), the Land Use and Building Act (132/1999) and Climate Change Act (609/2015).

Drafts have been submitted to parliament, but the proposal content can change during approval procedures so there is no certainty around any changes. Mawson does not believe any of the changes provided for in the drafts have a material impact on the physical configuration, economic viability or increase /decrease in the likelihood of the project permits being granted.

20.4.7 Project Approvals Status

Mawson has exploration permits for current drillings. No other project approvals have yet been applied.

20.5 Management Approach

Finland has rigorous regulatory processes with strict environmental standards and Mawson is committed at this early project stage to work with the regional and national authorities and broader stakeholder groups to develop the project in a responsible way. Mawson is a member of Finnish Business & Society (FIBS), the largest corporate responsibility network not only in Finland but also in the Nordic countries.

Mawson acknowledges that Environmental, Social and Governance (ESG) forms a comprehensive framework for our Company to successfully navigate and balance the benefits of our projects to the planet, people and profit. Mawson has had an active ESG program operating for many years, which is being constantly adjusted as its projects grow and develop. Its ethos is documented in the Company's Corporate Environmental, Health and Safety policy which is available on its website. As a listed company, the directors of Mawson have a legal obligation to ensure this policy is adhered to.

The Company complies with The Finnish Network for Sustainable Mining "Standard for Sustainable Exploration". The standard is comprised of Guiding Principles and three Protocols, which cover the entire lifecycle of exploration activities.

Mawson has identified other possible industry standards are which may be applicable:

- The Responsible Gold Mining Principles (RGMP).
- The "International Cyanide Management Code For the Manufacture, Transport, and Use of Cyanide In the Production of Gold" (Cyanide Code).
- The Global Industry Standard on Tailings Management (GISTM).
- The Cobalt Industry Responsible Assessment Framework (CIRAF) Mawson's project moves ahead, the company is committed to follow industry management tools for the project.

20.6 Stakeholder Issues and Stakeholder Engagement

Mawson considers stakeholder engagement and collaboration to be a critical part of the potential development of the Rajapalot project, and social aspects will be a key part of the EIA preparation process. In February 2021, Mawson announced that two key planning processes, the EIA and land use planning have been initiated for the Rajapalot gold-cobalt project in Finland. These processes represent a formalisation of the engagement processes that had already been ongoing.

The purpose of the EIA procedure is to generate information on the environmental impacts of a project, facilitate the consideration of environmental issues in planning and decision-making processes, and give the public and other stakeholders opportunities to participate in and affect these processes and planning.

20.6.1 Stakeholder Engagement

Stakeholder engagement is continuous with the locals. Mawson has created long-term relationships with people who live in the close by villages and cities. Interaction with different stakeholder groups is open and transparent. Company visits regularly in the meetings and events organized by the reindeer herders, village associations, municipalities, entrepreneurs, universities or other research bodies or any other important stakeholder group who acts in the area. Mawson documents all meetings and events participated to an excel file.

Mawson has offices in both municipalities, Rovaniemi and Ylitornio, and the interaction and communication with the municipal officials and residents is open and continuous. In addition, the company is in constant interaction with regional and national authorities, whose legal task is to contribute to the planning and implementation of the project. The interaction is recorded, and feedback received from the stakeholder groups is responded to.

Company has ordered studies from independent research bodies to get detailed information about the opinions of the stakeholders using or living in the nearby areas. Studies have been implemented for example regarding company's operations, stakeholder´s own expectations related to mining in the area and mining industry in general. The results are used for the continuous development and improvement of company´s operations and planning work. Mawson also collects feedback through an open on-line feedback channel on the rajapalot.fi website. Direct feedback received outside from formal permitting processes is being considered and integrated in planning and stakeholder engagement work.

20.6.2 Social acceptance studies

Social acceptance of the locals and their attitudes towards exploration and mining in their municipality of residence has been studied by two different independent research bodies already twice during 2020-2021 and in 2022. According to the studies Mawson has overwhelmingly strong support from the local stakeholders. The Ylitornio municipality, which mainly hosts the Rajapalot project, is a sparsely populated area with a decreasing population. Rovaniemi municipality is the administrative and commercial centre of Lapland. The Rajapalot project could create many opportunities for both the current population and for people who will settle in the area in the future.

The first study was implemented by Lapland University in the NEXT Horizon 2020 project (Figure 20-14). The study aimed at exploring factors affecting local actors' and citizens' attitudes to and acceptance of mineral exploration (Social License to Explore, SLE), and how attitudes to exploration relate to acceptance at later stages of the mining cycle (Social License to Operate, SLO). It explored local actors' and citizens' perception of and attitudes to exploration in three local case studies which differed in contextual conditions that are important to the level of acceptance, i.e. community mining background, socio-economy and existence of indigenous people and traditional livelihoods.

The latest study implemented by Taloustutkimus Oy in April 2022 (Figure 20-15) arrived with very similar results with the previous study. The survey highlighted popular support for mining across Finland, with the results of Rajapalot's local municipality Ylitornio recording above average sentiment across the suite of mining issues surveyed.

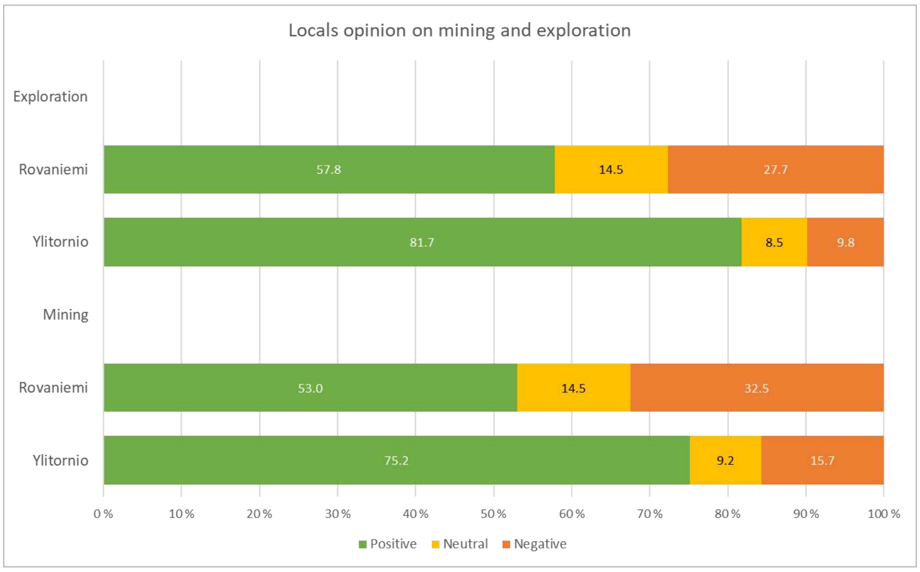


Figure 20-14: Local opinion on mining and exploration (Lapland University 2020)

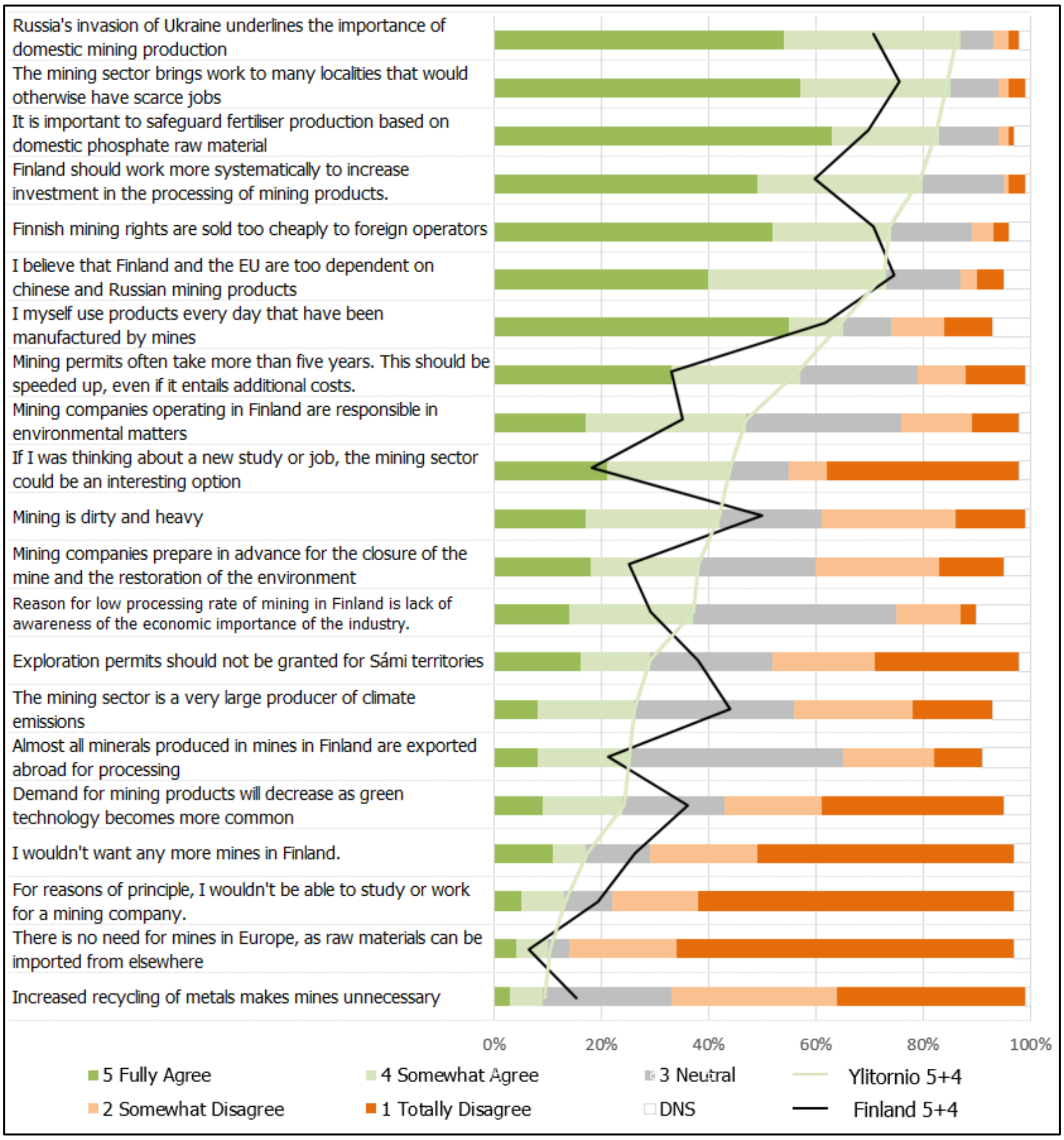


Figure 20-15: Results of survey by Taloustutkimus Oy in April 2022

20.6.3 Stakeholder Concerns

The following summarises stakeholders' concerns to date:

- Competition for land use, principally reindeer herding.
- Impacts on water quality
- Positivity about the potential for mining to maintain the vitality of the area; specifically potential for jobs, attracting new residents, as well as supporting domestic and EU self-sufficiency.

20.6.4 Contracts and Compensations

No agreements or compensations are in place currently.

20.1 Labour Conditions

In the Finnish labour market, the organisation level is high on both the employee and the employer side. Collective bargaining has a relatively important role in labour regulation. The basis of the regulation is in comprehensive and detailed labour legislation. Employment legislation covers for example contract issues, probation, severance pay, notice, hours of work, paid leave, maternity leave and maternity protection, sick leave, minimum age and protection of young workers, equality and trade union freedom.

20.2 Mine Closure

20.2.1 Closure Plans

Only high-level closure assessments are common practise at PEA study stage. The following study phases should include an iterative closure process. In such process information basis is systematically developed to reduce critical uncertainties and to enable gradual improvement of the closure plan. This cycle starts in early project development stages and continues over the whole life cycle of the mine. The closure plan needs to be confirmed by assessment of post closure impacts and assessment of closure and post closure risks. If impacts (or risks) are at an unacceptable level, review and partial re-planning of closure measures is required.

In Finland, mine closure requirements are largely based on European best available technology (BAT) definitions (EC 2018), but also relevance of post closure risk and impact assessments – which are also parts of the general BAT-definitions. Best closure planning practices are also generally iterative – repeating same risk and impact assessments in several study phases (ICMM 2019). ICMM also underlines the social perspectives of mine closure (stakeholder expectations).

20.2.2 Post-closure Conceptualisation (outlines)

Key objects remaining on the site after closure are the extractive waste facilities. Mawson aims to minimize extractive waste rock areas by utilizing waste rock in mine backfill and also in other infrastructure related projects. A project group from Lapland University of Applied Sciences is studying the possibilities of waste rock utilization.

Indicative review of drill core assays of Rajapalot-area (core ID PALXXX), not taking into account mining plan, shows that there is the potential for metal-rich, acid or neutral drainage from waste rock. This might be mitigated if two separate waste rock areas were established, one for less harmful waste rock and another one for waste rock of higher metal / sulphur content with impermeable basal structure. The cover structures are likely to be in line with the basal structures.

Precise closure structures suitable for tailings cannot be estimated at present, due to lack of geochemical data of chosen process option tailings. Based on the ore geochemical quality, and the suggested basal lining, it is likely that the tailings storage facility cover structure will have to be of a low net percolation type, as has been the basis for closure cost estimation in Section 21.2.6.

The aim of planning different structures for waste deposits should be minimal need of seepage water treatment post-closure. Seepage waters containing harmful substances in effective concentrations from waste storage facilities are collected and treated, if necessary, after closure.

Infrastructure will likely be demolished (unless subsequent use is discovered), but road networks typically remain to serve later land-uses, like forestry and reindeer herding. Water management and treatment systems from operational time serve until active closure implementation is completed.

Closure costs for the TSF and plant areas have been included in the project estimates. At this stage closure requirement considerations are only preliminary assumptions. The EIA and various permits may set additional requirements to the closure measures. Full assessment of closure costs will be completed when the needs are studied in future stages.

20.3 Environmental and Social Risks to the Project

20.3.1 Identification of Environmental and Social Project Risks

Nature and social risks related to mining projects in Finland include typically conflicts with the protection of species or habitats, watercourse impact issues, dust and air quality concerns, traffic and lasting physical impacts, and other noise and vibration issues. Also, land ownership impacts on property value and recreational land use, and land access for various purposes can become conflict issues. Project risks may materialize as a significant impact to management costs, difficult technical solutions, or permitting problems.

The following preliminary project risk assessment, as defined in Table 20-4, categories are considered especially important. The risk categories are mainly governed by Environmental Protection Act (527/2014) and Decree (713/2014) and Nature Protection Act (1096/1996). Risk prioritisation may change as the site-specific information increases.

Table 20-4: Preliminary summary of Environmental and Social Project risks

Risk category	Risk description	Consequence type	Recommended mitigation
Social, project acceptance, fishing and recreational values	Opposition to the project will be probably linked with fears of adverse environmental impacts and especially impacts on Natura 2000 and water courses.	- Project (mining /environmental) permitting delays/difficulties due to appeals.	- Open dialog with all stakeholders at all times - Present designed mitigation measures to the public
Terrestrial and aquatic nature	Stakeholders (like The Finnish Association for Nature Conservation) are likely to closely monitor the progress of the project and critically review all nature studies and nature and water impact assessments.	- Company reputation is locally compromised due to project plans within Natura area, and conflict potential increases - Project permitting delays/difficulties due to appeals	- Open dialog with all stakeholders at all times - Present designed mitigation measures to the public
Natura 2000 area	Mineral resources lie under Natura 2000 area and part of infrastructure needs to be built on Natura area. This may cause significant impact on protection grounds of Mustiaapa-Kaattasjärvi Natura 2000 area.	- Project permitting delays/difficulties due to appeals - Environmental permit must be accepted in Parliament	- Detailed design of infrastructure to minimize disruption leading to significant negative impacts. - Active compensation measures
Water quality	Acid, or even neutral rock drainage and metal leaching from high-sulphur tailings and possibly also from potential sulphide-bearing part of waste rock. Nutrient leach from all parts of the mine, especially nitrogen. Harmful substances in mine water are also expected.	- Cost risk related to extensive water treatment and water treatment sludge management, including some period after closure	- Water treatment design
Water bodies	Watercourses near project site are classified mainly to excellent or good ecological condition according to EU:s Water Framework Directive (WFD) classification system. Ecological status of the watercourses is not allowed to deteriorate.	- Project will not get environmental permit, if ecological status of classified watercourses below is predicted to deteriorate.	- Careful water impact assessment is needed, taking into consideration the LOM waste quantities. - Possible additional water treatment or discharge point in larger river may be required.
Dewatering	A groundwater drawdown in surrounding areas	- Project permitting delays/difficulties due to appeals - Potential effect on groundwater dependent sensitive vegetation	- Hydrogeological testing and dewatering /drawdown assessment is recommended to increase understanding on the risk. If necessary, mitigation measures can be studies if relevance of the impact or major risk becomes confirmed.
Data gaps concerning waste characterization	Basal and cover structures for waste storage facilities will have to be planned after complementary waste characterization data are available. From the available data, concerns are raised for As, Cu, Ni, and V concentrations of waste	- Initiation of waste characterization and source term assessments to serve the next study phases. Information is needed about the form of presence of critical elements and waste long	- Waste rock sampling selection based on current block model is needed to start the geochemical testing programme. At first stage, a set of static testing (ABA and NAG-

	rock, as well as sulphur content. The geochemical characteristics of tailings are unknown.	term behaviour. Source term assessments are needed after characterization. - Cost risks, planning of basal and cover structures is needed and costs cannot be defined with current level of information. - Another potential risk is higher water management costs.	tests, aqua regia soluble assays) is performed, and based on the results of these analyses, a series of samples for long term behaviour testing (e.g. humidity cell testing).
Closure	Relatively large waste deposits and harmful substances in seepage while receptors are sensitive watercourses. Difficulty to control the seepage direction after closure. Contamination transport is also possible from the underground mine via groundwater.	- Cost risks, may require rather effective closure measures even for the better quality wastes.	- Partition of waste rock by geochemical characteristics into separate areas. For tailings, possibility to process different types of tailings fractions to control the more high-risk material.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

Life of mine capital costs for Rajapalot are estimated as USD291M, comprising USD191M initial capital and USD100M sustaining capital (Table 21-1).

The LoM operating costs average USD55.9/t RoM (Table 21-2). The PEA cost estimates in this section have been completed by SRK, P&C, Sweco and Mawson.

Table 21-3 provides a summary of responsibilities of each contributor to the cost estimates.

Table 21-1: Summary capital cost estimate

Capital Expenditure	Units	Project	Sustaining	Total
Underground Mine	USD M	3.9	57.2	61.1
Capitalised Mine Operating Costs	USD M	10.5	-	10.5
Process Facilities	USD M	125.5	13.7	139.3
Backfill Plant	USD M	10.8	10.8	21.7
Residue / Tailings Management	USD M	11.1	7.4	18.5
Contingency	USD M	29.5	2.2	31.7
Closure	USD M		8.4	8.4
Total Capital Expenditure	**USD M**	**191.4**	**99.7**	**291.1**

Table 21-2: LoM Project Unit Operating Costs (including royalty)

Total Operating Cost	LoM Total (USD M)	Unit rate (USD/t mill feed)	Contribution (%)
Mining (including backfill)	353.0	34.9	62%
Processing (including TSF)	170.6	16.9	30%
G&A	40.5	4.0	7%
Royalties	1.9	0.2	0%
Total	**566.0**	**55.9**	

Table 21-3: Responsibility for Capital and Operating Cost Estimates

Description	Responsibility
Mining (including Materials Handling, Ventilation, Dewatering, Underground Infrastructure)	SRK
Backfill Plant	P&C
Process Facilities and Surface Infrastructure	Sweco
Tailings Storage Facilities	SRK
Product Transport and Treatment	Mawson
Indirect Construction Costs	Sweco
Rehabilitation/Closure Costs	Sweco/SRK

The battery limits for Sweco's scope of work (for both operating costs and capital expenditure) and hence SRK's work, are presented in Figure 17-2. Sweco's scope included the concentrating plant and surrounding infrastructure. The battery limit between mining operations and concentrating plant operations was placed so that mining operations take care of run of mine bins and feeding the grizzly. Crusher personnel is captured under concentrator plant personnel.

The other significant battery limit was defined between concentrator plant and mine backfill operations. The slurry limit was placed at the discharge to backfill. Return was placed in water treatment where tailings processing plant (which belongs to tailings operations) receives the untreated water from the concentrator plant, processes the water and delivers the processed water to the process water tank, tailings or paste to backfill.

The processing of the discharge water belongs to Sweco's scope i.e., both metal precipitation and settling and MBBR.

The estimate is reported in USD. Where capital costs were collected in EUR, they were converted to USD at a rate of USD1.0 to EUR, being the average rate during the estimate compilation. The majority of labour and processing operating costs were modelled in EUR and converted at a rate of USD0.91 to EUR to reflect longer term averages and provide modelling sensitivity capability.

Two years of project construction have been assumed, with all plant and associated infrastructure capital to be expended during this period. Backfill plant construction takes place partially during the first year of operations. Mine development works start in the second year of construction.

All capital expenditure to be expended from year 1 of operations onwards is considered as sustaining capital (despite some is related to construction). Mine project capital is low as the mine fleet during that period is covered under a lease to own scheme and hence covered under operating costs and spread out over a three-year period (part of which is hence covered under the capitalised mine operating costs).

21.2 Capital Expenditures

21.2.1 Mining

A mining cost model has been set up from first principles, using the life of mine physicals for the various underground operations combined with database, benchmark, and where available vendor quotes to derive capital expenditure and operating cost estimates. The model assumes the project will be owner mined.

Capital expenditures include the costs of the capital development (main headings which are used throughout the life of mine, with costs such as equipment running costs, drilling consumables, explosives, ground support and labour allocated), underground infrastructure and equipment purchase. Capital development occurs throughout the mine life, and sustaining capital requirements are therefor for ongoing capital development and equipment replacement. Capital expenditure as they occur during the two-year construction period are considered initial/project capital, with all capital to be spend during production labelled as sustaining capital.

Operating costs include the costs of excavating the operating development headings, and production costs (drilling, blasting, mucking, transport of stope material). As operational development does occur prior to production starting, operating costs are estimated during the construction period, which are capitalised for economic assessment purposes herein.

It has been assumed that the initial equipment fleet will be leased to own over a three-year period, and the cost thereof (upfront payment (20%), repayment and interest (9% per annum)) has been captured under operating cost herein. Any replacement equipment is bought as and when required and is treated as capital expenditure.

Table 21-4 summarises the LoM capital expenditure for the underground mining operations. This excludes the equipment purchase during years -1 and 1, which is treated as leasing to own under operating costs. Year -1 operating costs are included as capitalised operating costs.

Table 21-4: LoM Mine Capital Expenditure (incl capitalised operating costs)

Capital Expenditure	Units	Total	-1	1	2	3	4	5	6	7	8	9
Underground Mine	**USD M**	**61.1**	**3.9**	**5.1**	**7.0**	**8.4**	**6.3**	**3.6**	**4.3**	**9.4**	**8.1**	**5.0**
Contract Development	USD M	0.0	-	-	-	-	-	-	-	-	-	-
Mine Equipment	USD M	15.1	-	-	1.6	2.1	2.5	-	1.0	4.4	2.6	0.9
Mine Equipment Overhaul	USD M	25.4	0.4	2.6	2.9	3.2	2.7	2.7	2.6	3.0	3.3	2.1
Development Ground Support	USD M	4.4	0.3	0.5	0.5	0.8	0.2	0.2	0.2	0.5	0.7	0.5
Development Services	USD M	2.9	0.2	0.4	0.4	0.5	0.2	0.1	0.1	0.3	0.4	0.3
Development Drill & Blast	USD M	7.0	0.5	0.8	0.9	1.3	0.4	0.3	0.3	0.7	1.1	0.8
Mine Water Management	USD M	2.2	-	0.6	0.6	-	0.3	0.3	-	-	-	0.5
Technical Equipment & Software	USD M	1.5	1.1	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	-
Decline Portals	USD M	2.5	1.5	-	-	0.5	-	-	-	0.5	-	-
Capitalised Operating Costs	**USD M**	**10.5**	**10.5**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Total Mine Capital Expenditure	**USD M**	**71.6**	**14.4**	**5.1**	**7.0**	**8.4**	**6.3**	**3.6**	**4.3**	**9.4**	**8.1**	**5.0**

21.2.2 Processing Plant and Associated Infrastructure

A bottom-up capital cost estimate has been developed for the plant and infrastructure. Quantities have been derived from the process description, design criteria and detailed block flow diagrams, combined with plant and site layout drawing.

The expenditure estimate of major process equipment is calculated using the price estimates of major potential suppliers such as FL Smidth, Metso-Outotec, Sandvik and KPA Unicon, as well as local contractors for key installation activities such as buried pipeline and power line. The major equipment and service prices were updated in the autumn 2022. Minor equipment and services costs are estimated using Sweco;s in-house databases.

Costs for the structural steel, piping, electricity, construction work etc. are estimated using price information from the recent Sweco projects and Sweco's in-house databases.

The capital estimates include planning, manufacturing, packing, transportation, installation, and commissioning. Contingencies are covered under Section 21.2.5, intended to fully cover capital costs expected to complete the project.

Following assumptions were applied:

- LoM is expected to be 9 years;
- all equipment and material are new, and of local/European sourcing basis;
- implementation work will be continuous;

- an EPCM (engineering, procurement, construction management) project model will be applied; and
- costs incurred prior to a construction decision and land acquisition costs are excluded.

Process Equipment

Process equipment costs are summarized in Table 21-5, and include mechanical equipment and most bins and plate work. Crushing plant equipment and their costs are estimated according to Sandvik's budget quotation. Main process equipment is incorporated either according to FL Smidth's or Metso-Outotec's cost estimates. Some minor equipment is estimated using Sweco's in-house databases.

Table 21-5: Process Equipment Capital Expenditure

Process Equipment	USD M
Crushing	3.2
Additional Crushing Equipment	0.1
Fine Ore Bin	2.5
Grinding	6.0
Gravity Separation	0.5
Carbon in Leach	6.1
Intensive Leaching	0.5
ADR	3.5
Flotation	3.4
Thickening	0.3
Concentrate Filtration	2.1
Process Water Recycling	0.2
Reagent Preparation Systems	1.6
Process Pipes	4.8
Transportation	1.3
Installation	4.4
Total	**40.4**

Plant services

The plant services capital estimates (Table 21-6) includes all the goods around the process equipment excluding the buildings to complete the plant.

Table 21-6: Plant Supplementing Structures and Facilities Capital Expenditure

Plant Supplementing Structures and Facilities	USD M
Steel Structures	1.0
Process Automation and Instrumentation	5.2
Plant Electricity	5.4
Water treatment and Discharge	3.1
Building Electricity and HVAC	5.5
Emergency Power Station	0.3
Laboratory	0.6
Raw Water	0.5
Other	0.3
Total	**21.9**

Civil Works

The civil works capital expenditure estimate (Table 21-7) allows for earthworks and structures necessary to cater for the process plant and supporting facilities.

Table 21-7: Civil Works Capital Expenditure

Civil Works	USD M
Earth Works for Buildings and Structures	2.4
Buildings and Structures	15.2
Asphalted Areas	0.2
Total	**17.8**

Exclusions

- No ground survey has been carried out. Based on discussions about ground conditions (till/bedrock depth) with the Mawson team it has been assumed that there is no need for any piling at the Rajapalot project site.
- No large earth cuts nor transportations of excavated materials are allowed for.

Infrastructure

Infrastructure capital expenditure covers elements servicing the overall site and are summarized in Table 21-8.

Table 21-8: Infrastructure Capital Expenditure

Infrastructure	USD M
Domestic Water	0.5
Power Line	7.2
Area Electricity Network	1.2
Discharge Pipeline	4.9
Heating Plant	10.8
District Heating Network	0.2
Firefighting Water	0.7
Roads	0.5
Total	**25.9**

Note that the electricity supply is estimated assuming that Mawson will own and operate the power line connected directly to the national power grid in Valajaskoski. The option to amortize the capital cost of line construction into the power access charge will be evaluated in future studies

Indirect Construction Costs

Indirect construction costs are estimated for the two-year implementation period (Table 21-9). Execution is assumed via an EPCM model. All study engineering prior to this point is considered sunk cost and excluded from the estimate.

Temporary construction site arrangements cover all the activities needed in construction and start-up period.

Commissioning and start-up period is expected to last three months owing to a simple process which is well known and established. The costs for commissioning and start-up originate from use of Mawson's own personnel and EPCM consultants. Major vendor commissioning were included in the equipment supply estimates.

First fills are derived from the storage sizes for materials (chemicals, iron balls, fuel wood, etc). The same prices are used both in first fills and in the operating cost estimate.

Table 21-9: Indirect Construction Costs

Description	USD M
EPCM	15.1
Temporary Facilities	0.9
Commissioning and Spares	3.5
Total	**19.4**

Sustaining Capital Expenditure

Sustaining capital describes estimate of costs needed to maintain production at the planned level. The expected life span of the Rajapalot mining operations, and at the same time the concentrating plant is approximately 9 years. Most of the equipment are engineered to last 20-30 years with proper maintenance, however costs invariably arise during operations.

Annual sustaining capital is estimated to be 2% of Direct Costs when the concentrator plant is in steady operation mode (USD2.1 M per annum). Sustaining capital expenditure is expected to taper off in the final three years (75%, 50%, and 25%). Life of mine plant sustaining capital costs total USD13.7 M.

21.2.3 Backfill

Capital expenditure for the backfill plant and infrastructure are presented in Table 21-10. Benchmarking surface infrastructure is difficult due to the variability of processes required to manufacture backfill, however has provided a useful reference at this stage. Capital items such as filter presses, thickeners or positive displacement pumps can make a significant difference to the costs, as does the type of project (for example, expansion of an existing plant, sharing pre-existing facilities or a greenfield site).

Based on other projects undertaken by P&C, the capital cost estimate of USD21 M for the pastefill surface infrastructure appears to be acceptable for a plant of this production rate.

Table 21-10: Backfill Plant Capital Expenditure

Backfill Plant Description	Units	Surface Plant	Reticulation	Total
CRF	USD M	0.7	-	0.7
Pastefill	USD M	17.5	3.4	21.0
Total Capital Expenditure	**USD M**	**18.2**	**3.4**	**21.7**

21.2.4 Residue/Tailings Management

SRK prepared material take-offs (MTO) which allow for the development of a cost estimate for all capital and sustaining capital items associated with construction and operation of the chosen development option Ed. Capital costs include costs to construct the embankment, line the facility basin and tailings delivery system. Major capital cost items include:

- site clearance including clearing and grubbing of the embankment footprint area;
- restricted excavation (sub-excavation beneath the embankment, underdrains and for surface water diversion channels);
- embankment construction (mass fill using rock from a local borrow pit to construct starter embankments and waste rock and screened material from mining operations to construct the remaining embankments);
- tailings pipeline installation; and
- installation of monitoring equipment in the embankment.
- Of the total capital expenditure, 60% has been assumed as initial capital for the starter embankment. The relatively high capital results from substantial liner installation and starter embankment construction, which accounts for a large part of the overall costs, in the early stages of the project.
- The balance of the construction cost (40%) is assumed as sustaining capital and spread equally over the mine life. The phasing / height of the starter and subsequent stages will be developed further in future studies.

No allowance has been made for the following items in developing the capital and operating cost estimate:

- environmental, permitting, ecological and archaeological considerations;
- consequences from encountering different geotechnical conditions during future project phases than those upon which the existing design criteria and assumptions are based;
- force majeure events such as changes in government regulations, social disturbances, and industrial actions, whether legal or illegal, during the execution of the works; and
- social, sustainability, and community related items.

Earthworks unit rates have been benchmarked from similar projects in the region. Rates used were compiled based on projects of similar size and scope. No direct quotes from suppliers were obtained specifically for this project. Where comparable unit rates are not available from these projects, SRK estimated costs based on a first principle approach. Waste rock haulage rates (load-haul-place-compact) are based on typical rates for mining construction. All unit rates have been set in USD.

The quantities of each item have been estimated from the outputs of the Muk3D modelling work and estimating the distance between the TSF and process plant in a straight line direction.

A breakdown of the capital expenditure estimate is presented in Table 21-11, with initial project capital at 60% of total capital and sustaining capital the remaining 40%, spread out equally over the life of mine.

Table 21-11: Residue/Tailings Management Capital Expenditure

Description		USD M
Site Clearing & Preparatory Earthworks		3.1
Underdrains		0.2
Embankment Construction		8.2
Basal Lining System		5.3
Non-contact Water Management		0.3
Surface Water Management		0.5
Embankment Monitoring		0.1
Tailings Delivery Pipeline		0.8
Total Capital Expenditure		**18.5**
	Initial Upfront	11.1
	Sustaining	7.4

21.2.5 Contingency

In consultation with Mawson, Sweco assumed a contingency of 20% for the plant and infrastructure estimate. In Sweco's experience contingencies for scoping / PEA type studies vary from 20-30%. The use of the lower end of this spectrum reflects higher detail engineering, and levels of current, locally sourced pricing which has gone into the estimate.

The same level of contingency has been applied to the residue/tailings management and backfill plant initial/project capital.

21.2.6 Closure

After cessation of mining activities, the mining area must undergo rehabilitation. Rehabilitation aims to minimize and mitigate the environmental effects of mining. Rehabilitation management is an ongoing process to guarantee the healthy status of closed mining area. Rehabilitation /closure costs (Table 21-12) have been estimated in two parts: 1) by Sweco for landscaping and ongoing monitoring, and 2) by SRK for capping of the residue/tailings facility. TSF closure costs allow for a re-shaping via imported fill of the dam, such that a shedding surface is formed. This will be capped with clay and top soil to avoid water accumulation as suitable for permanent closure stability.

Whilst the active and passive care will occur over an extended period following mine closure, the costs have been accelerated to the final year.

Table 21-12: Rehabilitation/Closure Costs

Rehabilitation/Closure	USD M
Restoration (1 year period landscaping and rebuilding)	1.7
Active care (5 years period maintenance and monitoring)	0.5
Passive care (15 years period monitoring)	0.3
TSF capping	5.9
Total	**8.4**

21.2.7 LOM Capital

Two years of project construction have been assumed, with all plant and associated infrastructure capital to be expended during this period. Backfill plant construction takes place partially during the first year of operations. Mine development works start in the second year of construction.

All capital expenditure to be expended from year 1 of operations onwards is considered as sustaining capital (despite some it related to construction). Mine project capital assumes the initial mine fleet is acquired under a lease to own scheme and hence covered under operating costs and spread out over a three-year period (part of which is hence covered under the capitalised mine operating costs).

Table 21-13: Rajapalot LoM Capital Expenditure Summary

Capital Expenditure	Project USD M	Sustaining USD M	Total USD M
Underground Mine	3.9	57.2	61.1
Capitalised Mine Operating Costs	10.5	-	10.5
Process Facilities	125.6	13.7	139.3
Backfill Plant	10.8	10.8	21.7
Residue / Tailings Management	11.1	7.4	18.5
Contingency	29.5	2.2	31.7
Closure		8.4	8.4
Total Capital Expenditure	**191.4**	**99.7**	**291.1**

21.3 Operating Costs

Operating costs have been presented for the same disciplines as the capital expenditure, but the addition of general and administrative. The LoM operating costs average USD55.9 per RoM tonne as summarized in Table 21-14.

Table 21-14: LoM Project Unit Operating Costs (including royalty)

Total Operating Cost	LOM Total USD M	Unit rate (USD/t mill feed)	Contribution (%)
Mining (including backfill)	353.0	34.9	62%
Processing (including TSF)	170.6	16.9	30%
G&A	40.5	4.0	7%
Royalties	1.9	0.2	0%
Total	**566.0**	**55.9**	

21.3.1 Mining

As mentioned under Section 21.2.1, mine operating costs are based on first principles, using benchmark data as cost drivers.

Table 21-4 summarises the life of mine operating cost for the underground mining operations. This excludes the operating costs during year -1 which have been capitalised. For equipment leasing to own a three year term is envisaged with a 20% upfront payment and repayment plus 9% interest thereafter. Only the initial mine fleet is envisaged to be financed in this manner.

Table 21-15: LoM Mine Operating Costs (excluding capitalised operating costs)

Operating Costs	Units	Total	1	2	3	4	5	6	7	8	9
Equipment & Materials Handling	USD M	114.8	11.6	13.2	14.1	12.7	12.6	12.4	13.6	14.8	9.9
Development Ground Support	USD M	7.3	1.0	1.0	0.9	0.8	0.8	0.7	0.8	1.0	0.4
Development Services	USD M	4.6	0.6	0.6	0.6	0.5	0.5	0.5	0.5	0.6	0.2
Development Drill & Blast	USD M	13.4	1.8	1.8	1.7	1.4	1.4	1.3	1.5	1.8	0.7
Production Drill & Blast	USD M	10.4	0.7	1.1	1.1	1.4	1.3	1.2	1.4	1.3	1.0
Grade Control & Assaying	USD M	12.4	1.1	1.4	1.4	1.5	1.5	1.5	1.5	1.5	1.0
Mine Personnel	USD M	103.7	10.9	11.4	12.4	11.4	11.4	11.1	12.5	12.9	9.7
Mine Water Management	USD M	3.8	0.2	0.4	0.4	0.4	0.4	0.4	0.5	0.5	0.5
Ventilation Heating	USD M	12.5	1.2	1.5	1.4	1.5	1.5	1.5	1.4	1.5	0.9
Equipment Leasing	USD M	32.6	9.1	9.6	8.9	5.0	-	-	-	-	-
Total Mine Operating Costs	**USD M**	**315.6**	**38.1**	**42.0**	**42.9**	**36.5**	**31.4**	**30.6**	**33.8**	**36.0**	**24.2**
Mine unit costs per RoM tonne	USD/t ROM	31.2	39.7	35.0	35.8	30.4	26.1	25.5	28.2	30.0	31.8

21.3.2 Backfill

A backfilling cost of USD7.96 per tonne (dry) of pastefill is estimated for Rajapalot (Table 21-16), and USD7.20/t of CRF. When benchmarking the pastefill unit cost, the estimate is at the lower end of the expected cost range. P&C notes that the 2022 inflation rate applied to the benchmark dataset includes a significant increase between 2021 and 2022, potentially skewing the reference costs upward. Regardless, the Rajapalot mine plots within a cluster of reference projects which indicates the unit operating cost is broadly acceptable.

Table 21-16: Backfill Plant Average LoM Unit Operating Costs

Backfill Type	Units	Cost	Includes
CRF	USD/t CRF	7.20	Cement, power
Pastefill	USD/t paste	7.96	Cement, barricades, reticulation, power
Average backfill cost	USD/t ROM	3.70	

21.3.3 Processing

The operating cost estimate as described in this section includes all the cost items relevant to processing ore and keeping the plant and associated infrastructure running. An owner-operator operation has been assumed.

The operating philosophy is to batch process gold-cobalt feed separate to gold-only feed which bypasses flotation and therefore has lower variable costs, Based on the production schedule, approximately 40% of the RoM feed is gold-only.

The operating costs have been estimated by Sweco and are presented herein as either variable or fixed based on a nominal throughput of 1.2 Mtpa and are summarised in Table 21-17. Costs were estimated and modelled in EUR, and converted to USD at an exchange rate of USD0.91 to EUR.

Table 21-17: Processing Operating Cost Summary

Total Processing	Fixed (USD M/annum)	Variable (USD/Au-only t)	Incremental variable (USD/Au-Co t)
Labour	3.6	-	-
Reagents	-	7.3	1.9
Power	-	3.3	0.2
Consumables	-	1.2	0.1
Other	0.4	0.1	0.01
Total	**3.9**	**11.9**	**2.2**
Life of mine average		**USD16.7/t RoM**	

The following estimate basis and assumptions apply:

- Labour costs and plant general and administrative (G&A) type costs are fixed across operating modes.
- A manning estimate of 45 blue collar and 7 white collar process workforce has been developed, with salaries based on published Finnish salary averages for relevant level positions. The on-costs of employment were calculated according to Finnish legislation and the current (year 2022) collective agreements. Lower utilisation of labour during gold-only processing offsets vacation burden.
- Reagents for flotation, other concentrating processes and water treatment were incorporated in the estimate with current supplier prices from (verified in August-September 2022). Cyanide, SMBS and sulphuric acid were inflated from previous estimates.
- Electricity prices from Statistics Finland, current August 2022. Electricity transfer costs according to Fingrid's official price tables.

21.3.4 Tailings Residue Management

TSF operating costs are expenses associated with the operation, maintenance and administration of the tailings transportation to the TSF. Operating costs have been based (on USD/t) on similar projects in the region, and include:

- slurry pumping costs from the plant to the TSF;
- maintenance of the slurry pipeline; and
- monitoring of the embankment.

Annual operating costs have been estimated at USD0.14M, totalling USD1.27M over the LoM.

21.3.5 General and Administrative

No detailed assessment of expected G&A costs has been prepared, which is not deemed necessary at PEA level of study. An allowance of USD4/t RoM has been incorporated, which amounts to USD4.8M per annum once at steady state production throughput.

21.3.6 Summary Operating Costs

A summary of the LoM project operating costs is presented in Table 21-18, including the statutory landholder royalty which is further detailed in Section 22.5. A high-level summary is presented in Table 21-19, also showing the overall unit costs per main category.

Table 21-18: LoM Project Annual Operating Costs (including royalty)

Operating Costs	Units	Total	1	2	3	4	5	6	7	8	9
Mining	USDM	315.6	38.1	42.0	42.9	36.5	31.4	30.6	33.8	36.0	24.2
Mineral Processing	USDM	169.3	16.7	19.1	19.5	19.4	19.5	19.7	20.9	20.1	14.5
Backfill	USDM	37.4	3.6	4.5	4.4	4.4	4.4	4.4	4.4	4.4	2.8
TSF Management	USDM	1.3	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1
G&A	USDM	40.5	3.8	4.8	4.8	4.8	4.8	4.8	4.8	4.8	3.0
Royalty	USDM	1.9	0.2	0.3	0.2	0.2	0.2	0.2	0.2	0.2	0.2
Total Operating Cost	**USDM**	**566.0**	**62.6**	**70.7**	**72.1**	**65.5**	**60.5**	**60.0**	**64.2**	**65.6**	**44.8**

Table 21-19: LoM Project Unit Operating Costs (including royalty)

Total Operating Cost	LoM Total (USD M)	Unit rate (USD/t RoM)	Contribution (%)
Mining (including backfill)	353.0	34.9	62%
Processing (including TSF)	170.6	16.9	30%
G&A	40.5	4.0	7%
Royalties	1.9	0.2	0%
Total	**566.0**	**55.9**	

22 ECONOMIC ANALYSIS

22.1 Introduction

This PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

A technical economic model has been developed on an annual basis to assess the economic potential of the Rajapalot project.

22.2 Project Schedule

The current project schedule assumes a two-year construction period, followed by 9 years of production. Mine development starts in the second year of the construction period. Full production of 1.2 Mtpa of plant feed is accomplished in year 2 of operations. Whilst plant feed to the main circuit is kept constant at 1.2 Mtpa for the 7 years at full capacity, not all of this material is suitable for the cobalt circuit. Batch processing of gold-only, and gold-cobalt feed is undertaken. Annual feed to the extended cobalt circuit varies between 0.4 and 1.2 Mtpa and is dependent on feed availability coming from the mine plan.

22.3 Commodity Prices and Revenue Assumptions

As per Section 19, the following commodity prices have been applied in the PEA:

- Gold: USD1,700/oz; and
- Cobalt: USD60,000/t.

The following smelter and freight rates have been allowed for:

- Gold in doré:
    - 75% Au in doré;
    - payability of 99.85%;
    - shipment charge of USD5.00/kg doré;
    - fixed shipment costs of USD3,500/shipment, with 50 shipments per year; and
    - refining and sampling/assaying cost of USD27.5/kg doré.
- Cobalt concentrate:
    - cobalt grade deduction of 0.075% (cobalt grade is variable, as the sulphur grade in concentrate is the fixed parameter);
    - treatment charge of USD110/dmt of concentrate;
    - refining charge of USD0.22/kg cobalt; and
    - freight of USD55/wmt of concentrate (5% moisture).

22.4 Key Inputs and Assumptions

The following general assumptions have been made:

- All costs and revenues are presented in USD and are in real 2022 money terms.
- A 2-year pre-production period for construction, development and commissioning activities.
- Cash flows have been discounted to the start of construction using an end-year approach. Any cash flows, including cost of further studies, prior to the start of construction have been excluded from the analysis; however, refer to Section 22.5 regarding the tax loss opening balance as currently allowed for.
- A discount rate of 5% has been applied for NPV calculations. Exchange rates assumed are as per Section 21.
- Production rates, capital and operating costs are as set out in this report.
- The PEA mine production schedule (Section 16.11.4) is the main driver for the economic analysis, producing two products:
  - doré with gold recovery of 95%, and the doré consisting of 75% gold (for shipment purposes); and
  - cobalt concentrate: with Co recovery of 87.6%, S recovery of 88.0%, and a fixed S grade in Co con of 35%.
- A closure cost of USD8.4M has been included at the end of life.
- The cash flow model is presented post-tax and pre-finance.

22.5 Taxes and Royalties

A mineral royalty tax of 0.15% of net revenue is deemed payable on both gold and cobalt sales.

A corporate income tax of 20% has been allowed for. A simplistic depreciation calculation has been applied, assuming an annual depreciation of 25% of opening book value according to Finnish accounting practices. An amount of EUR32.0M has been incorporated as an opening balance for carried forward losses. This loss (structured as a loan) attracts interest and will increase as more pre-development activities are funded from equity, however conservatively no allowance has been made for increases in the model.

22.6 Economic Evaluation Results

Based on the inputs and assumptions in this report, key results of the cash flow model for the Rajapalot project are estimated to be:

- post-tax IRR of 26.5%;
- post-tax NPV of USD211M at a 5% discount rate;
- post-tax cash flows of USD 341 M;

- gold accounting for 92.2% of net revenue, with cobalt contributing 7.8%;
- C1 cash cost of USD670/oz Au, and All-in Sustaining Cost (AISC) of USD824/oz Au (placing the project in the first quartile on the cost curve as published by the World Gold Council, 30 June 2022); and
- undiscounted payback post-tax is 2.9 years since production start.

A summary of annual cash flow is set out in Table 22-1, with pre- and post-tax economic metrics presented in Table 22-2.

Note: Initial Capital Cost, Sustaining Capital, Cash Operating Costs, Total Cash Cost (C1) and AISC which are not measures recognized under International Financial Reporting Standards (IFRS) and do not have a standardized meaning prescribed by IFRS.

- Total Cash Costs and Total Cash Costs per Ounce: Total Cash Costs are reflective of the cost of production. Total Cash Costs reported in the PEA include mining costs, processing & water treatment costs, general and administrative costs of the mine, off-site costs, refining costs, transportation costs and royalties. Total Cash Costs per Ounce is calculated as Total Cash Costs divided by payable gold ounces.
- AISC and AISC per Ounce: AISC is reflective of all the expenditures that are required to produce an ounce of gold from operations. AISC reported in the PEA includes total cash costs, sustaining capital and closure costs, but excludes corporate general and administrative costs and salvage. AISC per Ounce is calculated as AISC divided by payable gold ounces.

Table 22-1: PEA Annual Cashflow Summary

Project Timeline	Units	Total/Avg	-2	-1	1	2	3	4	5	6	7	8	9	10
Physicals														
Plant Feed (Au circuit)	(kt)	10,121	-	-	960	1,200	1,200	1,200	1,200	1,200	1,200	1,200	761	-
Au Grade	(g/t)	2.26	-	-	2.51	2.73	2.30	2.32	2.47	1.97	1.95	1.86	2.34	-
Au Recovered														
Plant Feed (Co circuit)	(kt)	6,054	-	-	609	380	589	523	591	688	1,183	831	660	-
Co Grade	(%)	0.053%	-	-	0.037%	0.038%	0.040%	0.047%	0.049%	0.048%	0.057%	0.062%	0.081%	-
Co con produced	(kt)	314	-	-	31	18	26	26	28	31	63	45	46	-
Co con Grade	(%)	0.89%	-	-	0.63%	0.72%	0.79%	0.83%	0.91%	0.93%	0.94%	1.00%	1.02%	-
Co Recovered	(t)	2,806	-	-	197	127	208	217	252	290	596	449	470	-
Revenue														
Gross Revenue	USD M	1,341	-	-	135	177	155	156	168	138	154	140	118	-
TC/RC/Sampling	USD M	(36)	-	-	(4)	(2)	(3)	(3)	(3)	(4)	(7)	(5)	(5)	-
Freight	USD M	(20)	-	-	(2)	(1)	(2)	(2)	(2)	(2)	(4)	(3)	(3)	-
Net Revenue (Au + Co)	**USD M**	**1,286**	**-**	**-**	**130**	**173**	**150**	**151**	**163**	**133**	**143**	**133**	**110**	**-**
Operating Costs														
Underground Mine	USD M	(316)	-	-	(38)	(42)	(43)	(37)	(31)	(31)	(34)	(36)	(24)	-
Mineral Processing	USD M	(169)	-	-	(17)	(19)	(20)	(19)	(20)	(20)	(21)	(20)	(14)	-
Backfill	USD M	(37)	-	-	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(4)	(3)	-
Residue / Tailings Management	USD M	(1)	-	-	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	-
G&A	USD M	(40)	-	-	(4)	(5)	(5)	(5)	(5)	(5)	(5)	(5)	(3)	-
Royalty	USD M	(2)	-	-	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	-
Total Operating Cost	**USD M**	**(566)**	**-**	**-**	**(63)**	**(71)**	**(72)**	**(66)**	**(61)**	**(60)**	**(64)**	**(66)**	**(45)**	**-**
Capital Expenditure														
Underground Mine	USD M	(61)	-	(4)	(5)	(7)	(8)	(6)	(4)	(4)	(9)	(8)	(5)	-
Capitalised Operating Costs	USD M	(11)	-	(11)	-	-	-	-	-	-	-	-	-	-
Residue / Tailings Management	USD M	(18)	-	(11)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(1)	-
Process Facilities	USD M	(139)	(50)	(75)	-	(2)	(2)	(2)	(2)	(2)	(2)	(1)	(1)	-
Backfill Plant	USD M	(22)	-	(11)	(11)	-	-	-	-	-	-	-	-	-
Contingency	USD M	(32)	(10)	(19)	(2)	-	-	-	-	-	-	-	-	-
Closure Cost	USD M	(8)	-	-	-	-	-	-	-	-	-	-	-	(8)
Total Capital Expenditure	**USD M**	**(291)**	**(60)**	**(131)**	**(19)**	**(10)**	**(11)**	**(9)**	**(7)**	**(7)**	**(12)**	**(10)**	**(6)**	**(8)**
Economic Assessment														
EBITDA	USD M	720	-	-	67	103	78	86	102	73	79	67	65	-
Corporate Income Tax	USD M	(87)	-	-	-	(1)	(10)	(13)	(17)	(12)	(13)	(11)	(11)	-
Cashflow from Operations	USD M	632	-	-	67	101	67	73	85	61	66	56	55	-
Net Free Cash	**USD M**	**341**	**(60)**	**(131)**	**48**	**91**	**56**	**64**	**79**	**54**	**54**	**46**	**48**	**(8)**
AISC	(USD/oz)	824	-	-	1,046	772	911	797	643	788	760	860	612	-

Table 22-2: PEA Pre- vs Post-Tax Economic Metrics Summary

	Units	Pre-Tax	Post-Tax
NPV (5%)	USD M	271	211
IRR	(%)	30.2%	26.5%
Undiscounted Payback	(year)	2.8	2.9

22.7 Sensitivity

For illustrative purposes, key project assumptions have been flexed in the cash flow model to evaluate post-tax NPV_5 sensitivity. Result of a single variable sensitivity analysis is illustrated in Figure 22-1. This analysis calculates the post-tax NPV_5 by changing a single input value by ±30% from those assumed in the PEA for gold price, operating costs or capital expenditure. The Project is most sensitive to metal prices followed by capital expenditure. Select multivariate analysis is shown in Table 22-3.

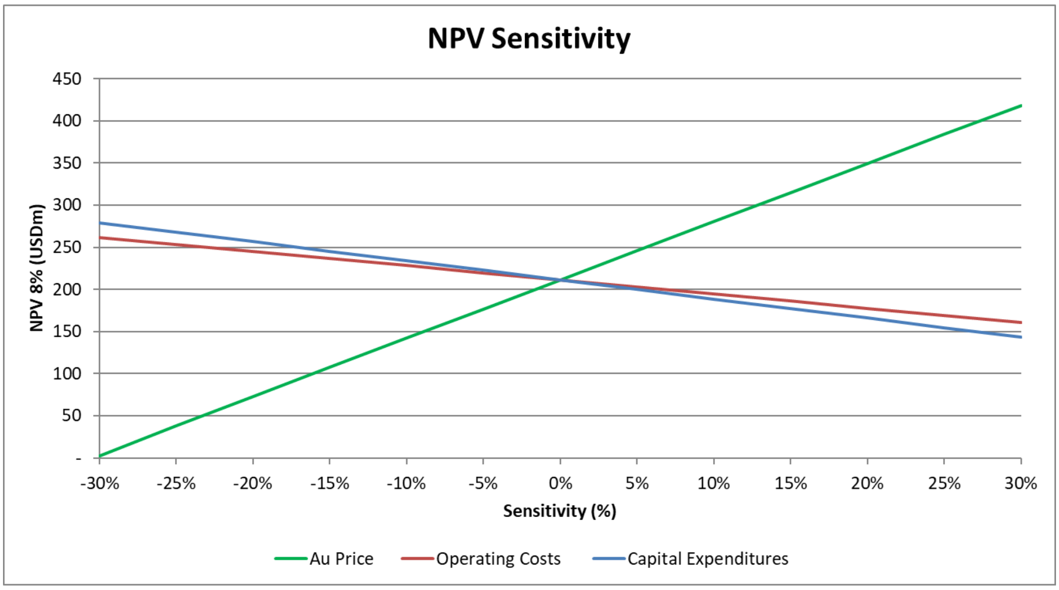


Figure 22-1: NPV (5%) Sensitivity to Key Project Inputs

Table 22-3: Key sensitivities to gold price

Gold Price (USD/oz)	Post-tax NPV_5 (USD M)					Post-tax IRR	Y1-5 FCF (USD M)
	Base Case	CAPEX - 10%	CAPEX +10%	OPEX -10%	OPEX +10%		
1,400	89	112	66	106	72	15%	234
1,550	150	173	128	167	133	21%	286
1,700	211	234	189	228	195	27%	338
1,850	272	295	250	289	255	32%	390
2,000	333	356	310	350	316	37%	442

22.8 Conclusion

Based on the PEA economic analysis, the project has positive operating margins, a 26.5% post-tax IRR and 2.9 year payback period. The Project operating life of 9 years results in an estimated net cash flow of USD341M and NPV_5 of USD211M. LoM AISC is calculated at USD824/oz Au, which places the project in the first quartile on the cost curve (as published by the World Gold Council as of 30 June 2022). These financial metrics indicate that the Rajapalot Project has good economic potential and warrants further studies.

23 ADJACENT PROPERTIES

The Peräpohja belt contains very few known mineral occurrences. No other gold-cobalt drill intersections are known from near the Rajapalot property held by other parties.

Mineralized occurrences near to Rajapalot include the following:

- Vinsa: (Mawson OY) is an orogenic copper-gold occurrence with no Mineral Resource estimate available. It comprises a 0.5-2 m wide, >250 m long quartz vein and enveloping alteration halo in a dolerite. Native gold (?) associated with chalcopyrite, pyrite and pyrrhotite. Mawson holds an exploration permit application over this area.
- Petäjävaara: (Mawson OY) is an orogenic copper-gold occurrence with no Mineral Resource estimate available. It comprises a set of quartz veins in a sheared, SW-trending, contact zone between dolerite and quartzite, and is chiefly hosted by the dolerite. Gold is possibly only within quartz veins. Mawson holds an exploration permit application over this area.
- Kivimaa: (Latitude 66 Cobalt Oy, owner) is an orogenic copper-gold deposit. In 1969, 18,600 t of ore was mined by Outokumpu Oy, and only 37 kg gold and 223 t Cu recovered. Kivimaa comprises a 1-6 m wide, >350 m long quartz vein and enveloping alteration halo in an E-W trending dip-slip fault in a dolerite. Native gold as inclusions in arsenopyrite and, possibly, as free gold. All gold appears to be in the quartz vein.
- Sivakkajoki: (Latitude 66 Cobalt Oy, owner) close to the Kivimaa deposit, is an orogenic gold occurrence with no resource estimate available. It comprises a set of carbonate-quartz veins and enveloping alteration halo in an E-W trending fault in a dolerite. Apparently, gold only in the quartz veins.

(see GTK data https://gtkdata.gtk.fi/mdae/index.html)

24 OTHER RELEVANT DATA AND INFORMATION

The Qualified Persons are not aware of any other information relevant to the understanding of this report. There are no further explanations required.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

Based on the PEA economic analysis, the project has positive operating margins with a post-tax IRR of 26.5% and 2.9 year payback period using a gold price of USD1,700/oz and cobalt price of USD60,000/t. The Project operating life of 9 years results in an estimated net cash flow of USD341M and NPV_5 of USD211M. These financial metrics indicate that the Rajapalot Project has good economic potential and warrants continued development.

25.2 Mineral Resources and Exploration Potential

The Inferred Mineral Resource estimate has been prepared as at 26 August 2021 under the CIM Definition Standards 2014 and considers the underground-only base case scenario which represents the most RPEEE. The MRE is considered suitable for the PEA and as a basis for further exploration and economic evaluation of the Rajapalot Project. Future drilling will focus on converting the inferred resource classification to indicated to use as a basis in future detailed mining studies.

Significant potential exists to expand the MRE, locally as well as in the regional Project area. The defined resource bodies are all open down dip, as highlighted by the deepest drilling into the largest body, Palokas, intercepting 30.8 m at 5.1 g/t AuEq from 553 m (3.9 g/t Au and 1,403 ppm Co).

Regionally, the 18,000 hectares contiguous 100% owned land package remains mostly undrilled, despite other significant gold occurrences defined on the property, including Rompas (highlight intersection 6 m at 617 g/t Au from 7 m) and North Rompas (highlight intersection 0.4 m at 395 g/t Au from 41 m), located 8 and 10 km, respectively, from the Rajapalot MRE. The wider property is relatively underexplored but has a significant number of anomalous gold occurrences that warrant follow up and present good potential for further discovery.

25.3 Mining

SRK concludes that the primary mining method of retreat LHOS with paste backfill selected for Palokas, Raja, The Hut and Rumajärvi deposits is appropriate and has the advantage of maximizing mining extraction and reducing tailings storage requirements on surface. The mining method selected for the Joki deposit is overhand C&F and considered appropriate due to its shallower dip angle with CRF for backfill support.

The overall production target of 1.2 Mtpa was based on an assessment of the RoM Inventory could be sustained over a 9-year period through mining three of the deposits at any time. The production strategy considers continuous mining of the larger two deposits (Palokas and Raja) over the LoM and mining the smaller three deposits (Joki, The Hut and Rumajärvi) sequentially with the order determined by higher gold grades, summarised as follows:

- Initial mining commences at Palokas (675 ktpa) + Raja (375 ktpa) + Joki (150 ktpa);
- When Joki is exhausted then production commences at The Hut (150 ktpa); and
- When The Hut is exhausted then production commences at Rumajärvi (150 ktpa).

In addition to the overall mining schedule, RoM material (on a stope-by-stope basis) is assessed against an NSR cut-off for cobalt extraction (USD2/t), to be separately stockpiled, and campaign processed. All feed will be processed for gold recovery but only a proportion, on a feed campaign basis, for cobalt recovery.

25.3.1 Geotechnical

The primary requirement to limit geotechnical uncertainty is for Mawson to conduct collection of the geotechnical parameters for calculation of the rock classification systems, Q and Modified Rock Mass Rating (MRMR). Currently this is not fully completed and is considered as standard for more detailed technical level studies. This will allow refinements and probable optimisation of stope dimensioning to larger spans based on actual geotechnical and structural logging data.

Currently there is an established practice of PLT testing but these cannot be converted with confidence to UCS values. A program of core sample selection is required for laboratory compression testing to establish the correlation between UCS and PLT. This will allow PLT testing to be conducted with conversion to calculate the UCS of the rock types in future logging. Additionally, core samples selected will be used to measure tensile strength using the Brazilian method (BTS) and triaxial compression strength (TCS). The three testing types establish the failure criterion for the rock mass which is used for underground stope, pillar and ground support design.

Major geological structures have not yet been modelled at the deposit scale. If present these will influence the underground mine design, access placement and also the extraction sequence.

25.3.2 Hydrogeology and Water Management

Based on the assumptions made in this study regarding the surface and groundwater regime at Rajapalot, it is likely that there will be no need for any active intervention pre-mining to advance dewater the rock mass around the underground operations. Whilst the low K and S properties of the formations that host the deposits still need to be confirmed through in situ testing, if they exist as expected, then conventional sump and pump arrangements should be adequate to dewater the underground mines. Additionally, the impact of run-off from the surrounding catchment, particularly during the spring freshet can be limited through the installation of berms and interception ditches around surface facilities. Although there appears to be little evidence of large-scale faulting at the site, the operators should allow some contingency for probe holing and advance grouting of declines and headings in case these structures are found to exist.

25.4 Mineral Processing and Metallurgical Testwork

For the purpose of the current PEA, the process design is based on maximizing recovery of gold and cobalt by gravity recoverable gold recovery and WoO leach for gold recovery as bullion, followed by bulk sulphide flotation from the leach tailings to produce a cobalt concentrate.

Based on the results from the testing, gold recovery of 95% is anticipated from the combination of gravity recovery for coarse gold followed by gold leaching at p80 of 75 µm and leach residence time of 30 hours.

Cobalt recovery and product grade are dependent on feed source and target concentrate product. Flotation testing of both mineral types demonstrated that high recoveries of cobalt can be achieved with appropriate pulp chemistry and collector additions.

As demonstrated by the test results, however, the mineralogical type represented by the Pal sample, with low cobalt head grade and only minor cobaltite content, will only produce a low Co grade concentrate, such that flotation processing after gold recovery may not always meet current economic criteria. Cobalt recovery and cobalt-in-concentrate grade is expected to average 87.9% and 0.89% respectively for the subset of mine ROM that is sent to the flotation stage.

25.5 Tailings Residue Management

A conceptual TSF design has been presented and costed for the storage of tailings at Rajapalot mine based on a trade-off assessment which considered the cost and permit risk perspective. The conceptual design included a lined TSF and ring dyke structure to contain tailings expanded using the downstream raise method. The following details need to be investigated further in future phases of study:

- Ground conditions beneath the footprint of the TSF including the depth of peat and loose unconsolidated till. Unfavourable conditions could lead to deep foundations or the need to move the location of the TSF.
- Geotechnical testing of representative tailings samples need to be completed to confirm the assumptions made in the volumetric model. If more or less favourable properties are reported, the size of the facility may decrease or increase respectively.
- The site is located within a cold climate with below-zero temperatures during the winter. During the winter period the tailings pond can freeze causing operational issues. Ice lenses that form within the TSF can also reduce the density of the tailings and affect the overall storage capacity of the facility.
- Geochemical properties of the tailings are currently unknown. Pre-treatment or additional layers within the basal lining system may be required. It is noted that the high sulphur recovery to cobalt concentrate should have a favourable impact in lowering acidification of the tailings.
- Settlement properties of the tailings are not known, unfavourable properties could result in the return water having a high sediment load requiring pre-treatment before reuse within the processing plant, or the return water being unsuitable for use within the plant and raw water required.
- Sources of materials for construction of the starter embankment have been assumed to be available from a local borrow pit. Sources of materials for construction of the underdrains and capping system have been assumed to be available on site, with minimal processing. If suitable materials are not available the costs associated may make the TSF option cost prohibitive.

25.6 Environmental Studies, Permitting, and Social or Community Impact

Mawson has completed a significant number of environmental studies and has been conducting baselining assessments across the relevant parts of its tenement package as well as surrounding areas, in support of its exploration activities and the evaluation of the impact of a future mining project. The studies have thus far confirmed that there are no such plant or animal species which would be unique to the project area or the larger vegetation zone area.

Finland has rigorous regulatory processes with strict environmental standards and Mawson is committed at this early project stage to work with the regional and national authorities and broader stakeholder groups to develop the project in a responsible way. Mawson has had an active ESG program operating for many years, which is being constantly adjusted as its projects grow and develop.

Mawson considers stakeholder engagement and collaboration to be a critical part of the potential development of the Rajapalot project, and social aspects will be a key part of the EIA preparation process. In February 2021, Mawson announced that two key planning processes, the EIA and land use planning have been initiated for the Rajapalot gold-cobalt project in Finland. These processes represent a formalisation of the engagement processes that had already been ongoing.

Closure costs for the TSF and plant areas have been included in the project estimates. At this stage closure requirement considerations are only preliminary assumptions. The EIA and various permits may set additional requirements to the closure measures. Full assessment of closure costs will be completed when the needs are studied in future stages.

Key objects remaining on the site after closure are the extractive waste facilities. Mawson aims to minimize extractive waste rock areas by utilizing waste rock in mine backfill and also in other infrastructure related projects. A project group from Lapland University of Applied Sciences is studying the possibilities of waste rock utilization.

26 RECOMMENDATIONS

26.1 Introduction

The main recommendations arising from the PEA study relate to collecting of more empirical data, particularly geotechnical and hydrogeological, and completion of more detailed engineering studies to increase cost estimate accuracy. Further gold and cobalt metallurgical test work is necessary and will be used to refine recoveries and operating assumptions, and alongside cobalt marketing studies, optimize the cobalt NSR. Upgrading the resource classification to indicated through drilling would be required to consider future Mineral Reserve assessments. Environmental baseline studies should continue in support of the in-progress EIA, including assessing opportunities to reduce reliance on fossil fuels and the carbon footprint of the Project.

26.2 Exploration and Mineral Resources

Approximately 60,000 m of drilling will be required to increase the drilling density of the resource wireframe to a level necessary to classify the resources as indicated (or better). Of the 60,000 m, 45,000 m (75%) is needed for Palokas and Raja, which represent 88% of the tonnes and 90% of the gold ounces reporting to the ROM.

The mineralized bodies remain open at depth. From a regional perspective, the recent recognition of the stratigraphic host position of the Rajapalot mineralization and the equivalents in the Rompas area has allowed Mawson to rethink the exploration strategy across its permits. New gravity data, combined with regional BOT, electromagnetics and IP-resistivity is further focussing the targeting. It is recommended that the surface sampling database be re-visited to combine it more effectively with the new geophysics and understanding of the stratigraphic location Rajapalot resources in order to identify new areas with economic potential.

26.3 Mining

The following should be considered for advancing the mining aspects of the Rajapalot Project in future studies:

1. Improve the geotechnical information available on the rock types for determination of localised extraction ratios and pillar requirements.

2. Backfill testwork to confirm the type(s), quantity and cost.

3. Materials handling trade-off studies considering the potential for Battery Electric Vehicle (BEV) and trolley-assist technologies to reduce reliance on fossil fuels and also to reduce greenhouse gas and carbon emissions.

4. Ground support requirements for boxcut/portal, underground access and ventilation raise requirements.

5. Once more information is available on the geotechnical and hydrogeological aspects of the project then further detailed mine planning work can take place to identify opportunities The mine design and schedule should be completed in line with the increased confidence of future mineral resources classification and in sufficient detail to provide accurate mine production rate estimates. Future more detailed planning is undertaken with consultation with equipment suppliers to understand the requirements (and costs) of reducing diesel-

powered mobile equipment and practically implementing developing BEV and trolley assist technologies at the individual mines.

26.3.1 Geotechnical

A proportion of newly drilled holes require full geotechnical logging before core cutting. As well as this, older cut holes can be revisited to collect joint condition and joint set (family) count, compared to core photos and combined with the current RQD and FF data. This will allow the calculation of geotechnical numeric classification from existing drilling.

At a minimum, a more thorough review of the potential stress regime is required which hinges on a robust structural geology interpretation. The stress orientation can be better estimated. Stress magnitude and orientation can be extracted from deep televiewer holes.

Future studies will require analysis of several aspects related to the underground access standoff distance and extraction sequencing. This will be interlinked with the revision of the extraction method and backfilling decisions. The ore access standoff and mine access rationale needs to be optimised based on stability risk and the best economic case. The ground support estimates will be further refined to suit the refined mine access design and extraction plans.

26.3.2 Hydrogeology

Site-specific in situ hydrogeological field measurements are recommended to be carried out in bedrock and possibly also in overburden. Hydraulic conductivity measurements according to depth in bedrock would be valuable initial data for groundwater flow modelling. Geotechnical parameters like RQD can be used together with hydrogeological testing data. A comprehensive groundwater table monitoring network should be established in overburden although groundwater levels are measured in drillholes. The conceptual hydrogeological and numerical groundwater flow models should be updated when more data is available and model parameters like K-values should be checked.

26.4 Mineral Processing and Metallurgical Testwork

As noted in Section 13.13, confirmatory testing will be undertaken to investigate the flotation response after gold recovery and cyanide destruction. Testing will target improving cobalt production economics, and therefore will be focussed on the Raja-South Palokas feed type for this flowsheet concept.

Investigation of response to alternative flotation conditions – post detoxification - will be undertaken with a view to:

- Optimising grade/recovery response for the different mineral components and reducing reagent requirements.
- Developing product specifications, and potentially samples from laboratory scale processing, as the basis for continued discussions with possible purchasers.

Preliminary testing of an alternate process flowsheet concept has been undertaken in parallel with the testwork supporting this PEA study. It is based on WoO flotation with high recovery of gold to the concentrate, along with a proportion of the cobalt containing mineralization. Gold would be leached from the flotation concentrate and the residual treated to produce marketable cobalt concentrate. A significantly smaller leaching plant would be required for this circuit, and power and reagent consumptions would be significantly reduced.

Testing is required to determine the gold recovery by combination of gravity recovery and flotation, and the effect on cobalt production from this process alternative, to be able to compare overall project economics.

26.5 Tailings Residue Management

The following opportunities need to be investigated further in future phases of study:

- Permitting of slurry/thickened tailings facilities are well documented in Finland and the regulator is well versed in these facilities, there are several case studies close by of similar technology being used. During further phases of studies, these operations should be reviewed and the opportunity taken to understand any operational difficulties or lessons learnt in operating a TSF in this area.

- At present there are no known dry stacks operational within Finland. However this project has an achievable throughput to explore the use of dry stacking technology. A full cost benefit trade off should be completed in future studies.

- The opportunity to change the depositional strategy before the end of the LOM to provide a shedding surface, and reduce the volume of imported material required should be explored. This would lower closure cost allowances.

- A borrow material options trade off study should be completed, large volumes of waste rock are required for starter embankment construction, which at present are not available from UG mining during start up. A trade off study should look at bringing forward waste mining to generate suitable fill materials, establishing a local borrow source, or adapting the construction sequence for the TSF to include additional cut beneath the foundations of the TSF to source competent bedrock.

26.6 Water Management and Treatment

Further work is recommended on the water management and treatments aspects of the Rajapalot Project considering:

- Geochemical investigation, analysis and modelling to estimate dewatering water quality and treatment requirements prior to discharge.

- Investigation into water quality of non-contact and potential contact waters as these will dictate the necessity for water treatment.

- A dewatering strategy to promote the recovery of non-contact water and thereby minimise contact waters.

- Advance the hydrogeological analysis for Project and complete a suitably detailed water balance covering all aspects related to mining, processing, tailings and backfill.

26.7 Environmental Studies, Permitting, and Social or Community Impact

Following issues are recommended to be taken especially well into consideration during the next study phases:

- Representative sampling of waste rock based on the current mining plan. All major waste rock types should be sampled for post-closure source (such as underground mine water, waste rock and TSF seepage, etc) term estimates.
- Representative sampling of future pilot testing of the ore processing. The sampling should include not only the tailings, but also representative sample of process water, which is needed in tailings storage area source term estimation (for wet deposit).
- Waste characterizations (Static geochemical testing of all waste fractions. Kinetic testing on selected samples based on static test results for long-term behaviour, form of presence for harmful elements).
- Source term assessments for extractive waste facilities at current planned dimensions (and waste quantities). Source term assessments are based on the long-term behaviour testing of the waste, and also take into consideration the field liquid/solid ratio.
- The Rajapalot area lies close to the former Litorina sea highest shoreline, and there are also some black schist areas north-west of Rajapalot area. These increase the probability for acid sulphate soils occurrence in overburden, especially in mineral soil covered with peat. The overburden should be geochemically assessed prior to removing the soil and plan the deposit area according to the analysis results.
- Hydrogeological testing and dewatering /drawdown assessment are recommended to increase understanding on the risk of drawdown impacts on habitats/flora/fauna, and also the rate of reflooding of the mine after closure to assess post-closure effects to ground and surface waters.
- Site-specific in situ hydrogeological field measurements are recommended to be carried out in bedrock and possibly also in overburden. Hydraulic conductivity measurements according to depth in bedrock would be valuable initial data for groundwater flow modelling. Geotechnical parameters like RQD can be used together with hydrogeological testing data.
- A comprehensive groundwater table monitoring network should be established in overburden although groundwater levels are measured in drillholes.
- The conceptual hydrogeological and numerical groundwater flow models should be updated when more data is available and model parameters like K-values should be checked.
- More detailed investigation of by-product markets. Especially usage of waste rock in other markets (Mawson has already received interest in usage of waste rock).

Preparation to climate change consequences and mitigation of negative contribution to climate change in Mawson's operations.

26.8 Cost of Recommendations

Table 26-1 outlines an indicative budget to facilitate a pre-feasibility level assessment.

Table 26-1: Indicative PFS budget

	CAD M
Drilling	11
Metallurgical testwork	0.3
Hydrogeology and geotechnical investigation	1
Technical studies	2
Total	**14.3**

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Impacts of exploration on vegetation (quadrats)	2014 Niina Lappalainen & Britta Hamari, Ahma Ympäristö Oy; Kasvillisuusvaikutusten seuranta Rajapalojen alueella 2014
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Polypores	2018 Juha Kinnunen; Raportti Ylitornion Mustiaapa-Kaattasjärven Natura-alueen kääpäinventoinnista 2018
Capercaillies	2018 Juha Kinnunen; Raportti metson (Tetrao urogallus) esiintymisestä Ylitornion Palokkaiden tutkimusalueella 2018
Moor frogs	2016 Juha Kinnunen; Viitasammakkoinventointi Mustiaapa-Kaattasjärven Natura-alueella 2016
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Otters	2015 Juha Kinnunen; Raportti Mustiaapa-Kaattasjärven Natura-alueen Palokkaan saukkoinventoinneista 2015
	2021 Juha Kinnunen; Raportti Mustiaapa-Kaattasjärven Palokkaan sekä Romppaiden Natura-alueiden saukkoinventoinneista 2021
Bats	2016 Juha Kinnunen; Raportti Mustiaapa-Kaattasjärven ja Romppaiden Natura-alueiden lepakkoinventoinnista 2016
Birds	2016 Karoliina Hämäläinen; Mustiaapa-Kaattasjärven ja Romppaiden pesimälinnustoselvitys
	2017 Karoliina Hämäläinen; Pöllökartoitus
	2019 Olli-Pekka Karlin; Mustiaapa-Kaattasjärven linnustoselvitys 2019

	2020 Olli-Pekka Karlin; Ylitornion Palokkaan ja Rovaniemen Kaattasjärven linnustolaskennat 2020
Golden Eagle	2019 Olli-Pekka Karlin; Maakotka lähetinlinnun paikannusten analysointi ja vertailu kairauspaikkoihin ja kulkureitteihin Ylitornion Palokkaan alueella 2019
	2021 Olli-Pekka Karlin; Raportti Ylitornion Mustivaaran maakotkan liikkeistä vuoden 2021 aikana
Insects	2017 Jukka Salmela & Lauri Paasivirta; Ylitornion Rajapalojen aapasuon hyönteisselvitys 2017
	2018 Jukka Salmela & Lauri Paasivirta; Ylitornion Rajapalojen aapasoiden hyönteisselvitys 2018
Calypso orchids	2020 Pia Kangas; Neidonkenkäinventointi Palokkaan alueella vuonna
	2021 Pia Kangas; Neidonkenkäinventoinnit Palokkaan alueella vuosina 2020-2021
Lady's-slippers	2020 Pia Kangas; Palokkaan alueen luontotyyppi- ja kasvillisuusinventointi 2018-2020
Flying squirrel	2015 Juha Kinnunen
Freshwater pearl mussel	2018 Juha Kinnunen
	2021 Jonna Autto; Raakkukartoitus Palokkaanjoki
Fish	2022 VAHANEN ENVIRONMENT OY; Exploratory fishing: Louejärvi, Kaattasjoki, Kuusijoki, Palokkaanjoki
Diatoms and benthic fauna	2021 VAHANEN ENVIRONMENT OY; Diatoms and benthic fauna: Louejärvi, Kaattasjoki, Katiskojoki, Ternujoki, Louejoki
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Groundwater level measuring (hydrometer)	2020-2022 Kimmo Neuvonen, Esa Pulliainen, Jonna Autto
Water quality	2011 Rompasoja-Rumavuoma, Geochemical study, GTK: report
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	2022 Groundwater monitoring, Eurofins

Glossary – Technical Studies

Feasibility Study	Means a comprehensive technical and economic study of the selected development option for a mineral project that includes appropriately detailed assessments of applicable Modifying Factors together with any other relevant operational factors and detailed financial analysis that are necessary to demonstrate, at the time of reporting, that extraction is reasonably justified (economically mineable). The results of the study may reasonably serve as the basis for a final decision by a proponent or financial institution to proceed with, or finance, the development of the project. The confidence level of the study will be higher than that of a Pre-Feasibility Study.
Pre-Feasibility Study	The CIM Definition Standards requires the completion of a Pre-Feasibility Study as the minimum prerequisite for the conversion of Mineral Resources to Mineral Reserves. A Pre-Feasibility Study is a comprehensive study of a range of options for the technical and economic viability of a mineral project that has advanced to a stage where a preferred mining method, in the case of underground mining, or the pit configuration, in the case of an open pit, is established and an effective method of mineral processing is determined. It includes a financial analysis based on reasonable assumptions on the Modifying Factors and the evaluation of any other relevant factors which are sufficient for a Qualified Person, acting reasonably, to determine if all or part of the Mineral Resource may be converted to a Mineral Reserve at the time of reporting. A Pre-Feasibility Study is at a lower confidence level than a Feasibility Study.
Preliminary Economic Assessment	A preliminary economic assessment (or PEA) means a study, other than a pre-feasibility study or feasibility study, that includes an economic analysis of the potential viability of mineral resources. Typically the accuracy of a PEA is in the range of -30% to +50% with a contingency of 25% to 50%. The confidence level of a PEA is low, below that of either an feasibility or preliminary feasibility study. Unlike the other two types of study, a PEA may contain result of an economic analysis that includes, or is based upon, inferred mineral resources. However, where that occurs, disclosure based on the study must contain prescribed cautionary language. In addition, it is important to note that a PEA should not act as a proxy for a pre-feasibility study or feasibility study. A PEA cannot demonstrate economic viably. A PEA is not meant to be a way to include a inferred resource in a pre-feasibility study or feasibility study or to alter such studies to include more positive assumptions. Just because a report is labeled a PEA does not mean that regulators will accept it as a PEA if it is done to the levels of a pre-feasibility study or feasibility study.

Glossary – Mineral Resources and Mineral Reserves

Mineral Reserves	Mineral Reserves are sub-divided in order of increasing confidence into Probable Mineral Reserves and Proven Mineral Reserves. A Probable Mineral Reserve has a lower level of confidence than a Proven Mineral Reserve. A Mineral Reserve is the economically mineable part of a Measured and/or Indicated Mineral Resource. It includes diluting materials and allowances for losses, which may occur when the material is mined or extracted and is defined by studies at pre-feasibility or feasibility level as appropriate that include application of Modifying Factors. Such studies demonstrate that, at the time of reporting, extraction could reasonably be justified. The reference point at which Mineral Reserves are defined, usually the point where the ore is delivered to the processing plant, must be stated. It is important that, in all situations where the reference point is different, such as for a saleable product, a clarifying statement is included to ensure that the reader is fully informed as to what is being reported.

Proven Mineral Reserves	A Proven Mineral Reserve is the economically mineable part of a Measured Mineral Resource. A Proven Mineral Reserve implies a high degree of confidence in the Modifying Factors. Application of the Proven Mineral Reserve category implies that the Qualified Person has the highest degree of confidence in the estimate with the consequent expectation in the minds of the readers of the report. The term should be restricted to that part of the deposit where production planning is taking place and for which any variation in the estimate would not significantly affect the potential economic viability of the deposit. Proven Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.
Probable Mineral Reserves	A Probable Mineral Reserve is the economically mineable part of an indicated, and in some circumstances, a Measured Mineral Resource. The confidence in the Modifying Factors applying to a Probable Mineral Reserve is lower than that applying to a Proven Mineral Reserve. The Qualified Person(s) may elect, to convert Measured Mineral Resources to Probable Mineral Reserves if the confidence in the Modifying Factors is lower than that applied to a Proven Mineral Reserve. Probable Mineral Reserve estimates must be demonstrated to be economic, at the time of reporting, by at least a Pre-Feasibility Study.
Mineral Resource	A concentration or occurrence of solid material of economic interest in or on the earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling. Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories.
Measured Mineral Resource	That part of a Mineral Resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of Modifying Factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A Measured Mineral Resource has a higher level of confidence than that applying to either an Indicated Mineral Resource or an Inferred Mineral Resource. It may be converted to a Proven Mineral Reserve or to a Probable Mineral Reserve.
Indicated Mineral Resource	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.
Inferred Mineral Resource	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.

Glossary – Development Status

Adjacent Property

Means a property (a) in which the issuer does not have an interest (b) that has a boundary reasonably proximate to the property being reported on, and (c) that has geological characteristics similar to those of the property being reported on.

Advanced Property

Means a property that has (a) mineral reserves, or (b) mineral resources the potential economic viability of which is supported by a preliminary economic assessment, a pre-feasibility study or a feasibility study.

Early-Stage Exploration Property

Means a property for which the technical report being filed has (a) no current mineral resources or mineral reserves defined, and (b) no drilling or trenching proposed.

Advanced Exploration Property

Properties where considerable exploration has been undertaken and specific targets have been identified that warrant further detailed evaluation, usually by drill testing, trenching or some other form of detailed geological sampling. A Mineral Resource estimate may or may not have been made, but sufficient work will have been undertaken on at least one prospect to provide both a good understanding of the type of mineralization present and encouragement that further work will elevate one or more of the prospects to the resource category.

Pre-Development Property

Properties where Mineral Resources have been identified and their extent estimated (possibly incompletely) but where a decision to proceed with development has not been made. Properties at the early assessment stage, properties for which a decision has been made not to proceed with development, properties on care and maintenance and properties held on retention titles are included in this category if Mineral Resources have been identified, even if no further Valuation, Technical Assessment, delineation or advanced exploration is being undertaken.

Development Property — Properties for which a decision has been made to proceed with construction and/or production, but which are not yet commissioned or are not yet operating at design levels,

Operating Mines — Mineral properties, particularly mines and processing plants that have been commissioned and are in production.

Care and Maintenance/Closed Properties

Mineral properties, particularly mines and processing plants which have been either decommissioned or placed on care and maintenance pending an improvement in economic and/or technical operating environments.

Abbreviations

2014 CIM Definition Standards	2014 Canadian Institute of Mining and Metallurgy definition standards for reporting Mineral Resources and Mineral Reserves
3D	Three dimensional
ADC	Arctic Drilling Company OY
AFRY	AFRY Finland Oy

AISC	All-In sustaining costs
All-UG	Model created to test the viability of an all underground operation (grade blocks were used where they were deeper than 20 metres below the base of the till.
ALS	ALS Laboratory, Piteå, Sweden; Sodankylä, Finland; Loughrea, Ireland; Vancouver, Canada.
AMC	AMC Consulting Group
AMV	At-Mine-Value
AREVA	AREVA Finland
AuEq	Gold Equivalent
BAT	Best Available Techniques
BATCircle	Business Finland funded battery metal research. Finland-based Circular Ecosystem of Battery Metals, BATCircle, aims at improving the competitiveness of the Finnish battery value chain.
BEV	Battery Electric Vehicle
BOT	Base of till drilling – small percussion drill rig with flow-through drill bit. Generally drilled to refusal through till and sample taken from top of bedrock.
BTS	Brazilian method
C&F	Cut and Fill mining method
C1	Initial Capital Cost, Sustaining Capital, Cash Operating Costs, Total Cash Cost
CAGR	compound annual growth rate
CDA	Canadian Dam Association
Chemical abbreviations	Gold, Au; cobalt, Co; iron oxide, FeO; sulphur, S; arsenic, As; nickel, Ni; bismuth, Bi; copper, Cu; tungsten, W, uranium, U; tellurium, Te.
CHM	Conceptual Hydraulic Model
CIL	cyanide in leach
CIRAF	Cobalt Industry Responsible Assessment Framework
CLGB	Central Lapland Greenstone Belt
CLGC	Central Lapland granitoid complex
CoV	Cut-off Value
CRF	Cemented Rock Fill
CRM	Certified reference material
CRM	Critical Raw Materials
CRS	CRS Laboratory, Kempele, Finland
CRU	Commodity Resource Unit
CSA	Canadian Securities Administrators
CSR	Corporate Social Responsibility
CSS	closed side setting
Cyanide Code	International law and standards for sodium cyanide preparation, handling, using and storage
DCS	Distributed control system
Deswik.SO	Deswik software Stope Optimiser module
Doré	Doré is the term used for the rough or unrefined gold produced in the mine's metallurgical plant. Its composition varies greatly depending on the mineralogy of the orebody and the type of processing plant.
DRC	Democratic Republic of Congo
EGL	Effective Grinding Length
EIA	Environmental Impact Assessment
ELY-Centre	Centre for Economic Development, Transport and the Environment for Lapland
EPCM	engineering, procurement, construction management
ERMA	European Raw Materials Alliance
ESG	Environmental, Social and Governance
ESIA	Environmental and Social Impact Assessment
EU	European Union
EV	Electric vehicle
FA-AA	Fire assay with AA finish
FA-GRA	Fire assay with Gravimetric analysis
FEM	Finite Element Modelling
FF	Fracture Frequency
Fingrid Oy	Finland's national transmission system operator
FMI	Finnish Meteorological Institute
FOS	Factor of Safety
FW	Footwall

G&A	General and Administrative
Ga, Ma	Abbreviations for billion year and million years respectively.
GCL	Geosynthetic Clay Liner
GIIP	good international industry practice
GISTM	Global Industry Standard on Tailings Management
GPS	Global Positioning System
GTK	Geological Survey of Finland (Geologian Tutkimuskeskus)
HDPE	High Density Polyethylene
HV	High voltage
HVAC	heating, ventilation, and air conditioning
HW	Hangingwall
ICMM	International Council on Mining and Metals
ICP-AES	Inductively coupled plasma atomic emission spectroscopy
IFRS	International Financial Reporting Standards
IROPI	Imperative reasons of overriding public interest
IRR	Internal Rate of Return
Joki	Used interchangeably with with Joki East and East Joki
KAIELY	Kainuu Centre for Economic Development, Transport, and the Environment
KKJ3	Finnish metric grid system also known as KKJ. Interchangeable and equivalent known as EPSG 3901 and EPSG 2392.
KPI	Key performance indicator
Lapin Liitto	Regional Council of Lapland
LeachWELL	ALS method
Leapfrog	May refer to Leapfrog Edge or Leapfrog Geo, part of the geological and modelling suite owed by Seequent Limited. Version 2021.1.2 used throughout this report.
LFP	lithium iron phosphate
LHD	Load Haul Dump (loader)
LHOS	Longhole Open Stoping mining method
LIDAR	Light Detection and Ranging (survey)
Liikennevirasto	Finnish Transport Agency
LIMS	Laboratory Information Management System
LME	London Metal Exchange
LOM	Life of Mine
LOMP	Life of Mine Plan
LV	Low voltage
MA	Mining Associates Pty Ltd
MAC	Mining Association Canada
MAP	Mean Annual Precipitation
Mawson	Mawson Gold Limited, Mawson Oy
Mawson Oy	The Finnish entity wholly owned by the parent company Mawson Gold Limited – all entities may be referred to in this Technical Report as “Mawson”.
MBBR	Moving bed biofilm reactor wastewater treatment process
MIBC	methyl isobutyl carbinol (frother)
Mineral species	See Appendix D.
MMW	Minimum Mining Width
MRE	Mineral Resource Estimate
MRMR	Modified Rock Mass Rating
MRT	Mineral Royalty Tax
MTO	material take-off
NEXT	New Exploration Technologies. Horizon 2020 funded EU project (Grant Agreement No. 776804 — H2020-SC5-2017)
NI 43-101	National Instrument 43-101 Report
NPV	Net Present Value
NSR	Net Smelter Return
NTK	Nivalan Timanttikairaus
OC	Open cut pit
OC-UG	Smaller pits defined using an opencut versus underground OPEX breakeven point – the method used to create the Resource on which this PEA is based.
OREAS	Ore Research & Exploration Pty Ltd Standards
P&C	Paterson & Cooke Nordic AB
PB	Peräpohja belt
PDC	process design criteria

PEA	Preliminary Economic Assessment
PFS	Pre-Feasibility Study
PLC	Programmable Logic Controller
PLT	Point Load Test
POX	Pressure Oxidation
PSD	Particle Size Distribution
QA	Quality Assurance
QAQC	Quality Assurance Quality Control
QC	Quality Control
QEMSCAN	Quantitative evaluation of minerals by scanning electron microscopy
QP	Qualified Person
QXRD	Quantitative x-ray diffraction
RC	Refining Charge
Rev-F-1	Revenue factor = 1 pit optimization model
RGMP	Responsible Gold Mining Principles
RMR	Rock Mass Rating
ROM	Run of Mine
Rompas-Rajapalot	Property area covering Rajapalot and Rompas projects. The resource descriptions in this report cover the Rajapalot project area within the broader Rompas-Rajapalot property. Exploration upside covers the broader property area.
ROPO	Recognised Overseas Professional Organisation
RPEEE	reasonable prospects for eventual economic extraction
RQD	Rock Quality Designation
R-SP	Raja-South Palokas
SD	Standard Deviation
SEM	Scanning electron microscopy
SEX	sodium ethyl xanthate (collector)
SFP	specific fan power
SGS	SGS Laboratory
SLE	Social License to Explore
SLO	Social License to Operate
SMU	Selective Mining Unit (used in the pit optimization process)
SO	Stope Optimiser
SoR	Slope of Regression
SRK	SRK Consulting (UK) Limited
SRK Group	SRK Consulting (Global) Limited
TC	Treatment Cost
TCS	Triaxial compression strength
TEP	Technical Economic Parameters
TSF	Tailings Storage Facility
TUKES	Finnish Safety and Chemicals Agency (Turvallisuus- ja kemikaalivirasto) is the Finnish Government agency responsible for exploration permitting.
UCS	Uniaxial Compressive Strength
UG	Underground
UNFCCC	United Nations Framework Convention on Climate Change
Vahanen	Vahanen Environment Oy
VG	Visible Gold
WAI	Wardell Armstrong International
WFD	Water Framework Directive
Whittle	Refers to the pit optimization software used.
WoO	Whole-of-Ore
WRD	Waste rock dump
XRF	X-ray fluorescence

Units

°C	Degrees centigrade
µm	Micrometre
CAD	Canadian Dollars

cm	centimetre
dmt	dry metric tonne
EUR	Euro
g	gram
g/t	grams per tonne
kg	kilogram
km	kilometre
koz	thousand ounces (troy)
ktpa	thousand tonnes per annum
ktpm	thousand tonnes per month
kV	kilovolt
kVA	Apparent Power in kilo-watts
kW	Actual Power in kilo-watts
kWh	kilo-watt hour
lb	pound (weight)
L/s	litres per second
m	metre
m/d	metres per day
m/s	metres per second
m^2	square metre (area)
m^3	cubic metre (volume)
m^3/d	cubic metres per day
m^3/s	cubic metres per second
Ma	Mega-annum, million years (geology)
mamsl	metres above mean sea level
masl	metres above sea level
mH	metres height
mL	metres length
mm	millimetre
MPa	Mega Pascals
mRL	metres reduced level
Mt	million tonnes
Mtpa	million tonnes per annum
MVA	Mega Volt-Ampere
mW	metres width
oz	troy ounce
P_{80}	80% passing size of the circuit product
ppm	Parts per million
s	second
t	tonne
t/m^3	tonnes per cubic metre (density)
TKM	tonne-kilometre
tph	tonnes per hour
USD	United States Dollar
wmt	wet metric tonne
Ns^2/m^8	Ventilation Resistance
K	hydraulic conductivity (m/s)
hr	hour
bar	unit of pressure
pH	potential of hydrogen, measure of acidity/basicity of an acqueous solution

APPENDIX A QP CERTIFICATES

SRK Consulting (UK) Limited
5th Floor Churchill House
17 Churchill Way
Cardiff CF10 2HH
Wales, United Kingdom
E-mail: enquiries@srk.co.uk
URL: www.srk.com
Tel: + 44 (0) 2920 348 150

CERTIFICATE OF QUALIFIED PERSON

I, Christopher Bray, B.Eng, MAusIMM (CP) do hereby certify that:

1. I am a Principal Consultant (Mining Engineer) of SRK Consulting (UK) Limited, 5th Floor, Churchill House, 17 Churchill Way, Cardiff, United Kingdom.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Rajapalot Gold-Cobalt Project, Finland" with an Effective Date of October 15, 2022 (the "Technical Report").
3. I graduated with a degree in Mining Engineering from Curtin University of Technology, Western Australia in 1997. I have worked as a Mining Engineer for a total of 23 years since my graduation from university and have been employed by SRK Consulting since October 2006 during which time I have been involved in a variety of engineering studies, valuations and technical reports and taken responsibility for mining and Mineral Reserve reporting aspects. I am a member of the Australian Institution of Materials Mining and Metallurgy (Membership Number 990571) and I am a Chartered Professional.
4. I have read the definition of "qualified person" set out in *National Instrument 43-101 Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited Mawson Gold Limited's Rajapalot property in Finland.
6. I am responsible for Sections 1 through 6, 15, 16, 18.1, 18.5 to 18.8, 19 and 21 through to 27, and overall preparation of the Technical Report.
7. I am independent of Mawson Gold Limited as described in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28 Day of November, 2022.

Sealed

(Signed) "*Christopher Bray*"

Christopher Bray, B.Eng, MAusIMM (CP)
Principal Consultant (Mining Engineer)

AFRY

ÅF PÖYRY

CERTIFICATE OF QUALIFIED PERSON

I, Ove Klavér, MSc, Eur.Geol., FAMMP do hereby certify that:

1. I am a Principal Geologist of AFRY Finland Oy. Jaakonkatu 3, FI-01621 Vantaa, Finland.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Rajapalot Gold-Cobalt Project, Finland" with an Effective Date of October 15, 2022 (the "Technical Report").
3. I graduated with a master's degree in Geology from Åbo Akademi University, Finland in 2011. I have worked as a Mine/Resource Geologist for a total of 12 years and have been employed by AFRY since June 2018. I am a member of the Fennoscandian Association for Metals and Minerals Professionals; I am also a member of the European Federation of Geologists (Membership Number 1775) and I am a Chartered Professional.
4. I have read the definition of "qualified person" set out in *National Instrument 43-101 Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have visited Mawson Gold Limited's Rajapalot property in Finland on 29 June 2021 for 4 days.
6. I am responsible for Sections 7 through 12 and 14 of the Technical Report.
7. I am independent of Mawson Gold Limited as described in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28 Day of November, 2022.

Sealed

(Signed) "*Ove Klaver*"

Ove Klavér, MSc, Eur.Geol., FAMMP
Principal Geologist

CERTIFICATE OF QUALIFIED PERSON

I, Eemeli Rantala, P.Geo, M.Sc. do hereby certify that:

1. I am a Senior Geologist of AFRY, Lilleakerveien 8, 0283 Oslo, Norway.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Rajapalot Gold-Cobalt Project, Finland" with an Effective Date of October 15, 2022 (the "Technical Report").
3. I graduated from the University of Turku with a B.Sc. Degree (2009) and M.Sc (2011) in Geology. I have worked as a Geologist in mining industry for a total of 13 years since my graduation from university and have been employed by AFRY since October 2019. I am a Professional Geoscientist (#169691) registered with APEGBC (Association of Professional Engineers and Geoscientists of British Columbia).
4. I have read the definition of "qualified person" set out in *National Instrument 43-101 Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have visited Mawson Gold Limited's Rajapalot property in Finland on 29 June 2021 for 4 days.
6. I am responsible for Sections 4.8, 4.9 and 20 of the Technical Report.
7. I am independent of Mawson Gold Limited as described in section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28 Day of November, 2022.

Sealed

(Signed) "*Eemeli Rantala*"

Eemeli Rantala, P.Geo, M.Sc.
Senior Geologist

RESOURCES ENGINEERING & MANAGEMENT Pty Ltd
ABN 94 082 112 221

CERTIFICATE OF QUALIFIED PERSON

I, Craig Brown, B.E., GradDipGeosci, FAusIMM, do hereby certify that:

1. I am Principal Consultant with Resources Engineering & Management Pty Ltd, 37 Spencer St, Corinda, QLD, Australia.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Rajapalot Gold-Cobalt Project, Finland" with an Effective Date of October 15, 2022 (the "Technical Report").
3. I am a graduate from the University of NSW, Sydney, Australia with a B.E. Degree in 1976 and have been professionally active since my graduation. I am a Fellow of the Australasian Institute of Mining and Metallurgy (Membership Number 102282).
4. I have read the definition of "qualified person" set out in *National Instrument 43-101 Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited Mawson Gold Limited's Rajapalot property in Finland.
6. I am responsible for Sections 13 and 17 of the Technical Report.
7. I am independent of Mawson Gold Limited as described in section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report, I was a QP for the "Mineral Resource Estimate NI 43-101 Technical Report, Rajapalot Property, Finland for Mawson Gold Limited" as issued by Mawson Gold Limited on the August 26, 2021.
9. I have read NI 43-101 and Form 43-101F1 and the sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
10. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the sections of the Technical Report I am responsible for contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 28 Day of November, 2022.

Sealed

(Signed) "*Craig Brown*"

Craig Brown, B.E., GradDipGeosci, FAusIMM
Principal Consultant

GOSSELIN MINING
consulting engineer

Mathieu Gosselin, P. Eng.
Gosselin Mining AB
Industrivägen 23
Solna, Sweden 171 48
+46 73 388 4583
info@gosselinmining.com

CERTIFICATE OF QUALIFIED PERSON

I, Mathieu Gosselin, P. Eng., do hereby certify that:

1. I am CEO, President and Industry Expert-Mining of Gosselin Mining with an office situated at Industrivägen 23, Solna, Sweden 171 48.
2. This certificate applies to the technical report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Rajapalot Gold-Cobalt Project, Finland" with an Effective Date of October 15, 2022 (the "Technical Report").
3. I graduated with a degree in Bachelor of Engineering, Mining from McGill University, Montréal in 2004.
4. I am a member of Ordre des ingénieurs du Québec (No. 135077).
5. I have worked as a mining engineer continuously for a total of 18 years since my graduation from university. I have relevant work experience in the evaluation and extraction of precious metals, base metals, industrial minerals, phosphate, coal, graphite and lithium mining projects. Similar Preliminary Economic Assessment projects specifically include those done for Bindal Gruver's Bindal gold project in Norway, Finnish Minerals Group's Sokli multi-elements project in Finland, Leading Edge Materials' Woxna graphite project in Sweden, as well as several due diligence gold projects in the Nordic countries done confidentially for various clients.
6. I have read the definition of "qualified person" set out in *National Instrument 43-101 Standards of Disclosure for Mineral Projects* ("NI 43-101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
7. I have not visited Mawson Gold Limited's Rajapalot property in Finland.
8. I am responsible for Sections 17 and 18.2 to 18.6 and part of Section 21 of the Technical Report.
9. I am independent of Mawson Gold Limited as described in Section 1.5 of NI 43-101.
10. I have not had prior involvement with the property that is the subject of the Technical Report;
11. I have read NI 43-101 and Form 43-101F1 and Sections of the Technical Report I am responsible for have been prepared in compliance with that instrument and form.
12. As of the aforementioned Effective Date, to the best of my knowledge, information and belief, the Sections of the Technical Report for which I am responsible for contain all scientific and technical information that is required to be disclosed to make the portion of the Technical Report for which I am responsible not misleading.

Dated this 28 Day of November, 2022.

Sealed

(Signed) "*Mathieu Gosselin*"

Mathieu Gosselin, P. Eng.

INGÉNIEUR - ENGINEER
Mathieu Gosselin
135077
QUÉBEC

APPENDIX

B DRILL COLLAR COORDINATE DATA

Table A 1: Drill collar coordinate data

Hole_ID	Hole Size	Depth	Prospect	East_KKJ	North_KKJ	Elev_KKJ	Azimuth	Plunge
PAL0001	NQ	111.0	Hirvimaa	3408871	7374467	175.5	135.0	-45.2
PAL0002	NQ	98.8	Hirvimaa	3408912	7374424	175.5	135.0	-45.8
PAL0003	HQ	14.1	Hirvimaa	3409829	7374326	176.8	150.0	-45.0
PAL0003B	NQ	160.5	Hirvimaa	3409832	7374326	176.8	150.0	-44.8
PAL0004	NQ	121.5	Hirvimaa	3409865	7374269	176.5	150.0	-44.6
PAL0005	NQ	51.6	Hirvimaa	3409944	7374362	176.4	150.0	-45.0
PAL0006	NQ	95.5	Hirvimaa	3409977	7374300	174.4	150.0	-43.8
PAL0007	NQ	104.3	Hirvimaa	3411300	7374683	175.0	150.0	-44.5
PAL0008	NTW	158.4	Hirvimaa	3409232	7374282	175.7	134.4	-60.0
PAL0009	NTW	201.5	Palokas	3408551	7373912	173.6	116.0	-60.0
PAL0010	NTW	286.0	Palokas	3408460	7373955	173.4	116.0	-60.0
PAL0011	NTW	11.8	Hirvimaa	3409406	7374222	176.1	135.0	-60.0
PAL0012	NTW	233.6	Palokas	3408516	7373837	173.5	111.7	-60.0
PAL0013	NTW	196.8	South Palokas	3408416	7373634	174.1	116.0	-60.0
PAL0014	NTW	20.0	South Palokas	3408332	7373665	174.0	116.0	-60.0
PAL0015	NTW	151.9	Palokas	3408571	7373725	173.5	116.0	-60.0
PAL0016	NTW	260.5	South Palokas	3408333	7373666	174.0	116.0	-60.0
PAL0017	NTW	221.2	Palokas	3408481	7373769	173.5	116.0	-60.0
PAL0018	NTW	240.7	South Palokas	3408447	7373692	173.4	116.0	-60.0
PAL0019	NTW	217.1	Palokas	3408559	7373999	173.6	135.0	-60.0
PAL0020	NTW	149.3	Terry's Hammer	3407945	7372994	173.0	135.0	-55.0
PAL0021	NTW	172.2	Rumajarvi	3408115	7372552	179.2	160.0	-53.5
PAL0022	NTW	100.5	Rumajarvi	3408175	7372409	181.8	160.0	-55.0
PAL0023	NTW	242.4	Raja	3408859	7372345	173.4	160.0	-55.0
PAL0024	NTW	256.2	Palokas	3408712	7374073	173.8	200.0	-65.0
PAL0025	NTW	170.4	Hirvimaa	3408990	7374176	174.5	330.0	-55.0
PAL0026	NQ2	71.2	Palokas	3408667	7373861	174.6	116.0	-60.0
PAL0027	NQ2	301.6	Palokas	3408670	7373860	174.6	116.3	-59.2
PAL0028	NQ2	92.3	Palokas	3408725	7373889	174.6	119.2	-60.7
PAL0029	NQ2	209.3	Palokas	3408628	7373987	175.4	114.3	-60.2
PAL0030	NQ2	194.8	Palokas	3408606	7373941	174.1	118.5	-59.6
PAL0031	NQ2	131.0	Palokas	3408703	7373952	174.2	123.1	-59.3
PAL0032	NQ2	174.2	Palokas	3408801	7374094	174.6	131.8	-59.6
PAL0033	NQ2	215.8	Hut	3408126	7373140	173.0	149.5	-59.9
PAL0034	NQ2	143.6	Hut	3408167	7373072	175.8	151.7	-59.7
PAL0035	NQ2	191.8	Terry's Hammer	3408095	7372898	176.9	137.1	-60.2
PAL0036	NQ2	115.1	Terry's Hammer	3408123	7372857	175.4	139.6	-59.4
PAL0037	NQ2	244.3	Rumajarvi	3408008	7372395	176.2	116.5	-60.0
PAL0038	NQ2	300.5	Rumajarvi	3407904	7372443	175.5	117.0	-60.1
PAL0039	NQ2	248.8	Rumajarvi	3408010	7372471	177.7	120.5	-49.3
PAL0040	NQ2	200.1	Rumajarvi	3407938	7372359	178.3	120.6	-49.6
PAL0041	NQ2	341.4	Rumajarvi	3407936	7372540	175.4	115.5	-50.1
PAL0042	NQ2	257.2	Rumajarvi	3407842	7372407	172.6	119.9	-48.6
PAL0043	NQ2	339.0	Uusisaari	3407843	7372798	174.4	117.2	-58.4
PAL0044	NQ2	250.6	Rumajarvi	3407645	7372424	173.1	90.6	-50.0
PAL0045	NQ2	352.1	Rumajarvi	3407533	7372697	174.5	115.7	-50.4
PAL0046	NQ2	108.1	Rumajarvi	3408153	7372322	179.5	138.5	-60.5
PAL0047	NTW	100.3	Regional	3410582	7373350	161.4	150.0	-50.0
PAL0048	NQ2	188.2	Raja	3408816	7372268	173.4	91.5	-48.1
PAL0049	NQ2	254.7	Rumajarvi	3408268	7372633	174.5	182.3	-59.9
PAL0050	NTW	103.5	Regional	3410617	7373308	161.0	141.2	-51.4
PAL0051	NQ2	153.9	Raja	3408810	7372200	172.9	95.5	-49.6
PAL0052	NTW	100.2	Regional	3410568	7373392	160.3	151.5	-48.4
PAL0053	NQ2	260.8	Rumajarvi	3408284	7372532	175.3	183.7	-60.2
PAL0054	NTW	154.5	Regional	3410651	7373254	163.0	150.0	-50.8
PAL0055	NQ2	190.7	Rumajarvi	3408380	7372319	176.3	151.8	-50.6
PAL0056	NQ2	268.2	Raja	3408708	7372201	174.6	93.5	-48.7
PAL0057	NTW	144.0	Regional	3410688	7373201	165.2	147.4	-49.5
PAL0058	NQ2	258.3	Raja	3408715	7372254	173.9	95.5	-49.9

Hole_ID	Hole Size	Depth	Prospect	East_KKJ	North_KKJ	Elev_KKJ	Azimuth	Plunge
PAL0059	NQ2	157.1	Rumajarvi	3408091	7372461	177.1	151.2	-59.1
PAL0060	NTW	153.0	Regional	3410987	7371862	138.0	70.1	-49.5
PAL0061	NQ2	256.7	Regional	3409764	7372753	158.2	150.8	-59.6
PAL0062	NQ2	237.0	Raja	3408753	7372464	175.3	156.1	-60.4
PAL0063	NQ2	173.9	Rumajarvi	3407949	7372717	172.8	114.2	-49.8
PAL0064	NTW	120.0	Regional	3411065	7371882	138.5	70.0	-49.6
PAL0065	NTW	97.5	Regional	3410951	7371900	138.1	70.0	-52.2
PAL0066	NQ2	252.2	Raja	3408970	7372539	174.2	159.1	-60.4
PAL0067	NQ2	203.0	Regional	3410017	7373124	162.0	133.9	-59.2
PAL0068	NQ2	255.7	Raja	3409010	7372419	171.9	162.5	-59.8
PAL0069	NQ2	85.7	South Palokas	3408450	7373348	172.1	112.0	-58.0
PAL0070	NTW	103.5	Regional	3410245	7372112	144.2	145.0	-50.0
PAL0071	NQ2	152.8	Raja	3408573	7372276	175.8	153.8	-50.0
PAL0072	NTW	121.5	Regional	3410288	7372052	144.0	144.3	-49.4
PAL0073	NQ2	445.9	Hirvimaa	3408968	7374400	176.0	173.2	-49.3
PAL0074	NQ2	142.1	South Palokas	3408407	7373279	172.0	112.5	-59.8
PAL0075	NQ2	178.2	Raja	3408930	7372245	172.7	287.8	-48.0
PAL0076	NQ2	254.4	Raja	3409032	7372290	169.8	181.9	-49.6
PAL0077	NQ2	25.3	South Palokas	3408311	7373328	171.7	116.0	-60.0
PAL0078	NQ2	237.1	South Palokas	3408309	7373329	171.9	115.8	-59.5
PAL0079	NQ2	206.0	Regional	3409673	7373283	172.7	300.8	-50.1
PAL0080	NQ2	161.5	Hirvimaa	3409416	7374396	178.3	160.0	-47.7
PAL0081	NQ2	167.2	Hirvimaa	3409463	7374241	176.4	162.4	-51.1
PAL0082	NQ2	292.4	South Palokas	3408299	7373425	173.9	112.3	-59.9
PAL0083	NQ2	101.7	Raja	3408869	7372212	172.0	59.2	-60.0
PAL0084	NQ2	191.2	South Palokas	3408481	7373565	174.4	119.6	-65.0
PAL0085	NQ2	215.7	Raja	3408764	7372324	173.5	58.6	-69.8
PAL0086	NQ2	135.0	Palokas	3408735	7373936	174.3	117.4	-60.0
PAL0087	NQ2	241.7	Raja	3408765	7372324	173.3	59.3	-49.6
PAL0088	NQ2	221.5	Raja	3408764	7372324	173.5	72.7	-88.3
PAL0089	NQ2	169.0	South Palokas	3408438	7373588	175.4	159.8	-60.0
PAL0090	NQ2	320.3	Palokas	3408591	7374004	173.7	118.3	-74.0
PAL0091	NQ2	352.8	South Palokas	3408412	7373657	174.1	157.5	-60.0
PAL0092	NQ2	323.9	Raja	3408707	7372440	175.0	60.0	-83.0
PAL0093	NQ2	329.8	Raja	3408707	7372441	174.9	57.7	-75.0
PAL0094	NQ2	191.0	South Palokas	3408527	7373605	173.5	120.7	-60.0
PAL0095	NQ2	370.0	Palokas	3408590	7374004	173.9	116.0	-88.0
PAL0096	NQ2	131.0	South Palokas	3408591	7373662	173.4	115.1	-60.0
PAL0097	NQ2	344.6	Raja	3408707	7372441	174.9	56.1	-69.1
PAL0098	NQ2	199.9	South Palokas	3408380	7373476	173.5	118.3	-60.0
PAL0099	NQ2	154.6	Terry's Hammer	3408190	7372764	178.7	120.1	-60.0
PAL0100	NQ2	343.8	Raja	3408707	7372441	174.9	55.1	-61.7
PAL0101	NQ2	182.7	Terry's Hammer	3408110	7372764	173.6	106.8	-60.0
PAL0102	NQ2	202.7	Palokas	3408754	7374037	174.6	117.7	-60.0
PAL0103	NQ2	173.3	Terry's Hammer	3408053	7372789	173.1	102.1	-60.0
PAL0104	NQ2	326.7	Raja	3408708	7372440	174.8	240.0	-88.0
PAL0105	NQ2	220.9	Rumajarvi	3407898	7372624	172.5	122.5	-60.0
PAL0106	NQ2	161.1	Palokas	3408864	7373985	174.8	127.7	-60.0
PAL0107	NQ2	335.1	Raja	3408775	7372487	176.2	58.1	-70.3
PAL0108	NQ2	226.9	Rumajarvi	3407961	7372405	175.8	120.1	-60.0
PAL0109	NQ2	289.9	Rumajarvi	3407964	7372403	175.8	62.9	-50.0
PAL0110	NQ2	128.2	Palokas	3408646	7373807	173.6	115.9	-60.2
PAL0111	NQ2	432.3	Raja	3408577	7372514	177.6	60.7	-69.0
PAL0112	NQ2	221.7	Hut	3408289	7373151	171.2	182.4	-55.0
PAL0113	NQ2	20.0	Palokas	3408533	7374097	173.6	111.1	-70.0
PAL0114	NQ2	218.4	Rumajarvi	3407875	7372383	174.5	118.8	-47.0
PAL0115	NQ2	318.9	Rumajarvi	3407900	7372520	173.1	123.8	-48.0
PAL0116	WL-76	186.7	Raja	3408860	7372371	173.5	235.4	-80.9
PAL0117	NQ2	148.9	South Palokas	3408481	7373336	172.0	117.2	-45.0
PAL0118	NQ2	445.5	Raja	3408577	7372514	177.6	60.8	-62.0

Hole_ID	Hole Size	Depth	Prospect	East_KKJ	North_KKJ	Elev_KKJ	Azimuth	Plunge
PAL0119	WL-76	178.2	Raja	3408916	7372341	172.6	240.0	-88.0
PAL0120	NQ2	170.7	South Palokas	3408530	7373319	171.2	117.7	-47.0
PAL0121	NQ2	251.0	Rumajarvi	3407986	7372584	176.8	120.9	-50.0
PAL0122	NQ2	209.6	South Palokas	3408354	7373580	175.1	122.9	-60.0
PAL0123	NQ2	199.5	Rumajarvi	3407937	7372656	173.6	120.0	-60.0
PAL0124	WL-76	132.6	Raja	3408563	7372192	173.7	73.4	-50.0
PAL0125	NQ2	112.5	Raja	3408577	7372514	177.4	60.3	-56.0
PAL0126	NQ2	8.9	Hut	3408089	7373033	173.2	90.0	-60.0
PAL0127	NQ2	157.7	Hirvimaa	3409495	7374571	178.1	140.1	-50.0
PAL0128	NQ2	305.7	Hirvimaa	3410577	7372673	150.1	34.6	-50.0
PAL0129	NQ2	305.0	Hirvimaa	3409604	7372112	151.5	38.9	-50.0
PAL0130	NQ2	212.0	Hirvimaa	3409439	7374639	178.5	141.2	-50.0
PAL0131	NQ2	149.5	Hirvimaa	3410497	7372437	145.7	124.2	-50.0
PAL0132	NQ2	300.1	Regional	3409525	7371980	163.4	37.5	-50.0
PAL0133	NQ2	167.3	Hirvimaa	3410336	7373235	163.6	136.7	-50.0
PAL0134	NQ2	281.2	Hirvimaa	3409377	7374729	181.6	139.9	-50.0
PAL0135	NQ2	196.3	Hirvimaa	3410402	7373172	161.0	135.3	-50.0
PAL0136	NQ2	293.3	Regional	3409441	7371857	165.2	218.8	-60.0
PAL0137	NQ2	212.0	Hirvimaa	3410476	7373095	159.1	136.8	-50.0
PAL0138	NQ2	221.2	Hirvimaa	3410582	7372990	156.6	134.3	-50.0
PAL0139	NQ2	139.4	Hirvimaa	3409645	7374573	180.6	142.5	-50.0
PAL0140	NQ2	440.5	Regional	3409356	7371737	159.7	214.8	-60.0
PAL0141	NQ2	143.4	Regional	3411012	7372820	159.7	141.5	-50.0
PAL0142	NQ2	157.3	Regional	3410965	7372856	162.4	139.6	-50.0
PAL0143	NQ2	196.8	Hirvimaa	3409600	7374623	180.2	142.5	-50.0
PAL0144	NQ2	110.5	Hirvimaa	3410155	7374827	180.4	160.7	-50.0
PAL0145	NQ2	450.0	Regional	3412561	7373159	171.6	177.3	-60.0
PAL0146	NQ2	259.9	Hirvimaa	3409475	7374728	181.7	141.8	-50.0
PAL0147	NQ2	203.6	Hirvimaa	3410098	7374933	181.0	154.4	-50.0
PAL0148	NQ2	266.3	Korkiakoivikko	3410602	7368194	131.2	149.5	-60.0
PAL0149	NQ2	130.1	Hirvimaa	3410585	7374738	178.4	150.7	-60.0
PAL0150	NQ2	136.6	Hirvimaa	3410548	7374792	179.6	151.8	-60.0
PAL0151	NQ2	15.1	Hirvimaa	3410959	7374848	179.4	170.0	-60.0
PAL0152	NQ2	200.5	Hirvimaa	3410960	7374846	179.4	167.4	-60.0
PAL0153	NQ2	140.5	Hirvimaa	3410999	7374408	172.5	141.1	-60.0
PAL0154	NQ2	91.1	Hirvimaa	3411029	7374367	172.7	139.8	-60.0
PAL0155	NQ2	169.9	Hirvimaa	3410864	7374328	172.4	139.9	-60.0
PAL0156	NQ2	140.8	Hirvimaa	3410907	7374278	173.5	139.9	-60.0
PAL0157	NQ2	149.2	Hirvimaa	3410648	7374381	172.8	140.2	-60.0
PAL0158	NQ2	219.9	Hirvimaa	3411510	7374818	180.1	163.6	-60.0
PAL0159	NQ2	473.8	Raja	3408545	7372603	179.4	57.6	-70.4
PAL0160	NQ2	447.1	Raja	3408486	7372581	178.0	66.6	-79.8
PAL0161	NQ2	407.8	Raja	3408695	7372555	178.9	53.7	-76.6
PAL0162	NQ2	482.8	Raja	3408446	7372648	180.3	46.0	-84.5
PAL0163	NQ2	467.0	Raja	3408486	7372588	178.6	65.0	-73.6
PAL0164	NQ2	441.7	Raja	3408545	7372603	179.4	59.9	-75.7
PAL0165	NQ2	167.9	Raja	3408612	7372312	176.9	60.0	-79.0
PAL0166	NQ2	238.6	Raja	3408897	7372385	172.5	240.0	-83.0
PAL0167	NQ2	398.6	Raja	3408486	7372587	178.3	95.7	-85.0
PAL0168	NQ2	45.6	Raja	3408557	7372808	176.1	233.4	-83.0
PAL0169	NQ2	545.8	Raja	3408556	7372807	176.1	232.9	-82.9
PAL0170	NQ2	200.2	Raja	3408714	7372255	174.0	59.5	-79.1
PAL0171	NQ2	497.6	Raja	3408604	7372635	180.3	58.0	-73.1
PAL0172	NQ2	491.9	Raja	3408445	7372648	180.5	49.6	-79.5
PAL0173	NQ2	427.9	South Palokas	3408256	7373707	172.4	116.3	-56.2
PAL0174	NQ2	8.3	South Palokas	3408255	7373707	172.5	116.0	-69.5
PAL0175	NQ2	120.1	Raja	3408830	7372237	172.7	60.0	-74.0
PAL0176	NQ2	140.0	Raja	3408937	7372300	173.4	204.0	-79.5
PAL0177	NQ2	250.5	Rumajarvi	3408433	7372388	176.3	240.0	-60.0
PAL0178	NQ2	237.2	Rumajarvi	3408226	7372340	177.5	60.0	-75.0

Hole_ID	Hole Size	Depth	Prospect	East_KKJ	North_KKJ	Elev_KKJ	Azimuth	Plunge
PAL0179	NQ2	209.0	Rumajarvi	3408106	7372351	180.7	59.9	-80.2
PAL0180	NQ2	778.7	Raja	3408127	7372706	174.0	41.0	-61.4
PAL0181	NQ2	161.7	Rumajarvi	3407953	7372245	176.2	149.9	-60.0
PAL0182	NQ2	439.7	Rumajarvi	3407944	7372476	177.0	60.0	-69.8
PAL0183	NQ2	170.0	Rumajarvi	3408094	7372423	177.4	159.8	-70.0
PAL0184	NQ2	211.8	Uusisaari	3407755	7372867	173.2	119.9	-49.8
PAL0185	NQ2	381.1	Rumajarvi	3407900	7372521	173.1	59.6	-68.1
PAL0186	NQ2	341.9	Rumajarvi	3407905	7372445	174.6	55.0	-75.0
PAL0187	NQ2	474.0	Raja	3408547	7372492	177.3	46.6	-63.4
PAL0188	NQ2	379.4	Raja	3408630	7372441	177.3	52.6	-63.5
PAL0189	NQ2	245.5	Raja	3408768	7372378	173.7	49.4	-76.8
PAL0190	NQ2	427.9	Raja	3408577	7372513	177.6	61.0	-65.4
PAL0191	NQ2	492.1	Raja	3408548	7372493	177.2	44.2	-58.5
PAL0192	NQ2	203.2	Hut	3408223	7373179	172.0	130.5	-60.0
PAL0193	NQ2	427.2	South Palokas	3408255	7373708	172.5	103.9	-53.2
PAL0194	NQ2	497.8	Palokas	3408312	7373980	173.8	74.5	-57.2
PAL0195	NQ2	245.6	South Palokas	3408354	7373580	175.2	64.8	-77.0
PAL0196	NQ2	317.4	Hut	3408091	7373032	173.5	90.5	-60.0
PAL0197	NQ2	466.8	South Palokas	3408272	7373630	173.9	61.6	-66.8
PAL0198	NQ2	296.2	South Palokas	3408414	7373660	173.9	116.0	-70.6
PAL0199	NQ2	386.7	Hut	3408124	7373140	172.7	215.1	-80.0
PAL0200	NQ2	536.8	Palokas	3408310	7373979	174.1	62.0	-61.8
PAL0201	NQ2	281.0	Raja	3408546	7372603	179.3	56.0	-67.3
PAL0201D	NQ2	524.6	Raja	3408546	7372603	179.3	56.0	-67.2
PAL0202	NQ2	769.6	Palokas	3408978	7374403	175.9	229.3	-45.2
PAL0202A	NQ2	826.7	Palokas	3408978	7374403	175.9	229.3	-45.2
PAL0203	NQ2	415.5	South Palokas	3408273	7373631	173.6	58.0	-62.8
PAL0204	NQ2	149.2	South Palokas	3408522	7373604	173.4	234.8	-85.0
PAL0205	NQ2	191.5	Palokas	3408587	7373803	173.5	58.1	-49.0
PAL0206	NQ2	326.2	Palokas	3408464	7373918	173.7	62.7	-57.5
PAL0207	NQ2	200.2	Palokas	3408610	7373895	173.6	57.1	-76.3
PAL0208	NQ2	555.4	Raja	3408541	7372693	179.2	52.2	-75.4
PAL0209	NQ2	200.8	South Palokas	3408471	7373638	173.5	58.0	-82.0
PAL0210	NQ2	198.0	Palokas	3408610	7373895	173.6	54.0	-86.0
PAL0211	NQ2	323.2	Palokas	3408464	7373918	173.7	62.7	-50.0
PAL0212	NQ2	492.6	South Palokas	3408256	7373711	172.1	59.0	-75.5
PAL0213	NQ2	509.3	South Palokas	3408273	7373631	173.6	60.0	-73.5
PAL0214	NQ2	154.3	Palokas	3408611	7373895	173.6	57.0	-52.0
PAL0215	NQ2	395.5	Palokas	3408676	7374105	173.8	237.0	-77.5
PAL0216	NQ2	344.6	Palokas	3408464	7373918	173.7	62.0	-65.0
PAL0217	NQ2	519.2	Raja	3408541	7372693	179.2	52.1	-79.5
PAL0218	NQ2	469.4	Palokas	3408308	7373979	173.7	74.5	-58.0
PAL0219	NQ2	419.7	South Palokas	3408273	7373631	173.6	59.2	-57.9
PAL0220	NQ2	501.1	South Palokas	3408255	7373710	172.1	61.5	-80.0
PAL0221	NQ2	280.4	Palokas	3408464	7373916	173.6	96.4	-53.5
PAL0222	NQ2	355.1	Palokas	3408464	7373917	173.7	65.9	-71.5
PAL0223	NQ2	404.1	South Palokas	3408273	7373631	173.6	61.0	-79.0
PAL0224	NQ2	560.6	South Palokas	3408168	7373754	171.5	63.0	-78.5
PAL0225	NQ2	490.9	South Palokas	3408255	7373710	172.1	69.5	-85.0
PAL0226	NQ2	487.8	Raja	3408540	7372692	179.2	53.0	-83.5
PAL0227	NQ2	359.4	Palokas	3408463	7373917	173.7	69.0	-77.5
PAL0228	NQ2	311.4	Palokas	3408463	7373918	173.6	110.0	-67.0
PAL0229	NQ2	635.4	South Palokas	3408168	7373754	171.5	56.0	-81.2
PAL0230	NQ2	631.4	Raja	3408487	7372776	177.0	47.0	-82.0
PAL0231	NQ2	395.6	Palokas	3408462	7373916	173.6	73.1	-82.7
PAL0232	NQ2	524.0	Palokas	3408270	7373876	173.4	57.0	-60.0
PAL0233	NQ2	167.5	Palokas	3408585	7373802	173.3	58.0	-70.0
PAL0234	NQ2	178.8	Palokas	3408271	7373876	173.4	54.0	-56.0
PAL0235	NQ2	522.0	South Palokas	3408208	7373668	172.7	47.0	-81.0
PAL0236	NQ2	530.0	Palokas	3408271	7373875	173.3	49.0	-56.0

Hole_ID	Hole Size	Depth	Prospect	East_KKJ	North_KKJ	Elev_KKJ	Azimuth	Plunge
PAL0237	NQ2	68.5	Hirvimaa	3409690	7374570	180.4	220.0	-61.0
PAL0238	NQ2	149.7	Hirvimaa	3409663	7374613	181.1	220.0	-77.0
PAL0239	NQ2	41.7	Joki East	3410303	7372643	151.0	60.0	-66.0
PAL0240	NQ2	281.7	Joki East	3410305	7372644	151.2	60.0	-66.0
PAL0241	NQ2	236.4	Joki East	3410338	7372661	151.3	60.0	-66.0
PAL0242	NQ2	236.8	Joki East	3410364	7372675	150.6	60.0	-66.0
PAL0243	NQ2	239.7	Joki East	3410309	7372708	151.4	60.0	-67.5
PAL0244	NQ2	251.7	Joki East	3410337	7372726	151.4	62.0	-68.0
PAL0245	NQ2	257.5	Joki East	3410275	7372690	151.4	60.0	-66.0
PAL0246	NQ2	287.6	Joki East	3410266	7372745	152.3	60.0	-71.0
PAL0247	NQ2	293.4	Joki East	3410212	7372729	151.5	61.0	-64.0
PAL0248	NQ2	323.6	Regional	3411715	7371405	124.9	65.0	-60.0
PAL0249	NQ2	269.6	Joki East	3410204	7372724	151.6	64.0	-72.0
PAL0250	NQ2	195.3	Joki East	3410404	7372632	151.2	60.0	-66.0
PAL0251	NQ2	179.9	Joki East	3410375	7372617	151.0	60.0	-66.0
PAL0252	NQ2	155.9	Joki East	3410435	7372651	149.5	60.0	-66.0
PAL0253	NQ2	359.7	Joki East	3410154	7372820	153.8	61.0	-78.5
PAL0254	NQ2	320.9	Joki East	3410153	7372821	155.0	61.0	-70.5
PAL0255	NQ2	347.9	Hut	3408126	7373140	172.5	90.0	-85.0
PAL0256	NQ2	272.6	Hut	3408126	7373140	172.5	88.0	-72.0
PAL0257	NQ2	230.4	Hut	3408127	7373140	172.5	87.0	-58.0
PAL0258	NQ2	389.8	Rumajarvi	3407835	7372450	172.3	39.0	-85.0
PAL0259	NQ2	299.9	Hut	3408064	7372937	173.4	57.0	-61.5
PAL0260	NQ2	320.6	Hut	3408089	7373033	173.1	58.7	-70.0
PAL0261	NQ2	311.7	Hut	3408064	7372937	173.4	57.0	-74.0
PAL0262	NQ2	361.9	Palokas	3408464	7373910	173.6	139.0	-73.0
PAL0263	NQ2	329.8	Hut	3408089	7373033	173.1	58.7	-84.0
PAL0264	NQ2	125.5	Rumajarvi	3407834	7372450	172.8	39.0	-68.0
PAL0265	NQ2	301.8	Hut	3407957	7373144	172.1	143.0	-49.0
PAL0266	NQ2	149.7	Rumajarvi	3407835	7372449	172.3	210.0	-78.0
PAL0267	NQ2	268.9	Rumajarvi	3407841	7372408	172.7	65.3	-48.2
PAL0268	NQ2	131.5	Terry's Hammer	3408186	7372768	178.7	60.0	-80.0
PAL0269	NQ2	268.5	Hut	3407957	7373144	172.1	126.0	-46.0
PAL0270	NQ2	289.8	Palokas	3408464	7373910	173.6	124.0	-59.0
PAL0271	NQ2	120.0	Terry's Hammer	3408186	7372768	178.7	210.0	-85.0
PAL0272	NQ2	302.6	Rumajarvi	3407841	7372408	172.7	65.0	-73.0
PAL0273	NQ2	82.1	Terry's Hammer	3408216	7372747	177.3	119.0	-54.0
PAL0274	NQ2	280.3	Hut	3407957	7373144	172.1	114.0	-45.0
PAL0275	NQ2	161.8	Hut	3408089	7373033	173.1	240.0	-81.0
PAL0276	NQ2	23.9	Palokas	3408468	7373868	172.0	128.0	-50.0
PAL0277	NQ2	257.3	Hut	3408091	7373033	173.6	135.0	-55.0
PAL0278	NQ2	280.0	Hut	3407957	7373143	172.1	150.0	-50.0
PAL0279	NQ2	287.9	Palokas	3408468	7373868	172.0	128.0	-50.0
PAL0280	NQ2	343.0	Rumajarvi	3407642	7372427	173.0	61.0	-38.0
PAL0281	NQ2	146.3	South Palokas	3408545	7373675	173.5	116.0	-60.0
PAL0282	NQ2	341.9	Hut	3407941	7373071	172.7	61.0	-67.0
PAL0283	NQ2	277.9	Palokas	3408468	7373868	173.5	141.0	-52.1
PAL0284	NQ2	146.6	South Palokas	3408521	7373606	173.6	62.0	-79.0
PAL0285	NQ2	316.4	Rumajarvi	3407642	7372427	173.0	61.0	-47.0
PAL0286	NQ2	149.4	South Palokas	3408521	7373606	173.6	240.0	-69.0
PAL0287	NQ2	346.7	Hut	3407941	7373071	172.7	61.0	-76.0
PAL0288	NQ2	172.8	South Palokas	3408521	7373606	173.6	240.0	-57.0
PAL0289	NQ2	305.2	Palokas	3408468	7373868	172.0	155.0	-52.0
PAL0290	NQ2	335.6	South Palokas	3408411	7373660	174.0	235.0	-78.0
PAL0291	NQ2	329.3	Hut	3407941	7373071	172.7	61.0	-85.0
PAL0292	NQ2	149.1	Terry's Hammer	3408112	7372770	172.4	60.0	-61.0
PAL0293	NQ2	347.3	Palokas	3408468	7373868	172.0	61.0	-68.0
PAL0294	NQ2	353.7	Hut	3407941	7373071	172.7	220.0	-87.0
PAL0295	NQ2	140.2	Raja	3408821	7372288	172.7	58.0	-80.0
PAL0296	NQ2	368.7	South Palokas	3408411	7373660	174.0	241.0	-71.5

Hole_ID	Hole Size	Depth	Prospect	East_KKJ	North_KKJ	Elev_KKJ	Azimuth	Plunge
PAL0297	NQ2	169.5	Raja	3408821	7372288	172.7	58.0	-66.0
PAL0298	NQ2	305.1	Palokas	3408467	7373867	173.9	128.0	-65.0
PAL0299	NQ2	394.7	South Palokas	3408411	7373660	174.0	241.0	-64.5
PAL0300	NQ2	142.5	Raja	3408821	7372288	172.7	245.0	-80.0
PAL0301	NQ2	335.1	Hut	3407999	7373194	172.1	115.0	-57.0
PAL0302	NQ2	163.8	Raja	3408913	7372341	172.3	238.0	-73.0
PAL0303	NQ2	629.2	South Palokas	3408090	7373644	172.7	44.0	-75.5
PAL0304	NQ2	125.3	South Palokas	3408526	7373603	173.6	160.0	-58.0
PAL0305	NQ2	281.5	South Palokas	3408411	7373660	174.0	50.0	-82.0
PAL0306	NQ2	280.6	Uusisaari	3407843	7372798	172.4	60.0	-45.0
PAL0307	NQ2	352.9	South Palokas	3408273	7373630	174.7	66.0	-85.0
PAL0308	NQ2	515.6	South Palokas	3408134	7373634	173.0	50.0	-77.0
PAL0309	NQ2	202.5	Rumajarvi	3407850	7372499	172.5	81.0	-74.0
PAL0310	NQ2	209.5	Palokas	3408610	7373895	174.9	167.0	-76.0
PAL0311	KATI100	78.9	Palokas	3408610	7373895	174.9	96.0	-55.0
PRAJ0001	EW	3.5	Hirvimaa	3408254	7375081	178.0	358.0	-60.0
PRAJ0002	EW	7.0	Hirvimaa	3408254	7375081	178.0	358.0	-60.0
PRAJ0003	EW	20.2	Palokas	3408695	7373822	176.5	118.0	-60.0
PRAJ0004	EW	19.6	Palokas	3408689	7373825	175.6	116.0	-60.0
PRAJ0005	EW	19.3	Palokas	3408681	7373831	175.0	116.0	-60.0
PRAJ0006	EW	29.0	Palokas	3408695	7373839	175.6	116.0	-60.0
PRAJ0007	EW	5.0	Palokas	3408713	7373829	175.8	116.0	-60.0
PRAJ0008	EW	13.3	Palokas	3408704	7373834	176.1	116.0	-60.0
PRAJ0009	EW	35.0	Palokas	3408703	7373856	175.7	116.0	-60.0
PRAJ0010	EW	13.0	Palokas	3408720	7373849	175.5	116.0	-60.0
PRAJ0011	EW	4.3	Palokas	3408774	7373870	175.2	140.0	-60.0
PRAJ0012	EW	3.4	Palokas	3408733	7373904	175.1	140.0	-60.0
PRAJ0013	EW	2.3	Palokas	3408738	7373898	174.2	140.0	-60.0
PRAJ0014	EW	7.3	Palokas	3408748	7373890	176.2	140.0	-60.0
PRAJ0015	EW	12.7	Palokas	3408730	7373869	174.4	116.0	-60.0
PRAJ0016	EW	28.4	Palokas	3408691	7373801	174.7	116.0	-60.0
PRAJ0017	EW	23.0	Palokas	3408685	7373777	173.1	116.0	-60.0
PRAJ0018	EW	20.8	Palokas	3408673	7373760	173.8	116.0	-60.0
PRAJ0019	EW	4.5	Palokas	3408666	7373788	173.3	116.0	-60.0
PRAJ0020	EW	31.2	Palokas	3408680	7373809	174.9	116.0	-60.0
PRAJ0021	EW	19.9	Palokas	3408703	7373818	176.8	116.0	-60.0
PRAJ0022	EW	37.5	Palokas	3408687	7373843	174.3	116.0	-60.0
PRAJ0023	EW	28.5	Palokas	3408695	7373862	175.2	116.0	-76.5
PRAJ0024	EW	37.0	Palokas	3408685	7373869	174.1	116.0	-76.5
PRAJ0025	EW	40.3	Palokas	3408710	7373874	175.9	116.0	-76.5
PRAJ0026	EW	35.2	Palokas	3408702	7373855	175.7	296.0	-75.5
PRAJ0027	EW	12.5	Palokas	3408795	7373926	175.0	140.0	-76.5
PRAJ0028	EW	22.3	Palokas	3408784	7373942	175.4	140.0	-76.5
PRAJ0029	EW	20.6	Palokas	3408772	7373959	174.8	140.0	-77.0
PRAJ0030	EW	27.9	Palokas	3408763	7373973	174.4	140.0	-77.0
PRAJ0031	EW	6.7	Hirvimaa	3409623	7375180	185.0	135.0	-60.0
PRAJ0032	EW	13.8	Hirvimaa	3409642	7375161	185.0	132.0	-60.0
PRAJ0033	EW	11.6	Hirvimaa	3409659	7375143	185.0	135.0	-60.0
PRAJ0034	EW	7.8	Hirvimaa	3409611	7375201	185.0	135.0	-60.0
PRAJ0035	EW	9.4	Hirvimaa	3409562	7374969	185.0	224.0	-60.0
PRAJ0036	EW	6.1	Hirvimaa	3409205	7374181	180.0	224.0	-60.0
PRAJ0037	EW	19.3	Hirvimaa	3409200	7374191	180.0	224.0	-60.0
PRAJ0038	EW	13.6	Hirvimaa	3409178	7374214	180.0	224.0	-60.0
PRAJ0039	EW	11.6	Hirvimaa	3409483	7374210	176.0	135.0	-60.0
PRAJ0040	EW	12.5	Hirvimaa	3409472	7374214	176.0	135.0	-60.0
PRAJ0041	EW	10.1	Hirvimaa	3409467	7374235	176.0	135.0	-61.0
PRAJ0042	EW	11.1	Hirvimaa	3409456	7374248	177.0	138.0	-60.0
PRAJ0043	EW	8.8	Hirvimaa	3409444	7374264	178.0	132.0	-60.0
PRAJ0044	EW	13.4	Hirvimaa	3409424	7374272	178.0	135.0	-60.0
PRAJ0045	EW	5.6	Hirvimaa	3409412	7374290	179.0	135.0	-60.0

Hole_ID	Hole Size	Depth	Prospect	East_KKJ	North_KKJ	Elev_KKJ	Azimuth	Plunge
PRAJ0046	EW	6.9	Hirvimaa	3409402	7374306	179.0	135.0	-60.0
PRAJ0047	EW	31.9	Regional	3410301	7373140	162.0	155.0	-58.0
PRAJ0048	EW	34.1	Regional	3410294	7373151	165.0	155.0	-60.0
PRAJ0049	EW	31.3	Regional	3410280	7373160	163.5	155.0	-60.0
PRAJ0050	EW	28.4	Regional	3410478	7373137	161.0	155.0	-60.0
PRAJ0051	EW	7.5	Regional	3410433	7373156	161.0	110.0	-60.0
PRAJ0052	EW	15.8	Regional	3410413	7373164	161.0	115.0	-85.0
PRAJ0053	EW	17.3	Regional	3410354	7373192	163.0	110.0	-60.0
PRAJ0054	EW	9.7	Regional	3410298	7373206	164.0	110.0	-60.0
PRAJ0055	EW	21.8	Regional	3410880	7372767	157.0	123.0	-60.0
PRAJ0056	EW	26.4	Regional	3410600	7372701	150.0	134.0	-60.0
PRAJ0057	EW	26.7	Regional	3409451	7372108	162.0	148.0	-60.0
PRAJ0058	EW	9.0	Regional	3409562	7372099	155.0	152.0	-60.0
PRAJ0059	EW	8.0	Regional	3409572	7372082	155.0	147.0	-60.0
PRAJ0060	EW	8.6	Regional	3409593	7372045	155.0	153.0	-60.0
PRAJ0061	EW	9.5	Regional	3409554	7372042	155.0	153.0	-60.0
PRAJ0062	EW	3.0	Hirvimaa	3409514	7371997	164.0	149.0	-60.0
PRAJ0063	EW	8.0	Regional	3409514	7371997	165.0	120.0	-90.0
PRAJ0064	EW	6.2	Regional	3409647	7372161	160.0	152.0	-86.0
PRAJ0065	EW	18.9	Regional	3410054	7372039	145.0	150.0	-61.0
PRAJ0066	EW	4.3	Regional	3410315	7372378	148.0	149.0	-60.0
PRAJ0067	EW	14.4	Regional	3410764	7372618	153.0	150.0	-60.0
PRAJ0068	EW	10.3	Regional	3410473	7373367	160.0	150.0	-60.0
PRAJ0069	EW	15.4	Regional	3410195	7373026	162.0	152.0	-60.0
PRAJ0070	EW	33.4	South Palokas	3408517	7373547	177.8	115.0	-60.0
PRAJ0071	EW	43.2	South Palokas	3408506	7373551	173.3	115.0	-60.0
PRAJ0072	EW	27.6	South Palokas	3408530	7373539	173.0	115.0	-60.0
PRAJ0073	EW	15.8	South Palokas	3408540	7373536	173.2	115.0	-60.0
PRAJ0074	EW	26.6	South Palokas	3408494	7373508	173.6	155.0	-60.0
PRAJ0075	EW	28.3	South Palokas	3408488	7373523	173.6	155.0	-82.0
PRAJ0076	EW	33.5	Terry's Hammer	3408157	7372825	179.1	59.0	-60.0
PRAJ0077	EW	35.0	Terry's Hammer	3408169	7372831	179.2	60.0	-60.0
PRAJ0078	EW	21.1	Terry's Hammer	3408172	7372821	179.2	115.0	-60.0
PRAJ0079	EW	23.0	Terry's Hammer	3408160	7372818	179.4	175.0	-60.0
PRAJ0080	EW	40.4	Terry's Hammer	3408261	7372787	180.2	115.0	-60.0
PRAJ0081	EW	41.4	Terry's Hammer	3408240	7372785	179.0	115.0	-60.0
PRAJ0082	EW	40.8	Terry's Hammer	3408267	7372780	180.1	115.0	-60.0
PRAJ0083	EW	33.2	Rumajarvi	3408318	7372377	176.6	115.0	-60.0
PRAJ0084	EW	15.2	Rumajarvi	3408230	7372359	176.8	115.0	-60.0
PRAJ0085	EW	43.4	Rumajarvi	3408391	7372273	175.9	60.0	-60.0
PRAJ0086	EW	32.6	Rumajarvi	3408454	7372339	175.5	20.0	-60.0
PRAJ0087	EW	8.9	Raja	3408852	7372282	172.5	115.0	-60.0
PRAJ0088	EW	5.0	Raja	3408815	7372268	173.5	115.0	-60.0
PRAJ0089	EW	5.7	Raja	3408815	7372268	173.5	115.0	-80.0
PRAJ0090	EW	43.3	Raja	3408818	7372227	172.8	90.0	-60.0
PRAJ0091	EW	37.4	Raja	3408840	7372188	173.0	115.0	-60.0
PRAJ0092	EW	35.5	Hut	3408146	7373054	175.9	115.0	-60.0
PRAJ0093	EW	41.0	Hut	3408133	7373061	176.0	115.0	-60.0
PRAJ0094	EW	36.6	Hut	3408134	7373080	174.8	180.0	-60.0
PRAJ0095	EW	8.4	Hut	3408257	7373121	174.1	135.0	-60.0
PRAJ0096	EW	31.4	Hut	3408114	7372972	172.8	115.0	-60.0
PRAJ0097	EW	37.4	Hut	3408105	7372982	173.0	170.0	-60.0
PRAJ0098	EW	27.7	South Palokas	3408443	7373470	178.0	160.0	-60.0
PRAJ0099	EW	35.0	South Palokas	3408436	7373485	173.3	160.0	-60.0
PRAJ0100	EW	31.0	South Palokas	3408428	7373501	173.6	160.0	-60.0
PRAJ0101	EW	48.0	South Palokas	3408419	7373516	174.5	160.0	-60.0
PRAJ0102	EW	48.1	South Palokas	3408413	7373537	174.5	160.0	-60.0
PRAJ0103	EW	49.7	South Palokas	3408403	7373553	174.5	160.0	-60.0
PRAJ0104	EW	32.2	South Palokas	3408469	7373507	173.9	115.0	-60.0
PRAJ0105	EW	46.6	South Palokas	3408452	7373519	174.4	115.0	-60.0

Hole_ID	Hole Size	Depth	Prospect	East_KKJ	North_KKJ	Elev_KKJ	Azimuth	Plunge
PRAJ0106	EW	45.7	South Palokas	3408433	7373530	173.9	115.0	-60.0
PRAJ0107	EW	97.4	Palokas	3408676	7373852	173.9	116.0	-60.0
PRAJ0108	EW	83.0	Palokas	3408664	7373822	173.9	116.0	-60.0
PRAJ0109	EW	71.7	Palokas	3408660	7373866	174.8	116.0	-60.0
PRAJ0110	EW	100.1	Palokas	3408623	7373882	174.0	116.0	-60.0
PRAJ0111	EW	71.8	Palokas	3408649	7373844	174.5	116.0	-60.0
PRAJ0112	EW	43.9	Palokas	3408654	7373768	174.0	116.0	-60.0
PRAJ0113	EW	96.1	Palokas	3408655	7373892	174.0	116.0	-60.0
PRAJ0114	EW	99.8	Palokas	3408662	7373909	174.9	116.0	-60.0
PRAJ0115	EW	81.3	Palokas	3408680	7373921	174.3	116.0	-60.0
PRAJ0116	EW	100.1	Palokas	3408624	7373831	173.9	116.0	-60.0
PRAJ0117	EW	115.2	Palokas	3408605	7373778	174.3	116.0	-60.0
PRAJ0118	EW	89.5	South Palokas	3408497	7373358	171.6	155.0	-60.0
PRAJ0119	EW	40.5	South Palokas	3408475	7373404	172.4	155.0	-60.0
PRAJ0120	EW	34.7	South Palokas	3408522	7373308	173.0	155.0	-60.0

APPENDIX

C SUMMARY OF DRILL INTERSECTIONS AT RAJAPALOT

Table B 1: Summary of drill intersections at Rajapalot

Drillhole intersections at Rajapalot ordered by grade times width AuEq, using a 2.0 g/t lower cutover a maximum of 1 metre.

Drillhole	From	To	Interval	Au g/t	Co ppm	AuEq g/t	g*w AuEq
PAL0093	252.2	261.8	9.7	23.1	1080	24.1	233.0
PRAJ0009	3.9	7.9	4.0	50.5	946	51.4	205.8
PAL0222	266.9	275.1	8.2	19.1	1572	20.7	169.7
PRAJ0006	1.3	16.3	15.0	9.2	769	9.9	149.1
PAL0228	251.4	258.4	7.0	17.0	2168	19.2	134.3
PRAJ0107	25.7	32.7	7.0	17.7	730	18.4	128.8
PAL0030	110.2	120.2	10.0	9.7	562	10.3	103.1
PAL0027	34.4	41.2	6.8	14.1	659	14.8	100.4
PAL0303	561.9	573.9	12.0	6.1	1926	8.0	95.8
PAL0188	321.6	328.6	7.0	11.9	1641	13.5	94.8
PAL0236	449.7	454.6	4.9	18.0	1236	19.2	93.3
PRAJ0003	0.0	3.0	3.0	27.5	851	28.3	85.0
PAL0203	303.0	311.0	8.0	7.9	2672	10.6	84.5
PAL0075	82.2	91.0	8.8	7.5	1229	8.7	76.9
PAL0190	381.8	387.8	6.0	11.8	949	12.8	76.7
PAL0297	74.0	78.2	4.2	18.3	83	18.4	76.2
PAL0213	294.0	304.0	10.0	6.5	1008	7.5	74.8
PAL0194	425.1	432.9	7.8	5.1	4454	9.5	74.4
PAL0092	246.0	249.0	3.0	23.3	1413	24.7	74.2
PAL0295	57.0	69.0	12.0	4.8	908	5.7	68.9
PAL0204	93.7	103.0	9.3	6.3	1018	7.3	67.6
PAL0297	90.3	94.7	4.4	14.3	148	14.4	63.5
PAL0213	317.0	323.0	6.0	9.0	1364	10.4	62.4
PAL0118	381.0	382.6	1.6	37.3	1143	38.4	61.5
PAL0188	307.7	315.6	8.0	5.9	1840	7.7	61.1
PAL0303	575.0	584.0	9.0	5.1	1356	6.4	57.9
PRAJ0114	61.1	68.1	7.0	7.1	947	8.0	56.3
PRAJ0004	2.0	10.3	8.3	5.9	454	6.4	52.8
PAL0190	374.0	378.0	4.0	11.2	1758	13.0	51.9
PRAJ0022	10.0	24.0	14.0	3.0	580	3.6	50.7
PAL0198	171.2	178.8	7.6	5.0	1484	6.5	49.5
PAL0241	168.6	170.2	1.6	28.3	1190	29.5	47.2
PAL0197	298.8	312.2	13.5	1.2	2236	3.4	46.2
PRAJ0109	42.7	49.7	7.0	6.0	494	6.5	45.4
PAL0085	124.0	131.9	7.9	5.0	751	5.7	45.1
PAL0235	447.5	453.5	6.0	5.8	1011	6.8	41.0
PAL0288	124.0	128.0	4.0	9.6	676	10.2	40.9
PAL0247	220.9	224.8	3.8	9.2	999	10.2	39.1
PAL0016	211.0	214.4	3.4	11.0	475	11.5	38.9
PAL0259	100.7	110.3	9.6	1.7	2192	3.9	36.8
PRAJ0111	42.1	44.9	2.8	11.7	1218	12.9	36.3
PRAJ0109	38.7	39.7	1.0	34.9	574	35.5	35.5
PAL0245	177.1	178.4	1.3	25.3	2327	27.6	34.5
PAL0301	207.7	211.2	3.5	7.4	2290	9.7	34.5
PAL0062	186.5	192.5	6.0	5.3	369	5.7	34.0
PRAJ0025	16.9	22.8	5.9	5.4	339	5.7	33.9
PAL0242	155.0	156.6	1.6	19.2	1478	20.7	33.1
PAL0173	276.1	281.0	4.9	4.6	1805	6.4	31.8
PRAJ0005	10.7	19.2	8.6	3.1	474	3.5	30.3
PAL0252	117.0	118.5	1.5	18.1	1696	19.8	29.7
PAL0296	256.0	266.0	10.0	2.4	571	2.9	29.5
PAL0227	296.2	299.1	3.0	9.3	604	9.9	29.2
PAL0206	262.2	263.2	1.0	28.0	377	28.4	28.4
PAL0182	87.0	93.2	6.2	4.0	553	4.5	27.9
PAL0290	242.0	249.8	7.8	2.9	566	3.5	26.9

Drillhole	From	To	Interval	Au g/t	Co ppm	AuEq g/t	g*w AuEq
PAL0194	420.7	423.9	3.2	7.3	1034	8.3	26.5
PRAJ0113	74.4	77.4	3.0	8.3	502	8.8	26.3
PAL0119	16.0	19.0	3.0	8.6	68	8.7	26.1
PAL0216	262.0	266.0	4.0	6.0	444	6.4	25.6
PAL0258	66.9	68.9	2.0	12.3	312	12.6	25.2
PRAJ0023	19.6	26.6	7.0	2.8	782	3.5	24.8
PAL0308	498.0	501.0	3.0	7.0	1004	8.0	24.1
PAL0191	424.0	425.0	1.0	23.1	717	23.9	23.9
PAL0297	99.7	106.2	6.5	2.9	677	3.6	23.5
PRAJ0024	28.1	32.0	3.9	5.3	580	5.9	23.1
PAL0093	269.5	275.7	6.3	2.1	1559	3.7	23.0
PAL0173	266.0	268.0	1.9	10.7	713	11.4	22.3
PAL0048	86.8	91.2	4.4	3.4	1649	5.0	21.9
PRAJ0109	51.7	56.7	5.0	3.9	428	4.3	21.7
PAL0205	101.0	104.0	3.0	6.4	606	7.0	21.1
PRAJ0108	18.3	23.4	5.1	3.3	857	4.1	21.1
PAL0109	17.1	23.0	5.9	2.8	743	3.5	20.9
PAL0197	316.9	321.0	4.1	3.5	1545	5.1	20.8
PAL0199	138.4	141.4	3.0	6.2	709	6.9	20.6
PAL0018	172.0	173.0	1.0	19.2	49	19.2	19.2
PRAJ0070	25.4	27.4	2.0	9.1	496	9.6	19.2
PAL0093	265.9	267.8	1.9	9.5	731	10.3	19.0
PAL0062	181.9	184.9	3.0	6.0	323	6.3	18.9
PAL0019	176.7	179.6	2.9	5.9	561	6.4	18.6
PAL0210	153.6	157.1	3.5	5.0	267	5.2	18.6
PAL0163	416.6	419.4	2.8	0.0	6507	6.5	18.5
PRAJ0005	6.7	8.7	2.0	8.9	370	9.2	18.4
PAL0116	144.0	148.0	4.0	4.0	617	4.6	18.3
PRAJ0110	87.2	90.2	3.0	5.3	598	5.9	17.8
PRAJ0114	83.0	85.0	2.0	8.5	358	8.8	17.6
PRAJ0113	59.4	64.4	5.0	2.8	748	3.5	17.5
PAL0033	152.5	154.7	2.2	7.7	94	7.8	17.2
PRAJ0073	8.7	11.7	3.0	5.1	493	5.6	16.8
PAL0091	155.0	155.8	0.8	19.9	494	20.4	16.3
PAL0048	54.0	59.0	5.0	2.1	1168	3.2	16.2
PRAJ0113	67.4	72.4	5.0	2.3	848	3.2	15.8
PRAJ0072	16.3	17.3	1.0	14.8	859	15.7	15.7
PRAJ0076	23.0	26.9	3.9	2.9	1168	4.0	15.5
PAL0221	235.9	236.9	1.0	15.2	328	15.5	15.5
PRAJ0097	17.7	18.0	0.3	49.6	88	49.7	14.9
PAL0216	319.0	321.0	2.0	7.4	3	7.4	14.8
PRAJ0107	20.5	22.9	2.4	5.9	295	6.2	14.8
PAL0302	97.4	99.4	2.0	7.1	96	7.2	14.4
PAL0197	323.5	326.3	2.8	4.0	1226	5.2	14.3
PAL0189	203.0	205.0	2.0	6.4	549	6.9	13.9
PAL0228	241.8	246.2	4.4	2.6	473	3.1	13.8
PAL0037	59.0	61.0	2.0	6.7	120	6.8	13.6
PAL0189	182.9	185.0	2.2	6.3	12	6.3	13.6
PAL0097	281.3	285.3	4.0	2.0	1441	3.4	13.6
PAL0245	191.0	191.5	0.5	23.0	3974	27.0	13.5
PAL0075	70.0	72.0	2.0	6.0	573	6.6	13.2
PRAJ0020	5.2	8.2	3.0	3.4	985	4.4	13.2
PAL0118	322.0	324.0	2.0	6.1	446	6.6	13.1
PAL0223	296.0	297.0	1.0	12.8	286	13.1	13.1
PAL0263	105.6	107.5	1.9	6.6	293	6.9	13.0
PAL0030	144.7	146.9	2.2	5.6	472	6.0	13.0
PAL0161	346.0	349.0	3.0	3.5	784	4.2	12.7
PRAJ0110	82.0	84.0	2.0	5.5	795	6.2	12.5
PAL0260	112.0	113.2	1.2	10.5	209	10.7	12.3
PAL0099	66.7	70.4	3.7	2.5	824	3.3	12.2

Drillhole	From	To	Interval	Au g/t	Co ppm	AuEq g/t	g*w AuEq
PAL0214	122.0	123.7	1.7	6.4	761	7.1	12.1
PAL0037	33.0	36.0	3.0	3.2	681	3.9	11.8
PAL0048	92.3	95.7	3.4	2.8	688	3.5	11.7
PAL0273	15.9	19.9	4.0	2.4	545	2.9	11.7
PAL0269	191.7	193.8	2.1	5.2	275	5.5	11.6
PAL0210	137.4	140.4	3.1	2.9	832	3.7	11.4
PAL0191	436.0	438.0	2.0	5.2	471	5.7	11.4
PAL0291	106.9	107.9	1.0	11.2	28	11.2	11.2
PAL0118	368.1	371.0	2.9	2.9	885	3.8	11.0
PAL0090	179.1	181.0	1.9	4.8	835	5.6	11.0
PAL0075	65.0	67.0	2.0	4.6	851	5.5	11.0
PAL0259	143.3	145.7	2.4	3.7	747	4.5	11.0
PAL0191	446.0	449.7	3.7	2.0	983	3.0	10.9
PAL0235	441.5	443.5	2.0	3.8	1691	5.4	10.9
PAL0293	260.2	261.2	1.0	10.1	622	10.7	10.7
PAL0280	248.0	251.0	3.0	2.0	1573	3.6	10.7
PRAJ0020	11.2	13.2	2.0	4.6	693	5.2	10.5
PAL0227	311.7	313.7	2.0	3.9	1249	5.2	10.3
PAL0159	452.0	454.0	2.0	3.9	1208	5.1	10.3
PRAJ0114	74.1	76.1	2.0	4.8	211	5.0	10.1
PAL0176	49.0	50.0	1.0	10.0	58	10.1	10.1
PAL0293	285.0	288.0	3.0	2.5	754	3.3	9.9
PAL0296	322.5	324.5	2.0	4.5	343	4.9	9.7
PAL0110	39.6	41.6	2.0	4.4	422	4.8	9.7
PAL0290	251.8	253.6	1.8	4.6	763	5.3	9.6
PAL0291	286.5	290.5	4.0	2.2	191	2.3	9.4
PAL0269	202.9	204.9	2.0	3.9	748	4.7	9.3
PAL0211	248.3	252.3	4.0	0.1	2227	2.3	9.3
PAL0264	104.9	106.9	2.0	3.4	1170	4.6	9.2
PAL0227	335.4	337.4	2.0	4.5	48	4.6	9.2
PAL0227	326.9	331.0	4.1	1.2	1012	2.2	9.1
PAL0189	213.2	214.3	1.1	7.3	1003	8.2	9.1
PAL0263	227.3	229.3	2.0	4.3	170	4.4	8.8
PAL0027	27.5	31.0	3.6	2.1	392	2.5	8.8
PAL0269	219.4	221.4	2.0	4.4	16	4.4	8.8
PRAJ0075	15.4	17.4	2.0	3.8	590	4.4	8.8
PAL0180	435.6	438.5	2.9	0.1	2863	2.9	8.6
PAL0027	45.2	47.2	2.0	3.7	484	4.2	8.3
PAL0283	222.8	223.8	1.0	8.2	52	8.3	8.3
PAL0308	443.5	445.5	2.0	3.7	419	4.1	8.2
PAL0191	429.0	430.0	1.0	7.7	425	8.1	8.1
PAL0093	247.0	248.8	1.8	3.3	1109	4.4	8.0
PRAJ0003	6.0	9.0	3.0	2.1	494	2.6	7.9
PAL0012	150.6	153.7	3.1	1.6	951	2.5	7.9
PAL0225	416.8	420.8	4.0	1.6	365	1.9	7.7
PAL0305	228.2	231.5	3.3	1.6	730	2.4	7.7
PAL0097	258.5	261.3	2.8	2.7	39	2.7	7.7
PRAJ0026	17.6	21.4	3.8	1.0	997	2.0	7.7
PAL0267	47.0	49.0	2.0	3.1	688	3.8	7.6
PAL0263	122.5	124.7	2.2	3.4	31	3.4	7.6
PAL0207	156.6	158.4	1.8	3.8	485	4.3	7.6
PAL0298	249.1	250.1	1.0	7.2	118	7.3	7.3
PAL0097	294.8	296.8	2.1	2.7	852	3.5	7.2
PAL0230	553.0	554.0	1.0	6.9	204	7.1	7.1
PAL0259	135.7	137.7	2.0	1.4	2120	3.6	7.1
PAL0093	280.4	281.4	1.0	6.8	206	7.0	7.0
PAL0258	97.0	99.6	2.6	1.5	1233	2.8	7.0
PAL0043	10.6	12.6	2.0	3.2	312	3.5	7.0
PAL0116	155.0	156.0	1.0	6.3	602	6.9	6.9
PAL0023	84.4	87.4	3.0	2.1	237	2.3	6.9

Drillhole	From	To	Interval	Au g/t	Co ppm	AuEq g/t	g*w AuEq
PAL0297	67.4	68.4	1.0	6.7	187	6.8	6.8
PAL0100	295.0	296.3	1.3	4.1	1116	5.2	6.8
PAL0225	354.0	356.0	2.0	3.0	395	3.4	6.8
PAL0159	435.0	436.0	1.0	5.4	1416	6.8	6.8
PAL0028	37.6	38.4	0.8	6.8	1590	8.4	6.7
PRAJ0080	15.0	17.4	2.4	2.6	228	2.8	6.7
PAL0246	188.6	189.2	0.6	10.3	725	11.0	6.6
PAL0297	86.2	87.2	1.0	6.4	47	6.5	6.5
PAL0201D1	450.8	451.9	1.1	3.8	2041	5.8	6.4
PAL0250	125.2	127.1	1.9	2.1	1162	3.3	6.3
PAL0293	278.0	281.0	3.0	1.1	993	2.1	6.2
PAL0190	366.2	368.0	1.9	2.4	883	3.3	6.2
PAL0291	295.4	297.4	2.0	1.8	1285	3.1	6.1
PAL0231	342.0	343.0	1.0	5.9	68	6.0	6.0
PRAJ0116	66.4	68.4	2.0	2.8	237	3.0	6.0
PRAJ0008	2.3	4.3	2.0	2.7	335	3.0	6.0
PAL0110	25.2	26.3	1.1	4.0	1438	5.4	6.0
PAL0118	373.4	375.2	1.8	1.4	1934	3.3	6.0
PAL0302	144.4	146.4	2.0	2.5	489	2.9	5.9
PAL0118	327.0	329.0	2.0	2.4	552	2.9	5.8
PAL0223	292.0	294.0	2.0	2.3	580	2.9	5.8
PAL0245	195.7	196.5	0.8	6.1	1530	7.7	5.7
PAL0308	439.5	441.5	2.0	0.1	2699	2.8	5.7
PAL0091	249.7	250.7	1.0	5.0	436	5.4	5.7
PAL0048	106.0	107.9	1.8	0.1	2973	3.0	5.6
PAL0179	6.0	8.0	2.0	2.1	678	2.8	5.4
PAL0213	250.2	252.0	1.8	2.8	150	2.9	5.3
PAL0091	145.9	147.9	2.0	2.1	553	2.6	5.2
PRAJ0009	26.4	27.4	1.0	4.4	637	5.0	5.0
PRAJ0020	23.0	24.0	1.0	4.6	384	5.0	5.0
PAL0228	248.2	250.2	2.0	2.2	284	2.5	5.0
PAL0023	109.9	111.9	2.0	0.3	2125	2.4	4.9
PAL0188	300.3	301.3	1.0	2.7	2079	4.8	4.8
PAL0098	35.6	36.6	0.9	4.9	25	4.9	4.7
PAL0293	274.2	276.0	1.8	1.0	1647	2.6	4.7
PAL0308	494.0	495.0	1.0	4.0	610	4.6	4.6
PAL0222	276.1	278.2	2.1	1.3	902	2.2	4.6
PRAJ0105	18.1	20.3	2.2	1.7	356	2.1	4.6
PAL0040	37.3	39.3	2.0	1.6	654	2.3	4.6
PRAJ0117	75.3	76.3	1.0	4.1	455	4.5	4.5
PRAJ0006	20.3	20.8	0.5	8.2	641	8.8	4.4
PAL0218	432.4	433.4	1.0	4.0	378	4.4	4.4
PAL0200	441.0	442.0	1.0	3.8	559	4.3	4.3
PRAJ0117	65.6	66.6	1.0	4.0	203	4.2	4.2
PAL0075	30.6	31.6	1.0	2.8	1400	4.2	4.2
PRAJ0109	65.6	66.6	1.0	0.1	4100	4.2	4.2
PAL0259	148.3	149.3	1.0	2.5	1685	4.2	4.2
PRAJ0107	42.6	44.2	1.6	2.4	144	2.6	4.1
PRAJ0108	32.9	33.7	0.8	4.7	379	5.1	4.1
PAL0246	208.6	209.6	1.0	3.2	766	4.0	4.0
PAL0176	30.7	31.9	1.3	2.2	902	3.1	3.9
PAL0296	269.0	270.0	1.0	3.0	947	3.9	3.9
PAL0267	75.9	76.9	1.0	3.6	154	3.8	3.8
PAL0206	306.3	307.3	1.0	0.0	3748	3.8	3.8
PAL0216	273.9	275.0	1.1	3.2	99	3.3	3.7
PAL0305	201.3	202.3	1.0	3.5	179	3.7	3.7
PAL0176	24.0	25.0	1.0	3.1	643	3.7	3.7
PAL0259	128.6	129.6	1.0	2.0	1671	3.7	3.7
PAL0250	87.5	89.2	1.7	2.0	159	2.1	3.6
PAL0210	146.4	147.4	1.1	2.9	437	3.4	3.5

Drillhole	From	To	Interval	Au g/t	Co ppm	AuEq g/t	g*w AuEq
PAL0199	287.0	288.0	1.0	3.4	22	3.5	3.5
PAL0118	377.1	378.4	1.3	0.2	2486	2.6	3.4
PAL0097	290.5	291.6	1.2	2.5	418	2.9	3.4
PRAJ0022	26.0	27.0	1.0	3.3	0	3.3	3.3
PRAJ0020	26.0	27.0	1.0	1.4	1940	3.3	3.3
PRAJ0107	36.7	37.7	1.0	2.9	407	3.3	3.3
PAL0074	32.0	33.0	1.0	3.2	82	3.3	3.3
PAL0223	322.0	323.0	1.0	1.9	1332	3.2	3.2
PAL0161	337.0	338.0	1.0	3.0	158	3.2	3.2
PAL0043	20.6	21.6	1.0	3.1	54	3.2	3.2
PAL0267	44.0	45.0	1.0	2.4	784	3.1	3.1
PAL0188	337.9	338.9	1.0	3.1	35	3.1	3.1
PRAJ0024	35.0	36.0	1.0	2.4	668	3.1	3.1
PRAJ0009	22.7	23.7	1.0	2.6	476	3.0	3.0
PAL0030	151.2	152.4	1.2	0.1	2430	2.5	3.0
PAL0263	98.7	99.9	1.1	2.2	473	2.6	3.0
PAL0090	173.1	174.1	1.0	0.0	2980	3.0	3.0
PAL0259	119.3	120.5	1.3	2.3	55	2.4	3.0
PRAJ0111	31.3	32.4	1.1	1.9	771	2.6	2.9
PRAJ0010	4.3	5.3	1.0	2.4	539	2.9	2.9
PAL0224	433.0	434.0	1.0	2.8	66	2.9	2.9
PAL0249	177.3	178.3	1.0	2.5	344	2.9	2.9
PAL0220	376.0	376.7	0.7	3.9	189	4.0	2.8
PAL0278	220.5	221.5	1.0	2.6	238	2.8	2.8
PAL0022	16.6	17.8	1.2	2.3	46	2.4	2.8
PAL0221	215.0	216.0	1.0	2.8	43	2.8	2.8
PAL0308	449.7	450.7	1.0	1.7	1098	2.8	2.8
PAL0210	133.4	134.4	1.0	2.5	258	2.7	2.7
PAL0256	112.0	113.0	1.0	1.6	1108	2.7	2.7
PAL0223	306.0	307.2	1.2	1.8	445	2.2	2.7
PAL0197	296.3	297.3	1.0	0.2	2423	2.6	2.6
PAL0189	210.0	211.0	1.0	1.6	1043	2.6	2.6
PRAJ0016	8.5	9.5	1.0	0.0	2600	2.6	2.6
PAL0083	45.0	46.0	1.0	2.6	55	2.6	2.6
PAL0118	388.0	389.0	1.0	2.4	202	2.6	2.6
PAL0159	424.0	425.0	1.0	0.2	2380	2.6	2.6
PAL0019	196.0	197.0	1.0	2.2	439	2.6	2.6
PAL0265	238.6	239.6	1.0	2.4	224	2.6	2.6
PAL0193	274.0	275.0	1.0	0.5	2103	2.6	2.6
PAL0095	232.0	233.0	1.0	2.3	233	2.5	2.5
PAL0259	96.6	97.6	1.0	0.5	2013	2.5	2.5
PAL0203	313.0	314.0	1.0	0.8	1736	2.5	2.5
PAL0243	193.0	194.4	1.4	0.8	966	1.8	2.5
PAL0009	172.0	173.0	1.0	0.2	2240	2.5	2.5
PAL0305	224.2	225.2	1.0	0.7	1725	2.4	2.4
PAL0053	68.7	69.7	1.0	0.9	1508	2.4	2.4
PAL0207	153.8	154.8	1.1	1.9	436	2.3	2.4
PAL0037	39.0	40.0	1.0	1.3	1160	2.4	2.4
PAL0220	419.0	420.0	1.0	2.4	19	2.4	2.4
PAL0294	216.9	217.9	1.0	0.2	2195	2.4	2.4
PAL0211	293.9	295.0	1.1	1.9	263	2.2	2.4
PAL0213	256.0	257.0	1.0	2.2	222	2.4	2.4
PAL0203	264.0	265.0	1.0	2.3	85	2.4	2.4
PAL0298	261.1	262.1	1.0	2.3	37	2.4	2.4
PAL0291	213.2	214.2	1.0	0.4	1945	2.4	2.4
PRAJ0081	21.6	22.5	0.9	1.8	701	2.5	2.4
PRAJ0024	22.7	23.7	1.0	1.5	912	2.4	2.4
PAL0204	88.2	89.1	0.9	1.7	881	2.6	2.4
PAL0286	114.6	115.6	1.0	0.2	2163	2.4	2.4
PAL0176	15.0	15.6	0.6	4.2	115	4.3	2.3

Drillhole	From	To	Interval	Au g/t	Co ppm	AuEq g/t	g*w AuEq
PAL0267	56.8	57.8	1.0	1.9	411	2.3	2.3
PAL0288	119.0	120.0	1.0	1.6	674	2.3	2.3
PAL0029	95.7	96.7	1.0	2.2	56	2.3	2.3
PAL0298	235.4	236.4	1.0	2.2	55	2.3	2.3
PAL0273	28.2	29.2	1.0	2.0	254	2.3	2.3
PAL0201D1	439.1	440.0	0.9	0.0	2505	2.5	2.3
PAL0091	151.2	152.2	1.0	1.1	1156	2.3	2.3
PAL0259	132.5	133.5	1.0	0.7	1589	2.3	2.3
PAL0193	280.0	281.0	1.0	0.7	1522	2.2	2.2
PAL0031	85.4	86.4	1.0	2.2	39	2.2	2.2
PAL0196	86.9	87.9	1.0	2.0	268	2.2	2.2
PAL0259	164.0	165.0	1.0	2.2	33	2.2	2.2
PRAJ0113	56.8	57.8	1.0	1.7	506	2.2	2.2
PAL0265	231.6	232.6	1.0	1.3	838	2.2	2.2
PRAJ0023	12.2	13.2	1.0	1.3	883	2.1	2.1
PAL0058	153.0	154.0	1.0	2.1	8	2.1	2.1
PAL0210	143.4	144.4	1.0	1.1	1043	2.1	2.1
PAL0049	163.0	163.9	0.9	0.0	2390	2.4	2.1
PAL0267	33.3	34.3	1.0	1.6	512	2.1	2.1
PAL0089	87.7	88.7	1.0	1.6	511	2.1	2.1
PRAJ0080	35.9	36.9	1.0	2.0	68	2.1	2.1
PAL0293	292.2	293.2	1.0	1.7	356	2.1	2.1
PAL0009	155.0	156.0	1.0	2.0	26	2.0	2.0
PRAJ0008	7.3	8.0	0.7	0.9	1900	2.8	2.0
PAL0202A	780.0	780.8	0.8	1.6	846	2.4	1.9
PAL0195	171.3	172.0	0.8	2.1	172	2.2	1.7
PAL0039	112.8	113.1	0.3	2.9	1650	4.6	1.6
PRAJ0071	35.7	36.5	0.8	1.5	640	2.1	1.6
PAL0207	148.0	148.6	0.6	2.0	720	2.7	1.5
PRAJ0097	3.4	3.9	0.5	0.1	2790	2.8	1.4
PAL0183	142.6	143.1	0.5	2.2	340	2.5	1.4
PRAJ0107	18.2	18.5	0.3	4.3	330	4.6	1.4
PRAJ0096	17.5	18.1	0.6	2.0	245	2.2	1.2
PRAJ0096	20.9	21.3	0.4	3.0	41	3.1	1.1

APPENDIX

D RAJAPALOT PROJECT AREA MINERAL FORMULAE

Rajapalot project area mineral formulae

Silicate Groups

Feldspars

Albite	$NaAlSi_3O_8$
K-feldspar	$KAlSi_3O_8$

Amphiboles

Actinolite	$Ca_2(MgFe)_5Si_8O_{22}(OH)_2$
Tremolite	$Ca_2(Mg)_5Si_8O_{22}(OH)_2$
Anthophyllite	$Mg_7Si_8O_{22}(OH)_2$ *(orthorhombic Mg-Fe amphibole group)*
Cummingtonite	$Mg_4Fe_3Si_8O_{22}(OH)_2$ *(monoclinic Mg-Fe amphibole group)*
Grunerite	$Fe_7Si_8O_{22}(OH)_2$ *(monoclinic Mg-Fe amphibole group)*
"Hornblende"	$NaCa_2(MgFe)_4Al_3Si_6O_{22}(OH)_2$

Micas

Annite	$KFe_3AlSi_3O_{10}(OH)_2$ *(biotite group)*
Biotite	$KMg_{1.5}Fe_{1.5}AlSi_3O_{10}(OH)_2$ *(biotite group)*
Phlogopite	$KFe_3AlSi_3O_{10}(OH)_2$ *(biotite group)*
Muscovite	$KAl_3Si_3O_{10}(OH)_2$ *(white micas)*
Phengite	$K(AlMgFe)_2(SiAl)_4O_{10}(OH)_2$ *(white micas)*
Chlorite	$Fe_{2.5}Mg_{2.5}Al_2Si_3O_{10}(OH)_2$ *(chlorite group)*
Clinochlore	$Mg_5Al_2Si_3O_{10}(OH)_2$ *(chlorite group)*
Talc	$Mg_3Si_4O_{10}(OH)_2$

Garnet group

Almandine	$Fe_3Al_2Si_3O_{12}$
Pyrope	$Mg_3Al_2Si_3O_{12}$
Andradite	$Ca_3Fe^{3+}{}_2Si_3O_{12}$
Grossular	$Ca_3Al_2Si_3O_{12}$

Other Silicates

Diopside	$CaMgSi_2O_6$
Epidote	$Ca_2Fe^{3+}Al_2Si_3O_{12}(OH)$
Clinozoisite	$Ca_2Al_3Si_3O_{12}(OH)$
Cordierite	$Mg_2Al_3(AlSi_5)O_{18}$
Sillimanite	Al_2SiO_5 *(as for kyanite, andalusite)*
Scapolite	$Na_4Al_3Si_9O_{24}Cl$ *(marialite end member)*
Tourmaline	General formula $XY_3Z_6(T_6O_{18})(BO_3)_3V_3W$ where X=Na, Ca, K; Y=Li, Mg, Fe^{2+}, Mn^{2+} Zn, Al, Cr^{3+}, V^{3+}, Fe^{3+}, Ti^{4+}; Z=Mg, Al, Fe^{3+}, Cr^{3+}, V^{3+}; T=Si, Al, B; V= OH, O; W= OH, F, O *(schorl is the Fe^{2+} variety, i.e. $NaFe^{2+}{}_3Al_6Si_6O_{18}(BO_3)_3(OH)_3OH$, dravite is the Mg end-member)*

Other Minerals

Sulphides

Pyrite	FeS_2
Pyrrhotite	$Fe_{(1-x)}S$ *(where S=0 to 0.2)*
Chalcopyrite	$CuFeS_2$
Cobaltite	CoAsS
Linnaeite	$Co^{2+}Ni,Co^{3+}{}_2S_4$
Pentlandite	$(Fe,Ni)_9S_8$ *(Cobaltian pentlandite Co_9S_8)* Molybdenite MoS_2
Galena	PbS
Sphalerite	ZnS

Oxides + miscellaneous minerals

Magnetite	$FeFe^{3+}{}_2O_4$ (Fe_3O_4)
Hematite	$Fe^{3+}{}_2O_3$ Scheelite $CaWO_4$
Wolframite	$(Fe,Mn)WO_4$
Rutile	TiO_2
Titanite	$CaTiSiO_5$
Ilmenite	$FeTiO_3$
Uraninite	UO_2
Altaite	PbTe
Nickeline	NiAs
Maldonite	Au_2Bi
Hunchunite	$(Au,Ag)_2Pb$
	+ Bi-Se tellurides of various compositions
Anhydrite	$CaSO_4$
Gypsum	$CaSO_4.2H_2O$